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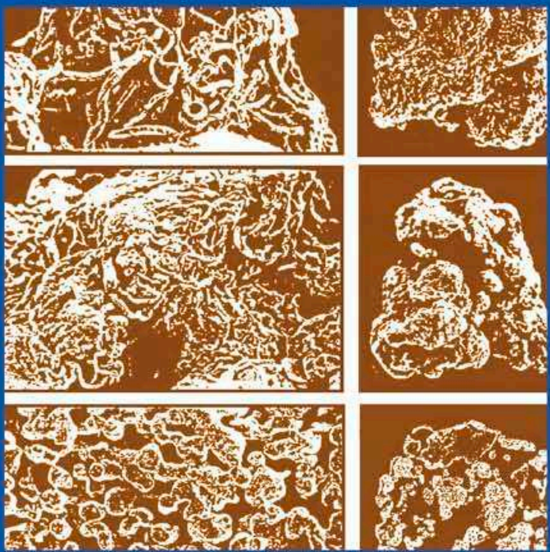
Handbook of gold exploration and evaluation

Eoin H. Macdonald



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Handbook of gold exploration and evaluation

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Foreword

I am honoured and extremely pleased to write the foreword for Eoin Macdonald's book *Handbook of Gold Exploration and Evaluation*. Eoin's first book on this subject, *Alluvial Mining – The Geology, Technology and Economics of Placers*, published in 1983, was aimed at a basic understanding of the fundamentals of alluvial mining theory and practice, primarily for beach sand minerals. In the subsequent years Eoin realised that a book based on alluvial gold deposits was warranted and indeed needed. *Handbook of Gold Exploration and Evaluation* is the result and represents the knowledge and foresight of a man with over 65 years of expertise and practical experience in the fields of exploration and mining of mineral deposits across the globe.

Rich alluvial gold deposits have been mined for centuries and continue to capture the interest and imagination of specialists and lay people in the search for the yellow metal. But with increases in demand and depletion of easily identified and won detrital gold deposits, exploration has entered into a new and exciting phase. It is thus pleasing to see a new book that recognises the common relationship of exploration geology, geochemistry and remote sensing techniques in the search for both alluvial and hard rock gold ores.

This book contains nine chapters covering topics as diverse as the nature and history of gold, geology of gold ore deposits, gold deposition in the weathering environment, sedimentation and detrital gold, gold exploration, lateritic and placer gold sampling, mine planning and practice, metallurgical processes and design, and evaluation, risk and feasibility. The breadth of subject matter contained in this book is outstanding and I know of no better and more up-to-date book on alluvial gold deposits, exploration and mining.

Discussions involving the methods, hazards and costs of conducting mining operations of primary gold ores at depth are beyond the scope of the treatment but an indisputable and most important conclusion is that both primary and secondary gold ventures profit equally from the same detailed field investigations. Detrital gold accumulations represent the weathered detritus of their host rocks hence neither primary nor alluvial gold surface features can be studied in isolation without neglecting possibly vital evidence from the other. Ground surface studies involve similar geochemical stream and soil sediment surveys. Similar geological and geophysical techniques also apply to sub-surface investigations in both primary and secondary gold settings including geomorphic reconstructions of palaeo-erosional surfaces. Remote sensing techniques employ similar sensors to map relevant geological and geomorphic features and dating of the same key rock types and minerals helps clarify the geological history of any area under investigation in both ancient and modern tectonic settings.

Future demands on gold will be very great and additions to resource quantities of both primary and secondary gold ores must increase in order to cope with increasing demand. Definition of that part of a resource that might be mined economically within the guidelines laid down for exploration is an essential requirement of economic evaluation.

This book will be of special interest and use to exploration and mining geologists, mining engineers, metallurgists, academics and students alike. However, the book is written in such a manner that non-technical people will find it informative and it will help them gain a thorough understanding of gold exploration.

Finally, congratulations Eoin on the tremendous achievement of publishing *Handbook of Gold Exploration and Valuation* in your 90th year, and may the ultimate compliment to you be the discovery and successful exploitation of a new alluvial gold deposit.

Assoc. Prof. J. Bruce Gemmell
ARC Centre of Excellence in Ore Deposits (CODES)
School of Earth Sciences
University of Tasmania

Preface

The intention of an earlier text *Alluvial Mining – The Geology, Technology and Economics of Placers* – was to provide a basic understanding of the fundamentals of alluvial mining theory and practice for placer engineers and geologists to build upon. The beach sand minerals rutile, zircon, ilmenite and middle density minerals cassiterite and tantalite took prominence over gold and other noble metals of higher density and greater unit value. The overall response was encouraging but the amount of fresh evidence emerging from ongoing studies showed that a fresh book was needed based solely upon gold as a unique metal in its own right. This book, the *Handbook of Gold Exploration and Evaluation* is designed primarily for professional geologists and engineers engaged in gold exploration-evaluation exercises, and as a text for under-graduate and graduate students in higher schools of learning. Since much of the treatment is empirical, it should also provide useful reference material for prospectors and small-scale miners.

Presentation of the text commences with a brief description of the nature of gold and of those properties that make it unique amongst all other minerals. The history of gold is reviewed from the earliest times recorded by man until the beginning of the 20th century when geologists were beginning to understand how the Earth's surface was formed and what causes it to change over time. New theories involving the metallogenic roles of orogenic-related volcanics and sedimentation within intracratonic basins were being examined. Geophysicists were investigating the structure of the Earth and its magnetic, electrical and magnetic properties. Classical geology, which could once tell only what happened in a past sequence of events but seldom how or why, was now finding answers in the new and revolutionary approach to mineral exploration, the global theory of plate tectonics. Resurgence of efforts seeking an understanding of universal processes involved with the origin and general structure of the Earth and its neighbours in space was stimulated by discoveries in planetary science.

Explanations based upon plate tectonic theory are offered for the evolution and inter-relationship of oceanic and continental crust and the supercontinental cycle in which continental masses continually join together, drift apart and rejoin to form other and larger supercontinents. Many aspects of Earth's geology and history are discussed, including the opening and closing of ocean basins, origins of mountain ranges, geological structures, hydrothermal activity and gold distribution in volcanic and tectonic settings. All past and present processes relating to the modification and release of gold grains in the weathering environment are shown to be paragenetically related to both primary and secondary rock types of all ages, and to the local and regional geologic setting at the time of their formation. Of particular relevance is recognition of a fundamental rock cycle in which all types of gold ores are seen as integrated parts of a process in which the rocks are related to one another and can be transformed one to the other.

Occurrences of detrital gold, saprolitic and lateritic-hosted gold ores are integral parts of a weathering cycle in which the global energy balance is largely dependent upon the influence of the Earth's oceanic-atmospheric heat circulation. The effects of climatic change and extremes of change on stresses that produce weathering and erosion have all played significant roles in the formation, sorting and deposition of secondary gold accumulations. Deep-seated lateritic weathering became a topic of great interest to geochemists as well as to geologists and new fields of gold exploration were opened up towards the close of the 19th century. The long-term weathering of regolith has delineated geochemical provinces and defined chalcophile corridors of strategic importance to explorers.

The basic concepts of viscosity and boundary layer flows are examined as they relate to sediment characteristics of size, shape and density and to sedimentation in natural stream channels, and expressions are developed for computing the relative magnitude of forces acting in solids/fluid flow. Examination of the complex relationships of channel geometry leads naturally to consideration of erosion, transport, sorting and deposition in the development of alluvial gold concentrations. As a function of topography, sedimentation is both space and time-related as demonstrated in the formulation of models, which establish the relationships between accumulations of heavy minerals, the dynamic conditions of transport and direction of sediment transport.

Ground geological surveys provide essential data for investigating the nature and extent of surface and sub-surface ore horizons; both for open-cast mine planning and treatment and for the exploration and development of any primary orebodies that might be revealed as a result of the surface operations. Changes in exploration methods have resulted from new theories of paragenetic relationships and the development of remote sensing and geophysical techniques for regional and sub-surface examination. From identification of the principal sedimentary controls on the distribution of pay zones in some Alaskan gold placers, the course is charted for further exploration of gold deposits in other areas of Quaternary and pre-Quaternary glaciations. Geophysical surveys in shallow offshore areas are typical of those used to investigate the extent of seabed features such as those that contain concentrations of gold in drowned beach sand deposits at Nome, Alaska.

A statement of the laws of sampling is followed by a brief description of the essential features of bench scale testwork required by mineral-processing engineers to write down all of the quantitative data needed for the design of each plant component. These data include all essential measurements of the gravel, sand and clay contents of a particular alluvial-type ore and the processing characteristics and percentages of gold grains in the different ore types involved. The vexed question of sample reliability and representivity is examined in relation to the methods employed in the computation of ore resource quantities. Space is devoted to some of the many geological and geographic constraints that may be encountered in the field.

Mine planning deals with various principles and factors involved in the choice between wet and dry systems of surface mining, development access, stripping and production requirements and environmental protection. Elements of the design and operation of selected methods are discussed in sufficient detail to make intelligent judgements of the most suitable types for any given case. In land-based operations the nature of surface features, topography and ground water, and the dimensions and location of pay horizons are of particular relevance. Offshore dredging is concerned additionally with the need to compensate for variable wind, wave and current conditions.

Determining the most effective recovery methods for specific ore types during the mine-planning phase enables gold-processing plant to be designed with a reasonable assumption of optimum performance within a designated product size and range. Theoretical aspects of the transport mechanism in natural stream channels are extended to examine the effects of sedimentation under controlled conditions of shear flow in gravity concentrating plant. Constraints to the recovery of finely divided gold particles in conventional gravity plant are examined, and three different types of gravitationally enhanced separators are compared in their respective abilities to recover micron size particles of gold. An example is given of a typical chemical leaching process for the treatment of lateritic gold ores. The carbon-in-leach process used in this arrangement was selected from the results of continuous pilot plant testing of batch samples of lateritic ore involving ancillary 'Bond Mill' grindability testing, thickening and carbon-stripping studies.

Ultimately the problems involved in the discovery of gold resources of the required order of magnitude have financial as well as geological implications and the process of evaluation is a necessary tool in the decision-making process. Basic principles of economics apply to gold-mining projects of all types and the full range of possible options is analysed to ensure that all costs and benefits are considered. Three inter-related disciplines of investigation are considered as being essential to any evaluation process: technical, economic and financial. Risks inherent to each of these techniques are indicated by demonstrating the sensitivity of the expected value of the project to predictions made in the forecast parameters. The final measure of feasibility is a bankable document containing all the ingredients to satisfy the requirements of government bodies, financial institutions and lending organisations for the scale of operation envisaged. A case history demonstrates the essential part played by risk and sensitivity analyses in decision making in a marginal evaluation exercise.

Acknowledgements

The author of a textbook is greatly dependent upon the encouragement and support of his colleagues and friends, particularly during those long periods when the mind blanks out and nothing seems to come together. Foremost amongst those deserving great affection and respect are my wife and family, who have been extraordinarily patient and supportive throughout. A special debt is owed to my dearest colleague and friend Roy Kidd without whose generosity of spirit, advice, kindness and help I could not have overcome the very great problems encountered during the first few years of writing. I must also record the patience and constructive criticism of Professor Bruce Gemmell, who helped mould my opinions and improve my knowledge in his particular field of expertise 'ore deposit research'. I have also drawn heavily upon a course entitled 'Mine Evaluation and Project Assessment' presented by Ed Malone for 'The Key Centre of Mines', University of New South Wales and a course entitled 'Gravity Concentration' prepared by Dr P.N. Holtham at the same university. The co-operation, understanding and ability of environmental geologist Rochelle Bading (my grand-daughter) and her husband's art and design company 'Artivation Studios' (www.artivation.com.au) gave vital assistance in the preparation of tables and diagrams.

My thanks are also due to the contributors of technical papers in many studies of exploration geochemistry, gravity concentration, and mine evaluation that I have drawn data from, and whom I gratefully acknowledge in the text. A comprehensive set of references provides the opportunity for readers to enlarge upon the necessarily basic treatment given to some aspects of geology and geophysics. I have corresponded or worked with many other engineers and geologists who have done much to help develop my opinions and improve my knowledge. Amongst these are Yam Sia Goh (Malaysia), Dr A. Royle (UK), Chamroom Usiriprisan (Thailand), Tony Farmer, (Australia), Wright Engineers (Canada), Joe Kovacik (UNDP), Dr Pierre Goossens (Belgium) and Professor L.J. Lawrence (NSW). The advice and assistance of Steve Proctor (Peak Computer, Launceston) is also gratefully acknowledged.

1
Nature and history of gold

Gold (Au) is a transition metal between Ag and Rg in the chemical series of the Periodic Table. Its atomic number is 79, and atomic mass 196.96655 (2) g/mol, and has only one stable isotope number 197. The gold isotope ^{198}Au (half-life 2.7 days) is used in some cancer treatments. The metal has been known and prized as an object of beauty and for its unique properties of chemical stability, electrical conductivity, malleability and ductility (trivalent and univalent) since mankind's earliest awakenings. As a standard of value against which to appraise the costs of labour, goods, currency and national economy, it has been the standard of many currencies since the world's first coinage in Lydia between 643 and 630 BC. The name for gold is derived from the historic English word 'geolo', for yellow and the chemical symbol for gold Au, from the Latin name for gold 'aurum' (glowing dawn).

1.1 Gold mineralogy

Natural resources of elemental gold are mainly contained in the mineral gold (plus 85%Au) and in seawater. The oceans contain a major resource of gold in solution but individual estimates are variable, depending upon the location of samples, which appear to range in gold content from as low as 0.1 to as high as 2.0 ppb by weight. Emery and Schlee (1963) note gold grades in the top 10 m of sediments in the Atlantis 2 Deep between 5 and 10 ppm. However, attempts to recover gold from seawater on a commercial scale have so far failed, mainly because of the large quantities of water involved; ion exchange appears to offer the present best avenue for research. Salt, bromine and magnesia are recovered from seawater on a large scale hence the oceans must be regarded as a potential gold source of major proportions. Element associations are broadly classified on the basis of their affinities for metals, sulphides, silicates or gas phases, and are referred to in Table 1.1 as siderophile, chalcophile, lithophile and atmophile respectively (from Goldschmidt, 1922). Basically a siderophile element, gold has some characteristics that relate it to chalcophile group elements. The general ubiquity of gold is

Table 1.1 Goldschmidt's geochemical classification of the elements

Siderophile	Chalcophile	Lithophile	Atmophile
Fe Co Ni Ru Rh Pb Re Os Ir Pt Au Mo Ge Sn C P (Pb) (As) (W)	Cu Ag (au)* Zn Cd Hg Ga In Tl (Ge) (Sn) Pb As Sb Bi S Se Te (Fe) (Mo) (Re)	Li Na K Rb Cs Be Mg Ca Sr Ba B Al Sc Y Rare earths (C) Si Ti Zr Hf Th (P) V Nb Ta O Cr W U (Fe) Mn F Cl Br I (H) (Tl) (Ga) (Ge) (N)	H N (C) (O) (F) (Cl) (Br) (I) Inert gases

* Parentheses around a symbol indicate that the element belongs primarily in another group, but has some characteristics that relate it to this group.

Table 1.2 Common element associations in some different deposit types (McQueen, 1997)

Element association	Deposit type
Au-As-Sb (CO_2-Si)	Mesothermal. slate-hosted quartz-gold veins (e.g. Bendigo, Central Victoria).
Au-As-W-Ag-Sb-Te-±Cu-Pb-Mo (CO_2-S)	Archaean greenstone-hosted lode gold deposits (e.g. Kalgoorlie, Eastern Goldfields WA).
Ag-Au-As-Sb-Te±Hg-Mn (S-Si)	Epithermal gold-silver veins in volcanic host rocks (e.g. Golden Cross, NZ, Gidginburg, NSW).
Au-As-Hg-Fe±Sb-Te Tl (Si-S)	Carlin-type disseminated pyrite-arsenopyrite gold-bearing systems.
Sb-Au (Si-S)	Quartz-stibnite veins in metasediments (e.g. Costerfield, Vic., Hillgrove NSW).
Au-Fe-As-Cu±Zn (S-Si)	Quartz-sulfide veins containing Au associated with pyrite and arsenopyrite.
Hg-Cu-Au-S±As-Bi-Co	Associated with ultrabasic rocks.
Cu-U-Au-Ag-REE (S-F)	Hydrothermal hematitic breccia complexes (e.g. Olympic Dam, SA).
Pb-Zn-Ag±Cd-Cu (S)	Structurally controlled lead-silver veins and hydrothermal replacement bodies (Northhampton WA).
Fe-Pb-Zn-Cu-Ag±Hg-Sb-Au (S)	Stratabound volcanic-hosted massive sulfide deposits (e.g. Woodlawn, NSW, Roseberry, Que River, Tas.).
Fe-Pb-Zn-Ag, Mn-Ba-Tl±Cu-As-Sb (S)	Shale-hosted stratiform lead-zinc deposits (e.g. McArthur River, NT).
Fe-Pb-Zn-Ag-Cu (S) Fe-Cu-Au±Pb-Zn (S)	Turbidite-hosted sulfide vein systems (e.g. Cobar deposits, NSW).

Table 1.2 Continued

Element association	Deposit type
Fe-Cu-Au±Ag-Bi-Mo Te (S) Fe-Mo (S)	Porphyry copper-gold and porphyry molybdenum deposits in subvolcanic acid to intermediate rocks (e.g. North Parkes, NSW, Climax, Co.).
Cu-Au-Bi (S) Fe-Cu-Pb-Zn-Ag W-Mo±Cu-Pb-Zn-Bi-As	Proximal contact replacement skarns (e.g. Browns Creek, NSW, Mt Biggendon, Qld, Old Cadia, NSW, King Island, Tas.).
Fe-Sn±As-Cu-Zn (O-S-F)	Replacement tin skarns in carbonate units (e.g. Mt Bischoff, Renison, Tas.).
Cu-Pb-Zn-W-S	Sulfides and scheelite occurring within sediments and volcanic rocks.
Fe-Ni-Cu-Co-PGE (S)	Nickel-copper sulfide deposits in mafic/ultramafic rocks (e.g. komatiite-hosted deposits, Kambalda WA, Sudbury, Canada).
Ni-Co±Mn (Si-O)	Ni laterites on ultramafic rocks (e.g. Greenvale, Qld, New Caledonia).
Cr-PGE-Ni-Cu (S-O)	Chromite lenses in layered ultrabasic rocks (e.g. Merensky Reef, S. Africa).
Fe-Ti-V (O)	Magnetite bands in layered mafic bodies and anorthosites (e.g. Bushveld Complex, S. Africa).
REE-Zr (CO_2-P)-Nb-Ta-Cu	Carbonatite deposits (e.g. Mt Weld, WA, Palabora, S. Africa).
Cu-U-V±Se-As-Mo-Pb	Redox front uranium deposits in terrestrial sediments (e.g. Lake Frome deposits, S. Africa).
U-V (K)	Calcrete uranium deposits (e.g. Yeelirrie, WA).
U-Au-Cu±Zn-Sn-Pb-Bi, Pt-Pd	Stratabound and structurally controlled uranium-gold deposits in carbonaceous sediments (e.g. Alligator River, NT).
Sn-W-As-B±Pb-Zn-Cu (O-S)	Porphyry style tin deposits (e.g. Ardlethan, NSW).
Sn-W-Mo-Cu-Pb-Zn-Au (F-B-Si-S)	Zoned vein systems in and around granites (e.g. Zeehan, Tas., Emmaville, NSW).
Ta-Nb-Sn-Li-Be (Si)	Pegmatites and complex veins associated with granites.
Al±Nb-Ti-Ga (O)	Bauxite deposits.

demonstrated in Table 1.2, which shows common element associations in a range of ore deposit types.

1.1.1 Elemental and native gold

The following physical properties of gold are based on normal temperature and pressure (20 °C @ 1atm).

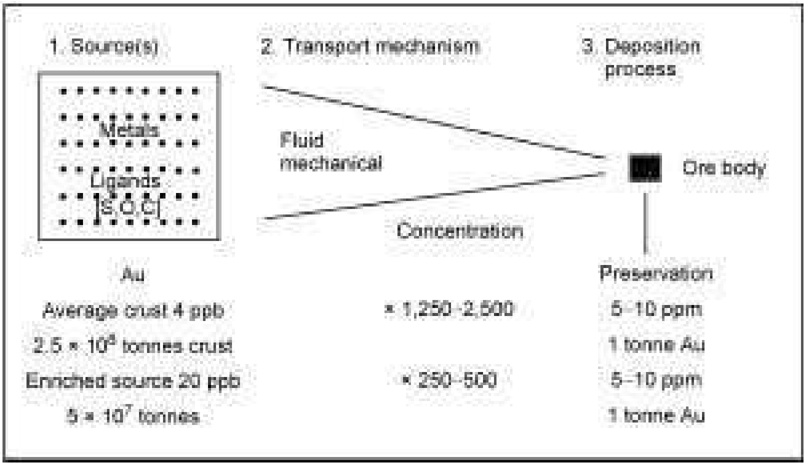
- Coefficient of linear thermal expansion
  – 0.0000142 cm/cm/ºC (0 ºC)
- Conductivity
  Electrical: – 0.452×10^6/cmΩ
  Thermal: – 3.17 W/cmK

Electronic processes create heat, and gold is essential to transfer heat easily from delicate instruments. A 35% gold alloy is used in the main engine nozzle of the space shuttle, where temperatures can reach 3300 ºC. It is the most tenacious and long-performing material available for protection at these temperatures.

- Reflectivity – high-purity gold reflects up to 99% of infra-red rays. This makes it ideal for heat and radiation reflection, as in life-saving face shields for astronauts and fire fighters.
- Density – 19.32 g/cc @ 300 K
- Melting point – 1064.58 ºC
- Molar volume – 10.2 cm^3/mole
- Specific heat – 0.128 j/gK
- Malleability – can be hammered into a sheet so thin that light can pass through it; one troy ounce of gold can be hammered into a one square metre sheet.
- Ductility – can be drawn into long thin wires; a single troy ounce of gold can be drawn into a thread approximately 5 miles long.
- Hardness – 2 to 3 (scale of comparative hardness of minerals from talc (1) to diamond (10) (Dana, 1890))
- Crystal System - isometric.

As a relatively rare element in the Earth's crust, gold is widely although not evenly distributed. It evolves as a siderophile element from the Fe-Ni core at crustal spreading centres and is present in Fe-Ni sulphides in the upper mantle. During partial melting of these materials the sulphides are partly consumed while the gold and other metals rise with basaltic fluids into the crust along mid-ocean ridges and at subduction zones. It is then associated with complex processes involving convection, subduction, partial melting, hydrothermal processing, weathering, erosion, and deposition before being returned to the mantle for recycling at subduction centres. Significantly large mineral deposits require the coincidence of particularly favourable processes and source parameters. Figure 1.1 depicts the basic requirement for the formation of any ore deposit with gold as an example. The degree of element concentration demonstrates the importance of an enriched source in minimising the concentration factor and hence the required efficiency and probability of the gold ore forming process. Fluids that can carry more than 10 ppb Au are excellent ore-forming solutions.

Gold occurs mainly in its native state, often alloyed with silver, copper, bismuth, mercury and platinum-group elements, and in tellurides and selenides. Native gold is distinguished from other minerals having similar yellowish coloration, often referred to as 'fools gold' (e.g. pyrite, chalcopyrite and



1.1 Basic requirements for the formation of any ore deposit. The degree of element concentration during ore formation is a critical factor (modified from McQueen, 1997).

millerite), by its strong metallic lustre and characteristic tint. Gold does not react with oxygen to form stable oxides, the principal oxidation states being +1 and +3. The most common compounds are $AuCl_3$ and the chlorauric acid $HAuCl_4$. Both are built up around the Au^{+3} metal ion. The main physical properties of metallic gold are listed in Table 1.3.

Table 1.3 The main properties of gold

Chemical symbol	Au
Atomic number	79
Atomic weight	197
Crystal form	Cubic
Colour	Golden yellow to silver white rarely orange red
Melting point	1064.43 °C (1948 °F)
Thermal expansion	14.2×10^{-6}/°C
Diaphansity	Opaque
Seawater abundance	4 to 8×10^{-6} ppm*
Hardness	2.5 to 3.0
Density	19.3 (when pure)
Magnetic susceptibility	Low
Resistivity	2.2×10^{-8}
Ductility	High
Lustre	Metallic
Tensile strength	138 megapascals when annealed
Crustal abundance	0.005 ppm
Volatility	Commences well below melting point

* The oceans contain about 1,350 million cubic kilometres of water and a seawater abundance of 0.000006 ppm (average value) represents a gold resource of about 8,200 cubic kilometres of fine gold.

Commercial applications

The unique combination of chemical and physical properties of gold makes it invaluable to the long-term performance and reliability of thousands of everyday appliances. The metal is completely recyclable and is a vital component in many medical, industrial and electrical applications. It is one of the least active metals chemically; it does not tarnish or oxidise in air, is unaffected by temperature change and is inert to all strongly alkaline solutions and to all pure acids, except selenic acid. Pure gold has a density 19.3 times that of water and weighs about 19,000 kg/m^3 (1,200 lbs/ft^3).

Gold alloys are used in industrial applications such as electroplating, granulation, pressing and lamination to keep costs down and make the product harder. The largest demand for gold comes from the electronics industry and the weapons and aerospace industry. Because of gold's resistance to corrosion and high electrical conductivity, it is used extensively in the manufacture of connectors, printed circuits, semi-conductors, relays, switches and a host of other electronic products. Properties of radiant energy unique to gold have led to the development of efficient energy reflectors for infra-red heaters and cookers and for focusing and heat retention in metallurgical processing. The extraordinarily high reflective powers of the metal are relied upon in the shielding that protects spacecraft and satellites from solar radiation. Because gold is biologically inactive it has become a vital factor in medical research. Tiny metallic gold particles are used diagnostically in prostate cancer treatment and gold complexes of thiols are used for the treatment of arthritis by injection into the site of inflammation. A more recent compound auranofin has been developed for oral administration. Radioactive ^{198}Au, with a half-life of 2.7 days is increasingly important for medical diagnosis and radiation therapy, and as a tracer in industrial applications such as monitoring sediment movement on the sea floor.

Gold is the most malleable and ductile of all metals and can be flattened into thin sheets less than one millionth of an inch thick and drawn into wire weighing only 0.5 mg/m length. Fabrication and other end uses of gold are for jewellery, electronics, coinage and other industrial and decorative purposes. Uses of gold for industrial and decorative purposes include brazing alloys, decorative plating, liquid gold for ceramics, rolled gold for fountain pens and gold used in the production of industrial laboratory chemicals. Italy has the largest gold jewellery manufacturing industry in the world and substantial portions of its gold imports are exported in the form of finished jewellery. The USA is the next largest producer of jewellery, then Germany, Japan, France, Switzerland and Canada.

The use of gold for the fabrication of medals, medallions and coins, after showing a decline in 1982, increased markedly in 1983 when South Africa became the largest individual manufacturer of these items. An undisclosed amount of gold is also issued (apparently with governmental consent) as coins for local use by business organisations and wealthy classes in some Asian countries. Although the cost of gold is only a small fraction of end-product costs in the electronics industry, consumers try to economise in its use by methods of spot plating, strip plating and reduction in the thickness of plated surfaces. Despite these economies, the rapid expansion of the electronics industry in keeping with increased demand for electronic appliances is expected to increase the overall demand for gold in that industry.

1.1.2 Gold geochemistry

Two important aspects of the aqueous geochemistry of gold are its chemistry and the particular properties of the matrix solution (e.g. acidity, pH and oxidation potential, Eh). Salinity can arise from various processes including rock weathering and dissolution of previously deposited halite, evaporation, seawater and aerosol deposition of seawater. Acidity, which is usually measured as pH and factors such as pH, Eh, and salinity have major effects on the speciation and solubility of gold. Eh (electrical conductivity) values less than 200 mV indicate reducing solutions, which tend to be rich in reduced species such as Fe^{2+} or SH^-. Values higher than 500 mV indicate oxidising solutions, which generally contain high concentrations of oxidised species e.g., $UO_2{}^{2+}$ or $AuCl_4{}^-$ (Gray, 1997a). The oxidation of pyrite and other sulphide minerals plays an important role in the generation of hydrogen ions (acidity) during weathering.

Particular complexing anions and/or solution processes are required to enable ground water mobility and the various complexes to become important under different chemical conditions as listed by Gray (1997b) in Table 1.4. Specific complexes are thio-complexes, halide complexes and organic complexes.

Thio-complexes

Sulphur forms a number of species with varying oxidation states from $^-2$ to $^+6$. Depending upon the concentration of reduced sulphur, the most important species for gold mobilisations appear to be (from lowest to highest oxidation-state):

- hydrogen sulphide (SH^-)
- solid sulphur, which does not mobilise gold
- thiosulphate ($S_2O_3{}^{2-}$)
- sulphite ($SO_3{}^{2-}$)
- sulphate ($SO_4{}^{2-}$), which does not complex gold.

The most important sulphur species for Au mobilisation appear to be hydrogen sulphide and thiosulphate.

Sulphide is oxidised to sulphate in the presence of oxygen, although the intermediate product, thiosulphate may be formed during pyrite oxidation under neutral to alkaline weathering conditions. Mann (1984a) calculates that 400–800

Table 1.4 Potentially important aqueous species (after Gray, 1997b)

Species	Possible origin	Solubility range
$Au(OH)_2^-$	Oxidative dissolution of gold under alkaline conditions	Oxidised pH > 8
$AuCl_2^-/AuCl_4^-$	Oxidative dissolution of gold under acid/saline conditions	Oxidised/saline/acidic
$Au(HS)_2^-$	Dissolution of gold by reduced waters during early supergene alteration, or by reducing solutions generated by biological action	Reduced/neutral Eh < −0.1 V pH 6–9 total sulphur > 0.02 M
$Au(S_2O_3)_2^{3-}$	Weathering of gold/pyrite in neutral to alkaline solution	Alkaline to weakly acid
$Au(CN)_2^-$	Interaction of cyanide with gold	Limited to cyanide content
Au – organic matter	Interaction of organic phases with gold	Not certain
Colloidal gold	Formed during reduction of gold by organic matter	Not certain

grams of $CaCO_3$ are required for every gram of FeS_2 to maintain alkaline conditions for thiosulphate production, and hence significant Au thiosulphate mobilisation.

Under highly reducing conditions gold bisulphide complex [Au $(HS)_2^-$] is particularly important for the hydrothermal transport of gold (Seward, 1973, 1982; Boyle, 1979) but has only a restricted occurrence in the supergene zone (Gray, 1997b). In the vicinity of sulphide deposits, sulphide may exist in solution at 10–1000 mg/L. [Au $(HS)_2^-$] solubility is greatest in neutral reducing conditions and assuming a total dissolved content of 2×10^{-6} M (700 mg/L) under optimum conditions, total dissolved gold will equal 6×10^{-6} M (1200 μg/L).

Halide complexes

The dissolution of Au chloride ($AuCl_2$) requires highly acid, saline and oxidising conditions:

$$2Au_{(S)} + 4Cl^- + \tfrac{1}{2}O_2 + 2H^+ \leftrightarrow 2AuCl_2^- + H_2O \qquad 1.1$$

Laboratory simulations of weathering in the presence of manganese dioxide have produced Au concentrations in acid (pH < 4) and highly oxidising (Eh > 680 mV) solutions containing one mole/litre chloride i.e., about twice seawater concentration (Cloke and Kelly, 1964; Lakin *et al.*, 1974). In this chemical environment the oxidation potential is controlled by the Mn^{2+}/Mn oxide redox couple as observed in water from an ore deposit (Panglo) near Kalgoorlie, Western Australia which has an Eh high enough for Au chloride dissolution.

Dissolution of Au chloride ($AuCl_2^-$) requires highly acid, saline and oxidising conditions as provided by deep continental brines; precipitation occurs under reducing conditions for example, in the presence of ferrous iron, which readily reduces Au:

$$AuCl_2^- + Fe^{2+} + 3H_2O = Au_{(s)} + Fe\,(OH)_3 + 3H^+ \qquad 1.2$$

With continued evaporation and increasing salinity, calcium is generally the first ion to be precipitated, either as calcite under neutral to oxidising conditions, or gypsum where there is an excess of Ca to CO_3^{2-}. Halide precipitates under high saline conditions as observed in the saline playas.

Organic complexes

Organic/biologically based complexes important for the mobility of Au in soils include cyanide complexes, organic complexes, colloidal gold and biological effects.

Cyanide complexes

Organic complexes capable of mobility of Au in soil profiles include cyanide complex Au $(CN)_2^-$. A highly organic horizon can contain high levels of cyanide and produce Au mobility. Gray (1997b) lists several authors including Watterson (1985), Korobushkina *et al.* (1974) and Rogers and Knowles (1978) whose studies of the influence of microorganisms on Au solubility, release or decomposition of cyanide and release of amino acid ligands may have value in Au exploration.

Au cyanide solubility is limited by the availability of cyanide. However certain plants and micro-organisms are known to release cyanide (Sneath, 1972) and can accumulate appreciable gold. Cyanogenic bacteria are frequently associated with plants, soil and organic matter and highly organic horizons or the immediate surroundings of an active cyanogenic plant root system could contain sufficient cyanide to promote Au mobility.

Colloidal gold

Gold readily forms molecular aggregations up to 5 μm in size (colloids or sols) and such chemical species have been known for centuries. Where stabilised by organic matter, colloidal gold has been observed in the laboratory by such workers as Goni *et al.* (1967), Ong and Swanson (1969) and Fedoseyeva *et al.* (1983) and has been postulated as an important mechanism for the mobilisation of gold. Being negatively charged, these colloids could be mobile in negatively charged soils, precipitating in contact with a soil horizon containing positively charged minerals such as Fe oxides. However, attempts to demonstrate the natural occurrence of colloidal Au have been unsuccessful (Boyle, 1979;

Kolotov *et al.*, 1980) due possibly to experimental difficulties at low Au concentrations. On this basis, Gray (1997b) suggests that Au soils may only be important for Au mobility in the presence of organic matter.

Biological effects

Some plants can absorb and accumulate gold but may also affect the re-distribution of gold by taking it up at depth and releasing it at the surface as litter (Erdman and Olson, 1985). In some cases it is believed that this hypothesis may account for a significant scale of gold depletion in sub-surface horizons of the regolith. However, while certain species of bacteria may either release or decompose cyanide (Smith and Hunt, 1985) thus potentially affecting Au solubility, the release of amino acid ligands by other bacteria could cause significant Au solubility (Korobushkina *et al.*, 1974).

1.1.3 Alloys of gold

Naturally occurring gold is never pure. Hydrothermal solutions leach other elements as well as gold from the rocks through which the solutions pass. Some of these elements are present in trace quantities only; the proportions of others such as silver and tellurium may be significant and materially effect fineness. Gold forms natural alloys with silver, copper, mercury and tellurium; less commonly with titanium, bismuth, palladium, lead and zinc. Varieties in primary ores include cuproaurite (copper gold), porpezite (palladium gold) and bismuthaurite (bismuth gold). Whilst these minerals are seldom found in alluvial detritus, their presence in a weathering zone may help unravel the geological history of an area under review.

Gold has a characteristic metallic yellow colour, but can be black or ruby when finely divided. In commercial operations mixing gold with other metals also changes its colour:

- blue – gold with iron
- green – gold with a higher silver content than copper
- pink (or rose) – 50% gold, 45% copper, and 5% silver
- white – gold with nickel, zinc, copper, and tin; manganese-nickel is often used because it bleaches gold
- yellow – 50% gold, 25% silver, and 25% copper.

The principal natural alloys of gold are native gold, electrum, amalgum and tellurides.

Native gold

Native gold is in itself an alloy, normally containing an isomorphous mixture of gold and silver in the proportions of 4 to 15% by weight, rarely less than 1%.

Gold in which silver is greater than 15% and less than 50% is classified as a separate mineral, electrum. It ranges in fineness units from about 800 to plus 990.

Electrum

Electrum, from the Greek 'elektron' (a substance that develops electricity under friction) is a common name given to all intermediate varieties in the isomorphous Au-Ag series. The physico-chemical properties of electrum vary with the silver content. Physically, with increasing silver the colour changes from yellow to near-white, the metal becomes less dense and decreases from about 800 to around 550 fine. Chemically with increasing silver, the lower fineness mixture becomes less stable than higher fineness gold and is thus more prone to alteration by weathering. Electrum is sometimes coated with halogens and sulphide compounds, which yield a thin film of native silver under suitably reducing conditions.

Amalgum

Mercury has a strong affinity for gold and may occur in nature as amalgum (Au_2Hg_3). Naturally occurring amalgum is much more widely distributed in gold-mining districts than previously thought. Fricker (1980) notes that 2% Hg has been identified in some South African gold, previously unrecognised because of its use in the gold recovery process. Dredgers in the Ampulit gold placer operation, Kalimantan, Indonesia recovered more mercury than was used in the treatment plant. However, this may have been due to the recovery of mercury lost in previous panning operations.

Cinnabar (HgS_2) belongs to the same type of mineralisation as most low-temperature, hydrothermal gold. It is generally more stable in oxidising conditions than most other sulphide minerals and due to its high density ($\rho = 8.09$) is often found in gold placer heavy mineral concentrates. Black films of metacinnabarite, native mercury and occasionally mercury chlorides may occur as secondary minerals of minor economic importance.

Tellurides

Gold tellurides occur widely in both epithermal and Archaean greenstone belt type deposits. The most common varieties are petzite $(AgAu)_2Te$, silvanite ($AuAgTe_4$) and calaverite ($AuTe_2$). Compounds such as aurobismuthite $(BiAuAg)^5$ S_6 are rare.

1.1.4 Measures of purity (fineness)

The purity of gold is measured in terms of fineness. Gold is commercially available with a purity of 99.999%. Gold of fineness 1,000 is pure gold, i.e.,

Table 1.5 Reported fineness of gold bars from early days of mining in Victoria, Australia

District	Fineness (carat system)	Fineness (1000 series)
Ararat	$23.0\frac{5}{8}$	961
Ballarat	$23.2\frac{5}{8}$	969
Dunolly	$23.1\frac{5}{8}$	965
Kingower	$22.3\frac{4}{8}$	931
Dry Gully	$20.3\frac{2}{8}$	847

equivalent to 24 K in the carat system used by jewellers. A 50% gold alloy is equivalent to 12-carat gold; 18-carat gold is $0.75 \times 1000 = 750$ fine and with a deeper colour. Alloys of gold with other metals provide mixtures of correspondingly lower fineness.

The advantage of the fineness system in mine evaluation studies is its elimination of fractions and decimal places. While gold of say 94.4% purity is designated 944 fine, it is an unwieldy 22.656 K in the carat system. The inconvenience of the carat system is demonstrated clearly in Table 1.5 which compares analyses of gold bars, reported in Fairfax's *Handbook to Australia* (in Smyth, 1869), in the early days of mining in Victoria, with the fineness of the bars as calculated by the fineness method.

Geochemical significance of fineness

Fineness is a rough and sometimes uncertain indicator of deposit type. For example, higher fineness Au is typically produced in larger grains by deep, high-temperature, high-pressure mesothermal solutions than by epithermal solutions, which tend to produce smaller particles of gold of lower fineness. But as a general model, this is often too simplistic; in addition to pressure differences at depths of formation, a number of other factors that may contribute to fineness include:

- the extent of adsorption of contaminating metals such as Ag and Cu along the flow path of the hydrothermal fluids and the conditions under which they are precipitated
- gradual facies changes in the mineral composition of the ores, due to merging of solutions of different composition
- the multi-stage nature of hydrothermal solutions
- reactions of fluids with the wall rock which vary as functions of the state of sulphur and oxygen in solution at various depths
- superposition of new mineralisation stages
- formation of veinlets of different ages within the same deposit.

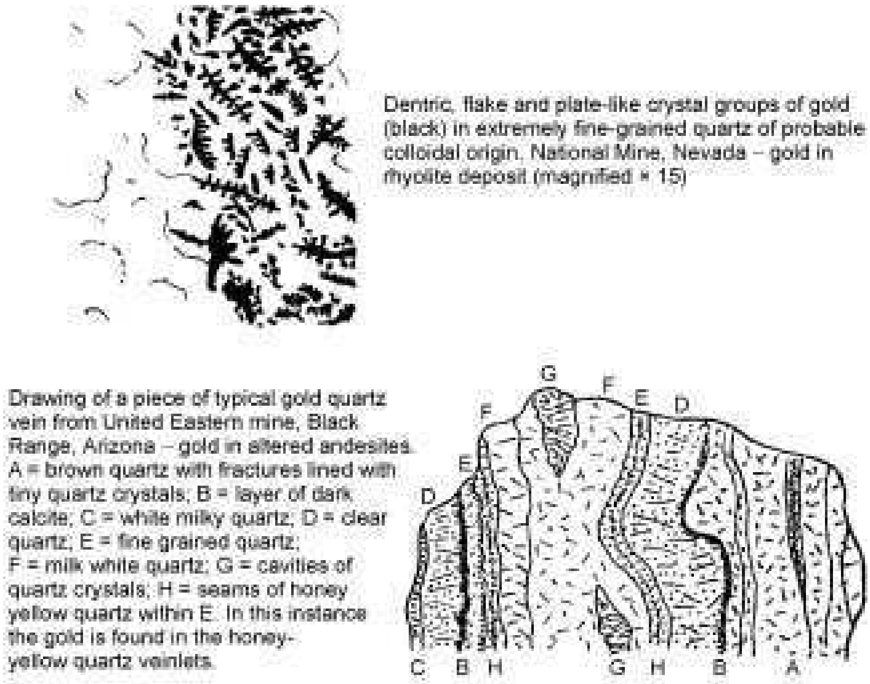
In fact, high fineness gold may occur in a wide range of rock types ranging from ultramafic through mafic and dioritic types to the granites, related porphyries and high sulfidation epithermal deposits. Gold in gold-rich porphyry deposits is mainly fine grained, less than 60 μm in size and is generally present as high-fineness native metal (Sillitoe, 1993). Of particular importance are high-fineness gold-bearing quartz veins derived from granitic rocks, often found in intruded rock (e.g. carbonaceous slates) well away from the parent intrusive. High fineness gold is also found in carbonate veins, particularly along continental margins and in gold-only veins in some skarn deposits, e.g. the Suian District, North Korea (Watanabe, 1943).

Regionally, some authors suggest that the nature of the basement may have some influence on fineness. Deposits that occur within the cratonic part of Guatemala and Honduras in the Central American epithermal gold belt are uniformly rich in silver while those of Costa Rica, Panama and Nicaragua, which are not underlaid by the craton are uniformly rich in gold. Similarly in North America, the deposits of Nevada and Colorado, which formed in cratonic sialic crust are pre-eminently silver-rich (Hutchison, 1985) whereas those lying outside the craton in California are gold-rich. From this he concludes that 'silver is of continental origin, whilst gold is of oceanic or mantle origin'.

1.1.5 Gold grain morphology

The morphology of a grain of gold is inherited from its primary state and to a large extent, irregularities of gold grains in source rocks predetermine grain morphology in an alluvial setting. Gold is one of the last minerals to crystallise out under hydrothermal conditions of deposition and thus tends to fill cracks and spaces between other minerals with which it comes into contact. The gold grains are moulded by the geometry of the opening into aggregates of irregular shape and size and commonly contain inclusions of quartz and other rock forming minerals. The Blanch-Barkly nugget (1,743 oz.) found at Kingower, Victoria Australia contained 2 lb of quartz, clay and iron oxide. In the Oso Peninsular, Costa Rica, Berrange (1987) describes flakes of gold less than 0.5 mm diameter containing quartz inclusions. He noted the presence, in both the vuggy interiors of quartz veins and smoother surfaced particles, of a variety of inclusions of syngenetic quartz, calcite, epidote and pyrite, together with adventitious Fe-Mg silicates, spinels and limonite veins picked up and included in the particles during transport. Micro-inclusions, although mainly quartz, may also be represented by heavy minerals such as ilmenite and corrundum.

Although physically, gold grains may be crystalline (octahedral, rhombo-dodecahedral or cubic) they occur more frequently in other forms. These forms include aggregates of irregular shaped grains in quartz and other hydrothermal minerals; reticulated dendrite, filiform, spongy massive and scales. Size, shape and surface texture are strongly influenced by their depositional environment



1.2 Two examples of gold grains in a primary state.

and associated minerals. Gold tends to be coarse grained when deposited in quartz veins with only small amounts of pyrite, but may occur in an extremely finely divided state in massive pyrite or other sulphide ores. Figure 1.2 shows two examples of the many ways that gold may occur in the primary state.

Gold grains in alluvial settings are distinguished by contrasting morphologies that reflect chemical and physical conditions in depositional environments that include regolith, glacial, fluvial, aeolean and shallow marine. When liberated, the type of gross deformation of a gold grain that the grain will undergo is predetermined by its original morphology. Pounding of equant grains produces flattening and cracking; flat grains are folded and pinched; elongated grains, which are transported by rolling, tend to develop into cigar-shaped particles. Lobate projections may represent gold grains that have been rounded by later supergene processes involving dissolution and re-precipitation of gold with limited fluvial or glacial transport. Table 1.6 summarises morphological and source rock relationships of some Asian placers.

Surface texture

Surface texture is a product of dynamic physical and electrochemical processes that control the modification and overprinting of secondary films or patinas of

Table 1.6 Morphology and size of gold vs. source (derived from Fedchuk *et al.*, 1978)

Source of gold	Type of gold	Morphology of colours	Size mm	Remarks
Gold from bedrock source	Hypogene (primary ores)	Xenomorphic angular, equant elongate, and laminar segregations. Hypidiomorphic crystallites glistening facets and dendritic intergrowths.	0.1–200	Coarse grains rare.
	Supergene (zones of oxidation)	Lumpy, spongy, with pimply pitted surface. Excrescences and projections of irregular shapes, octahedral crystallites and their intergrowths.	0.1–1000 up to 2000 and more	Fine grains retained.
Placer gold	Of hypogene origin	Flaky, drop-like, laminar wire-like, sometimes small crystallites with glistening or shagreen surface. Frequent impressions or crystals of hypogene minerals.	250–1000 dominant	Fine grains dispersed or agglomerated.
	Oxidation zones	Equant, thickly laminate with relicts of lumpy or spongy structure rarely incomplete crystalline shape; surface pimply pitted shagreen lustre uneven.	200–500 dominant up to 2000–3000 rarely nuggets	
	Hydrogenic	Dendrites, 'corals', aggregates and intergrowth of grains drop-like sinters with uneven lustre and rough surface.		Precipitation of fines from above.

pure gold over gold grain surfaces. Such patinas may either afford a measure of protection against wear by providing a more or less continuous cover over the exterior of the grain or, by gaining access along cracks, slip planes, etc., cause internal corrosion within the grain itself. A patina, once formed, is subject to removal or modification during recycling when exposed to physical wear or chemical weathering in an alkaline environment.

The nature of surface corrosion is, nevertheless, far from clear. Two conflicting schools of thought favour either variations of direct chemical plating of gold on grain surfaces or around nuclei or preferential leaching of silver from grain surfaces leaving behind a gold-enriched rim. Fisher (1935) argued for the removal of silver and the deposition of pure gold on the surface of gold grains by electrochemical corrosion. Based upon observations of gold grades in ancient terraces and conglomerate beds in the Bulolo deposits of New Guinea he concluded that average gold grades were enhanced as a result of this process. It was established later that while average grades were generally a few points higher in these deposits than in adjacent streambeds and flats, the gold was a mixture of both low and high fineness gold in various proportions and was derived from two different provenances.

Desborough (1970) analysed gold grains from 24 placers in the western United States and Alaska by micro-probe analysis. He concluded: ‘the low silver (high gold) rims of most of the gold grains examined, could best be explained by the apparent greater solubility and ease of oxidation of the metal silver in gold-silver alloy grains at low temperatures found in the placer environment’. Deposition of low-silver gold on the surfaces of gold grains was considered less likely, although chemically possible, because of the substantial quantities of gold that would be required from predominantly gold-deficient aqueous solutions. Chao (1969) measured the gold concentration in natural stream waters, including some of the streams from which Desborough’s samples were taken. These showed that, while the soluble gold content was generally less than 0.20 ppb, the silver content of the same solutions was two to 100 times that of the gold.

Boyle (1979) noted that ‘the weight of chemical evidence with respect to the relatively pure rinds of gold on nuggets points to the chemical precipitation of gold with some silver on the outer surfaces rather than that the silver was leached from the surface layer. All of the chemical evidence points to the probability that gold will be precipitated on any available gold nucleus provided that reducing agents are present’.

McDonald *et al.* (1990) examined mechanisms for the formation of gold rims on a number of gold particles from New Zealand, Australia and Alaska in Table 1.7. The tests suggested that formation of gold-rich rims on gold grains is a function of the silver content of the native gold, the chemical modification of low fineness gold (e.g., 20% Ag) being due to the preferential leaching of silver from grain surface. High fineness gold could have a rim of almost pure gold

Table 1.7 Composition of bulk and gold-rich phases in mature gold from 15 localities in Australia, New Zealand and Alaska (after Macdonald *et al.*, 1990)

Particle number	Particle source	Bulk phase (wt%) Au	Ag	Cu	Total	Gold-rich phase (wt%) Au	Ag	Cu	Total
1	Moliagul	95.6	2.91	0.03	98.5	98.5	0.37	0.00	98.9
1	Moliagul	96.1	3.41	0.02	99.5	99.8	0.59	0.02	100.4
1	Moliagul	94.4	3.29	0.00	97.7	96.5	0.68	0.00	97.2
1	Moliagul	94.9	3.42	0.03	98.3	98.3	0.31	0.00	98.6
2	Inglewood	90.4	8.67	0.03	99.1	98.4	1.57	0.03	100.0
2	Ingelwood	–	–	–	–	–	–	–	–
3	Ingelwood	–	–	–	–	–	–	–	–
4	Inglewood	92.1	8.04	0.01	100.1	99.8	1.25	0.00	101.1
5	Majorca	–	–	–	–	–	–	–	–
5	Majorca	94.6	3.77	0.00	98.4	96.1	3.69	0.00	99.8
6	Oberon	74.0	22.9	0.85	97.8	95.8	2.00	0.35	99.2
6	Oberon	–	–	–	–	–	–	–	–
7	Palmar R	85.3	10.8	0.11	96.2	97.9	0.04	0.00	97.9
8	Roc Palaeochannel	96.8	2.09	–	98.9	99.3	0.42	–	99.7
9	Brighton Terrace	90.7	5.43	0.00	96.1	93.2	3.76	0.41	97.4
10	Sherry River	90.3	6.13	0.62	97.1	93.6	0.11	0.29	94.0
11	Faith Creek	72.8	25.2	0.44	98.4	99.8	0.89	0.36	101.1
12	Faith Creek	72.6	23.9	0.16	96.7	97.2	1.25	0.45	98.9
13	Alaska	73.1	21.3	0.04	94.4	96.0	1.91	0.03	97.9
14	Alaska	88.4	9.05	–	97.5	97.9	0.72	–	98.6
14	Kaituna River	86.2	10.2	0.28	96.7	91.3	0.48	1.18	93.0
						76.9	0.00	22.0	98.9*

* Copper-rich rim around the particle.
– indicates 'not determined'.

deposited on it and deposition of silver and copper alloys may also occur. These mechanisms were held to explain observations by Haslam *et al.* (1990) of platinum on gold particles and Leake *et al.* (1990) of palladium on a gold grain.

In examining the possible economic effect of gold enrichment on the surface of gold grains, Bowles (1988) found no evidence to suggest that the overall effect on gold fineness of surface plating alone might be significant. This agreed with the Berrange (1987) comparison of microprobe analyses with chemical assays (Table 1.8), which also suggested that the higher fineness of the patina does not materially affect the sample grade as a whole. Other cases, in which the average gold fineness of secondary gold is substantially increased, are due to internal corrosion processes.

Internal corrosion

Internal corrosion develops along slip planes, cracks along planes of metal failure or other paths of access to the interior of the grain. Gold grains, as

Table 1.8 Composition of placer gold from Osa Peninsula (after Berrange, 1987)

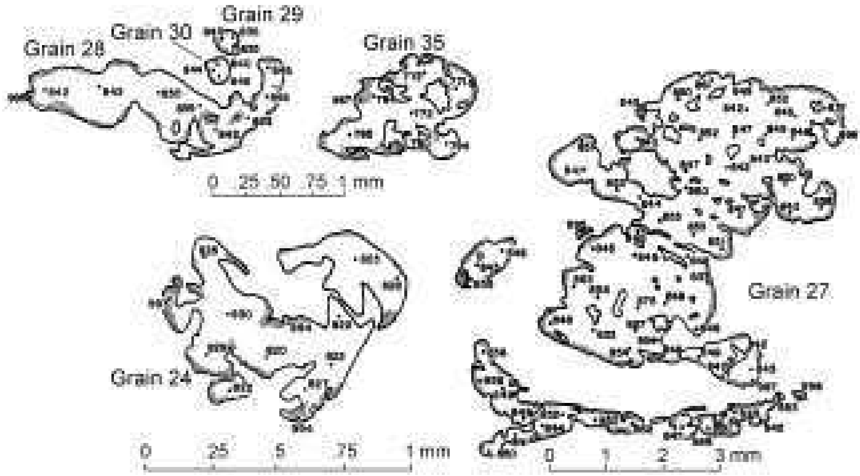
	Gold (%)			Silver (%)			
Analytical technique	Ave.	Max.	Min.	Ave.	Max.	Min.	Other trace elements identified (%)
Banco Central	93.8	94.5	92.4	5.4	6.8	4.6	Fe, P, Te, Pb, Zn, Ni, Mg
Microprobe (UL)	93.8			3.96			Cu=0.13 Zn=0.09
Microprobe (L)	97.1			0.46			Cu=0.06 Zn=0.15
XRD	95.96 (sd=2.08)			3.68 (sd=1.92)			Fe=1.045 PGE=0.244 Pb=0.163 Te=0.127 Hg=0.118 Zn=0.044 Cu=0.039 Bi=0.036 As=0.026 As=0.003?

UL = unleached area.
L = Ag-leached rim.

corroded in supergene or soil environments are protected against mechanical wear. Prior to achieving physical equilibrium the patina may seal the grain against both internal corrosion and physical decrepitation. Before this happens however, anodic effects in the depths of cracks may be intense because of the dissimilar nature of pure gold patinas and native gold. In a physically active environment such as an abrasive fluvial setting, development of paths of access may also involve significant and largely destructive processes entailing the formation of impact craters when small particles are pressed or impacted into grain surfaces. Cracks and striations may result from high-energy fluvial deformation or high-pressure glacial action. In some cases, the end result is fracturing of the grain along the line of corrosion and the liberation of small equant particles of supergene gold.

Microprobe analyses of a selection of coarse-sized, low-fineness gold from two placer deposits in Gold Creek, Granite County, Montana (Loen, 1994) produced data relating to the evolution of nuggets and effects of supergene processes on both silver depletion and gold enrichment. An ARL SEMQ electron microprobe was used for quantitative analysis. Semi-quantitative spectrographic data combined with the microprobe data suggested a trace element signature that could be related back to the type of lode source. Polished sections of placer gold grains showed areas of colour and reflectance that corresponded to differences in composition. Many of the grains are rimmed by 0.020 to 0.100 mm thick high-fineness gold whereas the core gold is of lower reflectance and higher silver content.

Figure 1.3 illustrates a tracing of polished sections of gold grains from the Master Mine showing location of rims (stippled) and gold fineness patterns determined by electron microprobes (dots representing analysis points). Grains



1.3 Tracings of polished sections of gold grains from Master Mine, Gold Creek, Granite County, Montana (after Loen, 1994).

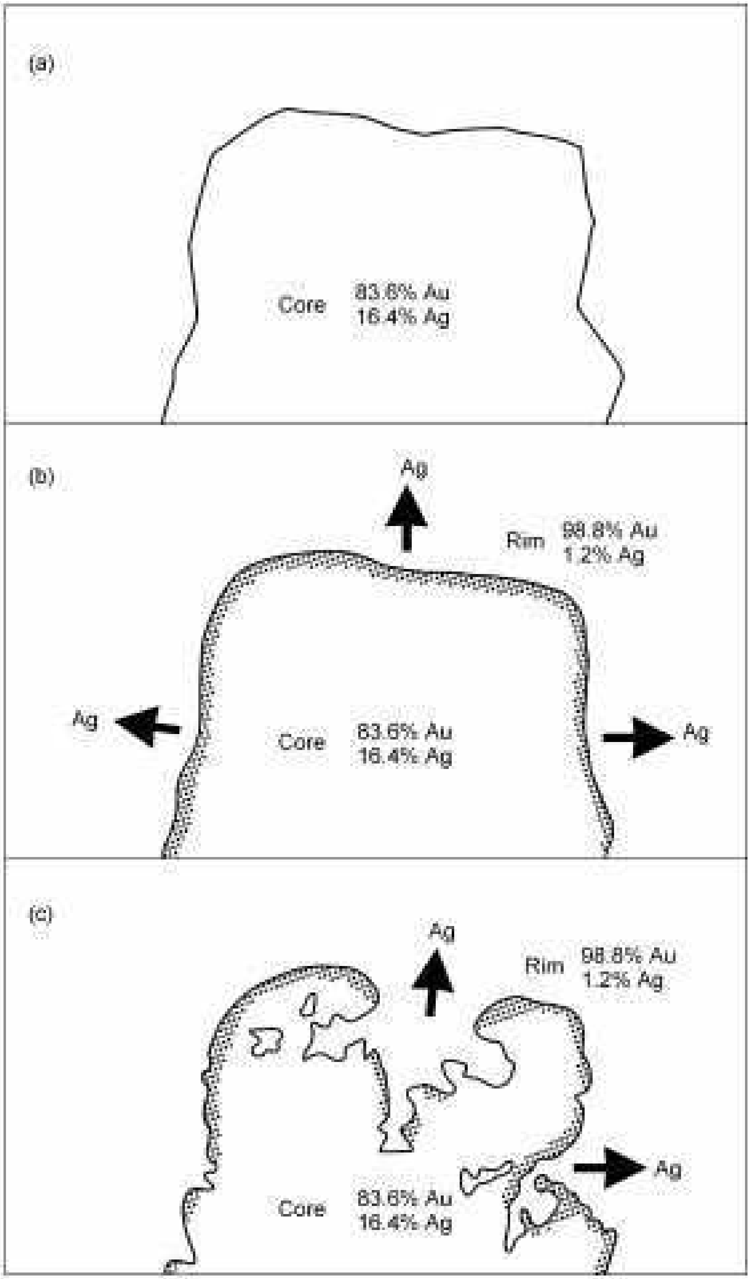
24 and 27 contain several elements that are separated as a result of polishing. Loen (1994) infers, from his examination of gold grains from sediments in Gold Creek, Granite County, Montana, that a vein of gold crystal develops initially rounded edges having high-fineness. Any weaknesses in the rim are penetrated and low-fineness gold-silver alloys within the grain are dissolved, possibly in multiple episodes of gold dissolution. Development of deep cavities is then followed by precipitation of nearly pure gold on the interiors of the cavities. Local penetration of the gold enriched border leads to dissolving of the low fineness Au-Ag interior causing development of cavities followed by precipitation of nearly pure gold layers in the interiors (Fig. 1.4c).

Giusti (1986) observed three major types of gold in his studies of the morphology of fine-grained gold in sediment of the North Saskatchewan and Athabasca Rivers, Alberta Canada:

- primary gold, often hidden in folds in the grain
- secondary gold related to re-disposition of 'new gold' in the grains
- secondary gold related to plastic deformation and internal recrystallisation.

Although none of the grains appeared to be of entirely secondary gold, all of the gold grains collected by the author showed the presence of gold-rich rims ranging in thickness between 1 and 30 μm. Giusti (1986) proposed a number of simple criteria (Table 1.9) as general aids to deciphering the geological history of gold placers recognising, however, that the assumption of a single cycle of erosion is an oversimplification.

Sketches by Giusti (1986) illustrate the nature of high-grade gold precipitation on fine-grained gold under the influence of mechanical and electrochemical



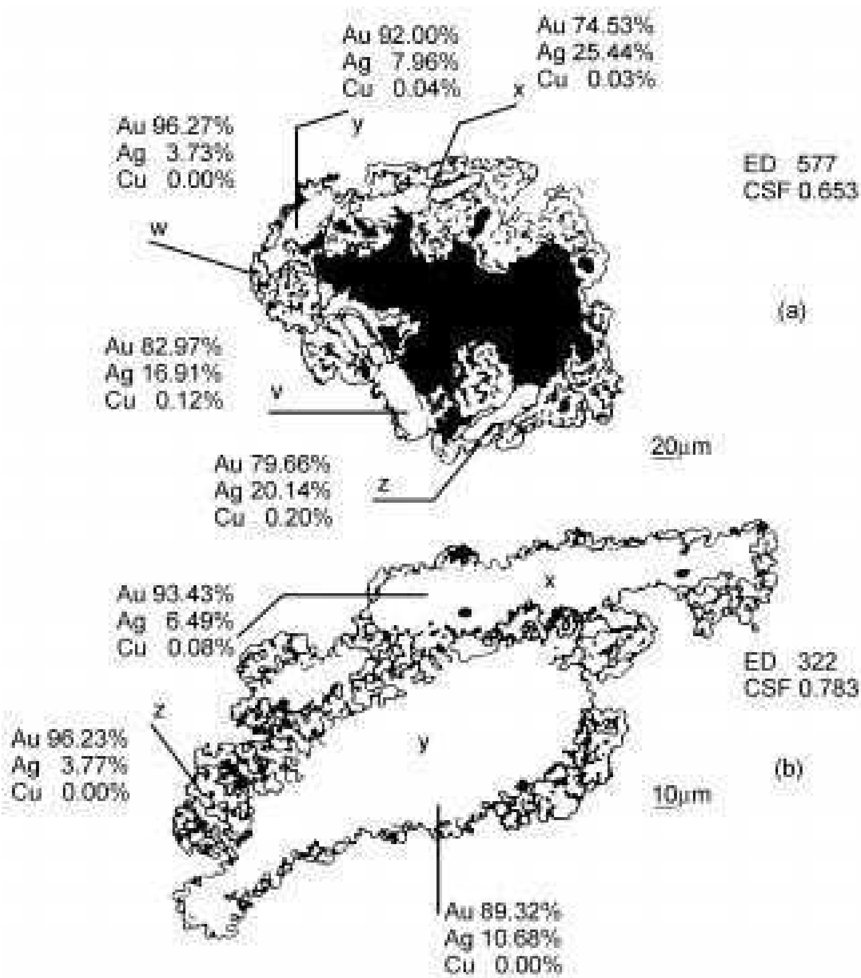
1.4 Inferred stages of development of rims and cavities of low-fineness gold grains (after Loen, 1994).

Table 1.9 Simplified model relating placer gold characteristics to environment and best recovery methods (from Giusti, 1986)[a]

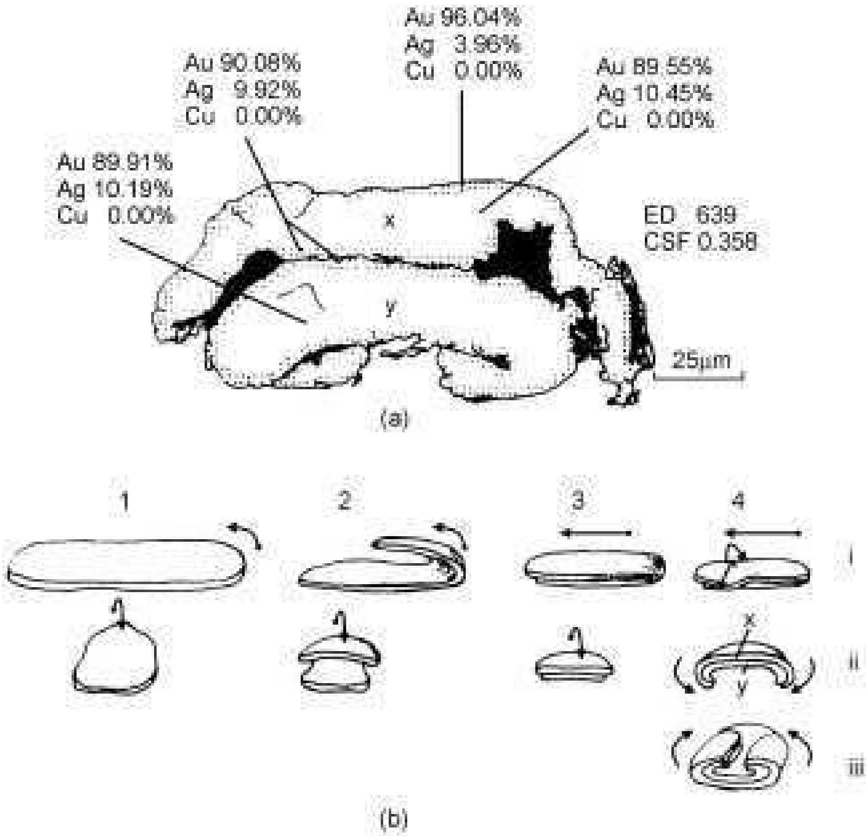
		(Increasing distance from source) →			
	Eluvial placer		Alluvial placer		
Gold grain morphology	Irregular; primary crystals still preserved; lots of inclusions; very high CSF	Irregular, rounded protuberances; some primary crystals still preserved, often in cavities or folded portions of the metal; mainly inclusions of quartz; high CSF	Flaky, jagged surface rounded outline; re-crystallised; plastic deformation; medium–small CSF	Flaky, rounded, multiple bending; re-crystallised; some secondary octahedral crystals on the surface; high CSF	Rounded, often porous; small ED; high–very high CSF
Rim effect	Porous rim is frequent	Porous to compact	Compact	Compact	Compact to porous
Abrasion	Moderate	Strong	Strong	Strong	Moderate
Chemical weathering	Very strong	Moderate–minor	Minor	Minor	Minor
Most-representative mesh size	+35	−35 to +120	−120 to +200	−200 to −400	−200 to −400
Prevalent sediment type	Cobbles, pebbles	Pebbles, sand	Sand	Sand, silt	Silt, sand
Environment	High energy	High–medium energy		Medium–low energy	Low energy
Suggested recovery methods	Mechanical (panning, sluicing, jigging, rocking)	Mechanical and chemical (sluicing, tabling, flotation)	Chemical and mechanical	Chemical and mechanical	Chemical and mechanical

[a]The model assumes (1) monocyclic gold and (2) relatively coarse-grained gold at the source.

factors. Figure 1.5(a) depicts a gold grain as a composite of four different gold particles x, y, v and z, held together by 'new' high-grade gold, the black zone represents a vugh in the central part of the grain. Figure 1.5(b) is a composite of two gold particles of different composition held together by new spongy gold of lower Ag content. The new spongy gold (z) has the lowest Ag content (average weight percent). Figure 1.6(a) sketches a polished section of a strongly bent gold grain, which has two different, consecutive generations of rims. Figure 1.6(b) is a sequential representation of the evolution of a gold particle in a placer assuming an originally larger unbent grain. This model is a simplified model, which assumes an original unbent grain. Giusti (1986) suggests that if the gold



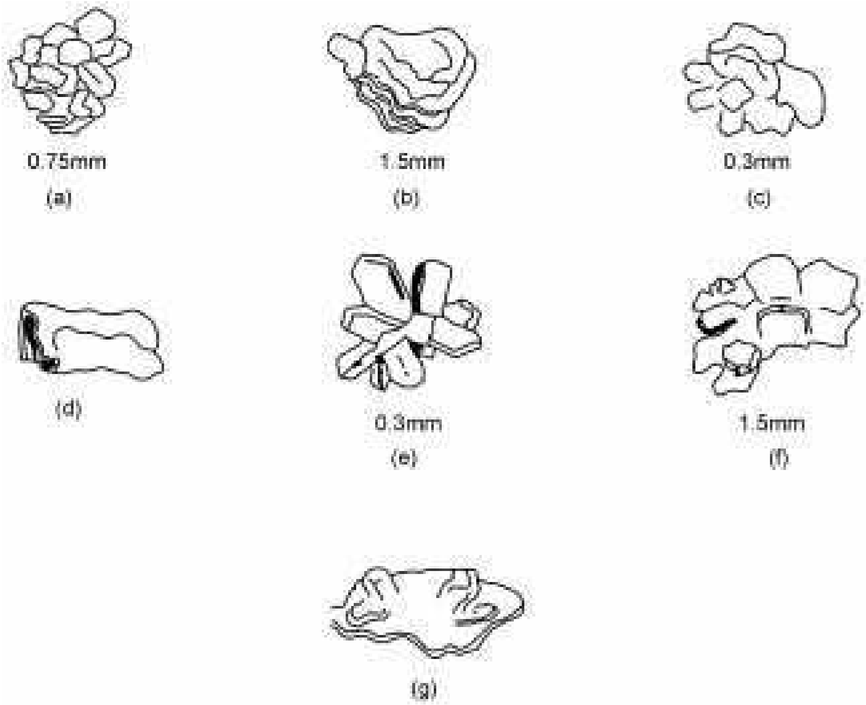
1.5 Sketches of polished sections of two composite gold grains (from Giusti, 1986).



1.6 (a) Sketch of polished section of gold grain showing two different consecutive generations of rims. The high-grade film around the upper surface of x has a lower Ag content (3.96 avg. wt %) than the one on its lower surface, or on the upper surface of y (9.92 avg. wt %). At the contact between x and y. (b) Sequential diagram representing four major stages (1 to 4) of the evolution of a gold particle in a placer: (i) side view, (ii) front view, and (iii) bottom view. At stage 4iii the shape of the gold particle is similar (upside down) to that of the grain in (a) (from Giusti, 1986).

grain is irregular in shape at the moment in which it is introduced into the placer, the morphological evolution presented in the sketch is even more likely to occur. Sketches of detrital gold grains recovered from Olipai flood sediments in the Lakekamu Embayment, Papua New Guinea by the author of this text are shown in Fig. 1.7.

Some geologists suggest that biochemical corrosion of gold by plant life is responsible for the entrapment of gold by chemical or biochemical means both inside and outside hyphae of plants or by both. Observation of 'hair roots' within openings in gold grains in some geochemical soil anomalies is thought to be



1.7 Sketches of detrital gold grains, Olipai River, Papua New Guinea: (a) crude crystal group; (b) layered gold flakes; (c) crude crystal group; (d) rolled up gold flake; (e) distinctly crystallised crystal group; (f) discoidal gold flake with outward projecting crystals; (g) discoidal gold flake with outward projecting crystals.

associated with gold precipitating bacteria and that the gold may have been biologically as well as chemically mobilised.

Origin of nuggets

The origin of gold nuggets has probably been debated since man first explored for the metal. Possible explanations have ranged from the supernatural to physical and chemical accretion, or simply to a primary origin. Arguments that gold nuggets may be formed by chemical accretion include the apparent occurrence of larger gold particles in the regolith than in underlying vein systems. Geochemical proof of the deposition of gold on suitable reductants and the occurrence of gold dendrites in zones of secondary enrichment is well known. The main arguments today seem to be based upon questions of scale.

In 1853, Hopkins (in Smyth, 1869) supported the aggregation concept: 'large masses of gold have been found near the roots of large trees and strong grass, often in very singular forms and evidently indicating the influence of roots in the formation and in the amount of gold drawn out of the rock below'. In the late 1850s, Selwyn and Ulrich (in Smyth, 1869) suggested cutting nuggets in half and determining, by appropriate analyses, differences between gold in central portions of reefs and those of surrounding masses, formed by deposition from meteoric waters. Another proponent of the growth theory, Dr Landsweert (1869) experimented with very dilute solutions of gold chloride using carbonaceous material (brown iron ore) as a decomposing agent. In his opinion, 'the occurrence of larger nuggets in gravel deposits that have been found in quartz ledges, with the fact that alluvial gold almost universally has a higher standard of fineness, would seem to imply a different origin for the two'.

There have also been many sceptics. Smyth (1869) refers to a paper by a Mr George Foord, reputedly the foremost authority of the day on all questions relating to the chemistry of gold. In this paper Foord writes, 'A good deal has been said and written concerning the formation of nuggets by the coalescence of grains in the alluvial state. No one can say that this has never taken place but if my opinion was asked, it would lean very little to the acceptance of this view. The physiognomy of nuggets points almost invariably to their position in the lode.'

Nor did Emmons (1940) find much in support of the growth theory, although it was much favoured by miners in the goldfields of California and Australia. He pointed to the size of the Holtermann nugget (Fig. 1.8) from a lode in Hill End, NSW, Australia, which weighed almost 2,800 oz. and to a piece of gold from Carson Hill, California, which weighed 2,300 oz. Both of these nuggets were heavier than the two largest alluvial nuggets from the Victorian goldfields 'Welcome' (2,218 oz.) and 'Welcome Stranger' (2,268 oz.).

In his review of arguments for and against the formation of nuggets by chemical accretion Boyle (1979) concluded that the gold in placers is of both detrital and chemical origin. In his opinion 'one should view the formation of nuggets in a dynamic sense, the agents forming them being both chemical and mechanical and their action being concomitant'.

In 1926, Johnson and Ugloo (1926) proposed a chemical explanation to account for the appearance of coarse gold in the Pleistocene alluvial deposits of the Cariboo Mining District, British Columbia, Canada. In their view, the gold deposits could have derived from deep weathering and supergene enrichment of quartz veins containing arsenopyrite and pyrite in Tertiary times. In support of this interpretation, Eyles and Kocsis (1989) compared the fineness of the lode gold (500–911) with that of the placer gold (775–950). They suggested that the increased fineness of the placer gold together with the crystal shape of the placer gold (dodecahedrons, cubes and octahedrons) is indicative of deposition from solution and the possible gradual accretion of gold particles by supergene processes. Supergene gold enrichment is reported from Bingham and Ok Tedi where a portion of the gold is substantially coarser than in the subjacent sulphide zones (Rush and Seegers, 1990). Under different environmental conditions,

1.8 Holtermann Nugget photograph from *Mineral Industry of NSW* (1928).

Lawrence (1984) remarks on the finely divided state of most supergene gold in the Pacific region of the Southern Hemisphere, stating that the gold is deposited characteristically as tiny, discrete, cubo-octahedral crystals scattered throughout the weathered rock.

The possibility that large gold nuggets may be formed by supergene accretion in the oxidation zone of a primary gold orebody cannot be entirely discounted because mechanisms for chemical accretion do exist. However, no examples are known of nuggets comprising central cores of primary gold encased in thick layers of high-fineness gold of sufficient proportions to materially increase the average grade of the nugget. Indeed, the greater abundance of nuggets found in residual rather than in lode material may simply be due to the enormous quantities of primary gold-bearing material from which the regolith has derived, compared with the very small quantities of hard rock mined in nugget producing areas. Whiting and Bowen (in Smyth, 1869) are noted as saying, in 1876, that 80 of 118 slugs of gold (each weighing more than 0.6 kg) recorded from Victorian quartz reefs were found within 50 cm of an intersecting quartz reef and indicator. The location of the others had not been recorded, but the authors had little doubt of a similar genetic origin. Figure 1.9 is a photograph of miners at Hill End, NSW displaying their gold nuggets.

1.9 Miners at Hill End, New South Wales, proudly displaying their gold-quartz nuggets (Holtermann Collection, Mitchell Library, Sydney Australia) (after Nolan, 1980).

Discovery and exploitation of supergene gold deposits in the lateritic regolith of the Yilgarn Block, Western Australia (Butt and Anand, 1997) provides further evidence of secondary gold enrichment in the regolith. A review by Wilson (1984) suggests that while many of the nuggets are clearly weathering resistates, primary in origin with Ag content >5%, others of higher fineness (Ag < 5%) have features that indicate secondary accretion. Mann (1984a) postulates nugget distribution similar to that for gold enrichment in general. Nuggets have also been found in laterite of the Cloncurry region of Queensland, often where little or no basement gold mineralisation is known.

However, while mechanisms for secondary gold enrichments in weathering profiles are well known, the origin of large gold nuggets within these profiles is still uncertain. In partly dissected terrain the nuggets may be residual from now eroded overlying horizons; even the apparent association of nuggets with the base of the pisolitic laterite horizon and with calcrete may originate in this manner (Butt, 1988).

The fine gold problem

The question of what constitutes fine gold has never been satisfactorily resolved because of the many different standards applied to finely divided particles by chemical, mining and civil engineers. To the early miners, fine gold was gold that could not be recovered easily in the prospecting dish, and little notice was taken of the loss of a few fine colours in samples dressed by panning. Good recoveries were not expected below 200 μm and generally the term fine gold referred to grains smaller than about 150 μm in size. In modern jigging practice, good recoveries of gold are reported down to 150 μm (Nio, 1988), and some jig manufacturers suggest 100 μm as a realistic cut-off point for fine gold from a well-classified feed. Early bowl type centrifugal separators used forces up to 60 g to enhance gravity differences between particles. Encouraging gold separations were made at sizes down to about 38 μm under laboratory conditions and to 100 μm from high slime feed materials in processing plants.

However, the finely divided state of the gold is only one consideration influencing its hydraulic behaviour, account must also be taken of factors other than that of particle size in gravity gold mineral processing. As discussed in Chapter 4, hydrophobicity, shape, particle density and texture and the slimes content of the feed all affect particles settling rates and transportability.

Influence of hydrophobicity

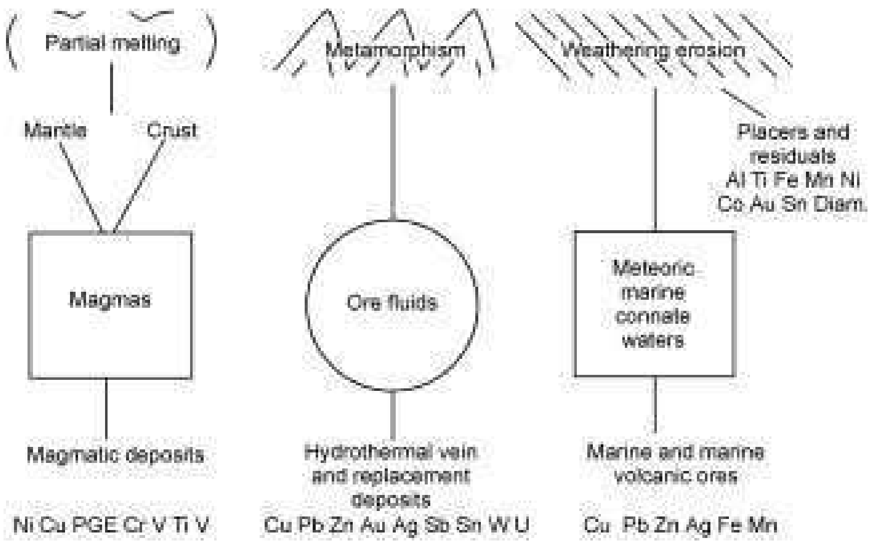
Hydrophobic surfaces have the property of repelling water, i.e., they do not easily become wetted in contact with water. The phenomenon is due to unbalanced molecular forces at the water/solids interface causing surface tension. In the case of gold, the surfaces are variably hydrophobic depending upon the purity of gold in the surface layer and whether the surface is clean or coated. Any process such as leaching or plating that increases the purity of the outer skin increases its hydrophobicity.

Finely divided gold particles, which have high surface area/mass ratios, tend to float on the surface of water because of surface tension and be lost where larger particles of gold with smaller surface area/mass ratios sink. The phenomenon thus poses considerable difficulties in sample dressing unless reagents are introduced into the water to lower its surface tension. Household detergents are in common use in most countries for this purpose. In Kalimantan, Indonesia, the Dyak miners use an extract from the leaves of chilli bushes to settle the gold. The leaves are pulped in water and after a short time, the panner steeps his fingers in the solution and then into the water in the panning dish. Any gold floating on the surface of the water immediately sinks to the bottom.

The problem of fine hydrophobic gold, as dealt with in bench-scale investigations, may not be as readily solved in a prototype treatment plant where it is neither practicable nor desirable to add reagents to the process water. No economic method has yet been devised to scavenge the hydrophobic gold content of slime material discharged from primary roughing circuits. The quantities of slimes are very large and desliming is necessary both to reduce the volume of flow and to provide a classified feed for primary concentration.

1.1.6 Classification of gold ore deposits

Figure 1.10 is a simplified genetic classification scheme for ore deposits showing the broad categories of processes and some of the common associated



1.10 Simplified classification scheme for ore deposits showing the broad categories of processes and some of the associated elements (modified from McQueen, 1997).

elements. The geological requirements of hydrothermal systems are linked together through processes and associated elements that link placer and residual minerals including gold with partial melting, metamorphism, weathering and erosion. An understanding of these linkages is fundamental to gaining a full understanding of ore deposition as a whole. Conceptual models of primary gold ore deposition (see Chapter 2, Section 2.3) are based upon a variety of classification schemes and may be related to:

- a source for the primary ore components (ligands and metals) and of energy to drive the system
- a transport mechanism that controls the flow of fluid between the source and discharge regimes and allows the appropriate concentration
- the chemistry of the fluid phase and a depositional mechanism to distribute the components in the orebody as ore minerals and associated gangue.

Assignment of residual and placer gold types to specific groups within one particular system of classification is fraught with much greater difficulty. Most classifications tend either to represent specific research interests or be more practically orientated as a result of field experience in different geological and climatic environments. Problems of interpretation arise due to the uncertainty of many of the processes and source parameters needed to complete the cycle. Included amongst these are:

- the wide spectrum of sediments generated by the activities of glaciers, rivers, wind and marine agencies (Chapter 3)

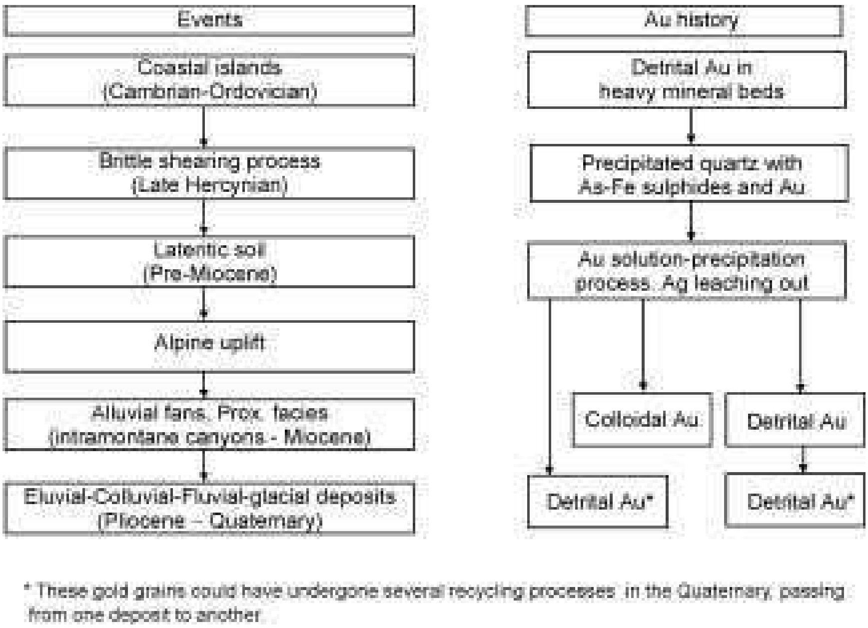
Table 1.10 Estimated earth movement and gold recovered in NW Spain during Roman times

Deposits	Earth movement (m^3)	Gold recovery (kg)	Average gold grade (mg/m^3)
Alluvial fan (Miocene)	203,000,000	10,200	50
Fluvial deposits (Pliocene)	20,000,000	1800	90
Fluvial deposits (Quaternary)	73,000,000	7300	100
Moraines and residual placers (Quaternary)	12,000,000	1200	100
Total placers (Neogene-Quaternary)	308,000,000	20,000	67
Quartz veins (late Hercynian)	290,000,000	170,000	600
Total gold		190,000	

- the diversity of conditions affecting the liberation and modification of gold grains in zones of weathering (Chapter 2)
- base level changes induced by isostatic adjustments (Chapter 4)
- the effects of climatic cyclicity and extremes on fluvial transport and sorting (Chapter 5).

Classifications in Western countries generally follow schemes based upon geological location and tectonic uplift or depression, as proposed by Lindgren and summarised in Table 1.10 by Boyle (1979). Lindgren (1911, 1933) taught that gold placers do not occur haphazardly or by chance, but as a result of particular sets of geomorphic processes in specific locations. In Sierra Nevada, he linked the effects of long periods of deep tropical weathering during the Eocene with the large-scale production of gold-bearing sediments and their deposition in valleys of the time. By tracing the subsequent history of volcanism and tectonic uplift, Lindgren provided cogent reasons for the restriction of the larger deposits to pre-volcanic valley systems and elucidated the genesis of deep leads. He demonstrated, moreover, that geomorphic processes leave their imprint upon the geological record, so that an original imperfect understanding of the evolution of a particular deposit may become clearer when fundamental principles are used to interpret observed facts.

Boyle (1979) stressed the great complexity of the geological history of many productive placers noting in particular the wide range of sources from which auriferous placers derive and the involvement of intermediate collectors of the gold such as quartzites and conglomerates. Geologists in Asian countries have generally followed upon Russian experience. Groupings by Smirnov (1962) consider aspects of both the mechanism of formation of placers and their geological environment at the time of formation. Considerable attention is paid to the geomorphic analysis of placer formation and preservation in the Urals.



1.11 Influence of pre-Miocene gold distribution on possible recycling processes of Quaternary placers in northwest Spain (after Perez and Garcia *et al.,* 2000).

In Spain, the geological history of the gold-bearing surficial deposits of NW Iberia is interpreted according to a classification (Fig. 1.11), which demonstrates the influence of the pre-Miocene age of gold distribution on possible recycling processes of Quaternary placers. The Miocene sediments (mainly alluvial fans) are derived from quartz veins in the Cantabrian and Leon Mountain uplifts of Cambrian and Ordovician sandstones and quartzite source rocks. Free gold was passed to alluvial fan deposits as detrital particles and/or as colloidal solutions (Perez-Garcia *et al.*, 2000). Ongoing Quaternary erosional/depositional processes produced deposits comprising principally regolith, fluvial terraces, moraines and glacial-fluvial fans.

Classification of gold ores in this book is subdivided into primary (Chapter 2), residual and alluvial type deposition (Chapter 3), firstly according to differences in formation and then in terms of topographic relief, climate and time. The following constraints are important:

- Limitations to the application of provenance in placer gold exploration are governed by the overall pattern of drainage or by the existence of a secondary source of provenance; for example, one tributary of a river may drain platinum-bearing ultramafics, whilst another might cut through gold-bearing terrain; ultimately this would bring together unrelated rock-forming minerals in which quartz would predominate.

- Economic minerals, including gold, of the various lode formations may not be located in sufficient quantity in a common drainage basin to form a viable placer deposit.
- The incidence of indicator minerals such as rock forming silicates in a placer is no guarantee that economic quantities of gold will be found in the alluvial or primary lode system.
- Placer geometry changes continuously due to structural alterations that occur as the result of changing rates of precipitation and run-off in the short term and by tectonic uplift, fluctuating base levels and climatic cyclicity and change in the long term.

1.2 Gold through the ages

In tracing the growth of scientific thought from its beginnings to present-day levels of understanding, it is important to note how many modern scientific concepts had already been foreshadowed and argued by ancient philosophers. Throughout history there have always been people who have sought explanations for how the Universe was formed, how it works and man's place in it. The Ancient Greeks were amongst the first to investigate the evolution and structure of the Earth. More factual than the Greeks, the Romans were probably the first to organise gold mining and exploration in a systematic fashion. Asian and Eastern scientists were in many ways more inventive than other world scientists were. Many significant scientific discoveries were made in China centuries or even millennia before they were made in the West. Important observations in physics, mathematics and medicine were made in India and clearly, scientific progress has never been impeded by genetic constraints; black, white, yellow and all races regardless of colour have made major scientific discoveries. Historically, only the religions, regardless of creed have imposed barriers to progress, either rejecting harshly or accepting only slowly and with great reluctance any ideas contrary to their particular religious dogma.

1.2.1 Gold in ancient times

Stone-Age people fashioned their tools and weapons from natural materials until about 10,000 years ago when atmospheric warming at the end of the last great ice age gave rise to melting of the ice caps and the widespread concentration of alluvial gold in streambeds of the day. Prehistoric man relied heavily upon meat and followed migrating reindeer as the ice sheets retreated more or less simultaneously in the Middle East, Orient and the Americas. Stone-Age implements were gradually replaced by tools made of native metals, (mainly copper and meteoric iron) which could be hammered into desired shapes. Gold was probably gathered for ornamental purposes because of its unique colour, and a few nuggets have been found in some later-period graves (Smirnov, 1962). The first evidence of large-scale fabrication of gold has been revealed by modern archaeologists who discovered highly sophisticated gold art objects and jewellery dating back to about 3000 BC in Sumerian tombs at Ur in Mesopotamia. By 1200 BC, goldsmiths of the Chavin civilisation in Peru were making gold ornaments and other artefacts by hammering and shaping.

Sumerian influence

The earliest recorded civilisations, Sumeria and Babylonia, were founded between the twin rivers Tigris and Euphrates in lower Mesopotamia some 4,500 years BC with priests as secular rulers. Engineering practice was directed initially towards the invention and manufacture of farming equipment and machines for the supply and control of water for agricultural purposes (Vargos and Gallegos, 1992). The foundation was laid nevertheless for the development of all branches of modern engineering practice. Civilisations in China and India may have been developed around the same time, but there was little contact between the East and the West and the influence of ancient technological development is traceable mainly to the Mesopotamian culture.

The many Sumerian achievements included discovery of the wheel, moulding and burning of mud bricks for building, and ploughs pulled by cattle. By inventing a form of writing (pictographs with 2,000 symbols) they introduced the first means of recording thought as well as history. Between 1900 and 1800 BC, Mesopotamian mathematicians are believed to have discovered the theorem later to be known as the 'Pythagoras Theorem' by the Greek mathematician and philosopher Pythagoras (580–500 BC). Multiplication tables appeared in Mesopotamia around 1750 BC. In 1600 BC, Chaldean astrologers in Mesopotamia identified the Zodiac. Sumerian engineering skills had a profound influence on the development of gold processing technology. Gold was smelted in Egypt and Sumeria about 3500 BC and around 2500 BC a form of soldering was developed by the Chaldeans in Ur, Mesopotamia, for joining sheets of gold.

Understanding of hydraulic principles made possible the construction of dams, and the design of water control mechanisms and channelling of water over long distances. Construction difficulties were resolved as they arose; e.g. the firing of mud bricks to give added strength for the formation of cantilevers and arches. This made possible the design of supports for both mining excavation and metallurgical structures. A ziggurat in Ur (Mesopotamia) 12 m high demonstrated the familiarity of Sumerians with columns, domes, arches and vaults (Hellemans and Bunch, 1988).

By around 2000 BC the early Minoan civilisation of Crete became a gold centre surpassed only by Egypt. Although there were few natural resources, the Minoans were superb goldsmiths, skilled in the most advanced kinds of gold fabrication. According to Greek tradition, Homer attributed the magnificently decorated shield of Achilles to Hephaestus of Crete. Eruption of the volcano of

Santorini is believed to have devastated the Island of Crete about 1645–1628 BC. After rebuilding, the Minoans rose to greater heights along with a sister civilisation on the island of Thera. A further eruption about 1450 BC was credited with destruction of the Minoan civilisation, the parting of the Red Sea, and the sinking of the island of Thera, popularly believed to be the legendary continent of Atlantis.

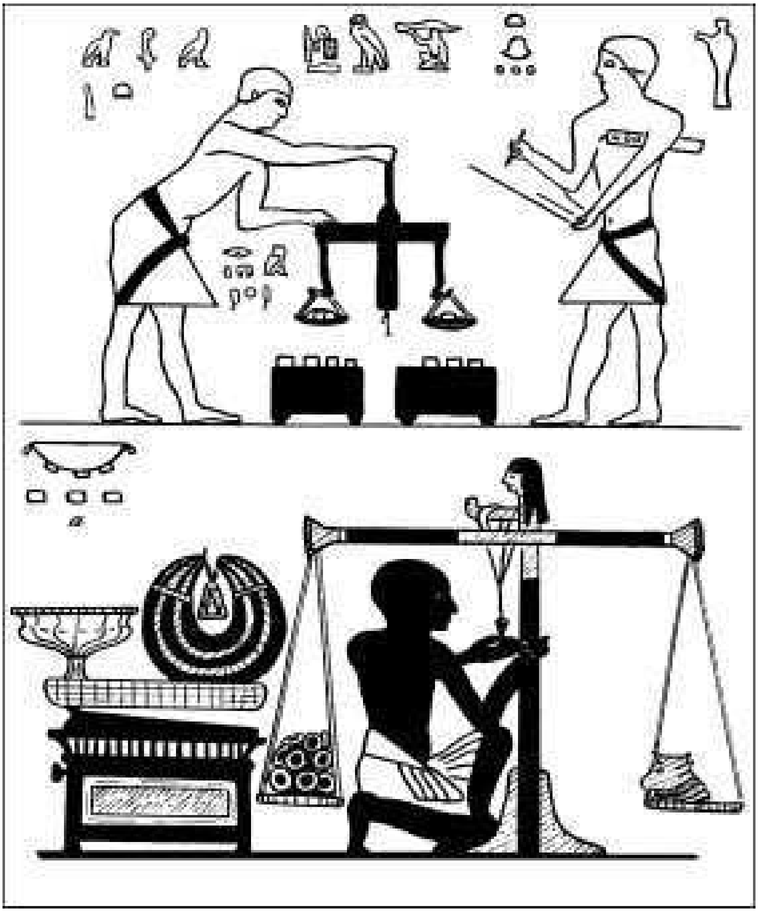
Egyptian era

Legend has it that it was from Ur of the Chaldes, some 350 km south-east of modern Baghdad, that Abraham led his people along the Euphrates valley to Haran, thence to Canaan and finally to Egypt (2100–1800 BC). The people of the Nile had become competent metallurgists by this time and Egypt was probably the first great power to contribute large amounts of gold to world markets. Gold mining was conducted in two large, rich goldfields. One of these, mainly alluvial, stretched along some 2,000 km of the beds and terraces of the Nile between Luxor and Kartoum. Recovery methods ranged from a simple hand picking of nuggets from river flats to a form of panning using woven reed baskets and wooden dishes. Roughly contiguous with the alluvial deposits and probably as provenances of the alluvial gold deposits, the second goldfield comprising auriferous quartz veins in ancient schists and crystalline rock formations was situated between the Nile and the Red Sea. These deposits were apparently exploited around 1250 BC.

Gold recovery techniques are displayed on Egyptian monuments dating back to about 900 BC and on wall paintings of the 20th Dynasty, 200 BC. Ancient Egyptian tomb drawings also depict the use of scales for weighing the gold. As shown in the upper part of Fig. 1.12, a scribe records the weight of gold bars; below, gold bullion rings are balanced against a symbolic bull's head. The hieroglyph for gold (inset *a*, centre left) depicts a gold collar similar to the richly adorned collar depicted on the table below.

Although less well documented than in Egypt, gold workings were common at about the same time in southern Russia, Africa and India. The rivers of western Turkey and streams in the mountains of Afghanistan and Turkey were said to be very rich in gold. By 2000 BC, trading in gold had spread throughout the known world. The Mycenaeans were established in Greece, via the Anatolian (Turkish) Highlands, and the Phoenicians, supposedly descended from Shem, son of Noah, were the dominant seafarers and traders. The ancient world, as envisaged by Herataeus a Greek traveller and historian of Miletus (a regional centre on the southern coast of what is now known as Turkey), is shown in Fig. 1.13. His conception of Europe and Asia was as semi-circles surrounded by oceans (Hellemans and Bunch, 1988).

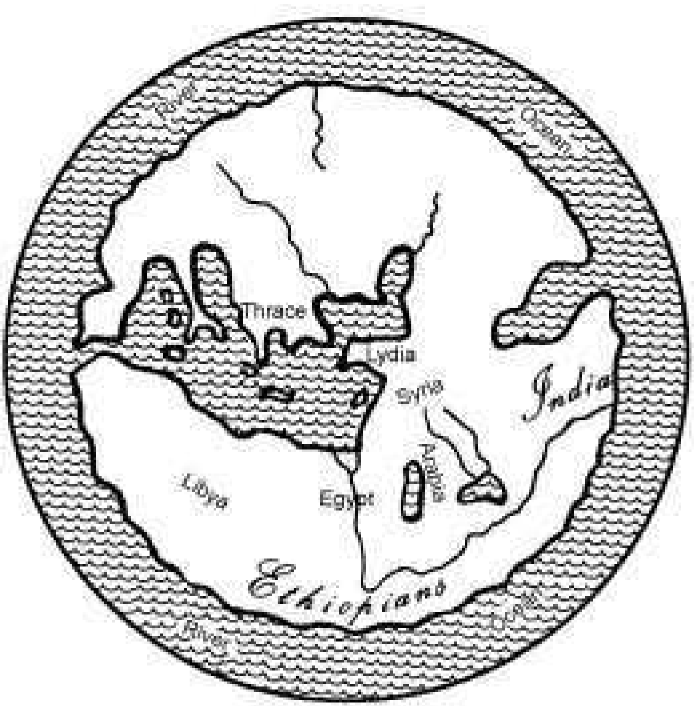
Strabo (63 BC to AD 24) travelled widely to collect first-hand information for his book *Geography* (published around 7 BC), which records the first known



1.12 Copy of ancient Egyptian tomb drawings depicting scales for weighing gold (after Nolan, 1980).

reference to the use of sluice boxes for gold recovery. He referred to an early method of sluicing attributed to miners in the Country of Saones in the Vooges Mountains: 'the winter torrents brought down gold which the barbarians collected in troughs pierced with holes and lined with fleeces'. The fleeces were hung on trees and, when dry, beaten to recover the gold. The method was well known in the Bosporous region in the first millenium BC when the process involved placing sheepskins, hair sides up, on the beds of streams. The sands were allowed to wash over the skins and gold, caught up in the wool, was recovered by panning after drying and beating or by burning the hides.

The well-known story of Jason and the Argonauts searching for the 'Golden Fleece' of Greek Mythology probably grew up around the exploits of adventurers who sailed around regions of the Black Sea and made forays on



1.13 The world as known by the ancients. Adapted from a map by Herataeus, 517 BC (after Nolan, 1980).

land to steal gold. The original legend held that Phrixus, son of Althamus and Nephele, escaped with his sister Helle from their wicked stepmother, riding into Colchis on the back of a golden ram, Chrysomallus. Helle fell into the sea, which was then called the Hellespont. Phrixus married the daughter of King Aeetes, and the Golden Fleece was placed, by the order of the king, in the custody of a dragon. According to the legend, it was from this dragon that the fleece was later taken and carried away by Jason and the Argonauts.

Most nationalities of the Egyptian era (e.g., Hebrews, Cretans and Phoenicians) engaged in trading rather than mining. Their chief source of gold appears at first to have been Egypt, although quartz gold-veins of Krissites and other areas in northern Greece were the backbone of the Greek economy. However, with the development of major trade routes, gold was brought in from Spain, Africa and south-east Asian countries as well as from various Eurasian mining centres. The weight systems used were those originally developed in Mesopotamia and gold eventually became a recognised standard of value against which to measure the worth of other commodities. The shekel, originally containing 11.3 g of gold was used as a standard unit of measure throughout the Middle

East around 1500 BC. The Greek Historian, Herodotus (484–424 BC), often called the 'father of history' tells of Croesus the last king of Lydia (Western Turkey) in Mesopotamia who fixed the value of gold and silver by issuing coins at a standard ratio of 10 parts of silver to one part of gold. Croesus was famous for his wealth ('rich as Croesus') and was also a devotee of the Oracle of Delphi, Greece on whom he bestowed great gifts of gold.

1.2.2 Scientific awakenings

Pursuit of a systematic and formalised approach to studies of the physical environment began in Greece in the 6th century BC when the first Ionian philosophers, Thales, Anaximander and Anaximenes sought general principles to explain natural observations. Thales (640–546 BC) a citizen of Miletus considered that questions relating to the nature of the world could not be explained by the whims of gods, questioning any concept that suggested that 'how things seemed were necessarily how they were'. He was not willing to accept an answer that invoked the gods but insisted that evolution was real and that its effects could be seen and touched in the world around us. For example, thunder and lightning were natural events and not simply due to an angry Zeus nor was 'broad bosomed Earth' created by Chaos. As a celebrated astronomer, Thales was apparently among the first to determine solstices and equinoxes, and his creation of a model of the Earth was based upon what he could actually perceive. He is also believed to have invented the sundial and to have produced the first geographical map but nevertheless he could not divorce himself from all ancient speculations and traditions. One difficulty was in explaining relationships affecting the movement of air and water. As observed by Adler (2002) in *Science Firsts* the Earth was to Thales a flat disc floating on water like a log. Earthquakes were waves formed by disturbance of the waters. A great river circled the Earth with the sun and other heavenly bodies being blown around the sky by winds created by the water's circulation.

Anaximander (610–546 BC), an illustrious pupil of Thales, shared his basic beliefs but soon devised a more sophisticated picture of the cosmos. By devising his own models and arriving at his own conclusions he postulated the existence of *apeiron* (the boundless), which had neither beginning nor end and whose basic principle could be compared with the 19th-century concept of 'ether'. To Anaximander, the boundless has always existed and is always in motion; 'the boundless spontaneously generated the rotating germ or seed of the world'. Once formed, qualities of hot and cold, wet and dry, light and darkness all separate out and begin to interact. He argued that while the Earth is finite in size it is limited in duration. He envisioned it at rest cushioned on air within an infinite symmetrical universe: 'It was not dominated by anything and thus was in equilibrium with no reason to move in any one direction than another' (Adler, 2002).

Anaximander also proposed an evolutionary theory 23 centuries before Darwin, arguing that the earliest forms of life arose from the interaction of primordial heat and water and that all land animals including humans evolved from fish-like ancestors. The first creatures lived in the sea, protected by shells and on the appearance of dry land gradually adapted to the new conditions. This was an idea that most people found disturbing; 150 years later Plato, in referring to the idea, said, ‘The fourth kind of animal, whose habitat is water, came from the most utterly mindless men’. Anaximander had, nevertheless understood that the same sequence of events that led to the formation of the Earth must also occur throughout the universe. Anaximenes (570–500 BC), who could have been a pupil of Anaximander, furthered his conception of the universe and what the world is made of. He believed that air is the basic principle of the universe and that the rainbow is a natural not divine phenomenon.

Pythagoras (570–490 BC) is credited with formulation of the ‘Pythagoras Theorem’ but is now believed to have copied the idea from Babylonian texts written a millenium earlier. Nevertheless, he or one of his followers discovered the relationship between the length of a plucked string and the musical note it produces. The substance of the string did not matter, only its length. Lengths in the ratio 2:1 produced an octave, 3:2 a fifth, 4:3 a fourth and so on. The undoubted numerical relationship suggested that the same pattern he had discovered in music would exist in the structure of the cosmos. Little else is known of Pythagoras except that he founded a politically influential religious brotherhood in Croton, South Italy, which became more controversial as its influence grew. Its tenets included immortality of the soul and transmigration. An ordered universe was created in which the Earth, Moon, and planets circled the central fire (Sun). This was rejected and eventually the Pythagorean Brotherhood fell apart and was disbanded at the end of his life.

Herodotus (484–425 BC) was amongst the first of the great philosophers to recognise the effects of climatic and tectonic change. He understood that land at the mouth of the Nile River had formed from sand and mud deposited by the river and proclaimed Egypt as ‘the gift of the Nile’. From observations of shells in Egyptian hills he concluded that at some time the sea must have covered the hills.

Socrates (469–399 BC) taught that true knowledge emerges through dialogue and systematic questioning, and an abandonment of uncritical claims to knowledge. Hippocrates of Cos (460–370 BC) to whom is attributed the ‘Hippocratic Oath’, and Alcmaeon, who discovered the optical nerve, and other physicians of the time, propounded ideas such as cleanliness between doctors and patients, moderation in eating and drinking and the requirements of a clean atmosphere.

Empedocles, in the 400s BC, thought that the Earth’s interior was composed of a very hot liquid and that all things come from earth, fire, air and water. Theophrastus, a pupil of Empedocles wrote a small paper *Concerning Stones*, which listed all the rocks and minerals known at the time. Aristotle (384–322 BC) whose works include 22 treatises dealing with logic, physics, astronomy, meteorology, biology, psychology, ethics, politics and literary criticism, believed that the Earth's structure was constantly changing. He noted that siltation in streams of the Black Sea region had necessitated a shift to smaller boats after only 60 years of sedimentation.

However, although early Christian theology absorbed many of the ideas of Socrates and Aristotle, the teachings of the church were also influenced strongly by Plato (428–347 BC). Plato was a pupil of Socrates but his philosophy was generally unfavourable toward empirical knowledge and rejected scientific rationalism in favour of arguments. Plato's 'ideal state of Utopia' (*The Republic and The Laws*) took a view of the world verging on mysticism and thereby discouraged secular knowledge, believing it to be a manifestation of heathenism.

The freedom of expression enjoyed by science had thus begun to be seriously challenged during the rise of the Christian Church. An organised priesthood developed a religious theory of creation, and differences between established religion and the irreligious attitude of the free-thinking philosophers became a dominant issue. Even the most renowned philosophers were persecuted from that time onwards. For example:

- Anaxagoras had to leave Athens in exile.
- Socrates was condemned to death by the Athenian authorities and died by drinking hemlock.
- Aristotle was forced into exile in Chalcis, where he died.
- St Augustine of Hippo (AD 354–430) maintained the Doctrine of Original Sin and the necessity of Divine Grace. He taught that all natural processes have a spiritual purpose.
- Bishop Theophilus (AD 390) destroyed the Library of the Temple of Serapis in Alexandria.
- St Cyril, Bishop of Alexandria, instigated the murder of the mathematician Hypatia (AD 415).

Following the conquering of the world and before his death in 323 BC, Alexander the Great spread Greek philosophy to many other countries including Egypt, where the new Hellenistic culture flourished. Under this culture:

- Mathematicians advanced the study of curved figures and algebra. Archimedes (287–212 BC) with his Archimedean spiral solved two of the classic problems, trisection of the angle and squaring of the circle.
- Hellenistic astronomers produced accurate observational results of the universe. Hipparchus (190–120 BC) invented trigonometry, calculated the length of the solar year and the lunar month, and discovered the precession of the equinoxes. He also made a catalogue of 800 stars and advanced the method of determining the location of places on the Earth's surface by lines of latitude and longitude.

From 146 BC onwards Greece was dominated by Rome and, as elsewhere during those times, scientific advancement was achieved only at the expense of opposing religious dogmas that had distorted and inhibited philosophic and scientific development since the most ancient times.

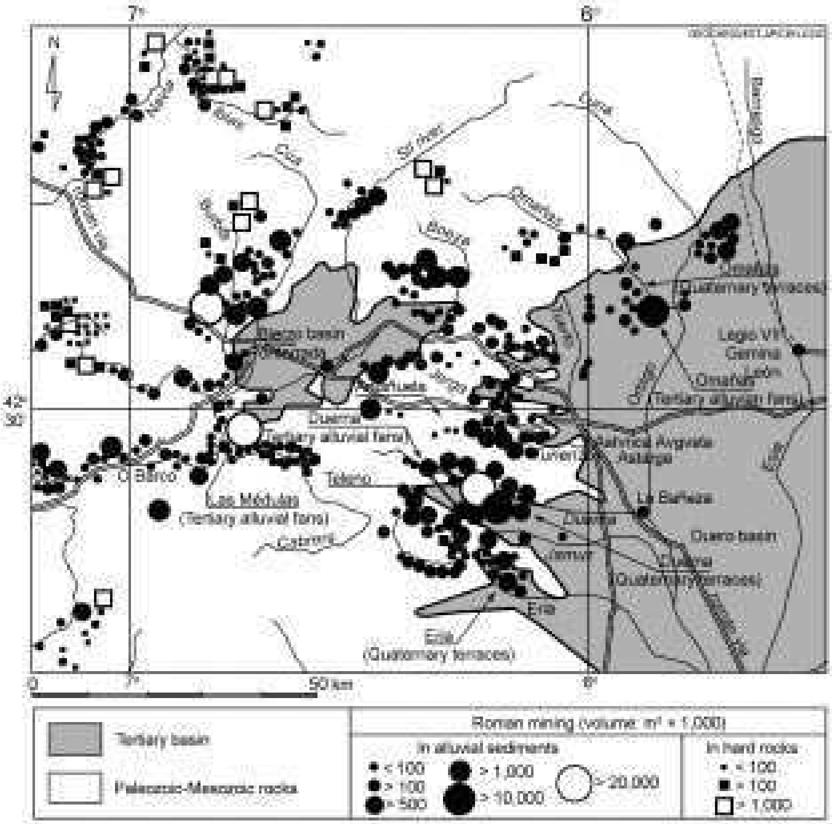
1.2.3 The Roman world

Rome became the dominant nation of the ancient Western world around 200 BC as a result of conquests in Macedonia, Thrace, Spain, France and Egypt, and vast resources of gold and silver came under Roman control. The upper classes became accustomed to adorning themselves with golden jewellery produced from Au-Ag alloys. Some of this work was done by Greek craftsmen in Alexandria and Antioch and in Greek settlements dispersed throughout the Roman Empire.

Writings contributed by the Romans were more factual than those of the Greeks but they also included much superstition and supposition. According to Pliny (AD 23–79), 'if the proportion of silver exceeds one fifth, the metal offers no resistance on the anvil and has the quality of shining more brightly than silver in lamplight. It also has the property of detecting poisons; for semi-circles resembling rainbows run over the surface in poisoned goblets and emit a crackling noise like fire'. Referring to writings by Homer, Pliny spoke of a goblet made of electrum that lay in the Temple of Athena, at Lindus of the island of Rhodes. Helen of Troy bequeathed the goblet to the Temple. History relates that it had the same measurement as her breast.

Strabo (63 BC–AD 24), who travelled widely to collect information for his treatise *Geography*, recognised that the rising and sinking of lands results partly from volcanic eruptions and earthquakes. In the AD 60s, the Philosopher Lucius Seneca wrote *Quaestiones Naturales*, which provided information on earthquakes, volcanoes, and surface and underground waters. The 37-volume *Historia Naturalis* by Pliny the Elder (AD 23–79) included all Roman knowledge about rocks, minerals and fossils.

The first of the Roman Emperors, Augustus (63 BC–AD 14) established the gold standard known as the 'Aureus' (Perez-Garcia and Sanchez-Palencia, 2000). Important amongst his reforms, the army was made a profession and mining engineers were attached to Roman armies to search for new deposits and upgrade existing operations. The engineers became expert surveyors and water races were constructed over almost inaccessible terrain to service the mines. Pliny (AD 23–79) describes examples of ditches constructed along mountain heights, frequently along a distance of 'a 100 miles or more, for the purpose of washing away the debris; and of gorges and crevasses bridged by aqueducts carried by masonry'. He noted that 'when the barriers are struck away, the torrents burst out with such violence as to sweep forward the broken rock. The trenches are floored with gorse, which is rough and holds back the gold'. A similar process, known as 'booming' was used in early Californian gold fields.



1.14 Roman mining works in the NW of the Duero Basin and Bierzo Basin (modified from Perez-Garcia and Sanchez-Palencia, 2000).

Following the Cantabrian War in Spain (19 BC) and during the 1st and 2nd centuries AD, more than 500 goldmines were operated by the Romans in the north-west portion of the Iberian Peninsular (Fig. 1.14). An estimate of the total volume of hard rock and alluvial gold material moved by mining operations in the NW of Spain during Roman times is given in Table 1.11. This estimate of a total of 600,000 m^3 is divided equally between quartz-vein and placer gold deposits.

The alluvial deposits occurred in Miocene sediments (mainly alluvial fans) derived from the Cantabrian Mountain and Leon Mountain uplifts, and Quaternary deposits (principally regolith, fluvial terraces, moraine and fluvio-glacial. The proximal facies targeted by the Romans is believed to have averaged only 100–150 mg Au/m^3. However, the gold was predominantly fine-grained); recoveries must have been very low (probably less than 40% overall) due to the primitive nature of the methods used.

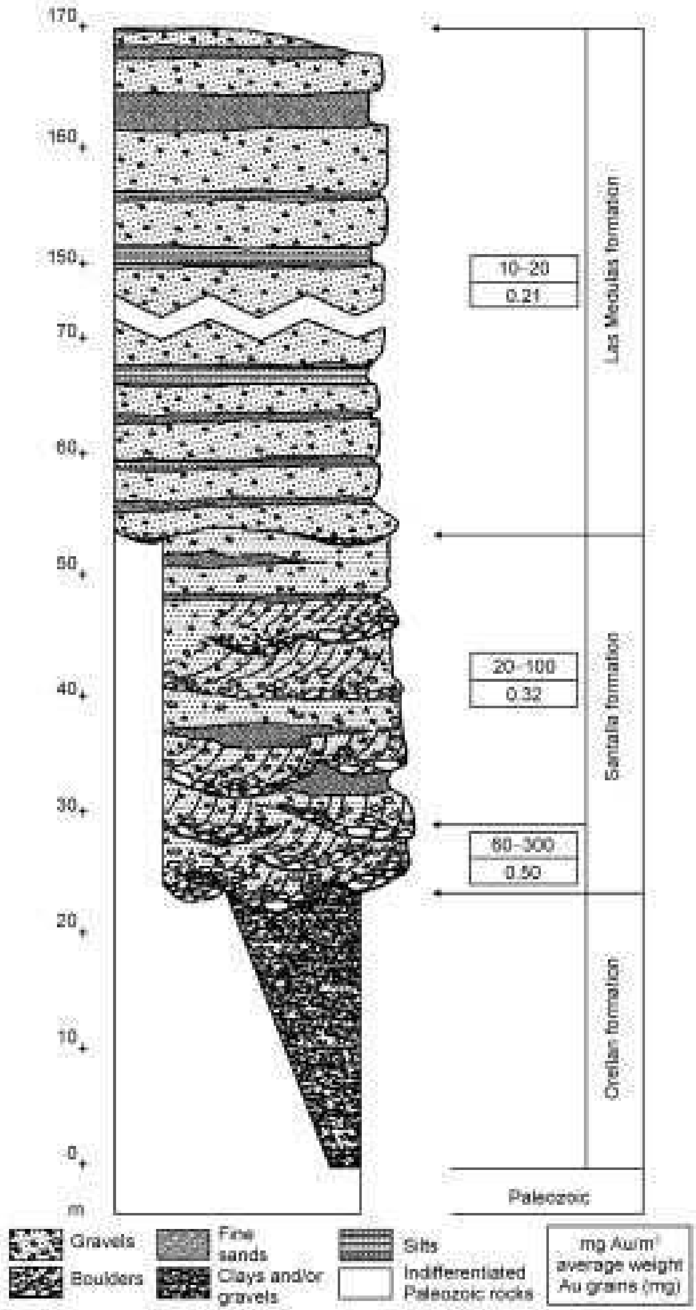
Table 1.11 Gold content and gold grain size in the different types of auriferous deposits (from Sanchez-Palencia, 1992)

Gold deposits	Size deposits	Gold content	Average grain size (μ)
Bed of heaving minerals in the Cambrian-Ordovician quartzites	Not determined	Not determined	230
Quartz veins and associated wall-rocks	Not well known <1 Mt (individual vein)	Not well known 0.1–10 g/t[a]	150
Proximal facies in the Miocene alluvial fan deposits	<60 Mm^3[b]	150–300 mg/m^3[c] (detrital Au > 50 μ)	300
Pliocene fluvial deposits ('Rana')	<7 Mm^3	80–100 mg/m^3[c]	200
Quaternary terraces	<70 Mm^3	70–250 mg/m^3[c]	220

[a] From small and isolated samples.
[b] Auriferous gravel, not including overburden.
[c] From large and systematic sampling.

Ditches constructed by Roman engineers in this part of Spain were typically 0.3–1.5 m wide, at average gradients of 0.5% and ranging in length from hundreds to thousands of metres (Fernandez-Posse and Sanchez-Palencia, 1988). Ditch water was obtained principally from seasonal snowmelt but there is no evidence of storage dam construction in the catchment areas. Instead, the water was accumulated periodically in small reservoirs at the tops of alluvial sections chosen for exploitation (Sanchez-Palencia, 1992). Shallow alluvium was removed by ground sluicing using trenches having average gradients of about 5%. The authors describe the dispersal of deeper, 30–100 m alluvium by a booming method known as 'Ruina Montium' (mountain collapse). The method, referred to as 'Bell-Pit Mining' by Pliny (AD 23–79), is reconstructed by Perez-Garcia *et al.* (2000) as it applies to collapsing and flushing piles of talus.

A water reservoir (piscina or stagnum) built on top of the talus is connected by a ditch (emissarium) to a vertical pit that extends to the base of the talus. A network of galleries, which radiate outwards from the bottom of the pit stop short of the outside surface. When the contents of the reservoir are discharged into the pit, water spreads into the galleries, which become saturated leading to collapse and flushing of the talus pile. The Las Medulas Mine, which was included in the World Heritage List in 1997, is the most typical of these deposits. The nature of the stratigraphic sequence (Fig. 1.15) is typical of the pulsatory nature of tectonic uplift during long histories of weathering under widely varying climatic conditions. Although some mercury appears with almost all of the gold grains recovered from current sampling of mine residues, it is probable that amalgamation would have been used only to upgrade final con-



1.15 Stratigraphic sequences, Au grade and average weight of gold grains in the Las Medulas Roman Gold Mine.

centrates. Amalgamation was certainly well known to the Romans. Theophrastus recorded the amalgamation of gold and mercury about 300 BC and Vitruvius described the use of mercury for recovering gold from gold-threaded cloth in 27 BC.

Alchemists around AD 300 defined chemistry (chemia, from the Greek Χηεια) as the art of transmuting base metals such as lead and copper into gold and silver. This process of transmutation involved using a hypothetical substance called the 'philosopher's stone', which was also supposed to be a universal solvent and a source of everlasting life. The Roman Empire rose, financed by gold, and fell amid political turmoil when the excesses of the ruling classes depleted defence coffers and destroyed its ability to survive. This resulted in a stifling of enterprise and learning in the Western World.

1.2.4 Dark Ages in Europe and the Eastern World

The fall of the Roman Empire in 476 saw the virtual end of scientific activity in Europe. The Byzantine Emperor Justinian closed great centres of learning such as the Academy and Lyceum in Athens in 529 and destroyed the Museum of Alexandria in 641. The 'Dark Ages', which followed and continued until the 12th century is believed by some historians to have been pre-empted by the catastrophic events which followed major volcanic eruptions around what is now known as 'The Pacific Ring of Fire'. Eruptions of Mt Vesuvius saw the destruction of Pompeii and the burial of Herculaneum, at the base of Vesuvius in 70 AD. As recorded by Pliny's nephew, Pliny the Younger, Pliny was overcome by the fumes and died at the town of Stabiae when he tried to rescue a friend.

Historians refer to similar geological and chronological records including the eruption of Mt Ilopango in Central America (260 AD), which is believed to have driven the ancient Maya Civilisation hundreds of kilometres away from their settlements, disrupting their characteristic culture for 200 years. The Mayan civilisation developed from its heartland on the Vera Cruz coastline of Mexico around 2000 BC with notable religious centres at San Lorenzo and La Ventes. Recent archaeological investigations suggest that the complete destruction of the Mayan Empire followed an ensuing catastrophic period of drought.

South East Asia and China

Written evidence of ancient scientific development in the Far East is sparse in the Western World. Recent discoveries in parts of South East Asia and China suggest that around 2400 BC the Chinese invented a method for taking observations of the sky based upon the Earth's equator and poles, a system not adopted and established in the Western World until the 16th century AD. Transition from the Stone Age to the Age of Metals also took place independently of the West at least 3,000 years BC. These discoveries show that certain aspects of physical science were more advanced at this time in the Far East than in the West. Amongst the discoveries unearthed at Ban Chiang close to the Mekong River were a profusion of metal objects including a bimetallic spearhead with a forged iron blade cast on a bronze socket. A socketed digging tool cast from pure copper was found nearby at Non NokTha. Thorne and Raymond (1989) date these tools at around 2,000–2,700 years BC. Discoveries of golden artefacts of those times are rare, but gold panning in the Mekong River has apparently been carried out for at least 5,000 years.

Between 1350 BC and the beginning of Christianity the Babylonians introduced fire assaying to determine the purity of gold. In China, squares of gold were legalised as a form of currency, and the first pure gold coins were minted in Lydia, a kingdom of Asia Minor. Julius Caesar ransacked enough gold from Gaul (France) to repay Rome's debts.

Around AD 330 the capital of the Roman Empire was transferred to Byzantium, beside the Straits of the Bosporus, by Constantine the Great who renamed it Constantinople. Centres of commerce then shifted to the East and the new Byzantium Nation (AD 395–1453) became enormously rich. When the Prophet Mohammed (AD 570–632) founded Islam, the destruction of Byzantium was an important goal for an empire based on religion and sustained by gold. Constantinople was twice besieged by the Arabs (AD 673–77, 718) but the Byzantines retained Anatolia. Syria, Egypt and North Africa were lost to the Empire during the 7th–8th centuries but under the Macedonian dynasty the Byzantine Empire reached the height of its prosperity around 1056. The Byzantine Empire was finally doomed when the Fourth Crusade sacked Constantinople in 1204 and set Baldwin of Flanders on the throne.

The few records that still exist of scientific discoveries by Arabian scientists during the period 4th to 12th century include one important thesis referring to weathering and erosion, in a work called *Discourses of the Brothers of Purity*, published by a group of Arab scholars somewhere between AD 941 and 949. Later, another Arab scholar Ibn Sina (AD 980–1037) expressed views on the effects of slow erosion over long periods of time. He classified mountains as 'those produced by uplifting of the ground, such as takes place in earthquakes; and those which resulted in hollowing out valleys in soft rocks'. He regarded fossils as unsuccessful attempts by nature to form plants and animals.

The development of agriculture in China, as in Mesopotamia, triggered the growth of engineering sciences, which revealed great inventiveness and led to revolutionary new technological achievements. These included 'The Grand Canal', stretching from Beijing in the north to Hangzhow in the south, which was started in AD 70 and completed in AD 1527 and is still working perfectly today. The Chinese invention of porcelain in AD 200 demonstrated a very efficient control of kiln temperatures and metallurgical abilities that was not matched in the West until the 15th century AD. China's geographical isolation and the wall of mystery it has always built up around itself have tended to hide its metallurgical achievements from others. Museum pieces in the two museums at Tonglushan in Hubei Province are evidence of a scale and development of copper mining that far surpasses technologies developed anywhere else at that time in the ancient world.

In *Geology and other Earth Sciences*, Shen Kua (AD 1086) outlined principles of erosion, uplift and sedimentation in his *Dream Pool Essays* that form the basis of modern Earth Science. In these essays, Shen Kua refers to a magnetic compass for navigation. Indeed, the world's first magnetic compass was made in China 2,200 years ago. A model of the original device now in the Beijing Museum, comprises a spoon carved from magnetic rock placed upon a bronze plate. If moved, the spoon always returns to a south pointing position. The first known reference to such navigational aids in Europe is made in *De Naturus Rerum*, published by Alexander Necham near the end of the 12th century.

Recent major contributions to an understanding of China's scientific development are found in two publications: *Science and Civilisation in China* written largely by Sir Joseph Needham, Cambridge University, and *Metallurgical Remains of Ancient China* by Noel Barnard, Australian National University, and Professor Sato Tamotsu of Tokyo.

In 1284, Venice introduced the gold 'Ducat', which remained as the most popular coin in the world for the next five centuries or so. The first major gold coin the 'Florin' was issued in Great Britain followed by the 'Noble', the 'Angel' and the 'Guinea'. The Black Death killed between one-third and one-half of the Western World population and plunged Europe into another deep depression in the 14th century. Lasting for about 100 years the disease finally died out bringing to a close about 1,000 years of virtual unenlightenment in the Western World.

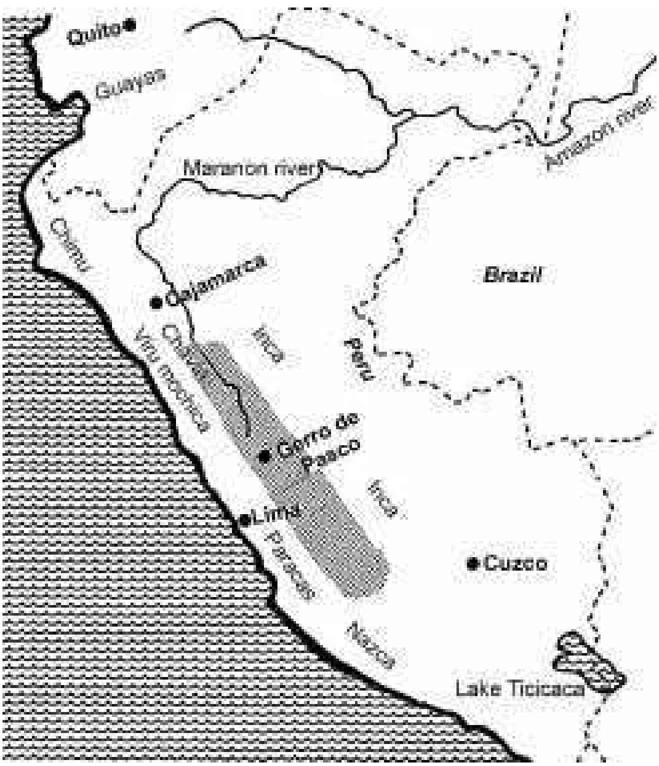
1.2.5 Renaissance and the New World

Renaissance (14th–17th centuries) marked the transition from medieval to modern times. It was a period of renewed interest in many fields of learning including geology. It began as a reaction to the massive depletion of Europe's population by the Black Death and flourished in Western Europe until about the 17th century. The survivors looked for new ideas and ways of compensating for lack of manpower. They also sought to enlarge the boundaries of learning, and global exploration was encouraged. Fresh ideas came from the release of scientific literature when Constantinople was threatened and finally fell to the Turks in 1453. Columbus reached America in 1492; Vasco da Gama sailed to India around the Cape of Good Hope in 1498 and Luther started the Protestant Reformation in 1517 by nailing his 95 Theses to the church door in Wittenberg.

The explorer Christopher Columbus (1451–1506) made four voyages to the New World, noting amongst other important observations, changes in the deviation of the magnetic needle from true north as he sailed across the Atlantic. Explorers identified further geographical discoveries during voyages made from the north to the south of the Americas but it was Inca and Aztec gold that initially attracted the first adventurers and sparked the Spanish-American invasion of the early 16th century.

Symbolically, in Inca and Aztec civilisations, the Sun God was believed to be the source of all light and life and was invariably depicted in gold. The first ruler of the Incas, about AD 1200, was thought to be the incarnation of the Sun. The Temple of the Sun in Cusco, Peru, was said to be literally covered by plates of gold; at harvest time it was adorned with artificial cornfields made entirely of gold. Important early civilisations of Peru including the main mining regions are shown in Fig. 1.16.

Most of the gold was alluvial and recovery methods were similar to those used in the Old World. Similar cultures evolved also. The Paracus people, who dwelt in Peru between 2700 and 2200 BC were primarily hunters and farmers. However, they developed other skills over time and amongst metallurgical



1.16 Important early civilisations of Peru. Shaded area is the main mining region. Most of the gold has come from the vicinity of Corro de Pasco (after Nolan, 1980).

discoveries, a gold-based solder (tumbago), was developed as an alloy of gold, silver and copper for connecting gold studs. Although recent archaeological work suggests a provenance earlier than Christ, gold metallurgy apparently took another 500 years in spreading to Panama and a further 300 years prior to its appearance in Mexico. Only when the Incas and Aztecs appeared on the scene in the 12th century AD were large-scale gold workings developed for the first time.

The invention of printing (mid-15th century) helped in the spreading of major new scientific discoveries. In his book *On the Revolutions of the Heavenly Spheres* (1543) the Polish astronomer Nicolaus Copernicus explained that the Earth rotates on its own axis, the moon revolves around the Earth and all planets around the sun. This was contrary to religious dogma and in 1552 Archbishop James Ussher calculated the origin of the Earth at 4004 BC based upon the *Book of Numbers* in *The Old Testament*. This calculation received academic approval four years later. John Lightfoot, Vice Chancellor of Cambridge University, pronounced that 'Heaven and Earth, centre and circumference, were made in the same instant of time and clouds of water and man were created by The Trinity on the 26th day of October, 4004 BC, at 9 o'clock in the morning'. Thus, when Bruno declared his belief that the Earth revolves around the Sun he was accused of heresy and other theories contrary to religious teachings, and burned at the stake in Rome on 17 February 1600.

Galileo invented an astronomical telescope and discovered that the same gravitational acceleration applies to both heavy and light objects. Leonardo da Vinci observed the rise of liquids in small diameter tubes, thus identifying capillary action. Of the engineers, Agricola (1556) (Latin pen name for the writings of Georg Bauer) systemised basic aspects of gold metallurgy and made outstanding contributions to science and technology with his descriptions of mining operations during the period 1546 to 1556. He described the diversity of factors of Earth Science, as studied by the miner, in the following terms:

> First there is philosophy, that he may discern the origin, cause and nature of subterranean things –, Secondly there is medicine, that he may be able to look after his diggers and other workmen – Thirdly follows astronomy, that he may know the divisions of the heavens and from them judge the directions of the veins – Fourthly there is the art of surveying – Fifthly, his knowledge of Arithmetic Science should be such that he may calculate the cost to be incurred in the machinery and the working of the mine – Sixth, his learning should comprise Architecture that he, himself, may construct the various machines – or so that he may be able to explain the method and construction to others – Next, he must have a knowledge of Drawing that he can draw plans of his machinery – Lastly there is the Law, especially that dealing with the metals, that he may claim his own rights — that he may not take another mans property and that he may fulfil his obligations to others according to the law.

Philosophia Naturalis Principia Mathematica, published by Isaac Newton in 1687 was the most important scientific work of the 17th century. Amongst others of his discoveries which included the binomial theorem, and differential and integral calculus, his three standard laws of motion and the law of universal gravitation form the basis of Newtonian mechanics, most of which is still in use today. His concept of 'uniformitarianism' suggested that the same types of river currents that produce characteristic sediment structures today must have operated similarly in the past, if the same structural types are visible in ancient sediments. Newton's 'First Law of Motion' is fundamental to all aspects of sedimentation in both field and processing plant operations.

Robert Hooke (1635–1703) originated Hooke's Law, which states that 'The tension in a lightly stretched spring is proportional to its extension from its natural length'. His inventions included a telegraph system, spirit level, universal joint and marine barometer. Hooke's discoveries led to the manufacture of clocks powered with springs instead of weights and watches that became precision instruments. Some of his hypotheses relating to planetary motion may have foreshadowed Newton's proposition that an inverse-square law of gravitational attraction to the sun will produce elliptical orbits for planets. In 1682 Edmond Halley observed the 'Great Comet' which was later named Halley's Comet in his honour.

1.2.6 Industrial Revolution

The Industrial Revolution (18th and 19th centuries) marked a period of enlightenment in which technology was to become of equal importance to pure science and philosophy. Philosophies of the ancients were revived in the renewed belief that simple classical laws could be used to explain problems in physics, chemistry and biology. Immanuel Kant, a leading philosopher of the 18th century reconciled empiricism and rationalism by asserting that knowledge is gained from both reasoning and experience. However, Goethe and von Schiller and some other philosophers were troubled by such extreme materialism and expressed the holistic view of nature as a single organism imbued with spirit. Diverse philosophical trends such as these fostered political revolution and USA Presidents Benjamin Franklin and Thomas Jefferson became part-time scientists.

The medieval science alchemy (Arabic al-Kimya) dominated most chemical thought in Europe until the Industrial Revolution and the birth of a new science termed 'Geology' by Horace de Saussure. In 1807 the Geological Society of London became the first scientific society devoted to the science of geology. During the same year a Swiss chemist, Berzelius, with his students, developed the underlying principles of mineralogy upon which the present classification of minerals depends.

In mathematics, Jakob Bernoulli (1654–1705) discovered a series of complex numbers used in higher mathematics (Bernoulli numbers). Johanne, brother of Jakob, developed exponential calculus and his son Daniel proposed the

Bernoulli Principle, which is central to the analysis of fluid flow. This principle allows the total energy at any point in a streamline to be written as a constant in terms of fluid density, fluid velocity, total pressure at the point, and the elevation of that point relative to an arbitrary datum (see Chapter 4). Hutton's *Theory of the Earth* (1788) presented firm geological evidence of a sedimentary cycle of erosion. His grasp of the immensity of geological time provided the key to understanding the formation and denudation of landscapes and the uplift of sediment deposited as mud and sand on the seafloor during the denudation of previous landscapes. His thesis, *The Present is the Key to the Past* proposed immeasurably long periods of time for great thicknesses of sediment to accumulate and suggested that the history of the Earth could be explained by what is happening now.

Although geology had by this time entered a new period of growth, the significance of Hutton's observations was largely overshadowed by a continued acceptance of Neptunian and Plutonistic concepts. Werner, a German scientist and a leading theorist of Neptunism, still argued that volcanoes were local phenomena caused by burning coal seams and that all rocks that would ever exist had already been formed and – that continents were slowly being washed into the sea and would eventually disappear. Plutonists, as represented by Hutton, thought that the Earth's heat would cause sediments to rise from the sea. In 1788, Hutton identified some rock formations, which appeared to consist of sedimentary rock that had been metamorphosed by heat.

But the seeds of understanding had been sown. In 1802, Playfair elaborated and expanded on Hutton's idea of small changes taking place over immense periods of geological time. In a publication *Huttonian Theory of the Earth* he asserted that 'no valley is independent of the rivers flowing in it, but instead, develops various characteristics governed by its geology and the environment within which it is placed'.

By this time, scientists such as Nicholas Desmarest, a French geologist, had shown that the basalt rocks of the Auvergne region of France were created from lava. At Moscow University, Professor Shchurovsky (1803–1884) deduced that 'plutonic rocks lifting the Urals enriched them in gold and other valuable minerals'. He also recognised the length and complexity of magmatic processes associated with mountain building and ore formation and the fracturing that subsequently led to the formation of vast detrital deposits in particular beds of stratigraphic successions. Extensive gold-bearing gravels discovered in 1829 in the Lena Basin, Eastern Siberia are still being worked today. Apart from the Witwatersrand lithified conglomerate deposits of South Africa, they are probably the largest gold deposits ever found in an alluvial setting.

William Smith, an English civil engineer, may have been the first person to use fossils to map rock strata. During the late 1700s, while surveying and building canals in southern England, Smith had examined layers of rock containing fossils. He observed that the same fossil types were specific to the same strata both locally and further afield. In 1815 he published the first geological maps showing the strata of England.

In 1830 Charles Lyell completed the first volume of a three-volume textbook called *Principles of Geology*. He began a study that showed the Earth to be at least several hundred million years old and two years later he identified the Recent, Pliocene, Miocene and Eocene periods of Earth history. Charles Darwin (1844) confirmed Lyell's work from geological observations on volcanic islands visited during the voyage of *HMS Beagle*, as being equally applicable to places that Lyell had never visited. He is remembered particularly for his part in the development and acceptance of a standard geological time scale based upon the presence of characteristic fossils in the rock.

In 1871 Dmitri Mendeleev asserted that 'the properties of elements are in periodic dependence upon their atomic weight'. The gaps in his periodic table represent undiscovered elements, which were filled by discoveries of new elements in 1875, 1879, and 1885.

1.2.7 Great gold rushes of the 19th century

The plundering of gold and destruction of the Inca and Aztec civilisations by Spanish and European financial and religious adventurers promoted a worldwide search for fortunes that could be gained by digging for gold in new unexplored areas. Locations of the great gold rushes of the 19th century are shown in Fig.

1.17 Locations of the great gold rushes:
1, California; 2, Klondyke; 3, New South Wales 4, Victoria; 5, Otago; 6, West Coast; 7, Queensland; 8, Western Australia; 9, Transvaal. Also shown are some of the main sources of gold found during the 20th century (after Nolan, 1980).

1.17 together with some minor gold areas. Gold rushes of the Americas and Australasia (Australia and New Zealand) followed. In December 1843 President Polk of the USA referred to the abundance of gold in California as 'of such extraordinary character as to scarcely command belief'. By 1848, mass migrations of the 'forty-niners', as they were called, took place from all over America and overseas. Some 50,000 people from all walks of life crossed the continent riding in wagon trains, on horses, mules and donkeys; fighting Indians along the way. Others braved the fever-infested jungles of Panama to make a faster crossing. Fifteen thousand or so accepted the rigours of passage around the 'Horn' and arrived in the goldfields some seven to eight months later. The 'Gold Fleet' from Australia carried more than 7,000 hopefuls. More than 18 million oz. of gold was won in the first four years, but of the 90,000 people of all nationalities, who reached the field, 18,000 died in the first six months mainly from privation. Figure 1.18(a) depicts stampeders arriving at the Gregory diggings Colorado in May 1859. Figure 1.18(b) shows the level of accommodation provided on the floor of a Colorado billiards saloon for newly arrived miners.

In Australia, Edward Hargraves, returning from California found gold in New South Wales in 1850 and the first gold rush took place at Ophir in 1851. The Ballarat alluvial goldfields were discovered a few months later followed by fresh discoveries over the whole of the Victorian gold belt. Victorian goldfields reached bonanza proportions, approaching those of California in terms of manpower and richness. In 1851, the population of Victoria was only 37,343 males above the age of twelve years. By 1854, the number had quadrupled to 144,803 of whom 65,763 were engaged in goldmining. By 1858, this number had risen to 223,604 even though 10 000 miners had left to join a new gold rush taking place in New Zealand. Records of production were not monitored closely but between 650 and 800 tonnes of gold are believed to have been produced from alluvial workings during the first ten years, including the worlds largest collection of nuggets.

Although gold discoveries in New Zealand were smaller than in Australia important finds were made in Otago in the South Island (McLintoch, 1966). Production between 1857 and 1867 amounted to something in excess of 100 tonnes Au. The first bucket dredge to be built in the world was commissioned in Otago in 1886.

In North America, Henry (George) Holt, who obtained the gold by trading with the Indians, brought out the first authentic samples of gold from the Yukon. The American government then provided troops to protect the trade routes in the area. Prospecting began shortly thereafter, at first in a desultory manner, then on a major scale in 1881 when gold quartz reefs were found at Juneau in the Yukon 'Panhandle'. The first large alluvial diggings were in the Stewart River, another tributary of the Yukon, some 96 km upstream of the Klondike confluence; the second was in the 'Forty-nine' about the same distance downstream. The last of

(a)

(b)

1.18 The gold rush days, Library of Congress, Washington DC: (a) Stampeders arriving at Gregory diggings, Colorado, in May 1856. From a wood engraving in *Beyond the Mississippi, 1867*. (b) Goldfield accommodation for newly arrived miners for 'two bits' per night in sawdust on the floor of a Colorado billiards saloon (*Leslie's Magazine*, 1879, in Nolan, 1980).

the ‘great rushes’ took place in the 1890s in the Klondike River, a tributary of the Yukon River, Alaska. Klondike placers that became particularly famous for their richness were ‘Rabbit Creek’ (later known as Bonanza Creek) and ‘Eldorado’.

The Klondike gold rush was, arguably, the most disastrous in terms of lives lost on trails leading into the various diggings and had a character of its own. Nolan (1980) epitomises two scenes, in particular, in the Klondike:

- newspapers crammed with advertisements for gold detecting machines; offers of trained rodents for burrowing for gold, and slot machines for use with gold nuggets
- after the rush ended – the frozen bodies of two men in a derelict hut on the Porcupine River. They had travelled in a great semicircle spanning 6,430 km but were still hundreds of kilometres from the goldfields. In a pot hanging over the long dead ashes of their fire, embedded in a cake of ice, was found a half-cooked pair of moccasins.

The gold of the gold rushes was mainly alluvial and, while many of the early techniques were little advanced from those of the Roman Empire, later developments took the form of improved sluicing arrangements, rocker cradles and horse-drawn puddling machines. In Australia, miners used devices called ‘flycatchers’. These devices were based upon the ‘Golden Fleece’ technique to catch finely divided and flaky gold floating across the riffles of sluices constructed across the downstream creek bed. The flycatchers consisted of weirs, on which were placed boards covered with blankets or sacking. The blankets were taken up at intervals and washed to recover the gold.

By the end of the 18th century the great alluvial goldfields of the Americas and Australasia were practically exhausted and the growing global demand for gold could be satisfied only by increased and sustained production. Problems associated with declining ore grades, more finely divided gold, increased overburden depths, rocketing labour costs and increasingly strict controls for environmental protection had to be faced up to for the first time. The Comstock Au, Ag lode was discovered in Nevada in 1859 and as a result, Nevada was proclaimed a State five years later.

Discovery of the Witwatersrand goldfield in South Africa in 1866 then opened up the largest concentration of gold ore deposits known to mankind. The main reef was apparently found by accident one Sunday morning by George Harrison and George Walker. Within a week it was recognised by some as being of unprecedented significance. But, as throughout Witwatersrand history, opinions on the origin of its gold and deposit potential differed widely. Only four weeks later, one expert reported that the auriferous reefs would not continue to depths below 10 to 15 feet from the surface. A year later another expert, Gardner Williams, stated categorically that the conglomerates were merely surface deposits along ancient shorelines and that no confidence could be placed upon a long life for the deposits. One hundred and forty years of intensive mining has resolved the question of size and value of gold deposits in the Witwatersrand, but the evidence is still inconclusive as to their source (see Chapters 3, 5 and 6).

1.2.8 Scientific advances of the 20th century

By the turn of the 20th century scientists such as the Curies, Einstein and Rutherford showed that radioactive elements within the Earth constitute an enormous source of energy that can be harnessed for mankind's good (or evil). Quantum mechanics, flowing from Max Planck's attempts to describe the behaviour of molecules, atoms and sub-atomic particles, provided a completely different kind of mathematics involving quantum theories. Discovery of the electron saw the birth of particle physics and, through the mathematics of quantum mechanics and experimental observation, several tenets of Newtonian physics were abandoned. Experiments showed that electrons could behave as waves when they are diffracted on passing through crystals. In 1900, Max Planck stated that substances emit light only at certain energies, and that electromagnetic radiation could be emitted only in specified amounts that he called quanta. Einstein in 1905 used the Planck theory on radiated energy theory to account for the discovery that light as electromagnetism travels in a vacuum at a fixed speed in every direction. His *General Theory of Relativity* introduced differential geometry into physics as a description of Nature. The science of particle physics emerged with the concept of particles as points moving through space along a line called the World Line. Two of the predictions of this theory have been the concept of an expanding Universe and black holes, both of which encapsulate issues in mathematical terms of reality and existence.

By 1911, Ernest Rutherford had established that the atom has a positive nucleus surrounded by orbiting electrons. In 1913, Niels Bohr calculated the quantum of the simplest case, hydrogen, in which a single electron orbits a proton. He showed that the quantum restricts the electron to particular orbits and that for each counting number (1, 2, 3, ...) there was one permissible orbit. Light is emitted when the electron changes from a higher quantum number to a lower one. There was not a continuous spectrum because the electron moves in jumps (quantum jumps) from one orbit to another. Scientists extended Bohr's model to explain both large and smaller lines in the spectrum and a second quantum number was introduced to explain the fine structure.

With some notable exceptions, the common view of most (although not all) geologists at the start of the 20th century, was that continents were locked in place and oceans were formed in areas where lateral compression causes sections of the Earth to subside. However, even as far back as 1596, Abraham Ortelius in his work *Thesaurus Geographicus* had suggested that the Americas were 'torn away from Europe and Africa – by earthquakes and tides'. Further, 'The vestiges of the rupture reveal themselves, if someone brings forward a map of the world and considers carefully the coasts of the three [continents]'. Francis Bacon (1561–1626) who noted the parallelism of shores facing one another across the Atlantic also remarked upon this. Similar ideas of the assembly and disassembly of continents, as originally argued by Suess and Snider-Pellegrini in the 19th century, were also rejected.

Orogenic processes were believed to be reactions of the Earth's crust to cooling and contraction. Geosynclinal concepts of crustal development had emerged as first attempts to explain the formation of fold mountains and the genesis of gold ore bodies. These theories left many important questions unresolved and it was not until 1912 that the movement of continents was seriously considered.

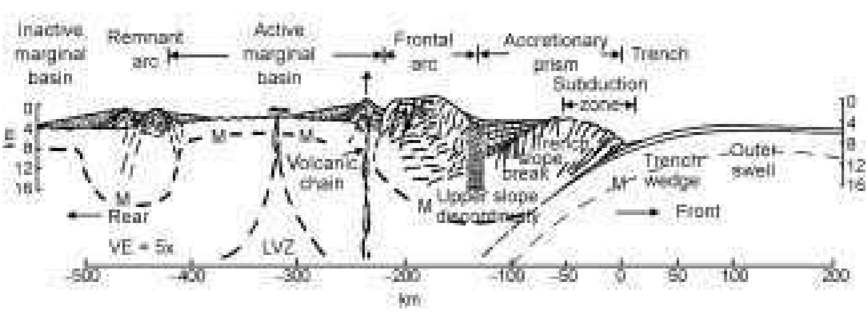
Plate tectonics

In 1915 a meteorologist, Alfred Wegener (1912), published the concept of continental drift and of a supercontinent comprising all of the world's continents merged into a single mass, which he called Pangaea. His proposition was that Pangaea had since split apart, the continents moving into their present locations. His theory was based upon evidence of similarities of rock structures and palaeoclimates when the continents are put together as he proposed. Lack of knowledge of the seafloor was a major constraint. It was only during the Second World War that underwater mapping and sounding techniques (developed for submarine warfare) began to uncover some of its secrets.

Recognition of the economic potential of the seabed led to expansion of wartime techniques for more general scientific purposes, including minerals and oil exploration. Observations and measurements were made of rifts in mid-ocean ridges, thicknesses and age of sediments on the ocean floor, palaeomagnetic reversals and the localisation of seismic activity and volcanism to specific crustal areas. Efforts to understand the processes that might have operated inside and at the surface of the Earth during its formation were intensified. In 1963, Hess wrote a paper called *Essay in Geopoetry* (summarised in Tarbuck, 1984) in which he proposed the theory that new crust is created at mid-ocean ridges and, due to seafloor spreading, is returned to the mantle at deep-sea trenches. From re-examination of echo-sounding data compiled during his experience as Commander of a ship during the Second World War, he later suggested that movement of convection currents in the Earth's mantle could be the cause of seafloor spreading. His evidence related largely to the age of the seafloor rocks, no part of the present seafloor being older than about 200,000 years. Seismic measurements of the speed of earthquakes, which showed that a low velocity zone (asthenosphere) exists at a depth of about 100–200 km, was recognised by Hess as being due to partial melting of the solid crust. It was deduced that the asthenosphere could constitute a lubricated layer over which the upper solid part of the lithosphere could travel.

Confirmation of seafloor spreading came in the 1960s when the US Geological Survey collected rocks of different ages from around the world and showed that rocks of the same age invariably had the same polarity. It was also noted that the striped reversal pattern was parallel to and symmetrical on both sides of mid-ocean ridges. As late as 1963, Emery and Schlee (1963) were to complain of the general lack of understanding of the essential interrelationships of marine and continental processes. They noted 'the tendancy for the work of the land geologist to stop at the waters edge and for that of the marine geologist to commence at the same line'. Other geologists were similarly concerned and in the late 1960s a synthesis of continental drift and seafloor spreading (plate tectonics) emerged as a single unifying mechanism for examining the major geological processes that are taking place on the Earth's crust. As briefly discussed in the following chapters, this theory relates virtually to every aspect of the geology of the Earth's crust, its rocks and structures and the tectonic conditions at the time of their formation.

Plate tectonic theory explains the great variety of geological activity that results from the interaction of lithospheric plates with one another at ridges, faults, trenches and subduction zones. It is illustrated in Fig. 1.19, which depicts essential elements of an active back-arc–interarc–marginal basin. The magmatic processes concentrate massive sulfide deposits, e.g. copper, lead, zinc, iron and gold in ocean crust. Hydrothermal fluid systems responsible for element concentration during ore formation are also sources of chemical energy utilised by heat-loving microbes to manage their food supply at the base of vent ecosystems hosted in the same systems. Certain microbes may relate to the base of the evolutionary tree of life. Scientific explanations are given for the layered structure of the Earth (core, mantle and lithosphere) and for structural movements such as mountain building and rock deformation on the continents, the location of earthquakes and volcanoes, and for heat flow, continental drift and magnetic reversals associated with seafloor spreading. Regionally extensive crustal structures occur within plate systems that may be thousands of kilometres long and hundreds of kilometres wide in both continental and marine volcanic settings.



1.19 Essential elements of an active back-arc–interarc–marginal basin and its associated arc-subduction complex (after Karig, 1974).

For the first time it is known why oceans open and close, mountains form and volcanoes erupt, and why new seafloor is created at seafloor spreading centres while old seafloor disappears at subduction zones.

Exploring the universe

Information was still sparse about the formation and very early history of the Earth prior to the concept of plate tectonics, which fundamentally changed scientific views of how the world was formed and how it works. Concepts of universal processes involved with the origin and general structure of the universe now suggest that the Earth was born about 4.5 billion years ago as a solidified mass of dust and ashes left over from the creation of the Sun. It was thought to be relatively cool at first, perhaps about 2,000 ºF, the main ingredients being iron and silicates with minor amounts of other elements including the radioactive minerals uranium, thorium and potassium. One currently popular ‘giant impact’ hypothesis is consistent with the idea that toward the end of its construction, a huge impactor hit a glancing blow to the Earth, heating it and then spinning off into orbit to form the moon. Other scientists also agree with the impact theory of planet formation although there are different views on the extent of melting and lines of evidence used to deduce it (Taylor, 1972).

Gravitational energy, together with energy from meteor bombardment and radioactive decay would have provided heat for melting, which then acted to concentrate the densest materials near the centre and the less dense near to the surface. A layered structure comprising core, mantle and crust developed as the result of magmatic differentiation. Core formation is believed to have been contemporaneous with the accretion of planetesimals at –4.5 Ga, as the result of temperature increase and melting caused by accretional energy and heat generated by short-lived nuclides. It has been deduced that the innermost molten Fe-Ni core with a radius of about 1,225 km had an average density of about 5.5 g/cc, with temperatures lying between 2,200 and 2,750 ºC at pressures 3 to 4 million times that of the atmosphere. Such highly siderophile elements as Au and platinum group elements were effectively concentrated into the core.

In 1923 the American astronomer Edwin Powell Hubble succeeded in observing Cepheid variables in the spiral nebula of Andromeda and other spiral nebulae. He was able to determine the distance of these nebulae from the apparent magnitudes of the variables, so discovering that they were much further away from the Earth than even the Magellan clouds. In comparing the distances of a number of galaxies with their Doppler shifts (red shifts) he discovered that with increasing distance from the Earth the faster it moves away. In 1927 a Belgian priest, Georges Lemaire, had proposed that the universe started by the explosion of a ‘primeval atom’, i.e., the concentration of all the mass of the universe in an extremely small space. Hubble’s constant ‘the ratio of the distance between the local group of galaxies and a receding cluster of galaxies, to the rate which the distant cluster recedes' then became the basis of the 'Big Bang' theory. Photography of stellar spectra became routine and in the mid 1940s, a German physicist, C. von Weizsacker, from theory and laboratory evidence proposed that a cloud of gas and dust rotating about a central body would develop vortices and eddies.

Discoveries such as these in planetary science stimulated a resurgence of efforts seeking to learn more about the Earth and its neighbours in space. Many theories have since developed including the 'steady state' theory, which assumes that matter is being continually created; the fresh matter forming new galaxies to replace galaxies that have moved to infinite distances. Models suggested by astronomers represent the universe as expanding, contracting and oscillating (expanding then contracting) or static (neither expanding nor contracting). Discovery of the cosmic background radiation of a body by Penzias and Wilson of Bell Telephone, while experimenting with a very sensitive microwave antenna, has been the strongest confirmation of the Big Bang theory. A hiss, corresponding to radiation of a body at a temperature of −454 ºF, was discovered at a wavelength of 7.34 cm with an intensity that seemed independent of the orientation of the antenna. This is the temperature to which the Earth is expected to have cooled as a result of its expansion since the Big Bang. There seem to be two possibilities (Brecher, 2002):

- If no more matter exists than is detected to date, the universe would expand indefinitely and stars would eventually exhaust the energy that makes them shine.
- If the universe contains large amounts of dark matter, i.e., material not yet detected, the expansion would stop in perhaps 20–40 billion years. The galaxies would then come together again and matter would approach infinite density. This cycle of collapse and expansion could then be repeated indefinitely.

The solar system is thought to have developed about five billion years ago from the accumulation of planetismals (meteorites and asteroids) at about absolute zero temperature. Earth has been kept alive by the flow of energy from its molten interior. This may continue for another 4–5 billion years before all of the heat supply has been dissipated. Virtually all of the other planetary bodies in the system are geologically dead except perhaps Mars. The first life forms on Earth, thermophilic (heat-loving) bacteria, have been found throughout its known geological history and Earth remains both geologically and biologically active. Mars is believed to have had a brief period of running water about 3,600 years ago; fossil thermophilic bacteria are recorded from carbonate inclusions, dated to 3,600 years ago within a Martian meteorite found in the Antarctic (Plimer, 1997).

The first individual radio telescopes had poor resolving ability. For example, the angular resolution of a 25 m radio telescope observing at 1 m wavelengths is almost 10,000 times smaller than optical telescopes. To achieve an optical resolution equal to that of a moderately sized optical telescope, a radio telescope observing at 1 cm would have to be 2 km across. The solution to this handicap was overcome by electronically combining radio signals from space using two or more telescopes in unison. Since the two telescopes are at slightly different distances from source there is a phase difference, which must be resolved. Merging the two signals electronically so that they are detected simultaneously by the same dish does this. Pairs of telescopes situated thousands of miles apart provide resolutions as small as 0.001 arc seconds or even less, much smaller than can be obtained using optical telescopes. The data from each telescope is recorded on magnetic tapes and played back into a computer that combines the signals as if they were from a single telescope.

However, although most visual and radio wavelengths penetrate the Earth's atmosphere, radiation at the following wavelengths is observable only by telescopes located in space. This is because:

- Most infra-red is absorbed by the atmosphere.
- Nearly all ultraviolet is absorbed.
- All gamma and X-ray radiation is absorbed.

In 1959 a space probe launched by the Soviet Union flew within 3,700 miles of the moon. Scientific data gathered by Luna 1 sensors was relayed by radio transmitters back to Earth. Since then several other space probes have orbited the moon or landed on it. Space travel became a new technology and a complete new phase of space exploration commenced. Gold-coated visors protected the astronauts' eyes from searing sunlight on the Apollo moon landing in 1969 when men landed on the Moon and returned safely to Earth. The charged-couple device, which used gold to collect electrons generated by light, was invented in 1970. This invention has subsequently been used in hundreds of military and civilian applications including video photography. Spacecraft missions provided and are providing information both within the solar system and around stars in outer space. Deployment of the Hubble Space Telescope provides a new means of taking measurements of other galaxies in outer space

Space telescopes

The Hubble Space Telescope launched in 1990 was the state of the art model at the end of the 20th century, working around the clock to investigate secrets of the universe. Hubble orbits 600 km above Earth and is the first scientific mission of any kind that is specifically designed for routine servicing by space-walking astronauts. It has a visionary, modular design that allows the astronauts to take it apart, replace worn out equipment and upgrade instruments. Periodic service calls ensure cutting-edge technology and views of the Earth that cannot be made using land-based telescopes or other satellites. Three such missions have been completed by the year 2002. Each time a scientific instrument is replaced the Hubble scientific power is increased by at least 10%. Hubble achieves 3 to 5 gigabytes of data daily, and delivers between 10 and 15 gigabytes to astronomers worldwide. As of March 2000, Hubble has:

- taken more than 330,000 separate observations
- observed more than 25,000 astronomical targets
- created a data archive of over 7.3 terabytes
- provided more than 2,663 scientific papers
- travelled about 1.489 billion miles in circling around the Earth about every 97 minutes
- received more than 93 hours of on-orbit improvements in three successful servicing missions.

The beginning of a new era of planetary science has since been marked by NASA's 'Spitzer Space Telescope', which can directly measure and compare 'extrasolar' planets. It was previously thought that the universe was a reasonably uncomplicated place containing relatively small star-forming galaxies. But researchers now think that this is far from true, even the early universe was extremely complex. A wide variety of galaxies are being found with properties that were not foreseen. The telescope's infra-red spectrograph captured the observation of nearby spiral galaxy M83 on February 16th 2005. Using the Infra-red Array Camera (IRAC) aboard the Spitzer Space Telescope, very distant old red galaxies post 2.5 billion years 'Big Bang' in the Hubble Deep Field South have also been found. The IRAC images have also displayed about a dozen very red galaxies at distances of 10–12 billion light years. These galaxies existed when the universe was only about one-fifth of its present age of 14 billion years. A feature of these images is that the young galaxies exhibit clouds of dust that are absent from the old galaxies. Some of the old galaxies had stopped forming new stars, raising the question of why they had died so soon, i.e., were red and dead.

2

Geology of gold ore deposits

Gold, because of its high density and attraction to iron, is believed to have been mainly concentrated in the Earth's solid and compacted Fe-Ni core during the Earth's accretionary stage along with Sn, Mo, and other highly siderophile minerals. These minerals are also present in Fe-sulphides in the surrounding mafic-ultramafic rock, the uppermost part of which is in a semi-molten state. During partial melting of the mantle, gold derived from the sulphides is contributed to the magmatic fluids and vapours that circulate to the surface through rifts in the crustal rocks in either extentional or compressional tectonic regimes. Tectonic regimes (cratons, ocean basins, divergent margins, convergent basins and transform boundaries) are related to one another through those earth processes encompassed by plate tectonic theory.

In the plate tectonic model, intensive research on the nature of continental and oceanic plate interactions and the associated geological processes that control the genesis of ores has made possible an understanding of many unresolved geological features and processes. The theory provides logical explanations for many aspects of Earth's geology and history including the opening and closing of ocean basins, origins of mountain ranges, geological structures, distribution of mineral resources, and palaeoclimates. Important deposit types are distinguished according to geological setting, host rock type, associated minerals and depths of emplacement. They comprise volcanic hosted massive sulphides (VHMS), mesothermal ore bodies, intrusion related porphyry and non-porphyry deposits, and epithermal deposits of both low and high sulphidation styles. Residual and detrital deposits are developed wherever the unroofing of a sufficiently large primary gold orebody contributes gold to the regolith under stable conditions of weathering erosion and deposition. This may be done directly, e.g. Palaeozoic to present or in stages, e.g. Palaeozoic to Mesozoic to present. Some authors also apply the same tectonic principles to explanations of certain aspects of Precambrian geology, particularly the genesis of greenstone belts, but this is a contentious issue.

Key geological features of gold ore geology discussed in this chapter are crustal evolution, tectonic elements of plate boundaries, hydrothermal gold systems, source rocks, provenance and the time rate of unroofing of orebodies.

2.1 Crustal evolution

No direct evidence exists of crustal evolution earlier than about 4.57 billion years ago but it is generally believed that while the Earth was still in a molten state gravitational forces acted to concentrate the densest material towards the centre and lighter material closer to the surface. The geological timescale (see Table 5.2) describes the timing and relationship between events that have occurred during Precambrian times in accordance with the dates and nomenclature proposed by the International Commission on Stratigraphy. Highly siderophile elements such as the platinum-group elements and Au were thus effectively concentrated into the Fe-Ni core during Earth's accretionary stage (Solomon and Shen, 1997). At this time the core was probably enclosed in a partly molten 'magma ocean' over the surface of which thin platelets of simatic and lesser sialic material were moved about and subducted under the influence of the hot convecting mantle fluids (Lowe and Ernst, 1992). When the surface layer cooled and solidified, a thin film of crustal rock formed around the Earth. Large cracks developed because of thermal stress breaking crustal material into individual rigid plates of 'lithosphere' of continental size, which moved as independent units across the hot plastic part of the upper mantle (asthenosphere). Depressions in the crust formed as natural basins filled with water rising up through fissures in the crust and from the Earth's early atmosphere to accumulate as oceans.

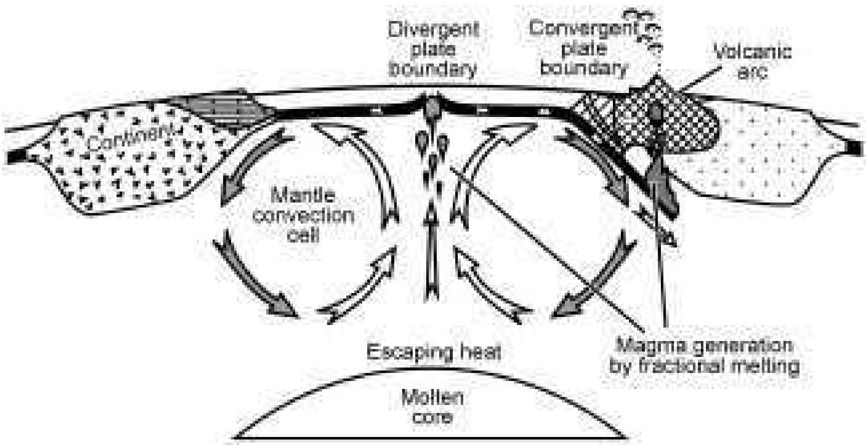
Various explanations are given for the observed spatial variations of the ore-forming elements in the mantle and crust and particularly of mantle heterogeneity generated during the Earth's accretionary stage of enriched mantle resources. One possibility is the addition of a small amount (<1 mass%) of oxidised accretional veneer from the core after its formation. The introduction of highly siderophile elements into the Earth's mantle might then account for the heterogeneous nature of distributions of Au and platinum group elements in the mantle (Kimura *et al.*, 1974; McDonough and Sun, 1995). Another speculation is that core-mantle interaction and rising of the boundary layer might have introduced important amounts of Au and Pt into the mantle through underplating of mantle plume and subducted lithosphere. Solomon and Shen (1997) suggest that while the evolution and final solidification of a circum-global magma ocean in the upper mantle might have resulted in large-scale mantle heterogeneity, these effects could have been counterbalanced by vigorous mantle convection. Available geophysical and geochemical observations point to a pyrolitic type mantle without major differences between the upper and lower mantle (McDonough and Sun, 1995).

2.1.1 Magma-forming processes

The molten or partly molten rock materials making up magma have varying compositions, temperatures, crystal contents, volatile contents and thereby varying rheological properties (McBirney and Murase, 1984). In their simplest form magmas are produced at mid-ocean ridges when hot mantle material rises from the asthenosphere to fill the gaps between diverging plates. The process begins with a mantle convection cell rising to the surface bringing with it ultramafic parent magma. The parent magma fractionally melts as it approaches the surface creating a mafic melt, which forms the oceanic crust while leaving an ultramafic residue behind in the mantle. Magma may reside for some time in high-level volcanic magma chambers, which are periodically replenished and tapped, and continuously fractionated to provide the mixing of individual magmas or development of composition zonation (Cas and Wright, 1995). Magmas filling the rift between spreading plates either solidify as vertical sheeted dykes or spill out at the seafloor to form pillow lavas.

Heat energy from the interior of the Earth rises to the surface due to the action of convection cells within the asthenosphere. The hot plastic rock cools and is turned over slowly at the base of tectonic plates carrying continents, and moves parallel to the Earth's surface at about 10 cm/year before descending back into the mantle at subduction zones to be reheated. In the plate tectonic model, the Earth's crust is broken into seven major and numerous minor lithospheric plates, which continuously jostle against one another as they move as independent, rigid units across the partly molten asthenosphere. The direction and rate of movement of any one plate is influenced by its size and shape and by the size, shape and motion of the surrounding plates. New oceanic crust is in process of formation by the upwelling of basaltic material at extentional plate margins (e.g., mid-ocean ridges, back-arc basins) constructive plate margins, while older crust is being consumed at convergent margins where subducting plates sink back into the asthenosphere. Figure 2.1 is a conceptual cross-section of the seafloor hydrothermal system showing the driving force of seafloor spreading when a plume of hot magma rises under the ocean rift forcing the plates to move apart, and the involvement of divergent and convergent plate settings.

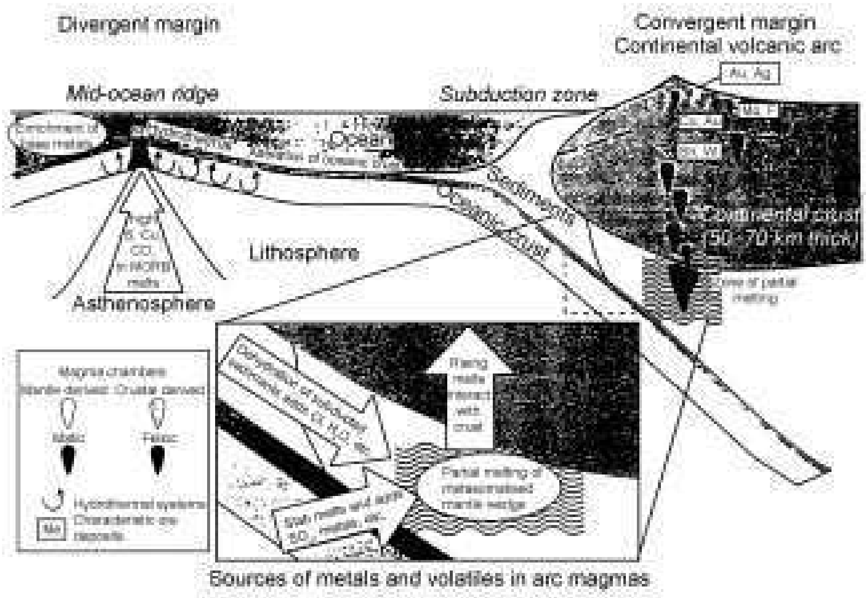
Composition and mineralogical characteristics of erupted magma are the end result of a complex history of processes causing chemical and physical change. Widely different geological histories include the degree of partial melting of the source rocks and other melting events and the nature and extent of the sedimentary cover. Additional factors are the amount of contamination from the wall rock and subducting slabs, periodic replenishment of fresh magma, and tapping and fractionating of magmas in a succession of magma chambers as they rise to the surface. Figure 2.2 is a schematic representation of the principal components of magma genesis, fluid flow and metallogenesis in convective plate settings where oceanic crust is subducted beneath continental lithosphere.



2.1 Conceptual cross-section of a seafloor hydrothermal system.

Influence of sediment on magma characteristics

Factors that link the assimilation of various sediment types to the melt during the magma-forming process strongly influence the composition and mineralogy of the hydrothermal fluids. Wind-blown sediments of all types are deposited at slow rates over the marine environment. The greatest influence of contamination by silicic materials is along continental margins where billions of tonnes of

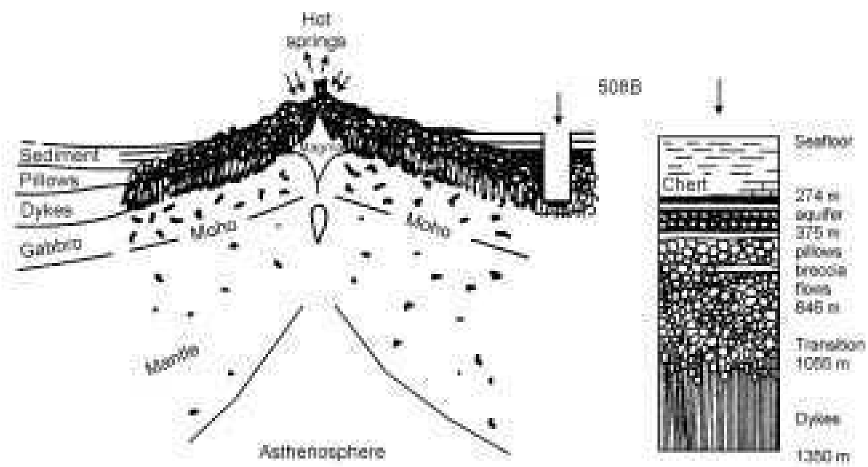


2.2 Schematic representation of principal components of magma genesis, fluid flow and metallogenesis (after Hedenquist and Lowenstern, 1994).

suspended fine to coarse bedload sediment are discharged by rivers into the ocean each year; and along the ocean trenches where turbulent currents distribute great quantities of pebbles, sand, silt and clay. Chemically formed sediments are precipitated as a result of reactions of the seawater to products of both on-shore erosion and off-shore volcanic action. In warm tropical waters, extraction of calcium carbonate from seawater by marine organisms and biological precipitation as shells may cause extensive carbonate sedimentation.

The mantle of sediments becomes increasingly silicic with distance from the magmatic source, and its assimilation into the melt during magma-forming processes in subduction zones strongly influences the composition and mineralogy of the hydrothermal fluids. Differences in nature and distribution of the rock types are functions of the average compositions of the continental and oceanic crusts. In modern settings, involvement of crustal magma is evident by the high percentage of silicic and intermediate volcanics as in the Andes and Cascades. In arc and back-arc settings, sediments are largely continental, e.g. fluvial, alluvial fan and lacustrine; and will range in the fore-arc area from continental to shallow marine to deep marine (Cas and Wright, 1995). However, silicic magmas may be generated in areas where the basement is oceanic as well as where it is continental; significant proportions of mafic magmas in continental settings are also not uncommon.

Based upon seismic properties, a complete section of oceanic crust has a depth of about 5–7 km (Anderson *et al.*, 1982). Figure 2.3 is a schematic representation of the section intercepted in hole 508B of the Deep Sea Drilling Programs DSDP/IPOD, which was drilled to a depth of 1,350 m on the southern flanks of the Galapagos spreading centre, south of Costa Rica. This hole is fairly typical of what might be expected of mid-ocean ridge volcanism where the only



2.3 Schematic representation of section intercepted in hole 508B of the Deep Sea Drilling Programmes DSOP/1POD.

source of overlying sediment for true open-ocean oceanic crust is of calcareous and/or volcanic pelagic origin. Pelagic (marine) sedimentation is dominant along the edges of median valleys, on ridges and abyssal plains at mid-ocean spreading centres.

2.1.2 Late Archaean (2.9–2.6 Ga)

With few exceptions only those parts of the Precambrian geological cycle relating to Archaean continental crust are preserved and the possibility of small relics of Archaean oceanic crust still being in existence is by no means certain. The mafic parent (Komatite Suite) generated oceanic lithosphere in Archaean volcanic arc systems that are quite different to those operating today. Rocks older than 2,500 Ga exist only because they are geologically stable and have survived episodes of faulting, uplift and erosion that have occurred over the past billion years. The oldest known rocks, of which the last unit has been dated from studies of isotropic systematics at around 3,827 Ga, occur in the Amitsoq Gneisses, Greenland. Judging from these deposits and similar occurrences in Canada, China and the USA, and of occurrences of detrital zircons aged 4.1–4.2 Ga in Western Australia the crust, atmosphere and hydrosphere must be several hundred million years older than rocks of the Isua Supercrustal Belt (Solomon and Shen, 1997). Surviving remnants of Proterozoic mobile belts may be cores of old mountain chains formed as a result of continental drift and plate tectonic processes similar to those that became widespread in the Phanerozoic. In the view of some authors the overall uniformity of both chemical composition and range of types of sedimentary and igneous rock from 3.87 Ga points strongly to the continuity of tectonic processes broadly similar to those of recent Earth (Lowe and Ernst, 1992; Windley, 1984).

The growth and stabilisation of continental crust, combined with high heat flow led to the formation of cordilleran and island arc volcanism and calc-alkaline magmatism, accretion of volcanic arcs in greenstone belts and development of cratonic and craton margin sequences (Taylor and McLennan, 1995; Solomon and Shen, 1997). Initially these movements and subduction would have taken place under the influence of shallow convection due to the hotter mantle. Deep penetration of the mantle by subducted material *was* probably inhibited by the shallow depth of melting so that a few crustal blocks could have formed to reach diameters of >1000 km in virtual isolation (McCulloch, 1993). Large volumes of crust, both ultramafic and felsic were periodically formed, recycled and reprocessed in the crustal environment to about 3.7 Ga through lithosphere subduction, meteorite impact, and crustal remelting. Extensive platforms were established with major rifts and passive margins in extensional settings and foreland basins in convergent settings (Lowe and Ernst, 1992).

With gradual cooling of the mantle, a reduction of melting of the subducted oceanic crust led to changes in the overall composition of the upper crust. The development of extensive continental margins and depositional basins heralded development of giant Au-U conglomerate and banded iron formations (Solomon and Shen, 1997). Major Ni-Cu, Au and base metal mineralisation in the Yilgarn Block of Western Australia and The Superior Province Canada was associated with orogenesis between 2.8 and 2.6 Ga (Barley and Groves, 1992). The late Archaean was a period of rapid growth of continental crust and by its close more than half of the Earth's continental crust may have formed (Taylor and McLennan, 1995).

Archaean greenstone belts

Diorite, andesite and eventually coarse-grained mainly granitic sialic rocks rich in silica and alumina (probably plagiogranite/rhyolite) dominate Archaean geology. The rocks themselves are of two main types: strongly deformed gneissic rocks that are characterised by the effects of immensely complex geological structural history, and groups of homogeneous intrusive igneous masses. Smaller groups of rocks, which formed on top of the crust (supracrustal rocks) comprise mixtures having volcanic and sedimentary associations. In some parts of the crust the rocks, commonly referred to as 'greenstones', are only partly metamorphosed and clearly distinguished volcanic components have a greenish tinge imparted by hydrous metamorphic minerals.

Greenstone belts are interpreted as remnants of very small island arcs (probable thin platelets) compressed closely together 3–4 billion years ago. The volcanic chains of island arcs were initially scattered across a single worldwide ocean, but through developing plate tectonic processes at divergent and convergent plate margins were swept together to form the first supercontinental type assemblages. The process, which probably involved extensive overturning of the convective cells in the upper mantle and very rapid growth gradually led to the formation of larger plates composed of continental crust. Extensive platforms were established with major rifts and passive margins in extensional settings (Lowe and Ernst, 1992). The gold-rich Witwatersrand Basin is thought to have formed as a foreland basin during the collision of two continents between 3.1 and 2.7 billion years ago (Burke *et al.*, 1986).

In addition to the Archaean greenstone belts of Southern Africa, greenstone belts intruded by granitic magmas and surrounded by extensive areas of granites and granitic gneisses are well represented in highly mineralised shield areas of Australia, Canada and North America. Notable examples of primary gold deposits are found in late Archaean rock belts including Timmins Camp and Albibi Belt, Canada; Kolar and Kanataka, India; and the Kalgoorlie-Norseman Belt of the Yilgarn Block, Australia. Typical examples of major mesothermal gold deposits occur in supracrustal Proterozoic rock belts at Homestake in the USA and Ashanti, Ghana. However, while mutual relations between supracrustal materials and granitoids in 'low grade' terraines are quite evident, areas in 'high grade' metamorphic terraines have been repeatedly deformed and metamorphosed at temperatures as high as 900 ºC and at pressures corresponding to depths of 20–50 km in the continental crust. Under these conditions the supracrustal materials would have lost their primary textures through recrystallisation and their relations to the granitoids would have been obliterated.

2.1.3 Proterozoic eon (2.6 Ga–580 mya)

Transition from the Archaean to the Proterozoic, 2.6 billion years ago was associated with gradual changes in overall composition of the upper crust, evolution of the hydrosphere-atmosphere and extensive sediment deposition in broad shallow seas. Many of the continental masses, with the exception of Australia, were apparently located near to or around the North and South Poles; widespread evidence of glaciation is found in late Archaean continental systems. The landmasses had grown significantly in extent by the accumulation of felsic rock produced in subduction zones. Changes in the geological framework would probably have occurred due to chemical overprints of metamorphism and hydrothermal alteration, combined with uncertainties of the geothermal gradient and the scale of the tectonic systems.

Basic, intermediate-felsic volcanic piles appear to be prominent from mid-Archaean times into the Proterozoic (Hutchinson, 1983, 1987). Deposits associated with these piles include zinc-copper and lead-zinc-copper-silver sulphides with lesser gold. Proterozoic sequences are dominated by collisional and obducted material, and by sediments in long-lived passive margins of the craton system. During periods of continental fragmentation and dispersal wide continental margins provided conditions suitable for the development of extensive non-clastic basins and a surge in BIF (banded iron formation) and Mn sedimentation. Hydrothermal Au-U-Pt group assemblages were formed in extensive fractured cratonic basins in Australia (e.g. Jabiluka) and Canada (e.g. Rabbit Lake) from about 1.7 Ga (Solomon and Shen, 1997). Important metallogenic events associated with the African continent include volcanogenic massive-sulphide and mesothermal gold mineralisation occurred during 900–600 Ga in opiolite and island arc terraines of the Arabian-Nubian Shield (Al-Shanti, 1979).

Deposition of the Witwatersrand gold-uranium deposits commenced at the end of the Archaean and continued through the early Protozoic. Similar deposits are known in other Proterozoic intracratonic basins such as the Archaean Yilgarn Craton of Western Australia. The Yilgarn Craton covers 20% of the Australian continent. Exposure to bedrock is extremely poor and most deposits occur either at or adjacent to sparse outcrops. Nevertheless, the craton produces two-thirds of the gold recovered from Australian mines. One of the most important deposits is the Roxby Downs Olympic Dam gold–uranium–copper ore deposit of South Australia. Olympic Dam, which is estimated to contain more than 2.3 million tonnes of ore and is the largest known uranium deposit in the world, the sixth largest copper deposit and contains one of the most significant gold–silver resources. Interpreted rocks of the region include granite, granitic gneiss, layered intrusions and sills. The Olympic Dam deposit was formed in a volcanic environment about 1,500 million years ago when magma was rising from the deep molten part of the Earth's upper crust. Some crystallising magma reached the surface before erupting explosively from volcanoes. Super-hot water-rich fluids were released containing dissolved iron together with copper, uranium, gold and silver. These fluids were under extreme pressure and caused fracturing and brecciation of the crust. The area was eroded flat by the scouring action of glaciers and covered by thick layers of sedimentary rocks including shale deposited on the ocean floor. A further uplift exposed the shales to erosion and the formation of the present land surface. The ore body was discovered 350 m below the surface in 1950 when boreholes identified copper and uranium mineralisation. On average, each tonne of ore contains about 13 kg copper, 0.4 kg uranium and 0.5 g gold.

The younger Proterozoic placers may be associated with Archaean provenances but are more likely linked with high-grade supracrustal gneisses having an ultramafic/plutonic affinity and greenstone belts with sedimentary, chemical and conglomerate affinities. The gold is generally subordinate to sulphide minerals in massive or stratiform deposits linked with major non-conformities involving high-grade sediments. Surviving remnants of Proterozoic mobile belts appear to be cores of old mountain chains that were formed as a result of continental drift and plate tectonic processes similar to those that became widespread in the Phanerozoic.

The distribution of landmasses at any time in the geological past is reconstructed by investigating such features as the distribution of rock types, fossils and geological structures, palaeomagnetism and radioactive age dating. Several lines of evidence suggest that plate motions at the present day and in the geological past can be correlated through changes in magnetic lineation on a worldwide basis, and linked to a geological timescale. With few exceptions only those parts of the above geological cycle that relate to Archaean sialic crust are preserved and the possible existence of small remnants of oceanic crust locked up in Archaean cratons is uncertain. Existing continental shield areas (cratons) consist of Precambrian rocks arranged in a series of elongate structures similar to lineal mobile belts. The structures are progressively younger with distance from the central craton, thus suggesting that over time, the attachment of mobile belts to continental belts could have obscured evidence of the mineralisation of those times.

The supercontinental cycle

Since the beginning of Earth's accretionary stage, the continents have been progressively joining together and drifting apart sending small pieces of continent across the ocean to collide and form other larger continental masses. Each new evolutionary trend generated new continental crust, the continents becoming larger each cycle. Fichter (1999) suggests that this cycle of break-up and collision has been a constant and evolutionary trend for some 4 billion years requiring about half a billion years for each cycle. Other estimates of elapsed time between each fully assembled supercontinent range between a few hundred million years to 2,000 million years (Gurnis, 1988). Worsley *et al.* (1985) recognised 500 million years tectono-eustatic cyclicity for the post Archaean. Megacycles of 210–325 million years are proposed by Vail *et al.* (1977) for the Phanerozoic. Hoffman (1989) interprets megacycles of similar length for the North American Proterozoic and Australian Archaean respectively.

Conceptually the supercontinental cycle begins with continental joining on one side of the Earth balanced on the other side by a superocean in which new oceanic crust is being created at spreading centres. In this scenario the formation of a supercontinent follows sequentially through:

- creation of small island arcs by convection cells and subduction zones
- volcanic arc collisions and evolution of protocontinents through many individual subduction zones
- volcanic-arc-protocontinent collisions to form microcontinents
- collision zones in which the microcontinents join together to form continents such as North and South America, Siberia and Australia
- further continental edge subductions leading to the formation of a supercontinent (Fichter, 1999).

The theory then suggests that the evolution of a major supercontinent is followed by a build-up of heat in the mantle; upwelling at hot spots, and subsequent break-up into separate continental masses. Metal deposits are associated with both the periods of orogenic supercontinent aggradation and the initial stages of break-up.

Rodinia

The oldest-known supercontinent, ‘Rodinia’ is thought to have been assembled about 1100 million years ago with Laurentia (Proto-North America) as its core. At that time, although the exact size and configuration of the supercontinent is uncertain, North America was adjacent to western South America on the east while Australia and Antarctica lay next to it on the west. Rodinia is thought to have split into two halves about 750 million years ago opening up the Panthalassic Ocean. North America rotated southward towards the ice-covered South Pole. The northern part of Rodinia, which comprised Antarctica, Australia, India, Arabia and a group of continental fragments that would one day meld and form China, rotated northwards in a clockwise direction across the North Pole. A third continent, the Congo craton, which was made up of much of north Central America, was caught in the middle between the two halves of Rodinia. Collision between the three continental masses about 550 million years ago (mya) resulted in the Pan-African orogeny and the formation of a supercontinent called Pannotia. These collisions took place when the continents comprising Pannotia already had active rifting.

Pannotia

Pannotia was the name given by Ian W.D. Dalziel in 1997 (see the *Encyclopedia Wikipedia*) to a hypothetical supercontinent that existed from about 600 mya to about 540 mya. Pannotia broke up at the end of the Precambrian and the Palaeozoic era began 540 mya with four continents, which lined the equator between latitudes 60º N and 60º S:

- Gondwana (which was composed of Antarctica, Australia, India, South America and South Africa)
- Baltica (northern Europe)
- Laurentia (North America)
- Siberia.

Polar regions were apparently land free.

Similarities in the patterns of Proterozoic mineralisation appear to reflect the amalgamation of northern Australia with Laurentia during the early Proterozoic (Hoffman, 1989). Barley and Groves (1992) record other important metallogenic events associated with the African continent. These events include the Bushveld Complex (platinum group elements) and the Palabora carbonatite (Cu-Fe-P). The Palabora deposits were emplaced at around 2 billion years ago (bya) within a piece of continental crust that had been only slightly affected by orogenic activity since the late Archaean (Gustafson and Williams, 1988). Volcanogenic massive-sulphide and mesothermal gold mineralisation also occurred in opiolite and island arc terrains accreted to a peripheral Pan-African orogen (900–600 mya) in the Arabian-Nubian shield (Al-Shanti, 1979).

2.1.4 Phanerozoic eon (580 mya–present)

The Palaeozoic era (225–580 mya) of the Phanerozoic eon was characterised by reduced physical changes and abundance of sedimentary rocks; geological periods were separated on the basis of intervals of continental uplift followed by submergence and encroachment of oceans onto the land. Polar regions during the Ordovician period (490–475 mya) were wide expanses of ocean and land-masses were located around the equator. Baltica and Gondwanaland shifted to the east and south. The Caledonian Orogeny reached its peak during the Silurian period (435–430 mya). The sequence of continental collisions and the formation of the supercontinent Pangaea resulted from:

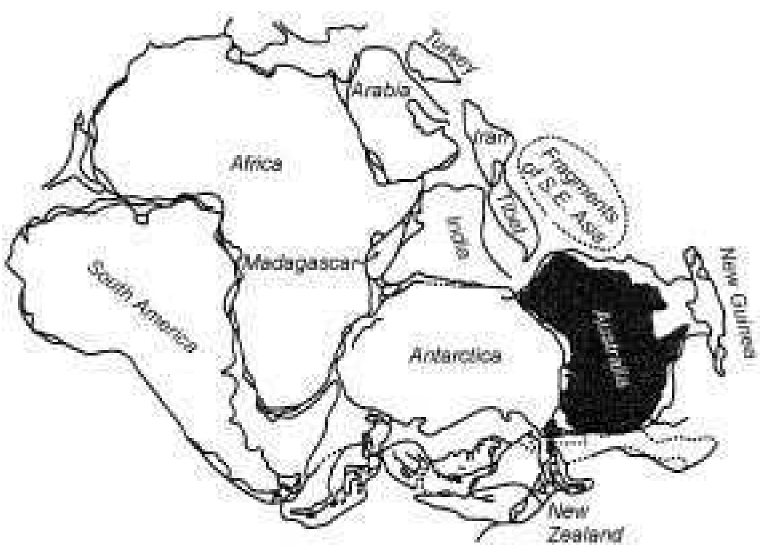
- the southward movement of Gondwanaland, which carried it over the South Pole and then northwards to where it would eventually form part of Pangaea about 350 mya
- closure of the ocean between Gondwanaland and other continental fragments, which commenced about 310 mya, and led to the westward movement of China, and that of Siberia from low latitudes to high latitudes, (merging with Kazakhstania to form the Ural Mountains).

The supercontinent Pangaea

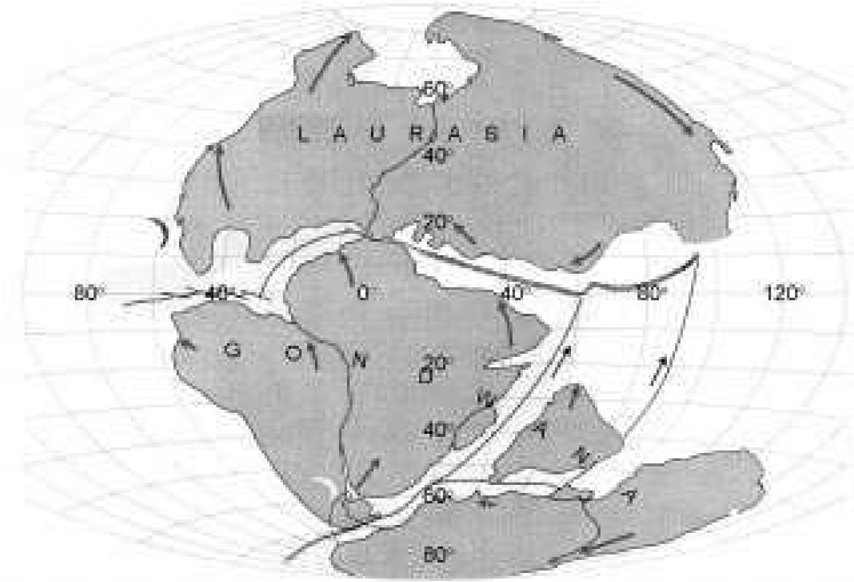
The successive break-up of the supercontinents Rodinia and Pannotia and their reassembly as Pangaea (250–300 mya) dominated Phanerozoic geological history. Reassembly of the landmasses occurred with the formation of the supercontinent Pangaea (all lands) surrounded by a single large ocean Panthalassa (all oceans) about 300–250 million years ago (Fig. 2.4). Two major rifts illustrated schematically in Fig. 2.5 were developed during a period of extensive orogenic activity about 200–180 million years ago (Hoffman, 1989). The Sea of Tethys divided Pangaea into 'Laurentia-Baltica-Siberia' in the northern hemisphere, and 'Gondwanaland' in the south (Hoffman, 1989; Moores, 1982).

Laurentia-Baltica-Siberia

The major northern continental mass, which evidently rotated in an anticlockwise direction as a single plate after an initial period of intense orogenic



2.4 The supercontinent Pangaea.



2.5 Pangaea after rifting (Wilson, 1976).

activity, commenced to break up. Oceans opened and closed. Laurentia split up into North America, Greenland and Europe and that part of Asia north of the Indian sub-continent. Sediments now in the Caledonian-Hercynian-Appalachian Mountains of Eastern Europe were laid down in an old ocean in the lower Palaeozoic. This ocean may have been closed long before the present Atlantic Ocean opened up and separated the Appalachian Mountains from their continuation in north-eastern Europe less than 150 million years ago (Bullard, 1969). In the Cainozoic period (65 million years to the present) the continents drifted to the positions they occupy today. North America is joined to South America through the Isthmus of Panama, the Indian landmass has continued its northerly drift and has collided with Asia, and Greenland has been detached from Europe and Asia by the mid-Atlantic rift.

The greatest abundance of sediment-hosted Pb-Zn and Cu minerals deposited during the formation of Pangaea (300–250 mya) appears to have been in either epi-continental or foreland settings of the Caledonian-Appalachian Orogen, e.g. Duane and DeWit (1998). Most types of gold deposits, including volcanic hosted massive sulphides, podiform chromite porphyry style Cu and Mo mineralisation occur in island arcs and basins along the Pacific margin (Sawkins, 1990).

Peripheral orogeny and related mineralisation is thought to have occurred along a continental margin from southern California to Scandinavia. Mineralisation includes Archaean style mesothermal Au (e.g. Homestake), VHMS and porphyry Cu deposits (see also Gaal and Gorbatchev, 1987). High sea levels

1.8–1.6 Ga resulted in the preservation of large marine platforms and intracontinental basins (e.g. the Athabasca Basin). Anorogenic magmatism preceded the break up of this supercontinent and renewed orogenic activity was accompanied by a period of continental aggregation between 1.3 and 1.0 Ga.

A gradual fragmentation took place during the Mesozoic era (235–280 mya). Laurasia and Gondwanaland drifted apart about 180 mya in the Jurassic Period, thus opening the way for the formation of the South Indian and North Atlantic Oceans. The South Atlantic Ocean was formed at the spreading centre between Africa and South America; the latter probably remained connected to part of Africa, Antarctica and Australia until the end of the Cretacious. The Urals were weathered during the Triassic and the first phases of Alpine and Andean folding introduced the great mountain ranges of modern times about 100 million years ago.

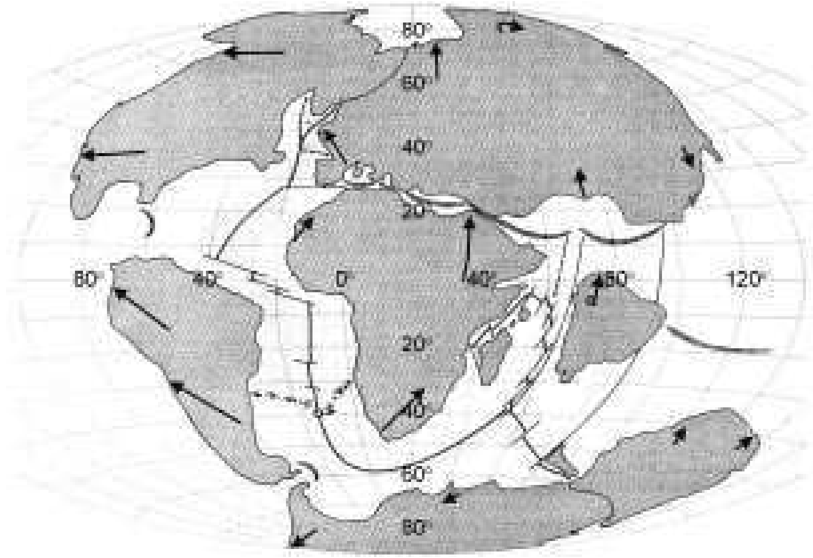
The supercontinent Gondwanaland

Evidence that the continents of New Zealand, Antarctica, Australia, Africa, India and South America were joined together as the supercontinent 'Gondwanaland' about 200 million years ago is shown by the geological make-up of the continents themselves. Fossil records of plants such as the fern Glossopteri found in the Karoo System in India, Africa and South America are very rare in the northern Hemisphere. Geologically, Upper Proterozoic to Cambrian rocks of the Adelaide orogenic belt continue into the Ross orogen of Antarctica and tillites, which occur all over the southern Hemisphere, contain many similarities in their sediments. Physically, the bulge of South Africa fits the impression in the western coast of Africa when fitting the individual continents together.

Fragmentation of the major southern continent Gondwana resulted in splitting South America and Africa as a separate landmass away from the remainder of Gondwana (Antarctica, Australia, and India). Figures 2.6 and 2.7 respectively, depict the postulated configuration of Gondwanaland after its split from Laurentia and the present configuration of the continents and plate margins (Plimer, 1997). The initial separation of Australia from Antarctica took place at a rate of about 12 cm/year and its current movement is 7 cm/year northwards away from Antarctica. The Pacific Ocean is becoming smaller along subduction zones under North and South America and Japan as North America and Japan get closer together. Ultimately Asia and America will collide to form another supercontinent. Judging from the past, this continent will also be comparatively short-lived in a geological sense.

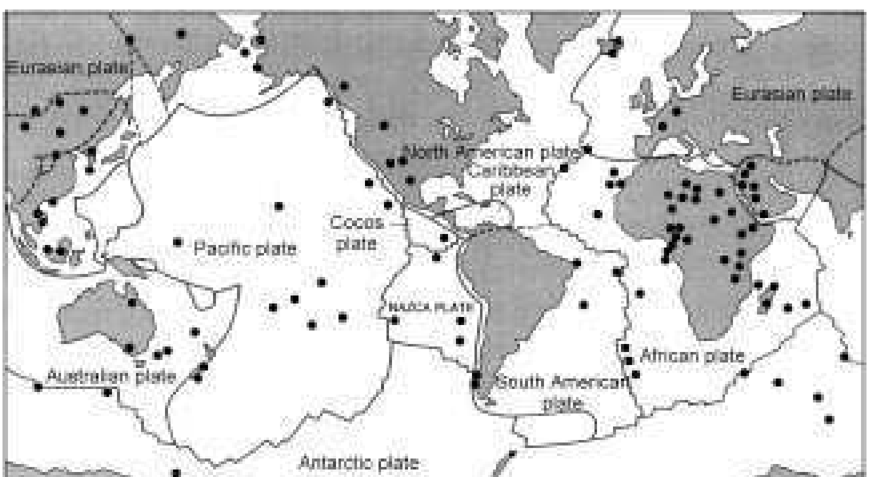
Cainozoic mountain belts

The Cenozoic era commenced about 65 mya at the end of the Cretacious with the northward movement of Africa and creation of the Mediterranean Ocean;



2.6 Postulated configuration of Gondwanaland (Wilson, 1976).

and about 45 mya with the separation of Australia and Antarctica. The collision of India with Asia results in the formation of the Himalayan Mountains. The North Atlantic Ocean is formed from the separation of Europe from Greenland and North America; and Europe fuses with Asia with the drying up of the Obic Sea. The probable connection between these continents is indicated by the occurrence of glacial deposits in present-day desert conditions in Africa, for example the Vaal River Valley, and in India and Australia.



2.7 Present-day configuration of continents (Wilson, 1976).

Cainozoic mountain belts are manifested in two broadly curved chains of volcanoes. One of these, the Circum-Pacific belt, rings the Pacific Ocean in what is known as the Pacific Ring of Fire. The other, the Eurasian-Indonesian belt commences in the west at the Atlas Mountains in North Africa and runs through the Near East to the Himalayas and thence to Indonesia where it joins the Circum-Pacific Belt. Gold-bearing ore zones are located within the igneous and metamorphic rocks of these settings at various depths ranging from near surface (epithermal) vein systems, to deep-seated (mesothermal) deposits at depths of 10 km and more. Deposits of economic significance are found mainly in stratovolcanoes and rhyolitic centres in continental and marine volcanic centres. Extensive sub-tropical weathering during the Mesozoic favoured the formation of lateritic and alluvial gold placer types.

Case history – the Chillagoe story

A scientific description of major geological processes that may have contributed to the formation of gold orebodies over long periods of geological time is provided by the geological history of the Chillagoe gold field (Fig. 2.8), as summarised by Ian Plimer in his book *A Journey through Stone* (1997). The book is a compilation of the work of geologist John Nethery who, together with his team, revolutionised the thinking on the regional geology of Chillagoe and its orebodies. In his book Plimer shows how the simultaneous action of all the contributing processes of landscape evolution and denudation is a balance over time between tectonic forces that create mountains and form orebodies, and the contributing processes of denudation that act to wear them down. The story serves as a general model to explain how present surface exposures of mineralised zones are controlled by factors related to continental drift and plate tectonic processes, reactivity of the host and country rocks to hydrothermal processing, and climate.

Many of the sediments and volcanics were formed while Australia formed part of Gondwanaland together with landmasses that are now parts of Africa, India, Antarctica and South America. In north-eastern Australia intense crustal forces were beginning to build up deep in the crust and the Dargalong Metamorphics first started to stretch and then broke. Some 1,570 million years ago, a very large folded mountain chain (similar to the Himalayas) was formed in a rift valley to the east of Chillagoe. The direction of compression changed about 100 million years later and the still hot and bent rocks were again pushed up into mountains. Five hundred million years later the third deformation of the Dargalong Metamorphics bent the rocks into a corrugated shape.

An ice age, which covered the Australian continent about 750 million years ago, left behind retreating glaciers, large masses of rock debris that yielded lake sediments containing thousands of layers of winter mud and summer sand. Intensification of crustal forces and stresses deep in the crust followed melting

2.8 The Chillagoe goldfield, North Queensland.

of the ice sheets and the redistribution of weight at the beginning of the Cambrian period. The Dargalong Metamorphics stretched and eventually broke and the Palmerville fault started to move. Continental Australia underwent intense weathering and erosion each time it was uplifted. At least 8 km thickness of crust was planed off the Dargalong Metamorphics in the period from 550–450 million years ago reducing the ranges into gently undulating hills which again rose due to isostatic adjustment. Melting took place at depth and the molten granitic rocks moved up along tensional fractures in the Dargalong Metamorphics to solidify some kilometres from the surface as the Nundah Granodiorite. The metal-rich steam released from the granodiorite formed a halo of water-rich minerals glinting with uranium, thorium and gold.

Compression from the Kanimblan Orogeny 370–325 million years ago was a major mountain building process, which pushed the sequence of rocks next to the Palmerville fault into slices, one on top of the other. The Nundah granodiorite then broke in half. Some of the uranium, thorium and gold in the halo was flushed out by hot circulating waters and moved along the Palmerville fault to be redeposited in rocks above the fault. Similar type compression is currently occurring from the flight of India from Australia to collide with Asia. The Himalayas have risen in the collision zone; Mt Everest is currently rising at the rate of two centimetres per year.

Because slices of crust had been humped up on one another, the Earth's crust at Chillagoe was very thick and the bottom of the crust was cooked at high pressures. Parts of the crust began to sink, and as sinking continued, the crust began to stretch, the rocks breaking into a sequence of sunken and elevated blocks. Molten rock again rose from deep in the crust via fractures and faults, some of it solidifying to form huge masses of granite, the remainder erupting at the surface via volcanoes. As the magma rose it also cooled and the resulting solid rock consisted of coarse-grained crystals in a fine ground mass (porphyry) when it finally froze.

Following cooling of the granite in the stress lull at Chillagoe 325–315 million years ago, geological stresses again increased. Compression gently folded all of the older rocks, such as the Chillagoe formation and reactivated the older faults, including the Palmerville fault. Blanketing of the whole area by volcanics acted on the rocks like the lid on a pressure cooker. The Dargalong Metamorphics and Chillagoe Formation again piled up on one another. The metals in the Chillagoe area underwent a second stage of natural recycling and again the gold was relocated and reconcentrated.

During the period 305–280 million years ago, the planet entered into another of its greenhouse/ice age cycles and an ice sheet, which covered Southern Australia and Eastern Australia underwent alpine glaciation as far north as Rockhampton. Australia was still a part of Gondwana, joined to the south to Antarctica, and to the west to India. The oceans covered much of Western Australia, Central Australia and Queensland. Continental glaciers disgorged enormous quantities of glacial debris to the sea or to the floors of lakes dammed by the debris. By contrast, the climate of equatorial Europe was arid.

One of the largest volcanic eruptions known to occur on Earth then happened around 280 million years ago, when 2,000 cubic kilometres of material erupted from the Featherbed Volcano located to the north-east of Chillagoe. The Featherbed Volcanics erupted from eight distinct volcanic centres over an area 200 km × 30 km wide. The centres of eruption were in a north-westerly direction parallel to the Palmerville fault. It is assumed that the rising molten rock followed fractures associated with the fault to get to the surface. The superheated blast material flowed at speeds of up to 200 km/h over the landscape devastating everything in its wake.

Another great crisis in the history of life on Earth occurred 245 million years ago, when 96% of all marine species on Earth became extinct. For some unknown reason, while almost all of the marine animals suddenly disappeared, very few terrestrial species were similarly affected. There was no evidence of any great climate or sea level change at that time. A massive Siberian volcanic eruption that may have triggered the Featherbed volcanic eruption added to the sulphurous gases that would have been released during that time but there is no evidence of land plants becoming extinct. Other possibilities, which have received little general acceptance, include a supernova bombardment from space and a global viral disease pandemic.

The geological history of Chillagoe is a continuing process. Weathering and erosion are still taking place; the land is rising in response and being reshaped by the elements. Climate is fluctuating between greenhouse and icehouse conditions. In time the plate tectonic cycle will move onto its next phase and small Earth tremors that result from release of stresses along fault lines will again be reactivated by further changes to the Earth's crust, oceans and climate.

2.2 Tectonic elements of plate movements

Specific variations in metallogeny associated with the aggregation and break-up of continents were not recognised until a few decades ago when it became evident that any explanations not involving plate tectonics were in most cases less satisfactory than those that do. In plate tectonic terms, the crust and upper mantle of the Earth consist of a number of lithospheric plates, comprising oceanic or oceanic plus continental crust (sima and sial) together with the adjacent upper mantle. The plates tend to move as rigid units, with most of the resulting deformation arising from interactions with other plates along their margins.

Plate boundaries show a variety of characteristics, which in part are due to the nature and motion of the adjacent plates. Some variation must also arise by the speed and angle of collision. However, differences in the features by which plate interaction is recognised are developed along the subducting boundaries, where earthquake foci define a zone of slippage dipping to depths as great as 700 km at angles between 15º and near vertical. Lines of evidence that indicate plate motions throughout geological time include magnetic lineations on the ocean floor that are the result of basaltic magma emplaced at spreading axes crystallising and becoming magnetised according to Earth's prevailing magnetic field. Reversals of this field produce marked changes of the remnant magnetism of successive parallel zones, thus recording the Earth's magnetic history and serving to date sections of the seafloor. Arc systems consisting of deep elongate oceanic trenches and volcanic islands are developed at the surface, on the majority of plate boundaries. Plate tectonic activity is best known in the Phanerozoic and to a lesser extent in Proterozoic settings. Some authors also apply the same tectonic principles to explanations of some aspects of Precambrian geology, particularly the genesis of greenstone belts, but this is a contentious issue.

2.2.1 Crustal deformation

Crustal movements are of two main types, orogenic and epeirogenic. Orogenic movements are spasmodic in nature due to such causes as the buckling of continental crust imposed by plate movement or telescoping of continental shelf strata during the collision of two continental plates. Mineralisation is associated with the sub-surface volcanic activity of the subducting plates. Important deposit types are distinguished according to geological setting, host rock type, associated minerals and depths of emplacement. The sporadic nature of the movements is due to the strength of the massive lithospheric layer, which resists uplift until at some critical time the pressures build up sufficiently to fracture or fold the overlying rock. Epeirogeny appears largely to influence the modification of existing landscapes and the re-working of pre-existing alluvial gold concentrations. Epeirogenic movements are slow, pulsatory mass movements of continental shield rocks, which produce extensive uplift or sinking of broad areas of crust in a vertical or radial direction. It is important to note that whilst orogeny engenders hydrothermal activity and is associated with the formation of primary gold ore bodies, epeirogeny appears largely to influence the modification of existing landscapes and the re-working of pre-existing alluvial gold concentrations.

Orogenesis

Orogenesis refers to major episodes of crustal deformation and uplift associated with tectonic activity at lithospheric plate boundaries. Colliding plates give rise to both compressional and tensional forces acting tangential to the Earth's surface. Compressional forces buckle the overriding plate into an elongated series of folded mountain ranges along the continental margins. The emplacement of large bodies of igneous rock enhances orogenesis and in metallogenic regions, granitic plutons may be associated with the deposition of gold in hydrothermal vein systems. Tensional tectonic activity occurs at divergent plate boundaries where oceanic plates are separating or continents are drifting apart. Vertical tension cracks that appear in the upper parts of anticlines formed by uplift of the overriding plate provide passages through which outpourings of lava build up to form volcanoes of the andesitic type. Orogenic belts, exposed at various levels of erosion, make up much of the world's continental area, either at the surface or under a sedimentary cover. With the possible exception of very old shield areas, most of the cratonic crust of the Asian region consists of orogenic belts of late Precambrian to Mesozoic age.

Orogenic episodes throughout geological time from the earliest Precambrian have combined numbers of old crustal elements into a generally rigid block. These elements, including Precambrian shield units represent earlier plates that have become progressively enlarged and welded together by successive orogenic cycles along belts of tectonic activity; presumably representing former collision or subduction zones between or marginal to the plates. In orogens, such as the early Palaeozoic fold belts of North America, collision and deep levels of erosion have erased the original topographic expression of the tectonic elements. In other Palaeozoic orogens, magmatic and fold belts fringe the shield areas, which include collision zones or sutures of late Palaeozoic age with later additions or re-mobilisations in the Triassic and Cretaceous. Caenozoic tectonic history is generally considered to closely relate to present-day plate interactions and both active and passive tectonic elements can easily be recognised. The major plate interactions were probably already established in the Mesozoic, but structures of this age in current arc-trench zones and generally overprinted by those of the main Tertiary orogenic phase, which is still active at the present day.

The Asian island arc system of the Pacific Ring of Fire is a type area for other such systems and may be expected to yield results of significance to similar systems in other parts of the world and throw additional light on the constitution of older arc systems. The implication is clear that detailed examination of active arc systems should provide models of processes at the higher crustal levels that can be extrapolated downward. The essential elements of an active back-arc–interarc–marginal basin and its associated arc-subduction complex are illustrated in Fig. 1.19.

Epeirogenesis

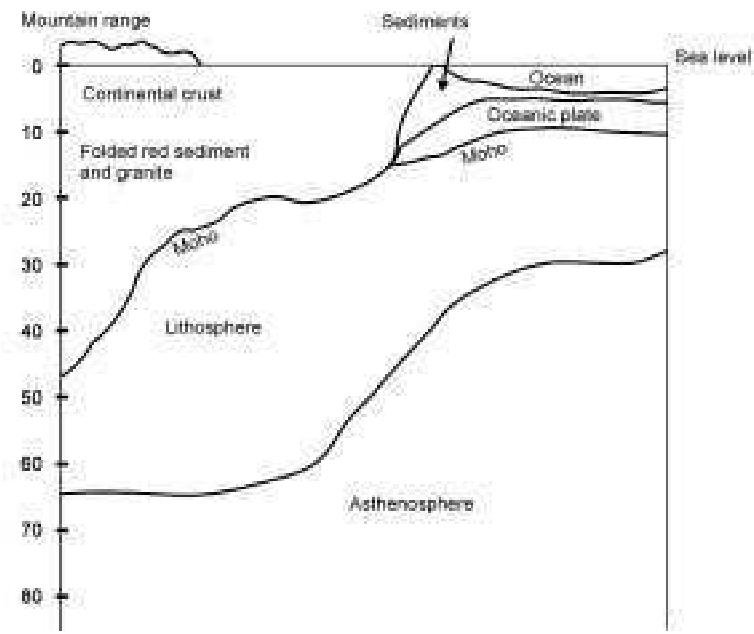
Rock masses affected by epeirogenic movements are typically warped or tilted in contrast to the more rapid folding and fracturing of orogenesis. Isostatic adjustment occurs in response to denudation or sediment accumulation. As denudation continues the material beneath the continent rises spontaneously to compensate for the loss of surface material, thus maintaining a state of isostatic equilibrium between the continental mass and the denser underlying sima. Retained stresses in the crustal rocks are periodically released by denudation of the overlying surfaces, so triggering a rapid isostatic compensation to restore equilibrium. Since the total landmass is several times greater than the mass lying above base level, uplift occurs at an amount less the thickness of the material removed.

Tilting of the crust produces a range of gradient changes that may include aggradation and degradation in the same fluvial system. Local decreased gradients in one sector may generate elevated pay streaks within an aggrading stratigraphic section. Increased gradients in another sector may segment otherwise payable pay streaks into potholes and other bedrock features. Downward warping of the Lakekamu Embayment in Papua New Guinea led to the slumping of some relatively high-grade pay streaks to presently uneconomical mining depths

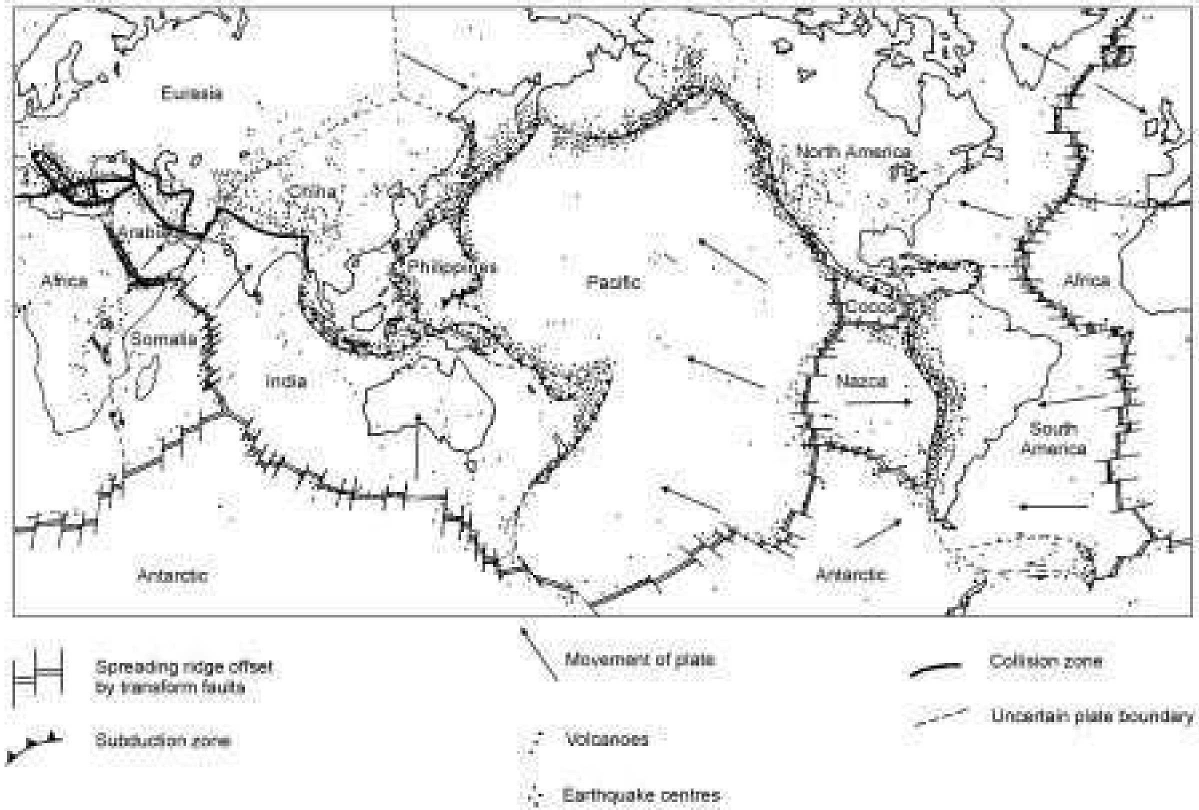
2.2.2 Lithospheric plates in motion

Lithosphere, which is composed of sial ($p = 2.65–2.7$) and sima ($p = 2.9–3.3$) is the general term given for the entire solid earth realm. Sial is the lighter granitic layer of the lithosphere that forms the continents and is rich in silica and alumina. Sima, which underlies the sial forms the floor of the oceans and is the denser mafic part of the lithosphere. The average thickness of the oceanic crust overlying basaltic lavas and intrusive gabbro is some 10–16 km. Continental crust is about 33 km thick with thicknesses of up to 50 km in high mountain regions. Figure 2.9 is a schematic representation of crust and mantle.

The first global system of lithospheric plates appeared in published form in 1968 but there have since been many minor changes and revisions. A recent diagram (Fig. 2.10) identifies seven major plates of continental dimensions and numerous minor plates ranging in size from intermediate to very small. The requirement for identifying a particular plate is good evidence of activity along all of its boundaries and of its interaction with adjourning plates. Continental plates are named for the continents embedded in them (North America, South America, Eurasia, Africa, Indo-Australia, and Antarctica). Oceanic plates are named for their oceans (e.g. Pacific, Nazca).



2.9 Schematic representation of crust and mantle.



2.10 Location of major plates of 'Pacific Ring of Fire' (derived from Mitchell and Garson, 1976).

The ‘Pacific plate’ occupies much of the Pacific Ocean and is almost entirely composed of oceanic lithosphere. Its relative motion is northwesterly so that it has an important converging boundary with the ‘North American continental plate’ along the coastal portion of California. An important smaller oceanic plate, the ‘Nazca plate’ subducts beneath the ‘South American continental plate’ leading to the formation of the Andes Mountains. A belt of oceanic lithosphere fringes the largely continental Eurasian plate on the north and east. The ‘Antarctic plate’ is almost entirely enclosed within a spreading boundary fringed by oceanic lithosphere. The continent of Antarctica comprises a central core of continental lithosphere completely surrounded by oceanic lithosphere.

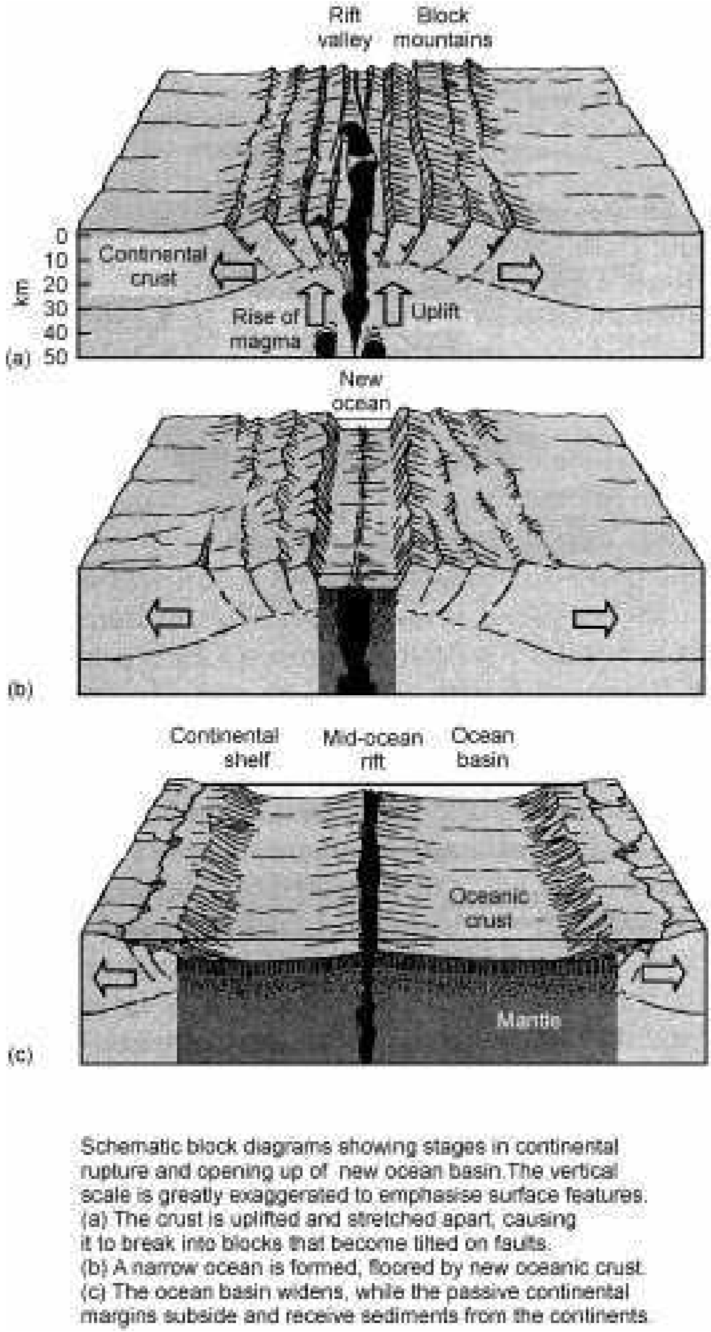
In their simplest forms, there are three types of plate boundaries divergent, convergent and transform (sliding) at which plates meet and interact.

- Divergent (extentional) boundaries are created where two plates separate and magma rises from the mantle to fill the gaps between them. New ocean crust is formed and with continued activity the ocean becomes wider.
- Convergent plate boundaries occur where plates thrust against one another with one plate bending and sinking down into the mantle to be destroyed in an oceanic trench subduction zone.
- Transform plate boundaries are where plates slide past one another without either creating or destroying lithosphere. Movement by sliding results in repositioning of different lithologies on each side of the plates and the creation of major earthquake activity along their boundaries.

Figure 2.11 shows three schematic block diagrams (Strahler, 1981) cited by Strahler and Strahler (1992) that demonstrate how continental rupture leads to the formation of an ocean basin.

Divergent plate margins

Divergent plate margins are found in both oceanic and continental settings where they occur most notably in the mid-Atlantic Ocean Ridge, East Pacific Rise in the Pacific Ocean, the Mid-Indian Ridge in the Indian Ocean and the Great Rift Valley of Africa. The Mid-Atlantic Ridge rises where American continents are separating from Europe and Africa at a late stage of rifting. Involvement of crustal magma is evidenced by the high percentage of silicic and intermediate volcanics as in the Andes and Cascades. In arc and back-arc settings, sediments are largely continental, e.g. fluvial, alluvial fan and lacustrine; and range in the fore-arc area from continental – to shallow marine – to deep. However, silicic magmas may be generated in areas where the basement is oceanic as well as where it is continental, and significant proportions of mafic magmas in continental settings are also not uncommon.



2.11 Stages in continental rupture and creation of oceanic spreading centres and related convergent plate margins (from A.N. Strahler, *Physical Geology*, Harper and Row, Publishers, Copyright © by Arthur N. Strahler).

Oceanic rifting

Magma is introduced into the crust at an oceanic divergent plate boundary (rift zone) or at any other divergent plate boundary generating ophiolite suite. Ophiolites are tectonic slices that have been tectonically emplaced during orogenesis. These slices comprise crystalline basement onto which the oceanic slice of sheeted dykes, lavas and an abyssal sediment sequence have been emplaced and post-emplacement deposits of laterite, reef limestone, or shallow marine or subaerial sediments have been developed. Seismic, gravity and magnetic studies suggest that the full stratigraphy of modern oceanic crust closely resemble ophiolite stratigraphy (Moores, 1982).

Creation of new oceanic crust begins along a string of hot spots in the Earth's crust. As a young ocean widens and matures mafic magmas rise repeatedly from the chambers, filling the rift between spreading plates and developing a layer of volcanic mountains as they spill out into the cold bottom waters at the seafloor. Hot spot volcanism at a submarine mid-ocean ridge is characterised by an axial rift (graben) in the centre of the ridge at its highest point, and is interrupted in many places by feeder dykes that cut through the crust. Enormous quantities of molten basaltic lavas are erupted from volcanoes at the edge of the rift and sometimes hundreds of kilometres to the sides of the axial rift. The process continues as the lavas move away from the spreading centre and fresh magma from the mantle is injected into the chambers

Divergent plate margins in back-arc basins are formed by small convection cells above subduction zones, fissure (flood) volcanism making the ocean basin wider. As the plate grows, the leading edge is destroyed as it descends back into the mantle. The locus of volcanism is typically confined to a line of volcanoes (some submerged and some emergent above sea level) lying parallel to a similarly linear deep-sea trench. Most 'young' island arc rocks are basalts or basaltic andesites of island arc tholeiitic type, derived either from the subducting lithosphere and/or the overlying mantle and crust.

Continental rifting

Continental divergent plate settings first appear as a terrestrial rift valley involved in the break up of some ancient landmass bordered by ocean basins. Prior to the onset of rifting the continent floats on the asthenosphere in a state of isostatic equilibrium. This passive condition is disturbed when a plume of hot mafic or ultramafic magma from deep in the mantle rises towards the surface creating a hot spot at the base of the continent. Heat from magma accumulating at the base of the continent causes lifting, stretching and pulling apart of the continental lithosphere as the result of extentional tectonics causing the crust to thin upward, crack and fracture on each side of the stressed area. A narrow basin is formed, floored by new oceanic crust (refer to Fig. 2.11). Cracks appear along rift valleys radiating away from the centre of the hot spot. Ideally there are three rift valleys that form a triple junction at 120º intervals. In practice the arms may diverge at different angles as the continent begins to split apart.

Heat rising to the surface from convection cells continues to be concentrated at the rifting centre and within a few million years the original continent is split into two continents, thousands of kilometres apart on separate tectonic plates. The newly formed continental margins are termed divergent or passive continental margins because they are no longer geologically active. They are, however, losing heat and the continental margins become denser as they move away from the rifting centre and cool. The crust sinks rapidly at first but more slowly with time and in about 5–10 million years horsts that had emerged above sea level become submerged, eventually reaching a depth of up to 14 km. The newly formed passive continental margin is now overlaid by a wedge of clastic sediment derived from the eroding continent and by carbonates resulting from chemical and biological activity. The Virginia coastal region of the USA, which was stabilised subsequent to rifting of the Atlantic Ocean almost 25 million years ago, gives an example of a modern passive continental margin.

Axial rifts that are typically tens of kilometres wide contain numerous smaller horsts and grabens of which the major graben is surmounted on both sides by horst mountain highlands 4–5 km high composed of felsic and high-grade metamorphic continental basement. The axial valley is initially on dry land except for lakes, which are created when graben floors subside, forming lakes in deep depressions. The Red Sea represents an early stage of continental rifting leading to the formation of an ocean basin. The Great Rift Valley of Africa represents an earlier stage of continental rifting.

With the East Africa Rift Zone as a model, Cas and Wright (1995) describe the effects of several uplift-updoming phases in the Cainozoic accompanied by widespread continental flood volcanism. At an early stage of ocean building in the Late Cainozoic, initial rifting and extension affected the areas of principal updoming with the development of fluvial, alluvial and lake environment within a normal-fault controlled topography. In future, more advanced rifting stages continued extension and axial graben subsidence should produce marked marine incursions leading in time to development of an overall transgressive succession. The basal parts of the succession containing volcanic materials would then slowly disappear up-sequence as the rift widens into a narrow sea with the formation of a mid-ocean ridge, such as in the Red Sea and Gulf of California. In time the axial graben subsides and fan deltas will develop as the invading sea creates a narrow marine basin. Turbidity currents will transport sediments toward the basin centre, which is still very deep and anoxic. After 10 million years or so the basin finishes filling, near shore deposition becomes the dominant process, and the process of ocean basin formation begins with a massive surge of mafic volcanic activity along one side of the axial rift.

Convergent plate margins

Since continental lithosphere is always less dense than the oceanic crust and upper mantle from which it derives, oceanic lithosphere is always subducted beneath continental lithosphere. Subduction may be either of the intraoceanic island arc type or take place along the edge of a continent (Cordilleran type). At the site of subduction, a descending slab composed of cold opiolite suite materials carries with it large quantities of seawater and is gradually heated due to the heat of friction and because of the geothermal gradient as it descends into the mantle. Fractional melting of the mantle material takes place just above the slab at a depth of about 120 km. With increasing heat, the lower temperature phases (Bowen Reaction Series) in the rock are melted to produce magmas of intermediate composition, which rise up through the crust to emplace and solidify as diorites, granodiorites, etc. The unmelted residue has a more mafic/ultramafic composition and is higher in the Bowen Reaction Series than the original rock.

Volcanic arcs form time-transgressive wider belts of volcanic rocks and associated granitoids. Small island arcs merge slowly together as the result of continual seafloor spreading; over time the continental crust becomes an amalgamation of small island arcs accreted together to form ever-enlarging continental assemblies. A string of relatively narrow (tens of kilometres wide) volcanic islands create a volcanic arc along the subduction zone that may be hundreds of kilometres long. Batholiths, emplaced early in the volcanic arc cycle, are fractionated by the heat released from later rising magmas to form alkali-granites and related rocks. The ultramafic residue, being very dense, remains in the mantle while the lighter melt rises to form intermediate and then felsic magma chambers from which magma reaches the surface as lava. Hydrothermal metamorphism occurs with the formation of pillow lavas at the interface of hot lava with seawater at the ocean floor.

Two kinds of metamorphism may occur:

- Low- to high-temperature–low-pressure metamorphism, which occurs within the volcano where rocks are subjected to heat from the batholiths, accompanied by intense folding and shearing; since the batholiths invade mafic oceanic crust, these rocks are converted into greenschist, amphibolite, and granularite facies rocks according to their proximity to the batholith and with depth.
- High-pressure–low-temperature (blue schist) metamorphism, which occurs at a convergent plate boundary where trench sediments are squeezed between two subducting plates; it is low temperature because the cool rocks are subducted too rapidly for them to have time to heat.

Japan, islands of the Malaysian Archipelago, and the Aleutian Islands of Alaska are ancient and modern examples of volcanic island arcs.

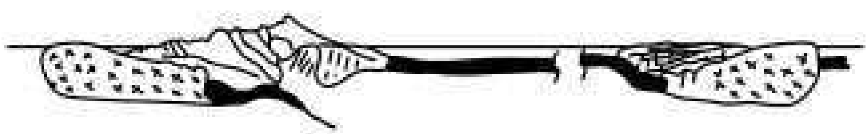
Island arc-continent plate collision

Although continents eventually collide with one another and crumple together, there is usually a prior island arc-continental collision (Fig. 2.12). During the collision the trench melange, as the first part of the island to be affected, is scraped from the descending plate and thrust up over the hinterland along a major thrust fault plane. The melange belt is all that remains of the intervening ocean basin after shearing and smearing in the suture zone between the two colliding blocks. It is also all that remains of an ocean basin that may have been thousands of kilometres wide. The hinterland volcanic island is thrust up higher into mountain peaks, which may be snow-capped. In the foreland, the thick wedge of divergent continental margin sediments is compressed and folded. Those closest to the island arc are depressed into the crust where they are metamorphosed forming marble, quartzite slate and phyllite. Volcanic activity behind the peaks may continue for a time but will eventually stop when mountain building ceases.

Cordilleran mountain building

On the seaward side of the island arc, processes of trench formation, subduction and fractional melting are the same as for island arc orogeny. In such cases all of the tectonic activity occurs along an old divergent margin that has accumulated a thick wedge of divergent continental margin sediments. This leads to the formation of another subduction zone. It may form another island arc, and repeat the island arc-continental collision process. Cordilleran mountain building assumes that decoupling of the oceanic crust will occur dipping under the edge of the continent as shown in Fig. 2.13.

The rising intermediate to felsic batholitic magma now injects into the thick wedge of continental margin sediments heating them to very high-grade meta-



2.12 Island arc-continental plate collision.

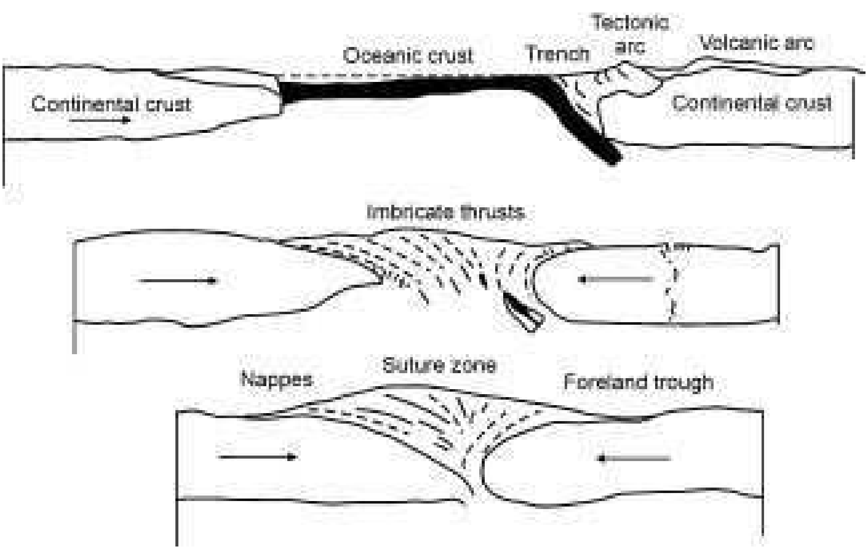
2.13 Cordilleran mountain building.

morphism. Immature sandstones and shales form phyllites, marbles and quartzites evolve from limestone and quartz sandstones. The basement batholiths are metamorphosed into gneisses and migmatites. The old divergent continental wedge of sediments and batholiths plus superimposed volcanoes is uplifted along major thrust faults to form towering mountains such as the Andes of South America and the Cascades in Northern America.

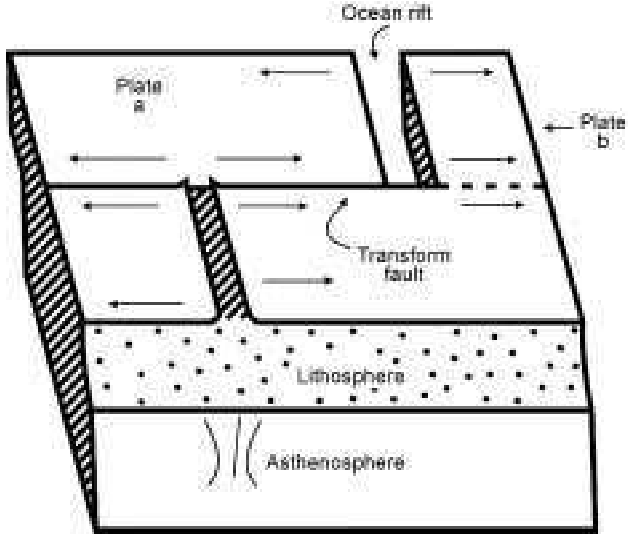
Continent-continent plate collision

Figure 2.14 is a schematic representation of a cross-section of the Earth showing the components of subduction types of convergence that result in the collision of two continents. The ocean has closed and mountain building has many of the same elements as island arc-continent collision, hinterland, foreland, suture zone, and a towering mountain range. There are, however, two foreland basin clastic wedges, one filled with sediments from a volcanic arc the other from a cordilleran mountain; and two suture zones of melange with a great variety of igneous and metamorphic rocks.

The Himalayan Mountains of India were formed as the result of collision between the Continent of India, which moved across the southern ocean to collide with the Continent of Asia. The Himalayas are still being pushed up; Mount Everest is rising two centimetres per year. Coincidentally with the rise, the Himalayas are being denuded by very severe erosion due to the extreme weather conditions.



2.14 Three stages of collision of two continents.



2.15 Diagrammatic sketch of a transform fault.

Transform plate margins

Two lithospheric plates in contact with one another along a common boundary provide conditions for the formation of a transform fault when one plate slides past the other with no motion that would cause plates to separate or converge. The plane along which a transform fault (Fig. 2.15) occurs is a nearly vertical fracture extending down through the entire lithosphere. The transcurrent motion may consist of two plates sliding past one another or pieces of the same plate moving at different speeds. They may be moving at different speeds in opposite directions or in the same direction. The boundaries are sites of intense seismic activity, with moderate to strong earthquakes. Most transform faults are found in the ocean basins, but important examples such as the San Andreas Fault in California and Mexico are found on land. The San Andreas Fault forms the boundary between the American plate and the Pacific Island plate in California. A huge rift zone, of which the Great Artesian Basin is now part, was formed along the eastern edge of Australia some 1,800 million years ago. Eyre Peninsular, Broken Hill, and Mt Morgan are all typical rift sequences.

2.2.3 Marine and continental volcanic settings

The foregoing sets the stage for examining the types and direction of the forces responsible for building large folded mountain belts, movement and growth of continents, global climatic change and the associated processes of metamorphism and plutonism. Differences in the concepts of orogeny and epeirogeny are explained as they apply to the evolution of landscapes and large-scale development of valley morphology, and the transport and deposition of sediment. Settings are provided in sequence from the generation of mafic igneous rock at oceanic divergent plate margins to intermediate and felsic rocks by fractional melting in a subduction zone.

As a starting point for gaining an understanding of the global tectonic framework, Cas and Wright (1995) divide areas of modern volcanic activity into the following tectonic settings:

- mid-ocean spreading ridge volcanism
- marginal sea-back-arc basin spreading volcanism behind oceanic island arcs such as the Marianas Trough
- intra-plate continental or flood volcanism, e.g. Cainozoic volcanism of eastern Australia
- continental rift volcanism such as the East Africa Rift zone and the Basin and Range Province of western USA
- young island arc volcanism associated with oceanic trench subduction zones as in Marianas, Aleutians and Tonga-Kermadic arcs
- micro-continental arc volcanism associated with oceanic trench subduction zones, e.g. New Zealand, Japan and Indonesia
- continental margin arc volcanism associated with oceanic trench subduction zones, e.g. Andes and Cascade volcanic belt, western USA.

Eruptions of calc-alkalic magma of andesitic to basaltic-andesitic origin at a subducting plate boundary range variably from compressive continental margins to extensional margins associated with island arc and back-arc systems. Marine stratovolcanoes differ from their continental counterparts by having a submerged foundation of oceanic or continental crust and, usually, a subaerial summit region (Cas and Wright, 1995). They have the potential for the formation of porphyry-type, copper-molybdenum-gold orebodies and higher-level epithermal gold-silver deposits. Prime exploration targets for gold-rich porphyry copper deposits are well defined in Circum-Pacific and Alpine-Himalayan regions (Sillitoe, 1993).

Ore zones located within the igneous and metamorphic rocks of Phanerozoic volcanic settings occur at various depths ranging from near surface (epithermal) vein systems, to deep-seated (mesothermal) deposits at depths of 10 km and more. Gold occurring in trace concentrations in volcanic hosted metalliferous sulphides may be chemically dissolved and re-precipitated or otherwise modified in a near-surface, oxidising environment (see Chapter 3). The following settings are of particular relevance to the genesis of gold-bearing ores.

Continental silicic volcanoes

Continental silicic type volcanoes occur typically in clusters of rhyolitic hills associated with shallow level hydrothermal systems. Epithermal deposits of mercury, arsenic, antimony, gold, silver, lead and zinc may occur both inside and outside the caldera margins and sometimes with diatremes and breccia pipes (Sillitoe, 1993). Porphyry copper-gold deposition may be associated with deeper level intrusives. Marine basaltic rift volcanism is a potential source of Cyprus type copper-pyrite mineralisation with minor associated gold. These deposits occur as massive sulphide bodies at the sediment-seawater interface and as subjacent hydrothermally altered stockworks (Rona, 1984).

Submarine felsic and silicious volcanic centres, which host Kuroko-type and Canadian Archaean base metal ore deposits occur as massive stockwork systems or base-metal sulphides, in some cases with significant gold mineralisation. Flood and valley fill non-explosive lava flows in continental basaltic settings, although not directly associated with the formation of primary gold deposits, may cover and preserve placer gold concentrations such as the Tertiary deep leads of California and Australia.

Stratovolcanoes

Stratovolcanoes are built up by eruptions of calc-alkalic magma of andesitic to basaltic-andesitic origin at subducting plate boundaries. Their settings range from compressive continental margins to extensional margins associated with island arc and back-arc systems. They have the potential for the formation of porphyry-type copper-molybdenum-gold orebodies and higher-level epithermal gold-silver deposits. Marine stratovolcanoes differ from their continental counterparts by having a submerged foundation of oceanic or continental crust and, usually, a subaerial summit region. Prime exploration targets for gold-rich porphyry copper deposits are well defined in Circum-Pacific and Alpine-Himalayan regions (Sillitoe, 1993).

Rhyolitic calderas

Multiple eruption points characterise rhyolitic volcanic centres, which occur typically in clusters of rhyolitic hills associated with shallow level hydrothermal systems. Epithermal deposits of mercury, arsenic, antimony, gold, silver, lead and zinc may occur both inside and outside the caldera margins and sometimes with diatremes and breccia pipes (Sillitoe, 1993). Porphyry copper-gold deposition may be associated with deeper level intrusives.

Rhyolitic volcanic centres

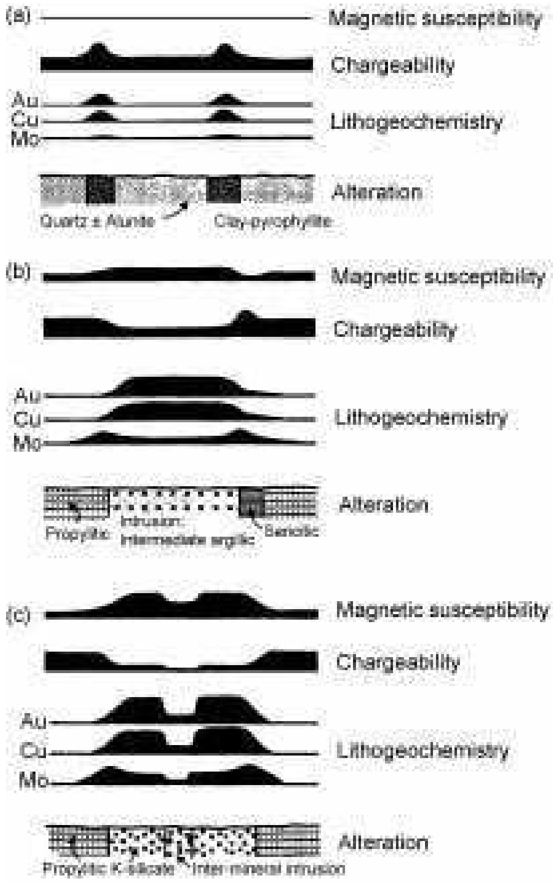
Submarine caldera types are sites for the formation of Kuroko-type massive stockwork systems of base-metal sulphides, in some cases with significant gold mineralisation (Ohmoto, 1978; Ohmoto and Takashati, 1983). Miocene mineral-isation in the Kuroko area of Japan is associated with rhyolite lava domes, and volcaniclastic rocks believed to have been emplaced at minimum water depths of 1000 m (Ohmoto and Takashati, 1983).

Observation of the above modern tectonic settings within which volcanism occurs provides a reasonably clear understanding of the overall regional geological framework of the volcanics concerned, combined with a consideration of the original tectonic controls that allowed volcanism to occur. However, caution is needed in trying to use modern global settings as analogues for past tectonic configurations. Ancient volcanic and tectonic settings offer little chance of such unambiguous clarification because of the effects of repeated deformation and magmatic intrusion over very long periods of geological time and masking of the original tectonic context by the chemical overprints of metamorphism and alteration. Furthermore, in Precambrian terraines where volcanic successions may have been associated with the eruption of similar type basaltic, andesitic and rhyolitic lavas as for modern volcanics, some of the larger scale tectonic controls, processes and settings would probably have been quite different. The continents have continually grown in extent since their earliest beginnings by the accumulation of felsic rock produced in subduction zones, and critical evaluation may be lacking of whether all the essential dynamically important tectonic elements of the modern analogue can be found in ancient configurations (Cas and Wright, 1995).

2.2.4 Minerals distribution through time

Metallogenic gold provinces usually contain clusters of similar type deposits or areas of host rock associations in which mineralisation has been active at one or more intervals of geological time. In summarising the distribution of minerals through time, Barley and Groves (1992) propose a relatively simple explanation for the uneven distribution of a wide range of deposit styles. They suggest that this could be provided by tectonic cyclicity resulting from the interaction between large continents, such as those that merged together in the formation of Pangaea, and mantle convection, combined with decreasing global heat flow. Since cyclicity in orogenic and anorogenic activity has important implications for eustacy, seawater chemistry and biotic activity (Worsley *et al.*, 1985), and for the generation and distribution of continental crust both the spatial and temporal distribution of metal may be affected. Figure 2.16 plots the distribution in geological time of some important types of mineral deposit from Meyer (1981, 1985), Lambert and Donnelly (1992) and Barley and Groves (1992). Meyer relates the uneven temporal distribution of metallic mineralisation to three major factors:

- evolution of the hydrosphere-atmosphere, secular decreases in global heat flow, and long-term tectonic trends
- evolution of the hydrosphere-atmosphere as the cause of restricted distribution of deposit styles (e.g., banded iron formations of the Precambrian) for

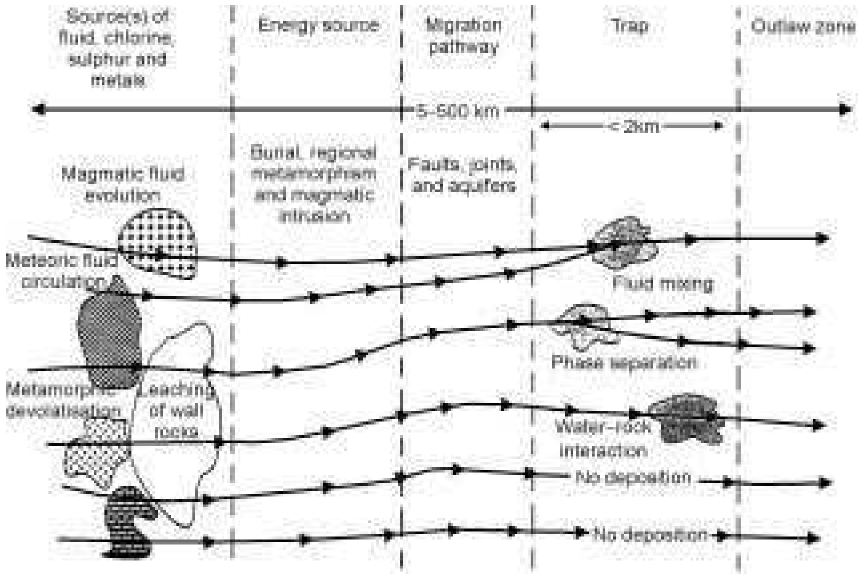


2.16 Uneven distribution in geological time of three important types of mineral deposit (derived from Meyer (1981), Lambert and Donnelly (1992) and Barley and Groves (1992)).

which the transport of metal is strongly dependent upon oxidation-reduction (e.g. Holland, 1984)

- secular decreases in global heat flow as responsible for the restriction of Kambalda-type, komatite-associated Ni deposits to the Archaean and Early Proterozoic; Archaean greenstone is typically associated with Precambrian gold deposition; continental silicic volcano arc successions host precious and other epithermal metal deposits; and porphyry copper-gold deposits occur with deeper-level intrusives.

The abundance of gold and volcanic-hosted massive sulphide and Kambalda-type komatite-associated Ni mineralisation in late Archaean greenstone belts and sedimentary basins may have been due to rapid growth and stabilisation of continental crust with high global heat flow. Important peaks in the abundance



2.17 Components of a mineral system (Huston, 1997, as modified from Knox-Robinson and Wybourn, 1997).

of sediment-hosted Pb-Zn and Cu minerals during the formation of Pangaea (400–200 mya) are interpreted to have developed in either epi-continental or foreland settings of the Caledonian-Appalachian Orogen (e.g. Duane and DeWit, 1998). From 200 mya to present, i.e., following the break-up of Pangaea, most types of gold deposits, including volcanic hosted massive sulphides, podiform chromite porphyry style Cu and Mo mineralisation occur in island arcs and basins along Pacific margins (Sawkins, 1990).

Important factors controlling the geometry and chemical behaviour of large-scale flow of hydrothermal fluids within the crust are illustrated in Fig. 2.17. The ore deposit-ligand source is suggested as an essential basis for all hydrothermal system models. Linkages between the various independent factors that control the overall geometry and the physical and chemical behaviour of large-scale flow of hydrothermal fluid in the crust are of fundamental importance; possibly more so than a detailed understanding of any one of them (Heinrich *et al.*, 1989b).

It is generally recognised that periods of major hydrothermal fluid flow and hence mineralisation are related to movement of the controlling faults and opening of dilatational structures within such fault systems. The controlling structures remain either weakly or non-mineralised and auriferous vein deposits are commonly found in normal or strike-slip systems perpendicular to the fault slip vector (Sibson, 1987). Amongst the best known are the shear zone deposits of Yellow Knife (N.W.T), Golden Mile, Kalgoorlie, Australia and the Mother Lode, California.

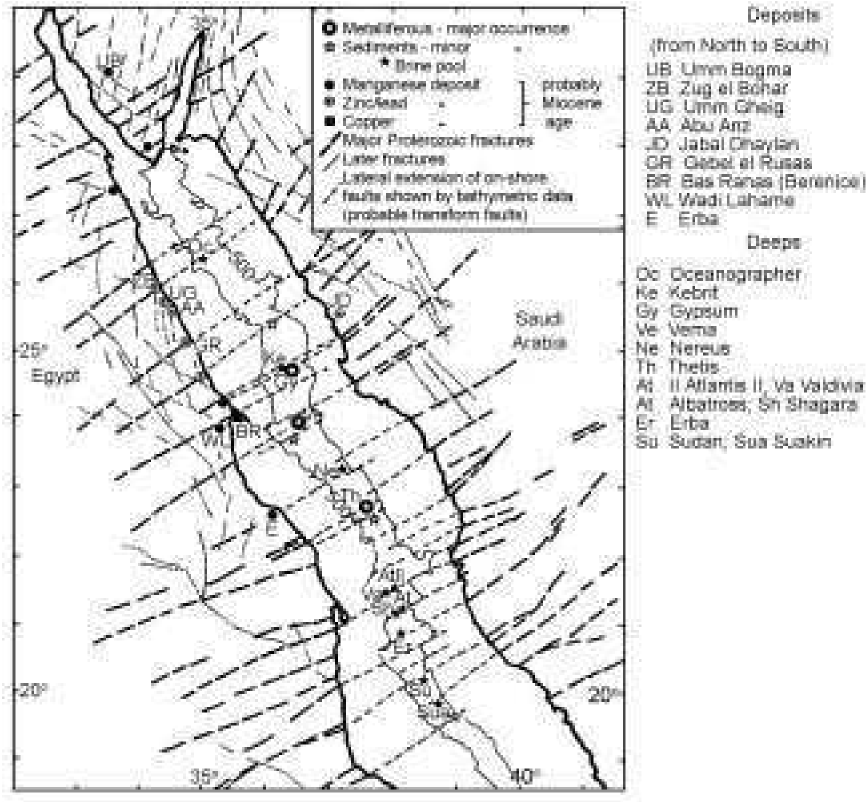
Discoveries of evolving gold-bearing polymetallic sulphide deposits in marine settings, made initially as a result of submarine investigations at mid-ocean ridges, have similarly been extended to other parts of the marine environment including volcanic and tectonic settings in shallow marine, island arc environments. Smaller marginal basins developed between island arc and nearby continental masses are somewhat analogous to the larger oceans in having spreading axes which involve separation of the lithosphere and upwelling of magma from the mantle to form new oceanic lithosphere. Differences in chemical composition of seafloor metal-rich sulphides and their precious metal contents are shown in Table 2.1 in relation to samples derived from a range of mid-ocean ridges and back-arc spreading centres.

A polymetallic sulphide ore body located in the Atlantis Deep, due west of Mecca in the Red Sea, represents the style of mineralisation that may occur at an early stage of seafloor spreading during the opening of an ocean basin (Fig. 2.18). The metalliferous muds of this deposit are estimated to contain 30 million tonnes of iron, zinc and copper, some 6,000 tonnes of silver and 50 tonnes of gold at an average grade of about 2 ppb Au (Rona, 1986). The Red Sea deposit

Table 2.1 Bulk chemical composition of seafloor polymetallic sulphides from mid-ocean ridges and back-arc spreading centres

	Mid-ocean ridges		Back-arc ridges	
	Volcanic-hosted[1]	Sediment-hosted[2]	Intraoceanic[3]	Intracontinental[4]
N	890	57	317	28
Fe (wt%)	23.6	24.0	13.3	7.0
Zn	11.7	4.7	15.1	18.4
Cu	4.3	1.3	5.1	2.0
Pb	0.2	1.1	1.2	11.5
As	0.03	0.3	0.1	1.5
Sb	0.01	0.06	0.01	0.3
Ba	1.7	7.0	13.0	7.2
Ag (ppm)	143	142	195	2766
Au	1.2	0.8	2.9	3.8

[1] Explorer Ridge, Endeavour Ridge: Main Vent and High Rise Fields, Axial Seamount: AHSES and CASAM, Cleft Segment: N and S Fields, East Pacific Rise: 11°N, 13°N, 21°N, 7°30′S. 16°45′S. 18°30′S. 21°S. Galapagos Rift, TAG: Active Mound, Mir and Alvin Zones, Snake Pit Mid-Atlantic Ridge 24.5°N.
[2] Escanaba Trough, Guaymas Basin.
[3] Mariana Trough, Manus Basin, North Fiji Basin: Kings Triple Junction, White Church, Vai Lili, Hine Hina Fields.
[4] Okinawa Trough,
Data compiled by Geological Survey of Canada and Freiberg University of Mining and Technology, Germany.



2.18 Distribution of metalliferous sediment and brine pools in the Red Sea at the early stage of seafloor spreading (derived from Mitchell and Garson, 1976).

ranks with the larger ancient orebodies found on land and is typical of an ore formed on the seabed, which may eventually be recycled in a subduction zone or obducted to form accreted terrain.

Potential source rocks on the seafloor range from mid-ocean ridge type basalts and clastic sediments to lavas of intermediate composition (basaltic-andesites, andesites) in intraoceanic back arcs. Felsic volcanics (dacite) are typical for young intracontinental back-arc rifts. Samples from white-smoker chimneys in the Lau back arc contain the first examples of visible primary gold (up to 18 ppm) discovered in polymetallic sulphides at active vents. Other finds include a deposit with epithermal characteristics in the crater of a submarine volcano 2,200 m below sea level in the Woodlark Basin, where seafloor spreading interacts with the continental crust of Papua New Guinea. Gold-rich (up to 24 ppm Au) polymetallic concentrations with veins averaging up to 3.8 ppm Au have also been identified in rifted continental crust resembling Kuroko type massive sulphides in the Okinawa Trough. Preliminary analyses of sulphides in the Eastern Manus Basin indicate an average gold content of more than 30 ppm Au with maximum 54 ppm Au. In contrast, sulphide deposits related to mature back-arc spreading centres associated with mid-ocean ridge basaltic type volcanics (e.g. North Fiji Basin, Mariana Trough) have gold contents of only 0.1–4.3 ppm Au, similar to deposits on mid-ocean ridges.

Volcanic dominated sediment-free deposits, e.g. high-temperature (350 ºC) black smoker chimneys composed of Cu-Fe sulphides typically contain less than 0.2 ppm Au whereas the 350 ºC member fluids contain about 100 to 200 ppm Au in solution (Hannington *et al.*, 1991). Hydrothermal reworking of gold during sustained venting of the mineralising fluids through the sulphide mounds may provide local enrichment comparable to that found on land, but sampling is at a very early stage. Because of sampling difficulties during submerged operations, most values reported from mid-ocean ridge deposits have been obtained from sulphide chimneys at the surface of the seabed. Such values are unlikely to represent the underlying interiors of the massive sulphide bodies from which only a few samples have been taken, none from any systematic deep drilling programme. More information may be obtained from a current feasibility study, which is investigating the possibility of mining Pacmanus deposits at a depth of some 1,600 m in the Eastern Manus Basin.

The likely preservation of Phanerozoic gold deposits is influenced by whether they become parts of peripheral or interior orogens. Peripheral orogens form adjacent to an ocean external to the continent (Murphy and Nance, 1991). As with the Pacific during the Phanerozoic, they do not undergo major continental collisions and are favourable for the preservation of richly mineralised, low metamorphic-grade island arc and marginal terraines. Deposits such as granite-associated and mesothermal Au-Cu mineralisation that form in the upper crust of continental magmatic arcs are locally preserved. Interior orogens (e.g. the Himalayas) on the other hand undergo intensive crustal thickening as the result of continental collision during ocean closure. Uplift and exposure of medium to high-grade rocks to rapid weathering and erosion gives them a much lower preservation potential.

With the general acceptance of seafloor tectonic and hydrothermal processes as being either analogous to or integral parts of geological processes operating in the continental crust, proposals postulating a seafloor origin for many land-based deposits now seem more plausible. For example, an overall picture of Mesozoic orogenic mountain building episodes may be developed from the identification of assemblages such as opiolites, blue schist series and melange characteristic of former plate margins or suture zones between major plates. Discovery of visible primary gold in white smoker chimneys at active vents in the Lau back arc has shown that seafloor hydrothermal processes are in many ways similar to hydrothermal processes involved with the development of some epithermal gold-only ores in volcanic arcs.

The technical feasibility of sea floor mining at depths up to several thousands of metres has been shown by a number of multi-national companies. Competitive proposals for dredging polymetallic nodules from the deep ocean floor were reviewed by Macdonald (1987) in the McGraw-Hill *Yearbook of Science and Technology*. Possible economic exploitation of the Red Sea muds has also been studied in small, pilot-scale testing programmes. However, problems of marketing, materials handling and economic feasibility have yet to be overcome and none of these proposals has yet been tested at a commercial scale. Not the least of the problems is the requirement of a very large, long-term and stable market for all of the contained metals. Studies of the feasibility of dredging gold-rich polymetallic deposits in back-arc basins are faced also with much more difficult sampling problems than those encountered in sampling polymetallic nodule and mud deposits (see Chapter 7).

2.3 Hydrothermal gold systems

Until the 1960s, and the advent of plate tectonic theories, the empirical approach to hydrothermal ore formation was based generally upon experiments that simulated natural conditions in trying to deal with the complexities of geochemical processes. Lindgren (1933) classified mineral deposits according to depth and ore deposits were classified in accordance with temperature–pressure relationships within the crust as 'epithermal', 'mesothermal' and 'hypothermal' depending upon the crustal level of ore formation. Since that time, the development of new analytical techniques such as fluid inclusion and stable isotype analysis has allowed much greater insight into the understanding of hydrothermal fluid evolution within the ore-forming environment. Sophisticated computer codes have been developed to simulate depositional processes such as cooling, boiling, fluid mixing and water-rock interaction, and to couple these with simulations of fluid flow in porous media.

Significantly large gold deposits require the coincidence of particularly favourable processes and source parameters. A critical factor is the degree of element concentration during ore formation. It is now recognised that hydrothermal mineral deposits are ultimately the result of chemical reactions and that they are localised by zones of higher permeability (e.g. faults and aquifers). Huston (1997) classifies these processes and the regions in which they occur as 'mineral systems'. The characteristics (e.g., T, P, pH, salinity, redox, sulphur content) of hydrothermal fluids defined by these processes within these systems determine the metal-carrying capacity of the fluid. Three general groupings of mineral deposits that contain Cu, Zn, Pb, Ag, and/or Au are:

1. Zn-Pb-Ag + or − Au,
2. Cu + or − Au,
3. Au + or − Ag.

However, some deposits are zoned and in these deposits individual zones are characterised by different assemblages. Huston warns that dynamic modelling requires both a high degree of computer literacy and a firm understanding of the basic fundamentals of thermodynamics. The programs, which are written to solve a particular set of problems, may not be able to cope with the particular geological program involved. Composition and mineralogical characteristics of the erupted magma are the end result of a complex history of processes causing chemical and physical change. Widely different geological histories include the degree of partial melting of the source rocks and other melting events, the amount of contamination by wall rock, periodic replenishing, tapping and fractionating of magmas in a succession of magma chambers as they rise to the surface.

2.3.1 Partial melting

The relevance of partial melting to the differentiation of gold-enriched silicic magmas has been demonstrated experimentally. It is shown that quartz-rich, water-rich silicic magmas such as those that produce quartz veins can be synthesised by partial melting of a mixture of wet sediments, volcanics and other materials as provided by a subducting plate. The composition of each melt varies according to the composition of the original mixture and the composition of individual partial melts produced sequentially at different temperatures and pressures.

It follows that the formation of similar type magmas may occur at certain critical depths in a subduction zone under similar conditions of mineralogy, temperature and pressure. Basaltic magma produced by partial melting of upper mantle material pushes and stopes its way up through the overlying crust. Material broken off or melted from the surrounding rock is assimilated into the rising magma by processes involving magma mixing, immiscibility and thermal segregation. Localised heat and pressure differences in magma chambers formed at higher stratigraphic levels provide complex and rapid changes in lithology, thus resulting in melt-crystallisation relationships producing differentiation of different rock types. Such rising magmas are less dense than mantle fluids because of their increasingly silicious nature; hence their tendency to rise buoyantly as intrusive and extrusive bodies into the arcuate systems of volcanoes and mountain belts developed at both divergent and convergent plate settings.

Partial melting of upper mantle sulphides contributes metals to crustal fluids that rise into the crust. Andesitic varieties are of major importance because of their common association with gold ores. All known mineralisation in the southwestern Pacific areas lies within regions of predominantly andesitic rocks (Liddy, 1972). The andesitic magma forms an array of island arcs that stretch along a belt of Mesozoic-Cainozoic mountain building in major zones of crustal instability. Intrusion-associated porphyry copper deposits, epithermal gold and

Table 2.2 Representative hydrothermal ore deposits associated with subduction related magmatism (after Hedenquist and Lowenstern, 1994)

Ore deposit type	Relation to magma	Temperature depth	Fluid	Associated metals	Example of active analogue
Porphyry	Adjacent to or hosted by intrusion	>600 to 300 °C 2–5 km	Hypersaline and immiscible vapour	Cu±Mo±Au, Mo, W or Sn	Shallow magma bodies beneath stratovolcano
Skarn	Adjacent to intrusion in carbonate rock	400–600 °C 1–5 km	Saline to moderate	Fe, Cu, Sn, W, Mo, Au, Ag, Pb-Zn	Shallow magma bodies beneath stratovolcano
Pluton-related veins	Fractures in and near intrusion	300–450 °C Variable	Moderate to low salinity	Sn, W, Mo±Pb-Zn, Cu, Au	Shallow magma bodies beneath stratovolcano
Epithermal (high sulphidation)	Above parent intrusion	<300 °C Near surface to >1.5 km	Moderate to low salinity, early acidic condensate	Au-Cu Ag-Pb	High-temperature fumaroles and acidic springs near volcanic vent
Epithermal (low sulphidation)	Distant (?) from magmatic heat source	150–300 °C Near surface to 1–2 km	Very low salinity, gas-rich, neutral pH	Au(Ag, Pb-Zn)	Geothermal systems with neutral pH hot springs, mud pools
	Distant (?) from magmatic heat source	150–300 °C Near surface to 1–2 km	Moderate salinity	Ag-Pb-Zn(Au)	Not observed, transient brine?
Massive sulphide	Near extrusive domes	<300 °C on or near sea floor	Near seawater salinity, gas-rich	Zn-Pb-Ag (Cu or Au)	Back-arc seafloor vents, black smokers

The term ‘fluid’ is used to refer to non-silicate, aqueous liquid and/or vapour. The salinities (Na, K chloride) of fluids in these environments vary from hypersaline (>50 wt%) to moderate (10–20 wt%), low (<5 wt%) and very low (0.2–0.5 wt%) salinity.

shallow formed epithermal and base metal deposits as shown in Table 2.2 are representative of hydrothermal ore deposits associated with subduction related magmatism (Hedenquist and Lowenstern, 1994).

Formed by gradual igneous replacement of country rock at great depths or by stoping during a mountain building period, batholitic intrusions provide important long-term sources of heat energy for the large-scale circulation of hydrothermal fluids. There is, however, some doubt that such large parent bodies can themselves host significant gold-bearing vein systems. Batholiths located along the gold-bearing cordillera of North and South America do not show any direct relationship with gold deposition. More suitable conditions for folding and faulting and the supply of heat for hydrothermal deposition are provided by smaller diapiric intrusions injected into relatively cold country rock. Metallogenic gold provinces usually contain clusters of similar deposits or areas of host rock associations in which mineralisation has been active at one or more intervals of geological time. The most productive areas of the Californian goldfields appear to be closely related to multiple smaller intrusions and/or apopheses of the larger parent granitic bodies.

2.3.2 Fluid phase

Hydrothermal fluids are of several different types. Magmatically related fluids emanate from felsic to intermediate igneous intrusives during late stages of cooling and crystallisation. Connate fluids comprise water trapped in the pores of sedimentary rocks or in the joints of igneous rocks. Metamorphic water, incorporated in silica structures, is released from rocks undergoing metamorphic change. Meteoric waters include rainwater, river water, lacustrine water and ground water. The metal complexing capacity of brines leads to a much greater exchange of chemicals between seawater and mantle basalts than occurs between meteoric waters and basaltic magmas.

According to recent studies, magmatic fluids are present in most active hydrothermal systems but are not always recognised because of masking by the large volume (>95%) of meteoric fluids convected during cooling of an intrusion at shallow depths in the crust (Cathies, 1991). Temperature, salinity and redox conditions are critical factors for a range of deposits. Tables 2.3 and 2.4 show temperature-salinity and temperature redox conditions respectively for a range of hydrothermal gold ore types. Meteoric and fluid inputs can usually be traced from oxygen and seawater hydrogen isotopes.

Despite the apparent dominance of magmatic water in the formation of many hydrothermal ore deposits, meteoric fluids are widespread throughout the crust and under suitable conditions gold and other soluble minerals may be leached from the rocks through which they pass. Rocks with high clay content and low in carbonates, carbon and pyrite are good sources of gold, the content generally ranging from about 0.005 to 0.01 ppm Au with some carbonaceous shale having

Table 2.3 Temperature–salinity conditions for selected hydrothermal ore deposits (after Huston, 1997)

Temperature	Salinity	Ore type
High (>300 °C)	High (>10 wt% NaCl)	Porphyry Cu-Au-Mo
Low (<200 °C)	High (>10 wt% NaCl)	MVT; Shale-hosted Pb-Zn-Ag
Moderate (200–300 °C)	Moderate (5–10 wt% NaCl)	VHMS Cu-Pb-Zn-Ag-Au
High (>300 °C)	Low (0–5 wt% NaCl)	Mesothermal Au
Moderate (200–300 °C)	Low (0–5 wt% NaCl)	Epithermal Au-Ag

Table 2.4 Relationships between temperature and redox conditions for selected hydrothermal type deposits (after Huston, 1997)

Redox conditions	Gangue	Temperature	Ore type
Highly oxidised (SO_4-rich fluids)	Hematite	Moderate (200–300 °C)	Cu-U-hemitate
Oxidised (SO_4-rich fluids)	Magnetite/ hematite	High (>300 °C)	Cu-Au-Fe oxide
Oxidised (SO_4-rich fluids)	Hematite/ pyrite	Low (<200 °C)	Sediment-hosted
Reduced (H_2S-rich fluids)	Pyrite	Moderate (200–300 °C)	Au + pyrite; no Cu
Highly reduced (H_2S-rich fluids)	Pyrrhotite	High (>300 °C)	Au-Cu-pyrrhotite

many times that value. The average values of selected country rocks from various sources are shown in Table 2.5.

Fluid composition

The relative concentrations of components such as sodium, potassium, calcium and magnesium reflect the equilibrium of fluid and rock within the hydrothermal system and can be used as geothermometers during geothermal exploration and exploitation (Giggenbach, 1992). Fluid compositions in steam heated waters in the surficial zone of active systems are strongly to moderately acid. Sulphate (derived from near surface oxidation of H_2S) or bicarbonate dominate over chloride as the principal anion (Heinrich, 1989a).

Phase separation resulting in the evolution of a gaseous systems phase is an important factor in gold ore formation due to significant changes to the chemistry of the residual brine, mainly through loss of volatiles such as H_2S, CO_2 and H_2 and cooling of the residual brine. H_2S loss causes Au deposition in dilute, low to moderate temperature, near-neutral fluids transporting Au as thio-complexes, for example in adularia-sericite epithermal systems.

Table 2.5 Average values for gold in country rocks (from various sources)

Rock type	Number of analyses	Weighted average, all analyses
Plutonic rocks		
Salic		
Granite	310	1.7
Granodiorite	380	3.0
Aplite (including dykes and veins of granite and greisen)	23	4.2
Intermediate		
Diorite, quartz diorite, monozonite, quartz monzonite, tonalite	261	3.2
Mafic		
Gabbros (including gabbro-diorite)	580	4.8
Ultramafic		
Mainly dunite and peridotite	149	6.6
Volcanic and hypabyssal rocks		
Salic		
Mainly rhyolite (including rhyoandacite, dacite, felsite, latite and obsidian)	188	1.5
Mafic and intermediate		
Mainly basalt, andesitic basalt, andesite (51) trachyandesite and trachyte (2)	696	3.6
Sedimentary and metasediments		
Rock type	Au (ppb) average	Au (ppb) high
Detrital		
Sandstone	7.5	
Shale	3.9	67.7
Metasediments		
Gneiss	1.8	
Schist	5.0	

Although some authors, e.g. Symonds *et al.* (1987), doubt the importance of the metal content of gases in relation to ore formation, Meeker (1998) has since estimated from snow analyses that about 1 kg of gold per day was emitted as volcanic gas from Mt Erebus (Antarctica) in 1986. At this rate, the emission of gold would amount to a million oz. in only 83 years. Estimates of the amounts of copper and gold discharged over short intervals of time from a number of hydrothermal systems including the White Island volcanic-hydrothermal system are listed in Table 2.6.

Modern analytical techniques now permit examination of ancient hydrothermal solutions trapped as fluid inclusions in such hydrothermal minerals as quartz, sphalerite and calcite. Detailed fluid inclusion studies have been applied to epithermal ore formation in both modern and ancient volcanic settings. Exploration

Table 2.6 Magmatic fluids discharged from volcanoes (after Hedenquist and Lowenstern, 1994)

	White Island NZ	Satsuma Iwojima Japan	Augustine	Alaska	Mt Etna Italy
Year	1988	1990	1976	1987	1975, 1987
Style of discharge	Eruption	Fumarole	Eruption	Fumarole	Eruption
Temp. (°C)	>859	877	>900	870	900
Flux (10^6 t yr^{-1})					
H_2O	1.9	5.2	ND	0.03	50
CO_2	0.5	0.04	ND	0.003	13
Cl	0.04	0.06	>0.6	0.005	0.1–0.5
S	0.06	0.09	>0.2	0.005	0.2–0.75
Cu (t yr^{-1})	110	0.16	>1200	0.011	480–580
Au (kg yr^{-1})	>36	0.02	ND	ND	80–1,200
Abundance ratios*					
Na/S ($\times 10^3$)	55	0.08	50–800	0.4	40
Cu/S ($\times 10^6$)	2,400	1.9	2,300–4,700	2.3	1,000

ND, not determined.
* For comparison with the erupted aerosoil, the Na/S and Cu/S of fumaroles at White Island are 0.7×10^{-3} and 2.8×10^6, respectively.

data for epithermal deposits (Berger and Eimon, 1983; Henley and Ellis, 1983; Henley, 1985; Heald *et al.*, 1987) show that such deposit types were also formed in ancient environments similar to presently forming active geothermal systems in volcanic regions. Comparison with older systems is made possible through the use of other techniques such as proton-induced X-ray emission and electron-microprobe. The information gathered has led to a better knowledge of ore formation and to the development of more plausible exploration models.

Volume of flow and time

Properties of competence and structure are the two most important factors affecting the movement of fluids through host rocks. Competence describes the physical response of rocks to the application of external forces. The more competent rock types tend to fracture under stress, thereby providing networks of passages for hydrothermal circulation. Less competent rock types, i.e. rocks having low resistance to shearing, tend to deform rather than fracture under stress and are thus unlikely permeable structures within which vein formation can occur. Approximate ranges of permeability of selected strata are given in Table 2.7.

Cleavage and schistosity make some rocks particularly susceptible to the passage of fluids. Slate, schist and phyllite all provide good structural control when brought into contact with actively rising intrusives. These rocks are all represented in the provenances of most of the major alluvial goldfields of the

Table 2.7 Approximate ranges of permeability of strata

Rock type	Permeability (Darcy units)
Clay, shale or dense rock with tight fractures	0.0001
Dense rock with few tight fractures	0.001
Dense rock, 0.13 mm fractures	0.5
Silt or fine sand	1
Dense rock and highly fractured	2
Sand	500
Gravel	1250

world. Wotjik (1984) (Table 2.8) lists 17 fields in which at least one of the above rocks is represented. Sedimentary interceptors have been recognised in all but three of the seventeen districts.

Whilst a mix of good physical and chemical properties is clearly desirable, very low grade source rocks can host major gold deposits given sufficiently large volumes of flow and time for the reactions to take place. The required volume of flow to leach and transport specific amounts of gold in solution can only be approximated because of factors such as dispersion into underground aquifers, but is clearly of enormous proportions. Some indications of flow rates have been obtained from measurements of total discharge of fluid from modern hot spring gold-bearing systems. Estimates of discharge rates approximating 75 kg/s from the Broadlands geothermal system, New Zealand (Heinrich *et al.*, 1989b) suggest that the system could have transported a minimum of 2.3 million ounces of gold to the surface during the past 10,000 years. Since only about one-tenth of large-scale fracturing this can be accounted for, the remainder could be disseminated in the subsurface 200–300 m of the system or perhaps located in some fracture zones not tested by drilling.

Until recently, deposition was believed to be a slow process. Radiometric studies show that millions of years separate the formation of host volcanic rocks from subsequent mineralisation. For example, host rocks in the Hauraki region of New Zealand are dated at 17 Ma whereas hydrothermal alteration associated with the mineralised veins gives an age ranging from 2.5 to 7.0 Ma (Robinson, 1974). Considerable evidence now supports the episodic nature of epithermal ore deposits thus suggesting that the ore forming fluids responsible for mineralisation may be present for only short periods in the very long life of a hydrothermal system. These periods probably correspond with specific tectonic or hydrothermal events (Hedenquist and Lowenstern, 1994).

2.3.3 Alteration assemblages

Paragenetic relationships also extend to alteration assemblages for which the main control appears to be the nature of the host rock lithology. Such relation-

Table 2.8 Gold placer districts of the world (from Wotjik, 1984)

District	Rock type	Age	Intrusive	Sedimentary interceptors
California Sierra Nevada	Slates, phyllites schists, hornfels limestones	Upper Palaeozoic	Granodiorite Diorite Aphite	Eocene Pliocene Pleistocene
California Oregon	Phyllite, chert metavolcanics limestone	Upper Palaeozoic Mesozoic	Granodiorite	Pliocene Pleistocene
Georgia	Mica-schist gneiss	Precambrian Palaeozoic	Diorite Granite	Not recognised
Guyana Fr. Guyana Surinam	Greenstones, slates, schists	Precambrian	Granite	Not recognised
Klondike Yukon, Can.	Schists	Precambrian	Granite Quartz Monz.	Eocene Pliocene
Yukon Alaska	Schists phyllites	Precambrian	Granite Quartz Monz.	Eocene Pliocene
Nome Alaska	Schists, slates phyllites	Precambrian	Granite Quartz Monz.	Not recognised
Cariboo B.C., Can.	Phyllites quartzite, slate	Lower Palaeozoic	Granite Granodiorite	Pleistocene
Choco Colombia, S.A.	Greenstones slates, phyllites quartzite	Palaeozoic	Granite Quartz Monz.	Pleistocene
Antiquia Colombia, S.A.	Slates schists	Palaeozoic	Granite Quartz Monz.	Pleistocene
Victoria N.S.W. Australia	Shales, slates graywackes sandstones	Palaeozoic	Granite	Eocene to Pliocene
Otago New Zealand	Schists	Mesozoic	Granite	Miocene to Pleistocene
Nelson New Zealand	Greenstones greywackes schists, slates	Mesozoic	Granite	Miocene to Pleistocene
Morobe New Guinea	Argellites schists, quartz por.	Palaeozoic	Granodiorite	Miocene Pliocene
China Mongolia	Schists gneisses	Not reported	Granite	Not reported
Lena River U.S.S.R/	Slates greywackes phyllites, schists	Precambrian Lower Palaeozoic	Granite Quartz, spar porphyry	Pliocene Pleistocene
Yakutia U.S.S.R.	Granularites, schists	Archaen pre-Mesozoic	Granitoid	Pre-Quaternary

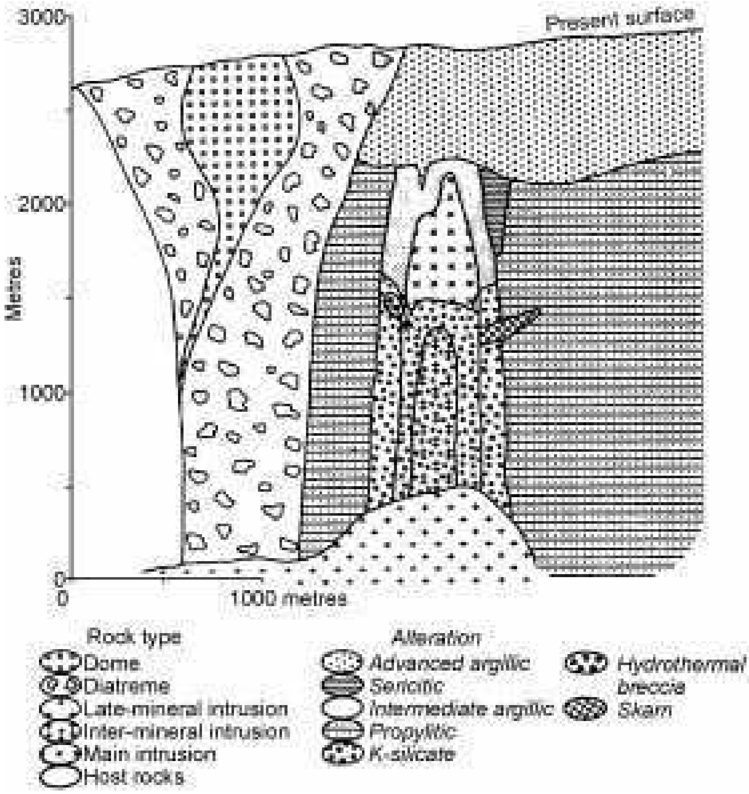
ships, if studied in conjunction with other exploration data, give useful information about the type of ore-forming fluid (pH, Eh, etc.) as a guide to identifying lateral and vertical zoning around individual veins. The intensity of the hydrothermal alteration varies from minimal in ores hosted by arenaceous wackes, to intense in some intermediate to mafic volcanics and ultramafics (Nesbitt and Muchlenbachs, 1989).

Ore-associated alteration is produced at decreasing temperatures both towards the surface and at increasing distances from the fluid conduits. This alteration mineralogy can be observed and the temperature measured in active systems to indicate the thermal stability of temperature alteration minerals (Henley and Ellis, 1983; Reyes, 1990). Information can then be deduced from the distribution of the alteration minerals to determine the locus of palaeo-flow and level of erosion. This is significant because most epithermal gold ores occur in conduit zones over the temperature range 180–280 ºC, equivalent to depths below the palaeo-water table of about 100 m to 800–1,500 m (Hedenquist and Lowenstern, 1994).

The nature of the alteration mineralogy changes with depth, thus suggesting that current levels of erosion can be predicted from the exposed rock. In high sulfidation systems, the ore is associated with a zone of intense acid alteration and is surrounded by mineral assemblages indicating less acid conditions. The gold in low sulfidation systems is typically transported as a chloride complex with different controls on precipitation, such as dilution and/or cooling (Giggenbach, 1991; Hedenquist, 1992). The ore is associated with the least-acid alteration and the gold is likely to be transported in solution as a bisulfide complex. Hence, epithermal orebody exposures with alteration assemblages indicating low palaeotemperatures are encouraging for primary gold exploration. Orebody exposures indicating alteration palaeotemperatures approaching 280 ºC are indicative of a long period of erosion with the possibility of significant placer formation.

Gold-rich porphyry deposits are generated in volcanic-plutonic arcs in both island arc and continental arc settings. They are associated with stocks and volcanic rocks ranging in composition from low potassium calc-alkalic, through high potassium calc-alkalic to potassium alkalic (Sillitoe, 1993). Introduction of the copper and gold took place mainly during potassium silicate alteration (± biotite, magnetite, amphibole and calcic minerals). High fineness gold is present in veins, stockworks and disseminations, mainly in zones of potassium silicate alteration but in some cases, with advanced argillic alteration as in the Phillipines at Santo Thomas 11 and Guinaoang. Hydrothermal magnetite is abundant in most gold-rich potassium silicate zones. Figure 2.19 illustrates the generalised intrusive and alteration relationships in and around gold-rich porphyry copper deposits.

The term ‘propylitic’ was originally coined by Becker in 1882 to describe the alteration of diorite and andesite beside the Comstock gold-silver lode, Nevada, USA where epidote, chlorite and albite are the main alteration products.



2.19 Generalised occurrence model showing intrusive and alteration relationships in and around gold-rich porphyry copper deposits (modified from Sillitoe and Gappe, 1984).

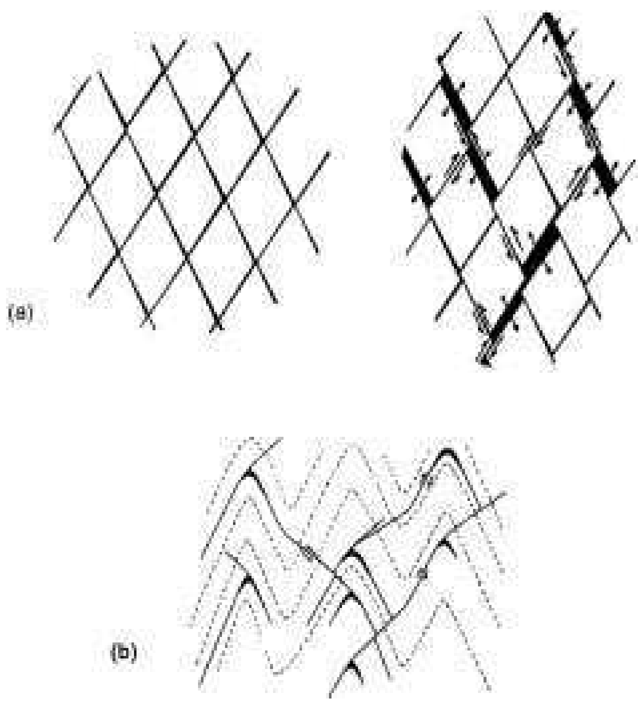
Carbonates, chlorites, epidote, zeolites and adularia are generally diagnostic in the immediate vicinity of veins containing quartz and chalcedony. Alteration of the wall rock adjacent to the gold-bearing veins commonly extends above the ore zone, occurring extensively in mafic volcanics but much less so in rhyolite. Chloritisation extends outwards for considerable distances from vein systems, while seritisation is closely restricted to fractures and mineralisation.

2.4 Gold deposition in volcanic terrain

The entrance of metals into magmas occurs in volcanically active basins. The composition and mineralogical characteristics of the magmas are affected by a wide variety of processes. These processes include the nature of the rocks where melting occurs; the history of previous thermal, metamorphic and melting events; degree of partial melting; addition of components from wall rocks and subducting slabs; and the effects of crystal fractionation on the composition of ore components in the magmas. Trace concentrations of gold in volcanically hosted metalliferous sulphides may be chemically dissolved and re-precipitated, or otherwise modified in a near-surface, oxidising environment.

Deposition occurs by replacement and filling in dilated openings in the country rocks when temperature, pressure and other depositional relationships reach certain critical levels. The development of a complex vein system is due to a combination of displacement parallel to and opening at a high angle to the length of individual shears as the result of bulk homogeneous flattening. This is demonstrated by Cox *et al.* (1991) in Fig. 2.20(a and b) in respect of the gold deposits of the Lachlan fold belt in Victoria, Australia. Figure 2.20(a) shows the development of a complex vein system consisting of non-parallel shear veins; (b) the pattern of intersecting limb thrusts and vein-filled dilated shear zones and fold hinges.

Orebodies are most favourably distributed if they shed gold into an alluvial setting from all around the catchment area. A cluster of small apophyses is more likely to provide wide-ranging hydrothermal plumbing systems than a single large pluton. The total amount of gold in the system and its distribution in an



2.20 (a) Diagrammatic representation of development of a complex vein system consisting of non-parallel shear veins as the result of bulk homogeneous flattening. (b) Pattern of intersecting limb thrusts and vein-filled dilated shear zones and fold hinges.

abundance of mineralised reefs and veinlets widely dispersed throughout the country rock is usually more important than the size and intrinsic richness of individual veinlets. The downgrading effects of dilution from all other rocks in the catchment area inhibit the formation of any but small placer concentrations from single lode systems, however rich.

2.4.1 Volcanic-hosted metalliferous sulphides

Volcanic-hosted massive sulphide deposits form mostly in back-arc basins and mid-ocean ridges. Gold is first taken into solution by hot brine solutions at active seafloor spreading centres when seawater penetrates through deep sections of the shattered oceanic crust to the underlying mantle. The seawater becomes heated by flowing near to localised heat sources (e.g. magma chambers) and at 'hot' spots in plate interiors (e.g. the Hawaiian Islands within the Pacific Plate). The surface manifestation of such dynamic hydrologic systems is manifested in seafloor hydrothermal vents located along the Earth's mid-ocean ridges at water depths of about 1,500 to 3,600 m within the upper few kilometres of very young oceanic crust.

Chemical reactions between the circulating hot seawater and the crustal rocks result in a two-way exchange of elements between the hydrothermal fluid and oceanic crust. Elements and compounds such as magnesium and sulphate are transferred from the water into the crust. Trace (ppb) concentrations of gold and other metals are dissolved from the mantle and crustal rocks by acidic solutions. Such solutions evolve from the release of hydrogen ions in the seawater and by the formation of hydrogen sulphide (H_2S) from the reaction of seawater sulphate (SO_4) with ferrous iron in the volcanic rock. The hot metal-rich hydrothermal solutions, having a low pH and redox potential, are transported away at enhanced metal concentrations in the hydrothermal solutions. Based upon data from the Broadlands geothermal fluid, the gold solubility window in mineralising solutions appears to be around a low of 1.5 ppb Au. By analogy with base metals, this suggests an upper limit of the window at about 100 ppb Au (Huston, 1997). Precipitation occurs during mixing and cooling in contact with the cold (around 2 ºC) bottom seawater at the sea-rock interface.

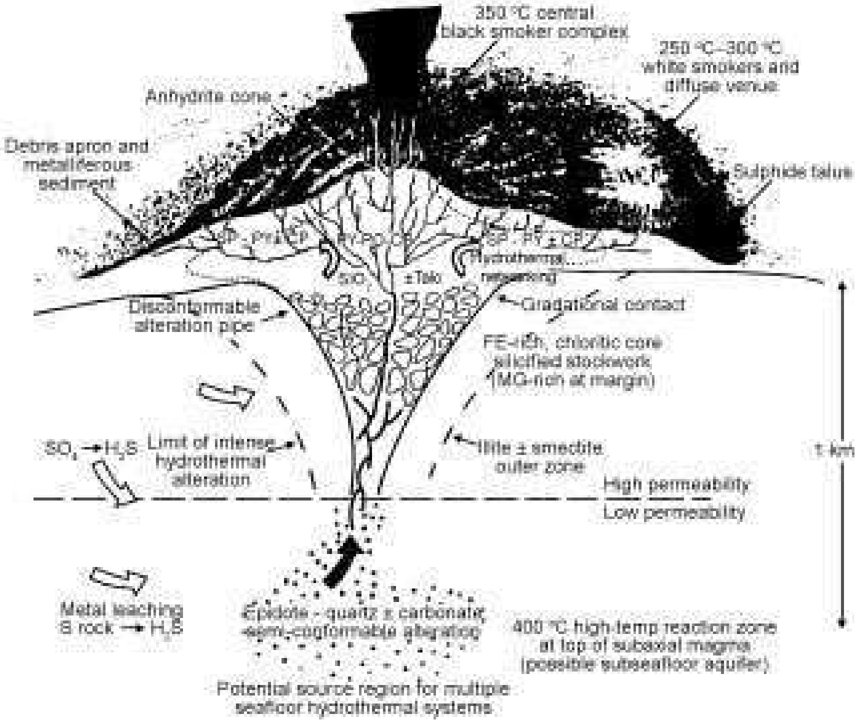
The mode of deposition is strongly influenced by the permeability of the rocks through which the solutions pass. Within a permeable crust, large massive sulphide deposits form by extensive alteration and replacement of volcanic rocks at or below the seawater-volcanic rock interface. An impermeable crust on the other hand causes the metal-rich hydrothermal fluids to emerge through narrowly confined channels and discharge directly onto the seafloor. Relative to seawater, the vent fluids are acidic, metal- and sulphide-rich and magnesium- and sulphate-poor. Mixing processes between the vent fluids and seawater result in the formation of massive sulfide deposits including black smoker chimneys and exhalation of plumes of particle-laden water that rise 200–300 m above the seafloor.

High temperature (350–360 ºC) black-smokers occur at the top of a hydrothermal mound; lower temperature (260–300 ºC) white-smokers vent from areas of hydrothermal activity around the apron of the mound. Other important sources of heat result from chemical reactions between the seawater and certain constituents of the volcanic rocks, and from radioactive decay. These hydrothermal systems play important roles in global heat budgets and geochemical balances. The smokers themselves are transient in nature; the crustal residence time of convecting seawater may be less than three years (Cann and Strens, 1982). Smokers grow continually by circulation of the hydrothermal fluids through the sulphide pile causing abundant re-crystallisation of sulphide minerals. Most occurrences of metallic sulphides amount to no more than a few hundred to a thousand tonnes; only where large numbers of black smokers are active simultaneously over longer periods of time are larger deposits formed.

Recent advances in deep-sea exploration technology have led to the direct observation of gold-bearing black-smokers at hydrothermal vents and fissures along active spreading ridges. The ore zones are located within the igneous and metamorphic rocks of these settings at various depths ranging from near surface (epithermal) vein systems, to deep-seated (mesothermal) deposits at depths of 10 km and more. Epithermal-like seafloor hydrothermal systems have been discovered in shallow marine island arc environments of the west and southwest Pacific. A conceptual cross-section of a seafloor hydrothermal system (Fig. 2.21) is given by Herzig and Hannington (1995) (cf., Ocean Drilling Program Leg 158: Trans Oceanic Geotraverse Survey 'TAG' Hydrothermal Field).

The depth and nature of sedimentation has important implications that reflect differences in fluid-sediment reaction on deposit styles. Sediment-hosted sulphide deposits tend to be larger than sediment-starved deposits and have lower concentrations of base metals. The larger cover of sediment close to the land produces longer-term heat retention and entrapment and insulation of vent fluids, which may account for the large size of the deposits (Kappe and Franklin, 1989). The sediment cover may also help to protect the sulphides from the effects of submarine weathering and oxidation.

The potential of ancient oceanic crust to survive is low because of the development of subduction systems and associated hydrothermal processes. Only remnants may still occur as accretionary prisms obducted over the edge of the overriding plate and its volcanic arc, whether oceanic or continental (Coleman, 1971; Moores, 1982). Rona (1984) demonstrated the significance of the impact of seafloor mineralisation on the development of land-based ore bodies by locating ancient sources of ore originally formed by seafloor hot springs and now on land. Gold occurring in trace proportions, but in fact in massive overall quantities, in volcanically hosted metalliferous sulphides may be chemically dissolved and re-precipitated or otherwise modified in a near-surface oxidising environment. The oldest base metal ores in the world are Zn-Pb-Cu deposits from the eastern Pilbara Craton.



2.21 Typical cross-section of seafloor hydrothermal system (derived from Hertzig and Hannington, 1995). (From *Ore Geology Reviews*, Volume 10, P Hertzig and M Hannington, Polymetallic massive sulphides at modern seafloor, pp 95–115, 1995, with permission from Elsevier).

2.4.2 Mesothermal gold ores

Mesothermal gold ores comprise mainly high fineness (low silver) quartz vein systems and disseminated replacement deposits. The orebodies are emplaced either in or near supracrustal belts dominated by volcanic rocks, or in supracrustal belts dominated by elastic sedimentary rocks. Table 2.9 offers a broad classification of mesothermal gold deposits for exploration purposes based upon host rock type and form of mineralisation. Whilst acknowledging the shortcomings of this grouping, Hodgson stresses that factors such as the geometrical and mineralogical-chemical characteristics of mineralisation commonly relate to host-rock type and point to the significance of the sulphide content in iron formation deposits. The main associated minerals in mesothermal ore zones are pyrite, pyrrhotite, base metal sulphides, arsenopyrite, tourmaline and molybdenite. In some areas, deposit size is significant. For example, in the Timmins-Kirkland Lake area, Toronto, Canada, Hodgson and Troop (1988) found that scheelite, tourmaline, arsenopyrite, tellurides, sphalerite and galena

Table 2.9 Broad classification of mesothermal gold deposits based upon rock type and form of mineralisation (after Hodgson, 1993)

Host rocks	Dominant form of mineralisation	Examples
	In or near supracrustal belts dominated by volcanic rocks	
Volcanic rocks, classic sedimentary rocks and intrusions	1. Auriferous quartz veins and veinlet systems	Sigma,[a] Kirkland Lake,[a] Hollinger-McIntyre,[a] Kalgoorlie,[b] Mother Lode,[c]
	2. Disseminated pyritic quartz-albite and (or) potassium feldspar- carbonate replacement zones	Hemlo,[a] Blanket[d]
Oxide-facies iron-formation	Breccia, stockwork and bedding-replacement zones	Mount Magnet[e]
	In supracrustal belts dominated by clastic sedimentary rocks	
Silicate-facies and (or) carbonate-facies iron-formation	Quartz stockwork and replacement zones	Morro Velho[f] Homestake[g]
Oxide-facies iron-formation	Breccia, staockwork and bedding-replacement zones	São Bento,[f] Cuiaba,[f] Raposos[f]
Clastic sedimentary rocks		Stawell,[h] Bendigo[h]

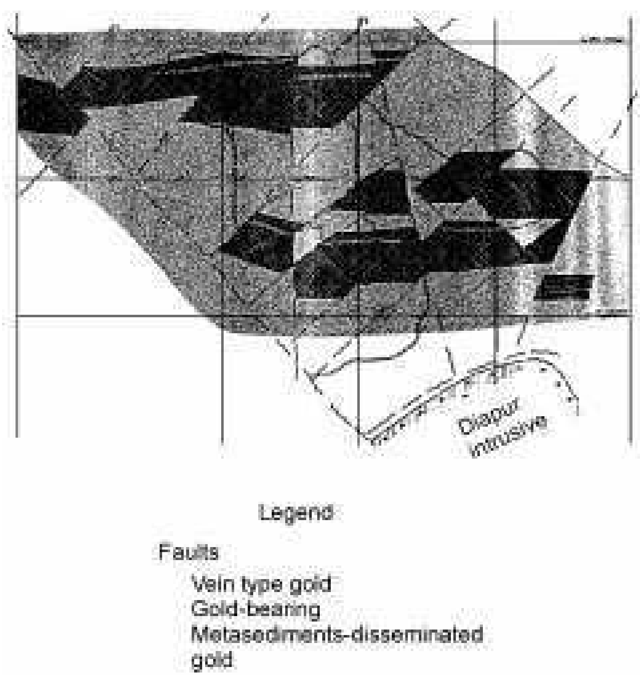
[a] Abitibi-Wawa Belt, Candian Shield. [b] Norseman-Wiluna Belt, Yilgarn Block, Australia. [c] Foothills metamorphic belt, California. [d] Gwanda Belt, Zimbabwe Craton. [e] Murchison Belt, Yilgarn Block, Australia. [f] Quadrilatero Ferrifero, Brazil. [g] Black Hills, South Dakota. [h] Lachlan fold belt, Victoria, Australia.

were more common in the larger deposits of the area than in the smaller. They noted also, that the mineral distribution varied between deposits associated with quartz-bearing and quartz-free porphyries.

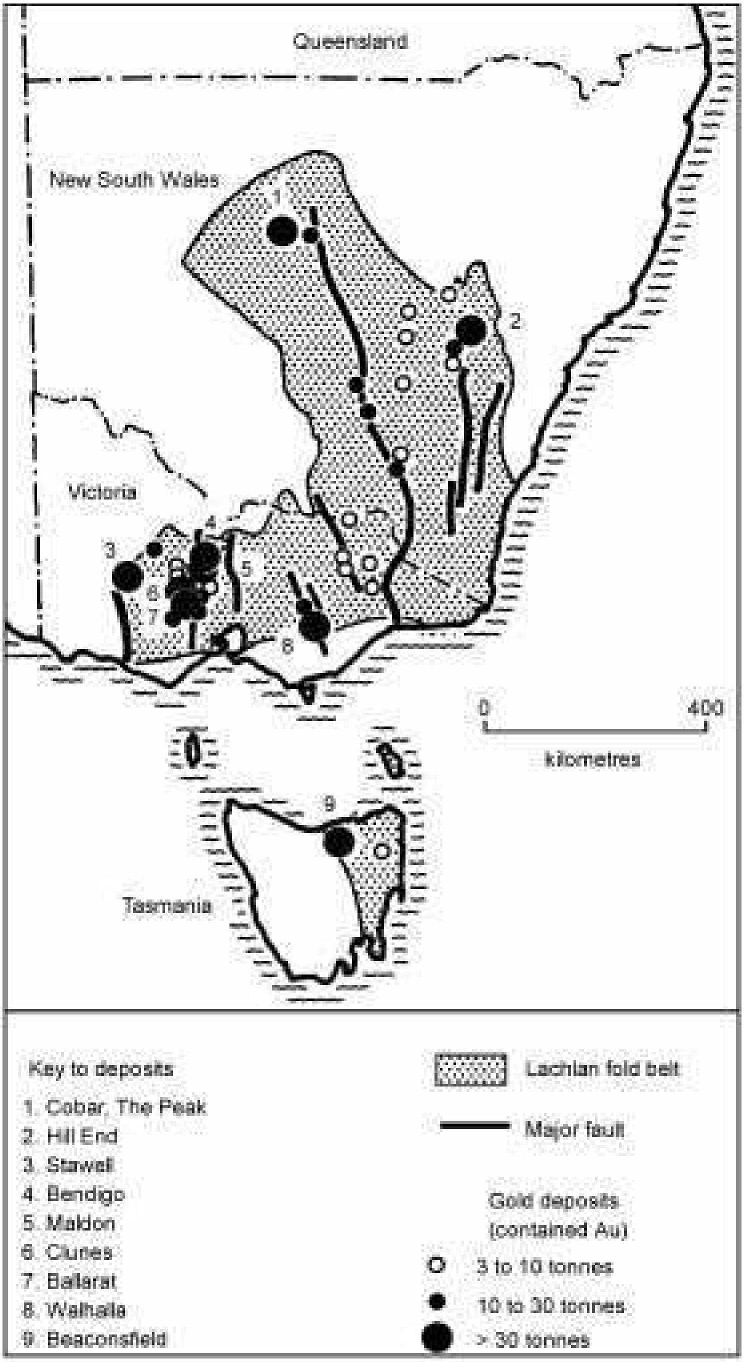
Mesothermal deposits occur mostly in low to medium grade metamorphosed supracrustal belts and mineralisation appears to depend upon intrusion by felsic to intermediate plutons. Important Mesozoic and Palaeozoic mesothermal gold deposits in northern China and Korea include metamorphic granitic to intermediate plutons and intermediate to high-grade Precambrian gneiss (Sang and Ho, 1987). Age dating has shown that the timing of mesothermal ore formation in both volcanic and clastic sedimentary belts is late in the orogenic history of a region and is relatively universal between principal mineralised districts. Host structures are mainly faults and folds and there is a regular association of the deposits with fluid passages that are fault-controlled. The faults are nearly always second-order structures, subsidiary to large barren strike-slip faults (Sibson, 1987). Strike-slip shear zones parallel to the fault slip vector are more conducive to dilation than the large and straight major faults.

This is demonstrated by the pattern of gold distribution in source rocks of the Gros Rosabel Alluvial Gold Concession Area, Suriname, South America (Macdonald and Wright, 1984). The control mechanism at Royal Hill (Fig. 2.22) was apparently built up around the thrusting of the diapur together with some strike fault development with normal movement at about 60º to 70º to the north. Gash veins were opened up at about 90º to the strike faults, probably in areas prestressed by the folding. These veins flattened and tightened as they extended away from the initial break. The openings so produced provided channels for the passage of hydrothermal fluids containing quartz, potash, iron sulphide, tourmaline, gold and other minerals. Flat-lying gash veins filled with quartz veins containing gold and pyrite make up 95% of the productive veins in the Royal Hill area.

Individual mesothermal gold deposits may comprise economically important parts of large-scale fluid flow systems in the crust. Provenances for major alluvial gold fields in Victoria and NSW, Australia (e.g., Ballarat, Ararat, Beechworth) in zones in the Lachland fold belt (Fig. 2.23), which comprised parts of larger deformation zones in which the faults acted as major fluid conduits (Woodall, 1990). In the Lucknow gold field of NSW, quartz veins were developed along a main fissure for about 900 m. A feature of deposits in this field is the invariable association of rich veins with calcite and its occurrence in



2.22 Zoning of a gold-bearing system around a diapur at Royal Hill, Suriname.



2.23 Provenances of major alluvial goldfields in the Lachlan fold belt, Victoria, Australia (after Woodall, 1990).

unoxidised zones with arsenical pyrites (mispickel), which occurs in masses of 'beautiful' stellate or radiating crystals known to the miners as 'Prince of Wales Feathers'. The gold content of the mispickel ranges from 50 to 500 oz./t (Kenny, 1928).

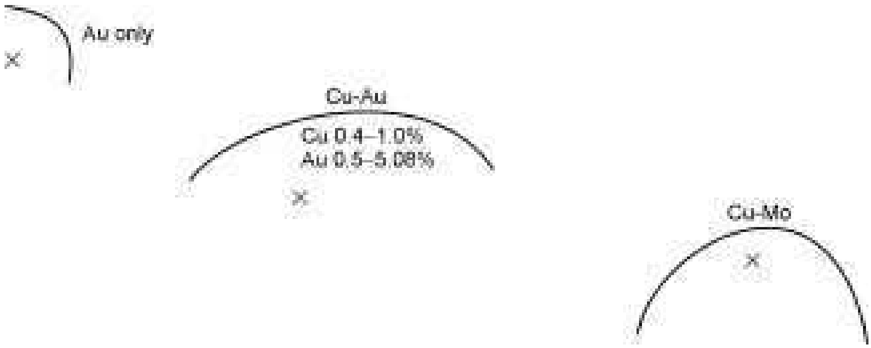
Differences in the vertical continuity of gold-bearing quartz vein systems are commonly reflected in the greater extent and richness of placers derived from mesothermal orebodies than from epithermal vein systems. Whereas epithermal ore bodies appear to be restricted to depths of about 600 m, mesothermal gold veins are characterised by their vertical and longitudinal continuity. They include the Braline deposit in British Columbia, which extends to depths of at least 2 km, and the Champian Reef mesothermal quartz-vein system, Kolar Goldfield, India which is continuous to a depth of 3.2 km with little change in mineralogy (Hamilton and Hodgson, 1986). Deeply eroded mesothermal ore bodies in Australia, American Rockies and the Urals of Asia have sourced most of the major gold placers worldwide. In most of these deposits the veins are discontinuous and a typical ore shoot structure consists of interspersed zones of high and sub-ore grade vein material.

2.4.3 Intrusion-related deposits

Most intrusion-related gold deposits are generated above zones of active subduction at convergent plate margins. The margins range from primitive, through mature island arcs to continental margins. Sillitoe (1993) lists a number of important types: gold-rich copper and gold-only porphyry deposits, intrusion-hosted stockwork/disseminated deposits, deposits in carbonate rocks, carbonate replacement deposits, stockwork disseminated and replacement deposits in non-carbonate rocks, breccia-hosted deposits and vein-type deposits. He notes that intrusion-related deposits commonly occur in juxtaposition and are locally transitional, one to the other.

Magmatic fluids commonly enhance porphyry type concentrations to ore grade (Hemley and Hunt, 1992). Contributing factors are water, metals (including gold), ligands (e.g. S and Cl) and other components of mesothermal intrusion-related porphyry copper deposits. Depths of formation vary generally from 2 to 5 km according to lithostatic pressures. Porphyry copper deposits tend to form in clusters and exploration potential may exist for several kilometres from known deposits. Gold in intrusion-related deposits clearly has the potential to precipitate both within and at various distances from a progenitor intrusion. According to Heald *et al.* (1987) the hydrothermal zoning is a function of temperature/concentration relationships. Zoning reversals are thereby suggested as the result of differences in the relative concentration of metals.

Gold-only porphyries, such as at Fort Knox, Alaska appear to occur at higher levels than Cu-Au porphyry deposits, which are themselves at higher levels than Cu-Mo porphyritic intrusives as, described schematically in Fig. 2.24. The fluids



2.24 Schematic representation of Au, Cu-Au and Cu-Mo porphyries, such as at Fort Knox, Alaska.

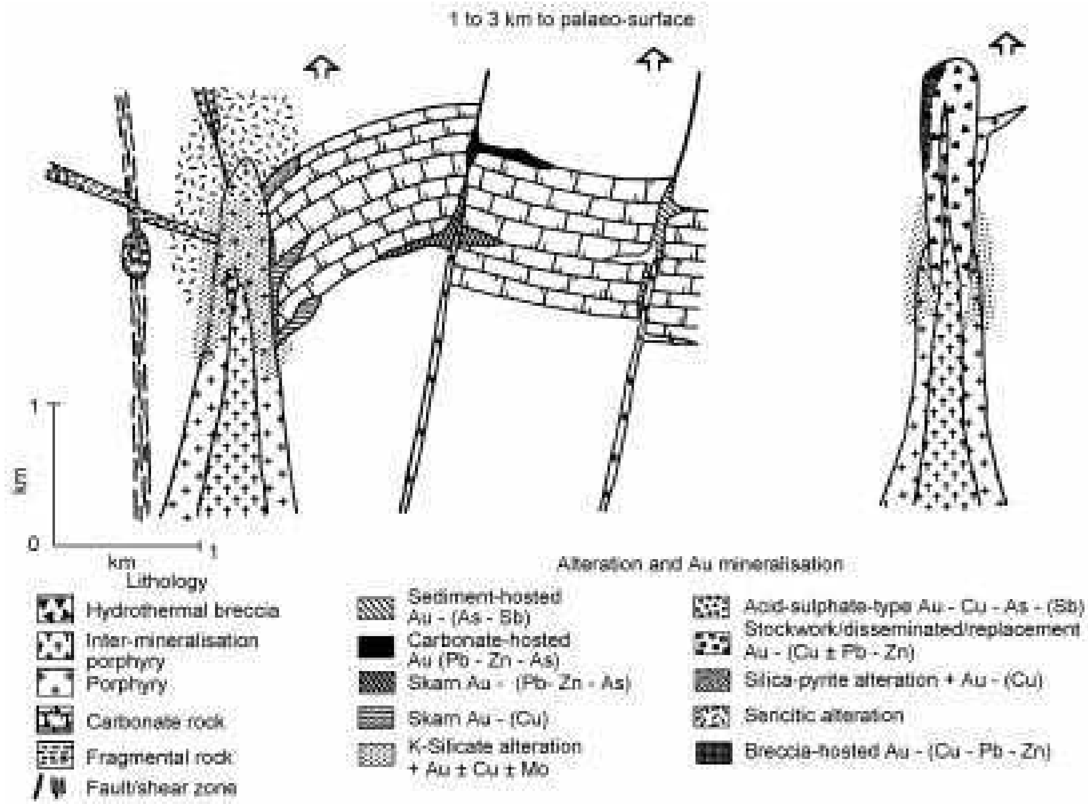
are thought to be initially hot (500–600 ºC) but zoned alteration assemblages are formed at lower temperatures as the metal-bearing fluid cools and reacts with the country rock as it moves away from the intrusion. Deeply eroded volcanic-plutonic arcs characterised by stocks and widespread volcanic rocks of intermediate composition may provide attractive targets for alluvial gold exploration.

Sillitoe (1993) reconstructs a typical intrusion-centred gold district characterised by carbonate wall rocks in Fig. 2.25. Geological evidence and fluid-inclusive studies in Sillitoe's text support depths ranging from 1 to 3 km beneath the palaeo-surface.

2.4.4 Carlin trend type gold deposits

The Carlin trend forms the largest and most prolific accumulation of gold deposits in North America (Teal and Jackson, 1997). Discovered in 1961, more than 40 separate deposits of disseminated gold mineralisation in carbonate rocks have contributed 25 million ounces of gold from 26 working mines. The Carlin trend is a 60 km long north-northwest trending alignment of gold deposits located in northeastern Nevada. Gold mineralisation is hosted in a variable package of Ordovician through lower Mississippian. The current north-northwest alignment of the Carlin trend reflects an apparent pre-existing zone of crustal weakness that transects present north-south trending basin and range topography. Inception of basin and range extension is interpreted to have begun with the onset of regional intrusive activity during the late Eocene (± 37 Ma) (Christiansen and Yeats, 1992). Tectonism may have begun during early Miocene (± 20 Ma), but remains a subject of debate (Teal and Jackson, 1997).

Carlin-trend geologists suggest a range of interactions involving system, structure and host rock as essential components for the formation of a gold deposit. Within this context, the major geologic parameters that have contributed to the genesis of gold deposition on the Carlin trend are:



2.25 Schematic relationships between progenitor intrusions and some gold mineralisation styles (Sillitoe, 1993). The presence of all styles in a single district is not necessarily implied. The environments are transitional upwards to the epithermal setting.

- a geologically long active zone of crustal weakness originating along a palaeo-continental margin, with development of major through-going fault systems
- a regional environment of crustal thinning with multiple intrusive episodes and sustained high heat flow
- multiple episodes of hydrothermal activity
- reactive and highly permeable carbonate host rocks.

Christenson (1993) summarises the salient alteration features that characterise Carlin trend deposits:

- carbonate dissolution
- argillic alteration of primary silicate minerals
- silicification
- gold-enriched sulfidation of reactive iron in host rocks to form gold-bearing sulphide (pyrite, arsenopyrite).

Teal and Jackson (1997) have adapted the alteration zonation pattern proposed by Kuehn and Rose to include the following major distal-to-proximal alteration assemblages:

- fresh silty limestone – calcite + dolomite + illite + quartz + Kspar + pyrite
- weak to moderate decalcification (dolomite halo) – dolomite calcite + quartz + illite + kaolinite + pyrite + gold
- strong decalcification – dolomite + quartz + illite kaolinite + pyrite gold
- decarbonisation – quartz + kaolinite/dickite + pyrite gold.

Carlin type gold is also being discovered elsewhere. The gold occurs as sub-micron particles primarily within the lattices of pyrite and arsenopyrite. Fluid inclusion studies suggest that the metal was transported as hydrogen bisulphide complex (Kuehn, 1989) by gold-bearing fluids of mixed meteoric-magmatic origin. Due to high CO_2 content in fluid inclusions, Kuehn estimated a depth of formation of the Carlin trend deposits of 4.4 ± 2.0 km within a temperature range of 180–245 ºC. Deposits that appear to be similar to those documented for the Carlin trend ores were discovered at Salaman (Leon, Spain) by BP Minera Espana in 1985 (Paniagua *et al.*, 1996). Hydrothermal alterations at the Salamon deposit are fundamentally decarbonatisation-dolomitisation, silicification and argillisation, and vary according to the type of host rock. The gold is refractory, hosted by pyrite and arsenopyrite, crystals measuring a few dozen microns. Mineralised bodies are made up with mineralised tectonic breccias, veins and pockets, and disseminated sulphides in a quartz-carbonate gangue. The host rocks consist primarily of carbonate rocks with very bituminous perlitic intercalations from the Lena Group. Salamanca University, in collaboration with SIEMCALSA, is carrying out further work (at Castilla-Leon Autonomy Community) in the framework of the metallogenic research on hydrothermal gold deposits.

2.4.5 Epithermal ores

The term epithermal is given to deposits that form at shallow depths (surface to about 2 km) in the Earth's crust over the temperature range 150 ºC to 300 ºC (Berger and Eimon, 1983). Epithermal gold tends to have a more distinctive primary provenance than mesothermal gold, occurring mostly in the volcanic rocks and shallow depth intrusives of convergent tectonic settings. Two distinctive styles of epithermal gold mineralisation are developed from fluids of contrasting geochemistry. Each style has its own characteristic mineralogy and wall rock alteration signature:

- low sulfidation systems are formed from reduced near-neutral pH fluids with large meteoric water input
- high sulfidation systems form from oxidised acidic fluids generated in the magmatic-hydrothermal system.

White and Hedenquist (1995b) list the characteristic forms of the two styles in Table 2.10 to help distinguish between them in the field. The authors point to the necessity of making this distinction because 'although the two styles have similar alteration mineralogies, the distribution of the alteration zones is different and the economic potential is associated with different parts of the system'. Alteration zoning, but only where the style is correctly identified, indicates the prospective parts of a system. The authors in Table 2.11 give examples of epithermal gold deposits. The ore minerals in epithermal gold-rich ores are shown in Table 2.12 in order of frequency of occurrence. This table is based upon compilation of mineral data from more than 130 epithermal deposits in the south-west Pacific region (White, 1995a) and 47 deposits in North and Central America (Buchanan, 1981).

Table 2.10 Examples of epithermal deposits (after White and Hedenquist, 1995b)

Low sulphidation	High sulphidation
McLaughlin, California, USA	Goldfield, Nevada, USA
Round Mountain, Nevada, USA	Summerville, Colorado, USA
Hishikari, Japan	Iwato, Kasuga and Akeshi, Japan
Emperor, Fiji	La Coipa, Chile
Gold Cross, New Zealand	El Indio, Chile
Waihi, New Zealand	Pueblo Viejo, Dominican Republic
Lebong Tandai, Indonesia	Chinkuashih, Taiwan
Kelian, Indoensia	Rodalquilar, Spain
Porgera Zone VII, Papua New Guinea	Lepanto, Philippines
Pajingo, Australia	Lahoca, Hungary

Table 2.11 Characteristic forms of low sulphidation and high sulphidation alteration assemblages (after White and Hedenquist, 1995b)

Low sulphidation	High sulphidation
(Adularia-sericite)	*(Acid sulphate)*
Open-space veins dominant	Veins subordinate, locally dominant
Disseminate ore mostly minor	Disseminated ore dominant
Replacement of minor ore	Replacement ore common
Stockwork ore common	Stockwork ore minor

Low-sulphidation ores

Mineralising fluids in low sulphidation gold systems are typically dominated by deeply circulating meteoric waters which have absorbed acid magmatic gases (CO_2, SO_2, HCl). Such fluids equilibrate with their host rocks in a low sulphidation, epithermal gold environment, thus becoming reduced and developing a near-neutral pH (Giggenbach, 1991). The ore association is produced by the near-neutral pH solutions boiling at shallow depths, and generating CO_2 and H_2S-rich vapours. Condensation of these vapours in the vadose zone above the water table, leads to the formation of steam heated acid-sulphate waters with increasing pH. Eventually, the loss of H_2S from the solution causes the solubility of gold to decrease, thus leading to its precipitation (Henley *et al.*, 1984).

Topography is of major importance. Major low sulphidation deposits formed in shallow conduits in low relief areas are likely to be distributed above a feeder

Table 2.12 Ore minerals in epithermal ores (after White and Hedenquist, 1995b and Buchanan, 1981)

	Low sulphidation	High sulphidation
Pyrite	ubiquitous (abundant)	ubiquitous (abundant)
Sphalerite	common (variable)	common (very minor)
Galena	common (variable)	common (very minor)
Chalcopyrite	common (very minor)	common (minor)
Enargite-Luzonite	rare (very minor)	ubiquitous (variable)
Tennantite-tetrahedrite	common (very minor)	common (variable)
Covellite	uncommon (very minor)	common (minor)
Stibnite	uncommon (very minor)	rare (very minor)
Orpiment	rare (very minor)	rare (very minor)
Realgar	rare (very minor)	rare (very minor)
Arsenopyrite	common (minor)	rare (very minor)
Cinnabar	uncommon (minor)	rare (very minor)
Electrum	common (variable)	uncommon (very minor)
Native gold	common (very minor)	common (minor)
Tellurides-selenides	common (very minor)	uncommon (variable)

zone extending into the basement. Such deposits are commonly distributed in a roughly symmetrically pattern due to mushrooming of the ascending hydrothermal fluids. In high relief areas (andesitic stratovolcanoes) a large degree of lateral flow takes place, which results in strongly asymmetrically altered zones, relative to the upward flow.

The ores are texturally diverse in banded, crustiform quartz and chalcedony veins, and in druse-lined cavities and multiple episodic vein breccias (Berger and Eimon, 1983). They are associated with the least acidic alteration minerals, e.g., calcite and adularia although calcite, formed as a result of boiling, may be replaced by quartz as the system cools. White and Hedenquist (1995b) stress the importance of determining the origin of alteration minerals that indicate acidic conditions. These conditions include hypogene activity due to magmatic HCl and SO_2; steam heated acid-sulphate waters formed near the surface; and post-hydrothermal weathering of sulphide minerals.

High-sulphidation ores

A well-documented genetic association exists between magmas and epithermal gold deposits where the deposits are formed by high sulphidation (acidic and oxidised) fluid, typical of acidic springs near volcanoes. Isotopic studies have shown that reactive components in the high sulphidation environment are derived from a relatively oxidised magmatic source, with little wall-rock interaction at depth as they rise to the surface. According to Rye (1993), SO_2 and HCl vapour absorbed by ground water causes SO_2 to disproportionate to H_2SO_4 and H_2S followed by dissociation of the H_2SO_4 and HCl. This results in hot (200–300 °C) highly acidic (pH 0–2) oxidised solutions which react with and leach wall rocks at shallow depths; as distinct from low-sulphidation fluids, which rise from great depths and react extensively with deep seated rocks (Gray, 1997b).

High sulphidation ore deposits are distinguished from their low-sulphidation counterparts by features, which relate to differences in physico-chemical conditions of formation and zoning of their alteration products. Both deposit styles are associated with economically important orebodies. But whereas low-sulphidation deposits usually comprise veins, breccias and stockworks of veins in which the filled cavities have sharp edges, high-sulphidation deposits are typically disseminated ore bodies, usually in leached zones of most acidic alteration extending outwards in the surrounding country rock from the fluid conduit. The gold is transported mainly as a chloride complex, with dilution and/or cooling as controls on precipitation. Suitable conditions for the deposition of high grade ore bodies are sometimes provided by localised veining or brecciation, but the dominant texture overall, is massive vuggy quartz caused by leaching at pH 2 (Stoffregen, 1987). Such vuggy quartz bodies may be cut by massive to banded sulphide veins consisting of pyrite and enargite (White and Hedenquist, 1995b).

Alunite $KAl_3(SO_4)_2(OH)_6$ and clays such as kaolinite, picilite and pyrophyllite are stable under acidic conditions and are common, but minor, constituents of high-sulphidation gold systems. Quartz is the principal gangue mineral and propylitic alteration, controlled by rock composition, occurs outside the conduit zones in both low and high-sulphidisation styles. Albite, calcite, epidote and pyrite are typical minerals in regions of low water–rock ratios. Alteration varies both vertically and laterally and high-sulphidation orebodies in zones of most acidic alteration are surrounded by mineral assemblages indicating less acid conditions as the acidic water is neutralised by reaction with the host rock away from the fluid conduit.

2.5 Provenance

The term 'provenance' is derived from the French 'provenir' and the Latin 'provenere' meaning to originate or come forth. In a residual gold setting, provenance refers to both the source and country rocks from which gold-bearing placers and lateritic regoliths are derived. In order to link economic concentrations of detrital gold to broad groups of gold-bearing rocks in the immediate hinterland:

- There must have been an adequate catchment of source rocks.
- The rocks must have been sufficiently weathered to release the gold.
- The regime must have induced efficient methods of transport.
- Concentration must have taken place under lateritic conditions of weathering, or in alluvial deposits by some form of fluvial or aeolean transportation.
- Detailed petrology may be needed to trace detrital gold that has been recycled through sediments such as beach concentrations back to their original source.

2.5.1 Significance of provenance

Residual gold deposit types have unique features and locally, the weathering of source rocks provides for the movement of chemically altered and partly fragmented gold-bearing material into the drainage system. The detritus is diagnostic of the source and associated country rocks from which it is derived. Fragments of unstable rock varieties close to the source, have sharp edges and are only partly weathered. A lesser amount of fresh rock is incorporated in the next alluvial cycle and so on until ultimately little or no rock pebbles remain in the stream sediment. At this stage, although the altered country rocks could contain distinctive assemblages of secondary or hydrothermally altered minerals as haloes around the parent ore bodies, the initial source may be rendered difficult or impossible to trace by conventional panning methods.

The free gold content of the sediment is determined by the extent to which the gold grains have been released and/or modified during the weathering processes.

The gold grains themselves are liberated sequentially by chemical, physical and biological weathering effects in the physical environment at different depths of the weathering zone and on slopes and along drainage lines. Following stage one deposition, further chemical and mechanical decomposition ensues and one of the most important features of provenance is that of being able to account for the various non-economic minerals and rock particles found with the gold. Provenances have generally predictable gold-rock paragenesis and regardless of age certain indicator rock-forming minerals act as pointers to particular rock types that may represent the source or origin of the gold. Quartz is dominant in most mineralised zones. Other important minerals are Ca, Fe, Mg carbonates, and sulphides (pyrite, arsenopyrite and chalcopyrite) less commonly galena and sphalerite. Such minerals as graphite, scheelite, pyrrhotite, tourmaline and tetrahedrite usually appear in trace quantities only. Gold in mesothermal ore bodies is paragenetically late (Nesbitt and Muchlenbachs, 1989) and is commonly associated with quartz, carbonate, galena, and sphalerite plus or minus tellurides in fractures in early sulphide ore bodies.

In this book, placer gold provenances are divided between Phanerozoic and Precambrian rock types on the basis of age and recognition of similarities and limitations. Points of similarity include:

- host rocks that provide a zoned pattern of hydrothermal gold-ore formation
- gold-bearing, base metal sulphides in both Phanerozoic and Precambrian environments
- mesothermal deposits of both age groups that may be considered as part of one metamorphic change in an environment of tectonic uplift age-independent class
- that, while settings and conditions of Phanerozoic volcanic processes may have been different from those of the Precambrian, the same basic principles should be applicable to both ancient and modern volcanic successions (Cas and Wright, 1995)
- some evidence to suggest that gold-rich deposits, possibly of porphyry style, are widespread in Archaean greenstone belts (Symonds *et al.*, 1987) and that they, like mesothermal deposits which form much deeper in the Earth's crust, also had a long preservation potential.

2.5.2 Limitations of provenance

Important limitations of the provenance principle are:

- Not all rocks of apparent paragenetic relationship carry the expected gold mineralisation.
- The primary gold deposit must be sufficiently large for a viable secondary gold deposit to form.
- The incidence of indicator minerals (rock-forming silicates) in a regolith is no guarantee that an economic gold deposit will be present.

- The source of minerals such as gold cannot always be predicted from available evidence, where that evidence is geographic and there is no predictable mineral-rock paragenesis.
- The application of provenance in placer exploration is governed by the drainage pattern but it may also be influenced by the existence of a secondary source for the gold-bearing minerals.

For example, one tributary of a river may rise in platinum-bearing ultramafics, whilst another may cut through granite or sandstone terrain. Ultimately this would bring together unrelated rock-forming minerals in which quartz would predominate. The author's experience of such an example includes the Uraido River gold/platinum placer, Colombia, South America.

Because of these limitations, any model for reconstruction of the geomorphic history of a residual placer gold environment must take account of the variable nature of past climates and the wide range of timescales within which individual changes may have occurred. All past and present processes that relate to the modification and release of gold grains in the weathering zones of orebodies will be critical to investigations of both primary and secondary gold deposits. Neither primary nor alluvial gold deposition can be studied in isolation without neglecting possibly vital evidence from the other. A detailed knowledge of a source area (petrology, structure and geological history) is thus of inestimable value to both primary and placer gold explorationists.

2.6 Time rate of unroofing ore bodies

During the active life of a volcano, the aggregation of magma exceeds the enormous amounts removed, the principal effect being to continuously raise the surface of the land against the base level of erosion to which the erosional processes operate. Uplift occurs in stages and is rapid during the most active stages of mountain building. Only during extended periods of tectonic calm, e.g. in later stages of orogeny will erosion rates exceed those of upward movement and thus eliminate the majority of the surface relief overlying the mineralised zones. On a global scale, orogenic belts exposed at various levels of erosion make up much of the world's land area, and differences in the vertical continuity of gold-bearing ores reflect differences in the geology of their formation, as well as their levels of emplacement and regional associations. Interpretation in terms of active arc systems and collision belt processes requires detailed geological investigation and careful comparison with presently active systems.

2.6.1 Volcanic uplift

Volcanic eruptions are short-lived and occur only intermittently during the total history of a volcano. Regionally extensive crustal structures include volcanic arc, fore-arc region and trench and, in some cases, back-arc basins. Back-arc basins are underlaid by oceanic crust and lithosphere between island arcs and continental margins or between two island arcs (refer to Chapter 1). Folded mountain chains and sedimentary basins are created either by the extrusion of magma as in island arc orogeny, tectonic uplift of continental crust at subducting plate boundaries, or by a combination of both. Deposits of economic significance in these belts are found mainly in stratovolcanoes and rhyolitic centres in continental and marine volcanic centres.

The active life of stratovolcanoes and rhyolitic volcanic centres is made up of short-lived eruptive periods in which enormous volumes of rocks are erupted, and much longer repose periods between eruptions when some of the erupted material is eroded away. However, the total time of all of the periods of quiescence far exceeds the total time of all of the eruptions during the active life of the volcano and surface processes operate at very high rates. Consequently, when volcanic activity ceases, erosion is capable of eliminating the majority of volcanic surface piles within a geologically short period of time.

Repose periods between eruptions occupy the bulk of its active lifetime. Large volcanoes such as stratovolcanoes and those of rhyolitic centres are most affected. In general, stratovolcanoes have active lives of less than 100,000 years, but overlapping and closely spaced stratovolcanoes of a polygenetic complex may be developed over time periods of 10 Ma and more (Cas and Wright, 1995). For example, Martinique has at least six volcanic centres; K-Ar isotope dating indicates their development over a period of at least 20 Ma (Briden *et al.*, 1979). Individual cycles in the history of the Guatemalan volcanic chain appear to be marked by four distinct phases of activity (Vessell and Davies, 1981):

1. an inter-eruption phase of 80 to 100 years characterised by low rates of sediment deposition, erosional incision of meander rivers and reworking of deltas
2. an eruptional phase lasting less than one year
3. a fan building phase dominated by debris flows and coarse fluvial sedimentation lasting up to two years
4. the influx of large volumes of sediment into stream systems, transforming them from incised meandering into flood-prone braided systems; this phase lasts from 20 to 30 years.

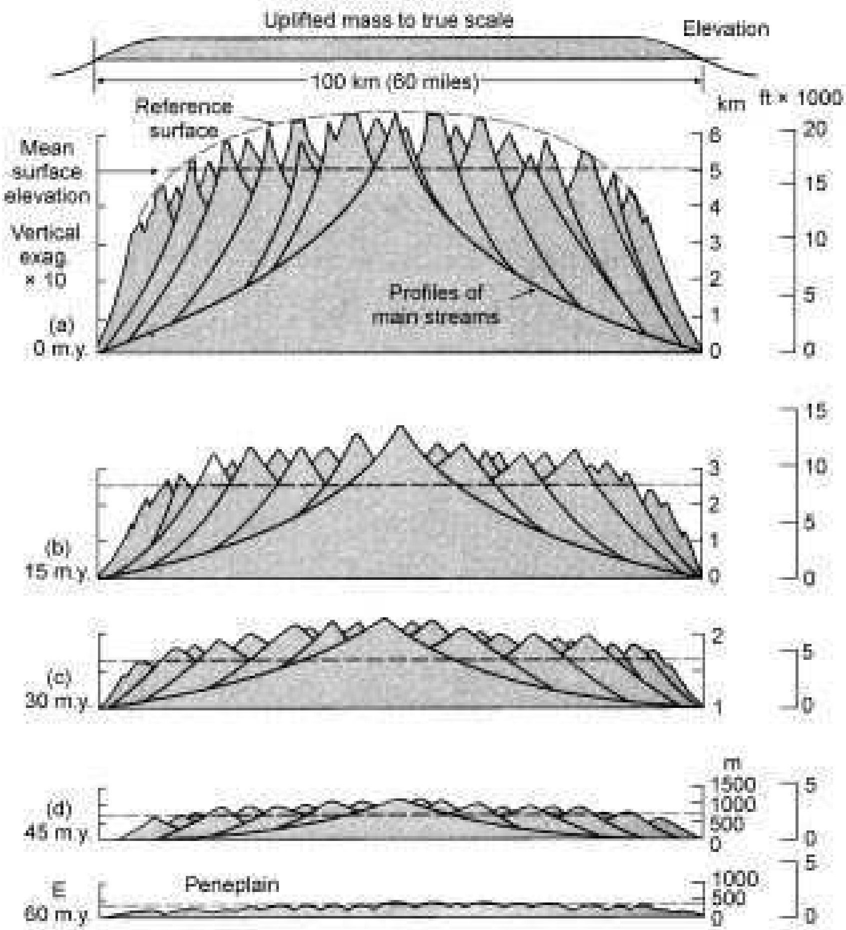
Similarly in other volcanic environments, all measurements and observations (Kuenzi *et al.*, 1979; Mills, 1976; Francis, 1983) show that the duration of epiclastic sedimentary processes is typically far greater than the eruptive phases and that they are rejuvenated by renewed volcanic activity. Based upon global geological evidence, rates of recent tectonic uplift vary globally from about 4–12 m/1,000 y in mountain belts above active subduction zones or along lithospheric sutures (Strahler and Strahler, 1992). The authors suggest that by assuming an average rate of uplift of 6 m/1,000 y, a mountain top might rise to an elevation of 6 km in about a million years less what might be lost in that time by erosion. Francis (1983) summarises known erosion rates within the context of volcanic terrains, citing downcutting rates varying between 0.1–1.0 m per thousand years in areas of high relief.

2.6.2 Denudation

Denudation is the sum total of the combined efforts of weathering, mass wasting and erosion involved in the lowering of the Earth's surface and transport of sediments to the sea. The preservation potential of orebodies depends to a large degree upon the stage of denudation reached at the time of their exposure to the atmosphere. On a geological timescale, magmatic arcs and collision type mountain belts are rapidly exhumed and eroded away. Erosion rates of modern gold-rich porphyry copper deposits, which form in volcano-plutonic arcs in subduction-related settings, are also high. Epithermal ore bodies are shallowly emplaced in tension fractures in the rocks and, although values are often laterally extensive, they appear to be restricted to vertical depths of about 600 m (Hutchison, 1985). Being closest to the surface, these deposits are exposed to atmospheric weathering and erosion during the most violent stages of uplift and derived sediments thus have low preservation potential, particularly after long exposure to landscape denudation. Climatic conditions play an important role in chemical decay of rock, as does topography. Humid conditions accelerate decomposition and areas of low relief allow the reactants a longer time to proceed to completion. Rapid run-off of surface water is effective in removing the products of chemical change but militates against reaction time. For all of these reasons, gold deposits derived from epithermal ores are preserved mainly in Tertiary and younger sedimentary successions.

Large-scale fracturing in intrusive bodies stems from stresses set up during cooling where the force of rising magma below causes the roof or hood zone to yield; such stresses are transmitted to the immediate country rock. Concomitant tectonic forces also play a significant role on a regional scale causing all rock formations to fracture over a wide area. Intrusive rocks themselves exist within a uniform stress field, in addition to external or regional forces, when confined by surrounding rocks prior to erosion. As erosion progresses the stresses within the rock are dissipated. Each type of rock-forming mineral has a different modulus of elasticity and this leads to the development of miriads of microfractures and to the loosening of the constituent grains – rock disintegration on a small scale (grain by grain) is initiated. Deposits, which form much deeper in the Earth's crust, have a much longer preservation potential. There is some evidence to suggest that gold-rich deposits, possibly of porphyry style, may be widespread in Archaean greenstone belts (Symonds *et al.*, 1987).

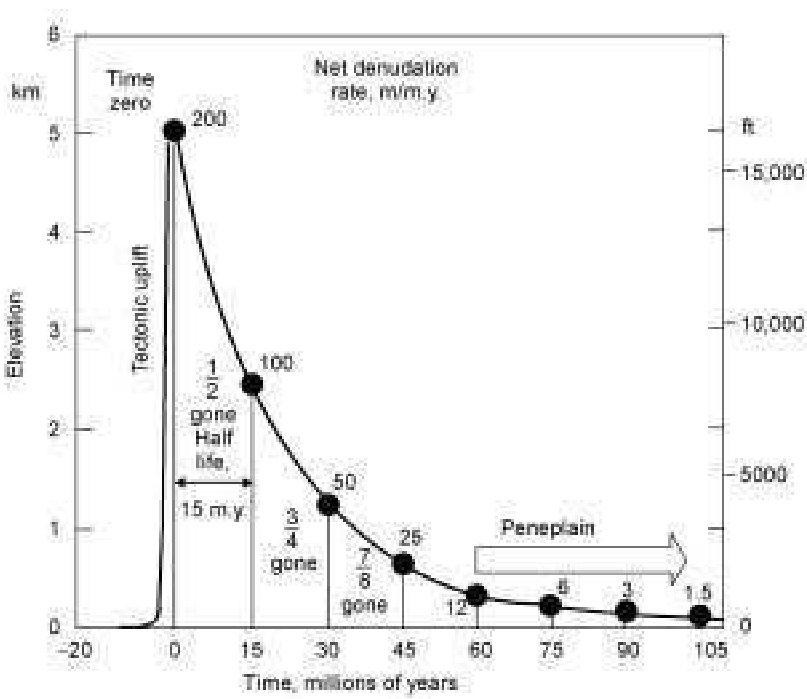
The course of erosion can be expressed in terms of a ratio of mechanical to chemical weathering. In high rugged terrain mechanical erosion is dominant because of the steepness of slopes and lack of vegetation. As time passes and land surfaces are reduced in altitude, the slopes flatten until ultimately, the topography will be so lowered in level that only a low undulating land surface remains. The form and depth of weathering is controlled by the water table and the permeability and chemical reactivity of the country rock. The effects of various physical and chemical conditions at different depths of the weathering zone are recorded by the development of gold grain morphology including metamorphic growth, hydrothermal deposition and alteration, supergene deposition and biological change. Eluvial and colluvial gold placers, which represent the first significant stages of lateritic placer formation are usually both formed



2.26 Schematic diagram of landmass denudation. In this model, the average surface elevation is reduced by one-half every 15 million years. (From A.N. Strahler, *Physical Geography*, Harper and Row, Publishers. Copyright by A.N. Strahler.)

and consumed during the same cycle of erosion. They are restricted to regolith that is typically shallow and geologically short-lived. The total depth of weathering and thickness of the residual regolith overlying an ore zone is typically restricted to a relatively immature and transient surface layer of eluvial rubble and a thin colluvial mantle of sandy-clay saprolitic subsoil, locally with exposures of fresh bedrock.

The Strahler model of landmass denudation assumes that the average surface elevation is reduced to one half every 15 million years as shown conceptually in Figs 2.26 and 2.27. Within this scenario, an epithermal gold-vein system emplaced in a zone 1 km below the ground surface would be subject to much greater erosive forces than a mesothermal ore body emplaced at a depth of 5 km below the surface. The epithermal orebody could be unroofed within a few million years of uplift. The mesothermal orebody would probably not be uncovered until at least 105 million years of uplift. It would also have a much better chance of being deeply weathered chemically on flatter slopes prior to mechanical erosion following renewed uplift. The Raigarh alluvial gold field in Madya Pradesh, India, is an example of almost total erosion down to the lower level of primary gold mineralisation. The Raigarh field occupies an area of about



2.27 Graph of increase in average surface elevation with time, as shown in Fig. 2.26. (From A.N. Strahler, *Physical Geography*, Harper and Row, Publishers. Copyright by A.N. Strahler.)

1,000 km^2 and has yielded large quantities of placer gold for hundreds of years with no evidence or record of underground workings.

With decreasing relief the stability of slope material increases in response to the decreasing amount and size of the material in transport. Slopes then tend to be provided with increasing levels of plant growth and in regions of heavy sub-tropical rainfall a profusion of tree and plant roots shatter the near-surface rocks thereby increasing infiltration and decreasing surface run-off. The crumbling effects of plant growth and biological weathering processes increase contact between rock and circulating water and expose the surface rocks to deeper and more extensive alteration. Ultimately, the topography will be so lowered in level that only a low undulating land surface remains and sediments of all types may overlie a deeply weathered regolith. In such regoliths, the chemical mobilities of elements are determined by the stability of their primary host minerals, the presence or absence of secondary host minerals and the changing weathering environments to which they have been subjected. Differentiation of the remnants of weathered ore zones from the rest of the landscape is made more difficult by the continued action of other processes, which continually modify the landscape.

3

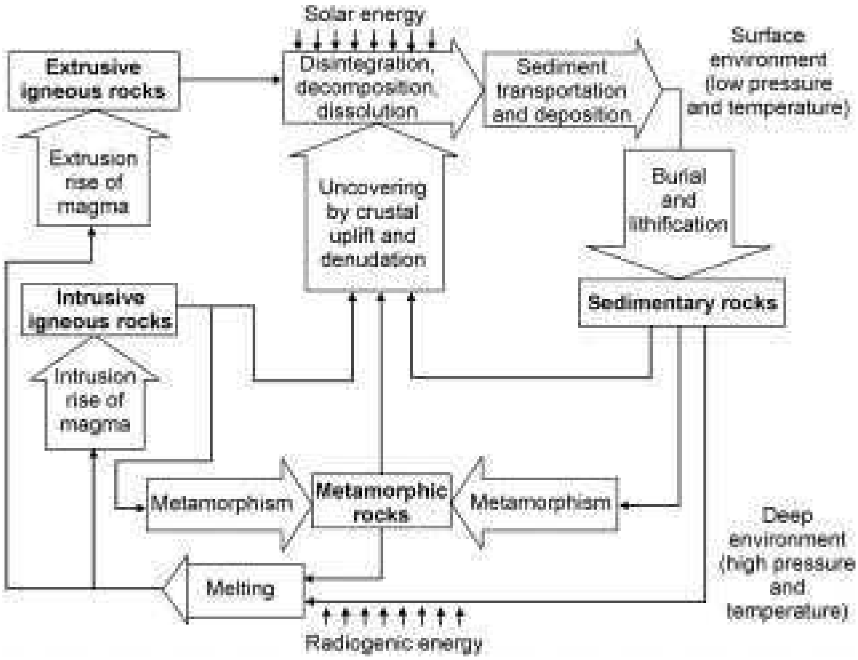
Gold deposition in the weathering environment

Alluvial and lateritic gold deposition in the weathering environment involves a complex and interrelated sequence of chemical, physical and biological weathering processes within the regolith of waste *in situ*, above or in the immediate vicinity of gold source rocks. Climate and topographic relief govern the rates of operation of the different weathering processes at the atmosphere-lithosphere interface; various rock types are progressively exposed to changing environmental conditions and as time passes and relief diminishes zones of mineralisation undergo complex sequences of change. The gold grains are liberated and modified by largely chemical means at various depths in the weathering zone and on slopes. Long-term changes and reversals of climate are accompanied by changing water tables and deep-seated chemical weathering. Eluvial and colluvial gold placers, as short-lived products of weathering, are related to processes of the present environment in terrains of moderate to low relief. Saprolitic and lateritic placer types are formed on flat slopes by processes that relate to past climatic regimes of long duration in deeply weathered regoliths.

3.1 The plate tectonic rock cycle

The plate tectonic rock cycle asserts that the Earth's physical/chemical evolution is driven by energy from the interior and that every geological feature is formed under a specific set of tectonic conditions. Minerals and rocks are stable only under the conditions at which they form and as conditions change so too do the rocks change in order to achieve equilibrium with the new conditions. Igneous rocks evolve into sedimentary rocks when exposed to weathering and erosion at the Earth's surface, sedimentary rock changes under stress into metamorphic rocks, and metamorphic rocks eventually melt to once more become igneous rocks.

A fundamental concept is that all rocks are related to one another, and can be transformed each to the other. Figure 3.1 suggests how these transformations may take place, distinguishing between a surface environment of low pressures and temperatures (sedimentation and weathering processes) and a deep environ-

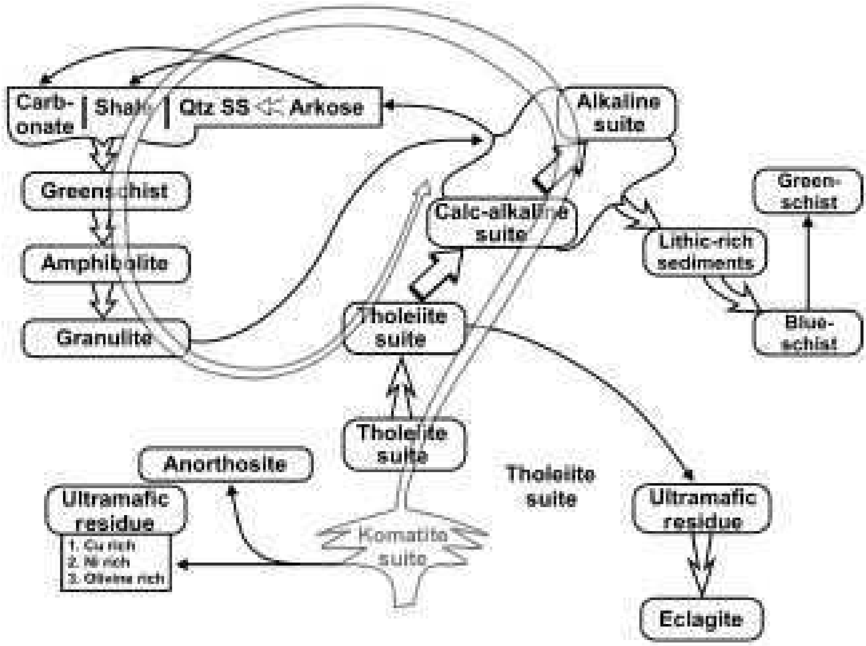


3.1 Cycle of rock transformation (after Strahler and Strahler, 1992).

ment of high pressures and high temperatures (a realm of igneous intrusion and metamorphism). In its complete form this diagram stresses that mineral matter is continually recycled through the three major rock classes as an introduction to the dynamic geological system (plate tectonics), which provides an additional perspective to the cycle of rock transformation. Occurrences of detrital gold and primary gold are now seen as integral parts of a cycle in which the rocks are related to one another and can be transformed one to the other.

The plate tectonic rock cycle, as illustrated in Fig. 3.2 is a complete summary of the processes that lead to the evolution of the Earth. The cycle begins with the generation of mafic oceanic crust by fractional melting of ultramafic magma at an oceanic or any other divergent rift zone. A rising mantle convection cell brings the ultramafic parent magma to the surface creating a mafic melt, which forms the oceanic crust, leaving behind an ultramafic residue rich in Cu, Ni and olivine. The parent mafic magma is designated the komatite suite. The major paths of flow through the cycle are arrowed. The komatite suite passes through the tholeiite >> calc-alkaline suites, to the sedimentary processes through the Barrovian metamorphism (greenschist >> amphibolite >> granularite), and turns back towards the calc-alkaline and alkaline suites.

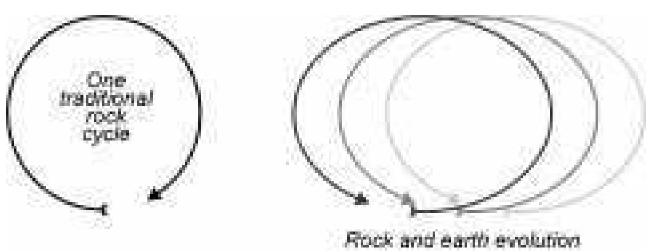
As illustrated diagrammatically in Fig. 3.3 the tectonic rock cycle appears to operate in a neverending cycle with each round of the cycle increasing the



3.2 Plate tectonic rock cycle (Derived from Fichter, 1999).

diversity of the rocks and increasing the volume of felsic igneous rocks. The composition of the rocks thus gets progressively lower on the Bowen Reaction Series scale with each repeated fractionating of material by the parent Earth processes. Indeed, although no record remains of the original rocks of the Earth's crust the parent mafic in the Archaean must have generated an oceanic lithosphere quite different from the parent mafic magma today. Similar changes occur in the chemistry of the sedimentary processes each time the sedimentary rocks are metamorphosed to the metamorphic stage, the result being an igneous rock lower on the reaction series.

Note that most of the evidence for the above interpretation is based upon the geological history of the Cainozoic Era and more particularly upon the erosion



3.3 Multiple tectonic rock cycles.

and recycling of Tertiary placers during rapid climatic changes in the Pleistocene. Because of the effects of multiple metamorphic phases, repeated deformation and discrete thermal events (e.g. magmatic intrusion) are not always observable on Precambrian rocks by petrography. Whether a particular rock is of Archaean or Proterozoic age may be determined only by its syngenetic concentrations. 'It is not enough for a gold bearing quartz vein to be within Archaean rocks for the mineralization to be Archaean, it could be anything from Archaean to Recent' (Goossens, 1983). Anhaeusser (1981) even suggests that post 3.8 billion-year Archaean ore deposits are essentially secondary in origin. The principal factors determining the resistance of the rocks to chemical and mechanical decay, are the chemistry of the rocks and physical properties of hardness, toughness, cleavage and texture.

3.1.1 Igneous rocks

Igneous rocks constitute about 80% of the Earth's crust. They solidify from molten or partly molten silicate magma in which the oxide of silicon ranges from about 45% to 75%. Two main categories are distinguished by the abundance and composition of the major phases particularly quartz, felspars and ferro-magnesium minerals, and whether the rocks are intrusive or extrusive. Extrusive varieties include lava flows that reach the surface in either a molten or partly molten state, and molten ash that has been blown apart by the explosive action of dissolved gases as pressure is released. Intrusive igneous rocks crystallise from magmas that do not reach the surface.

Rocks are 'felsic' if they are high in silica and 'mafic' if they are low in silica and high in ferro-magnesium minerals (e.g., pyroxenes, amphiboles and olivines). Rocks in which the silica content exceeds about 60% by weight contain quartz and alkali feldspars with or without muscovite. They are more resistant to weathering and hence are better represented in sediments than are the darker-coloured mafic varieties.

Colour is an important diagnostic tool. The colour index of a rock is defined as the volume percentage of dark or ferro-magnesium minerals; the lower the index, the more felsic and silicic the rock. Table 3.1 lists the colour indices of common igneous rocks having quartz contents of less than 10% and more than 10%.

Intrusive varieties

Because of slower cooling, intrusive igneous rocks are coarser grained and generally retain a higher proportion of hydrous phases than do their extrusive counterparts. The rock varieties range from the most felsic (granite) through granodiorite, diorite, gabbro, and peridotite to the most mafic (dunite).

Massive discordant igneous bodies are called subjacent because they form in a

Table 3.1 Typical mafic minerals and colour indices of common igneous rocks (after Ernst, 1969)

Rock type	Mafic minerals	Colour index
Less than 10% quartz		
BASALT, gabbro	olivine, augite ± hypersthene	35–65
ANDESITE, diorite	hypersthene ± augite ± hornblende	20–45
LATITE, monzonite	hornblende ± biotite	10–30
TRACHYTE, syenite	biotite ± hornblende	0–20
More than 10% quartz		
DACITE, granodiorite, quartz-diorite	hornblende ± augite ± biotite	20–50
QUARTZ-LATITE, quartz-monzonite	hornblende ± biotite	10–25
RHYOLITE, granite	biotite ± hornblende	0–15

lower situation but not directly underneath overlying strata. They are termed batholiths if their present outcrop areas exceed about 100 km^2, or stocks or pendants if they are less extensive. The Sierra Nevada batholith of California, the Boulder batholith of Montana and Idaho, and the Coast Range batholith of British Columbia are all associated with clusters of stocks and other inclusive bodies that are directly connected with extensive and massive gold mineralisation.

Tabular intrusions comprise discordant dykes and sills. Dykes occur along fissures that cut obliquely across pre-existing country rocks. The largest dykes are major features; the Great Dyke of Rhodesia stretches for 500 km in length and ranges from 3 to 12 km in width. As with dykes, all principal magma types occur in sills. Laccoliths are small intrusions of the more felsic rock types that have a planar bottom and are domed due to arching of the overlying country rocks. Lopoliths are very large floored intrusions, concordant with the upper contacts. They result typically from the intrusion of predominantly mafic magma.

A third type of intrusive body includes relatively small, discordant plutons as represented by the filled conduits of eroded volcanoes. They consist of any rock type that occurs as lava and are appropriately referred to as volcanic necks. Each of the intrusive rocks represents the emplacement of more than one body; volcanic necks are associated with dyke swarms.

The weathering characteristics of individual crystals differ largely due their variable grain size. Coarse-grained feldspathic rocks of the more basic varieties are particularly susceptible to chemical decay either due to the alteration of individual crystals or of the matrix binding them together. Of the felsic varieties, granitic rocks are either very resistant to chemical or physical change or weather rapidly, depending on the coarseness of their grain size and the relative abundance of felspar and ferro magnesium minerals. The crystals of fine-grained granitoids, being closely interlocked, are less affected by chemical attack than the coarser-grained varieties and hence less prone to mechanical disintegration.

Extrusive varieties

The physical properties of extrusive igneous rocks depend largely upon the viscosity of the liquid phase of erupting magma and the nature and abundance of its gas content. Viscosity is a function of the chemical composition and temperature of the magmatic fluid, its content of rock fragments broken off as it rises to the surface and its gas content, either dissolved or as bubbles. In order of increasing viscosities, the magmas are mafic (basaltic), rhyolitic and felsic.

Sudden cooling and solidification of lava at the surface is accompanied by an equally sudden and sometimes violent release of pressure. Major eruptions are explosive in action, forming typically steep-sided cone-shaped volcanoes. Minor eruptions occur as bubbles combine in the magmatic fluid and burst their way out at the surface. Basaltic lava, being of low viscosity, releases its gas content easily and tends to flow rapidly and evenly away from the volcanic centre. The greatest volumes of lavas are issued from fissures in the ocean floor. On the continent, lavas accumulate to form broad plateaux, which approach a kilometre or more in thickness and tens of thousands of square kilometres in area. In some major goldfields (e.g., California, USA; Victoria and NSW, Australia) basaltic lava flows cover large areas of valley floors, thereby protecting any underlying Tertiary gold placer deposits from erosion. These buried placers described in Chapter 4, are usually referred to as 'Tertiary' deep leads.

Evidence for mineralisation is small within presently active arc regions, where large andesitic volcanoes dominate the landscape, although some exhalative deposits can be important. Being more viscous than basaltic magmas, andesitic lavas retain a higher gas content than basaltic lavas because the gas does not escape so readily. However, in terrains where superficial deposits have been cut through by erosion, evidence of mineralisation becomes more common. Slightly altered lavas, which are intermediate in composition between basalt and rhyolite are the principal rock types associated with primary gold-silver mineralisation of epithermal type, while porphyry copper deposits are found associated with intrusive diorites and granodiorites in the sub-volcanic environment.

Although rhyolitic magmas may originate from the simple differentiation of basaltic magmas, processes of melting of crustal rocks and magma at colliding oceanic-continental or continental-continental plate boundaries also contain the essential ingredients for producing rhyolitic type magmas. Completely crystallised rhyolites consist largely of alkali felspar and quartz or other polymorph of SiO_2, minor mafic minerals and subordinate oligoclase. Porphyritic varieties contain numerous large crystals (phenocrysts) disseminated through the matrix. Lesser amounts of glassy rhyolite include such rocks as obsidium, pumice and perlite. The large variations in composition, which occur between rhyolites and basalts, suggest that some association with an intermediate granitic stage may be necessary to their formation. The melting of some mixtures of crustal rocks and sediments in a subduction zone gives melts of granitic origin.

Residual products of weathering of volcanic rocks especially basaltic and andesitic rock types include a variety of ultra-fine sediments such as clay minerals and silt. Difficulties in predicting the hydraulic behaviour of such particles pose many problems in commercial mining operations (see Chapters 7 and 8). Size is only one consideration. Physical properties such as dilatance and plasticity affect such problems as rates of carry over of finely divided gold particles and the provision of ample space for predicted volumes of slimes disposal. Dilatance relates to wave motions set up during settling. Plasticity affects the rheology of the slime and the ease with which the solid/water mixtures deform. Because of differences in such properties, slimes fractions derived from andesitic rocks tend to have higher settling rates than basaltic slimes thus causing fewer plant operational problems.

3.1.2 Sedimentary rocks

Although igneous and metamorphic rocks together comprise about 95% of the total volume of rock types in the Earth's crust, they are predominantly covered by sedimentary rocks and thin surface layers of sediment. About three-quarters of the surface of continental platforms and almost all of the marine basins are covered by sediments that have been reworked chemically or mechanically from pre-existing rocks of any origin. The processes involve weathering of source material, transportation by water and airflows, deposition and lithification.

The components of sedimentary rocks are simpler than the rocks from which they derive but are still variable in their resistance to forces imposed by weathering, erosion and transport. These forces are mainly dependent upon topography, climate and geochemistry. Topography is a major factor influencing the nature of sediment movement from source to the area of deposition. The depth of weathering is a function of the rate of *in-situ* chemical weathering compared to the rate of mechanical removal. Organic acids that promote the chemical breakdown of rock materials are released by the decay of plant and animal waste. Elevated temperatures and the availability of adequate water for chemical activity and the mobilisation of waste products enhance most of the chemical reactions.

Two separate types of sediment (continental and marine) reflect the sometimes rapidly fluctuating depositional conditions of their environments of deposition. Continental sediments grade gradually from rapid mass movements of mud and debris flow down to the slow accumulation of sands and muds on river flats. Marine sediments grade from the shallow granular deposits of beaches and deltas and continental slopes, to deposits of wind-blown dust and chemical sediments flooring the ocean basins.

The clastic end products of weathering of pre-existing rocks are quartz sands, clays and hydrated iron oxides, which eventually pass into shales, sandstones and occasional ironstone. Chemical rock types, which originate as chemical precipitates are formed either as supersaturated solutions or by the biochemical actions of marine organisms. Chemical precipitates are roughly homogeneous in composition and are characterised by an interlocking crystal texture in which the crystal sizes vary widely. The components of sedimentary rocks are simpler than the rocks, from which they derive, but still vary in their resistance to weathering. The most important textural property is the size of the individual particles. Based upon particle size, sedimentary particles are divided into conglomerate (2–4 mm) gravel, (larger than 2 mm), sandstone (0.0625–2 mm), siltstone (0.0004–0.0625 mm), and claystone (smaller than 0.0004 mm). Sedimentary rocks containing grains of different sizes are classified on the basis of the dominant particle size. Conglomerate is the name given to a consolidation of water-worn pebbles in a matrix of constituent rock and mineral fragments, agglomerates and fine cementing materials such as carbonates and iron oxide.

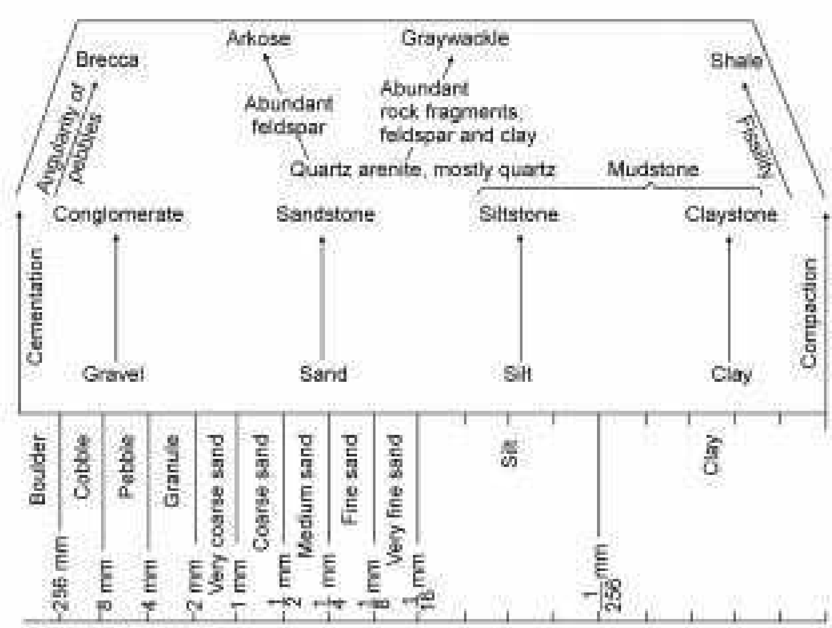
Of the various sedimentary rocks, shales and sandstones take up water most easily and are susceptible to mechanical breakdown by frost action. High density jointing in some sandstone promotes rapid weathering by both chemical and physical means. Being weakly compacted, they break down into irregular fragments and eventually revert to clays and silts. Basal conglomerates associated with each fresh cycle of erosion are the most important deposits in many alluvial gold areas. Notable examples are the Banket conglomerates of the Witwatersrand, South Africa, which have contributed more than one-half of the gold produced in the world today. Figure 3.4 is a sedimentary classification of rocks and their derived sediments.

3.1.3 Metamorphic rocks

Metamorphic rocks comprise igneous and sedimentary rocks that have been altered from their primary states at higher temperatures and greater pressures than are normally present at the Earth's surface. They represent products of both mechanical shearing and crushing for which chemical and mineralogical changes are negligible and recrystalised rocks in which new crystals are formed although hindered in their growth by old minerals. Their structures reflect the physiochemical environment in which they form and thereby the genesis and history of the metamorphic rock (Barlow and Newton, 1974).

Metamorphism may result from orogeny and the intrusion or extrusion of magma, or by interaction with migrating fluids from an external source. All gradations of change are displayed according to differences in the type and intensity of the metamorphic processes:

- Dynamic metamorphism due to pressure along dislocations in the Earth's crust, is local and restricted in occurrence.



(a) Detrital sediments

Rock	Limestone	Dolomite	Iron formation	Evaporite	Chert	Organics	Phosphate
Chemical composition	$CaCO_3$	$CaMg(CO_3)_2$	Fe silicate -oxide carbonate	NaCl $CaSO_4$	SiO_2	Carbon	$Ca_3(PO_4)_2$
Minerals	Calcite (aragonite)	Dolomite	Hematite Limonite Siderite	Gypsum Anhydrite Halite other salts	Opal Chalcedony Quartz	Coal Oil Gas	Apatite

(b) Chemical sediments

3.4 Sedimentary rock classification (a) by grainsize on the horizontal scale, (b) by chemical composition (after Press and Siever, 1972).

- Contact metamorphism occurs in response to increased temperature from adjacent magmatic intrusion; chemical changes take place due to magmatic exhalation and/or pressure effects.
- Regional metamorphism occurs as a result of an increase in temperature and pressure during the formation of folded mountains; pressure is also generated by shear forces, which accompany the orogenic movements.

Tectonic movements, whether orogenic or epeirogenic, result in the fracturing and breaking of country rocks. Surface rocks are broken apart under conditions of great stress during mountain building and tend to fracture thus enhancing the actions of weathering and erosion. Deeply buried rocks, already subject to high

Table 3.2 Metamorphic rocks (from Whittow, 1984)

Metamorphic grade	Sandstone	Limestone	Mudstone	Clay	Basic lava
Low	quartzite	marble	slate	slate	greenstone
Medium	quartzite	marble	mica schist	amphibole	amphibole
High	quartzite	marble	gneiss	amphibole	amphibole

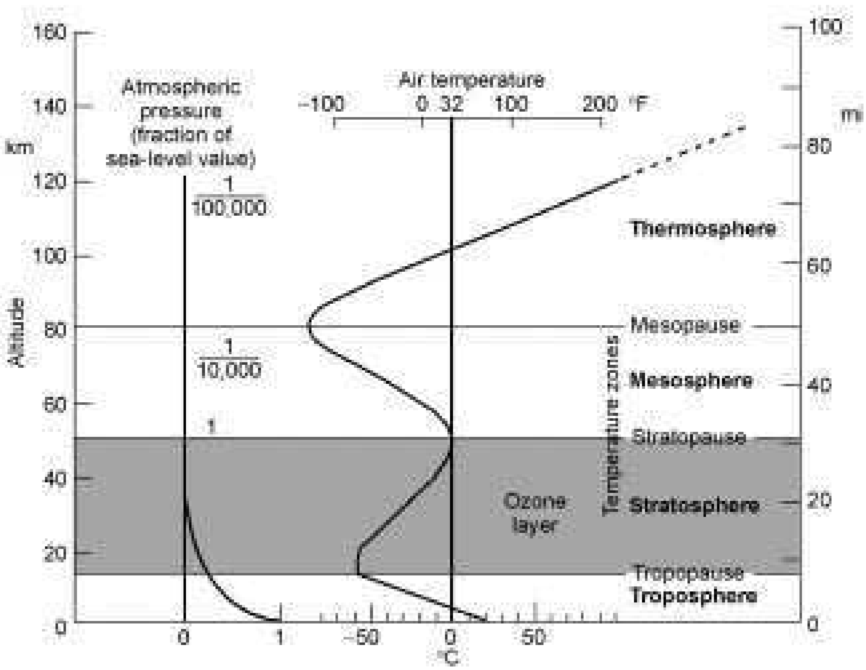
temperatures and pressures, tend to become plastic and warp and fold when subjected to additional prolonged stress. Because there is little recrystallisation and chemical action, any structures present will have been formed principally through granulation and shearing and the moving out of fragments of pre-existing material. Friction breccias, which may result from crustal deformation, comprise abundant fractured and angular rock fragments with dimensions ranging from about a millimetre to a metre in length. Gneiss forms as the result of more intense shearing and recrystallisation and the development of almond-shaped grains or grain clusters set in finely grained matrix.

Contact metamorphic rocks form as the result of pronounced temperature increases in the virtual absence of increased stress. Physical change occurs when magmatic fluids, which often bring about recrystallisation, permeate pre-existing minerals. These rocks are characterised by alternating bands of darker minerals such as chlorite, biotite mica and graphite. The areas of contact called aureoles or concentric shells surround hot igneous bodies emplaced at upper levels.

The most durable metamorphosed sediments (metasediments) are silica-cemented quartzites. Resistant quartzitic types comprise an almost pure quartz sandstone with quartz cement, other quartz sandstones (e.g. carbonated sandstone) and a chemical variety of chert (monomineralic quartz rock composed of interlocking cryptocrystalline quartz veins). Of other common rocks in a gold placer environment, limestone is mechanically resistant but is easily broken down chemically, slate is chemically durable but splits easily along well-defined cleavage planes, and coarse grained metamorphic rock such as gneiss exhibits a weathering pattern similar to the coarser grained granites. Schistose rocks, particularly those containing large plates of mica, are more susceptible to weathering than most other schists because of the easy penetration of aqueous solutions along the micaceous layers. The above rocks are described in terms of metamorphic grade in Table 3.2.

3.2 Earth's atmosphere and climate

The Earth's surface comprises atmosphere, land and sea each one exerting an influence upon the properties of the others. Atmosphere is a shallow mixture of gases surrounding the Earth held by gravity. The atmosphere is subdivided into



3.5 Temperature structure of the atmosphere (after Strahler and Strahler, 1992).

four major environmental layers: 'troposphere', 'stratosphere', 'mesosphere', and 'thermosphere', based upon zones of temperature change. The atmopshere is densest at sea level, thinning rapidly upward (Fig. 3.5). Climate is a generalised term that describes prevailing conditions in a given region, e.g. temperature, humidity, cloud cover and height, wind speed and direction, and precipitation. These are also functions of weather but whereas weather, as a small component of climate, refers to meteorological conditions at the time of reading, climate is the average of meteorological conditions over long periods of time and takes account of extremes, averages and frequencies of individual readings.

3.2.1 Sun–Earth space energy relationship

Solar radiation

Atmospheric temperature is a function of the modification of solar radiant energy by air, clouds, land, sea and other water surfaces. Solar radiation is discharged from the Sun in the form of electro-magnetic radiation divided roughly into:

- ultraviolet and shorter wavelengths (7.8%)
- visible light (47.3%)
- infra-red light (44.9%).

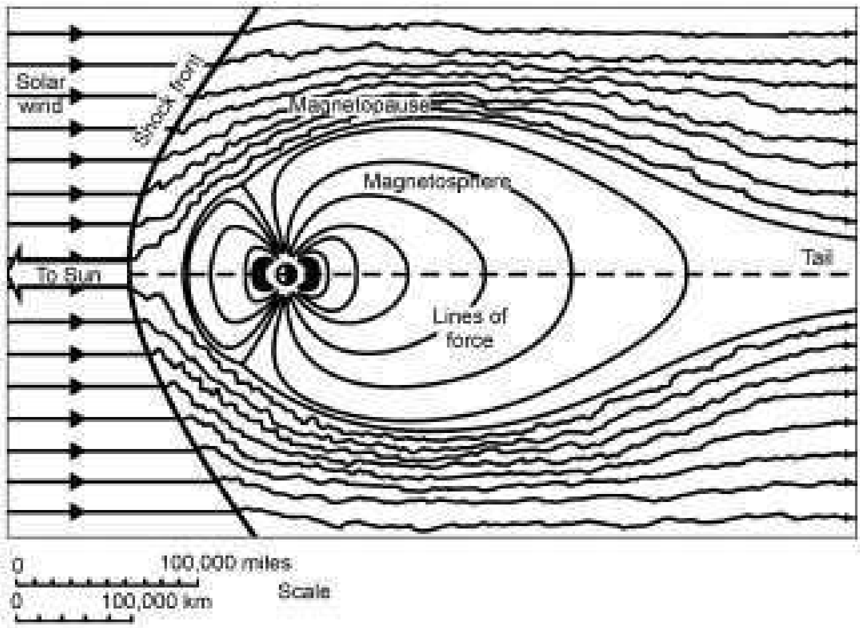
Temperature drops uniformly in the troposphere at around 6.4 ºC per 1000 m of ascent. It then changes abruptly at the tropopause (transition into the stratospheric zone) at about 14 km. After rising in the stratosphere, another abrupt change takes place in the stratopause at about 50 km before again falling sharply in the mesosphere. At about 80 km elevation the temperature levels out again in the mesopause, again rising rapidly in the thermosphere before entering into an electrified region, the ionosphere, then through the exosphere, into space. The heights of the various boundaries vary with latitude and season. In the troposphere, where most of what happens of direct importance to mankind occurs, the height of the tropopause is 5 to 10 km higher in equatorial regions than in polar regions where it is fairly stable at an elevation of about 10 km.

The intensity of solar radiation when the Earth is at its mean distance from the Sun just outside of the Earth's atmosphere is termed the 'solar constant'. It is measured as the strength of the energy received on a unit area of horizontal surface perpendicular to the Sun's rays in unit time. The mean value of the solar constant is 1.94 langleys/min, the lower values occurring in the shortest part of the spectrum. Changes have also been recorded up to 2.04 langleys/minute (Whittow, 1984) due, probably, to deviation from regular motion of the Earth's orbital elements as a result of gravitational attraction from other bodies in the solar system (e.g. Jupiter and Saturn).

The magnetosphere

The Earth also has a magnetic atmosphere from which lines of force of the Earth's magnetic field pass outward into surrounding space thus creating the Earth's external magnetic field. The entire region within the limit of the magnetic field is called the 'magnetosphere'; its outer boundary is termed the 'magnetopause'. The plane of the magnetosphere lies in the plane of the magnetic equator. The Earth occupies a position that is determined by the action of the solar wind, a more or less continual flow of ionising radiation from the sun, which acts to press the magnetopause close to the Earth on the side nearest to the Sun. The result is that lines of force are concentrated in this region while the force lines on the opposite side are greatly attenuated (Fig. 3.6). The magnetosphere shields the inner atmosphere and surface of the Earth from the fast-moving stream of electrons and protons of the solar wind that would otherwise destroy all life on Earth. These particles are trapped and retained within the force lines of the magnetic field upon encountering the magnetosphere and concentrated into a belt of intense ionising radiation surrounding the Earth termed the 'Van Allen Radiation Belt'. Trapped particles are continually discharged from the tail of the magnetosphere.

Solar flares of ionised hydrogen gas produced by sunspots on the Sun's surface reach the Van Allen Belt about 13–26 hours after they are first sighted. Electrons trapped by the Earth's magnetic field are then directed downward in



3.6 Magnetopause and magnetosphere, Van Allen Belt in black on either side of the Earth (after Strahler and Strahler, 1992).

corkscrew fashion along the lines of force suddenly increasing the magnetic field strength many fold. An ensuing magnetic storm simultaneously generates electrical currents in a layer of ionised atmospheric gases (the ionosphere) that sets in at about 50 km from the surface and extends to an altitude of 1000 km. Voltage fluctuations may reach 1,000 to 2,500 volts during violent storms. Dynamic currents measured in thousands of amperes flow in enormous horizontal circular patterns around the Earth giving rise to a series of electrical currents in a shallow layer of the Earth's surface. Technologies that rely upon electronics are affected by magnetic storms in two different ways. Circulating electrical currents disrupt radio communications, which depend upon the reflection of radio waves in the ionosphere; and television systems, which affect the functioning of navigation satellites on which modern global positioning systems rely (refer to Chapter 5). Induced electrical currents in high-tension power transmission lines at ground level overheat transformers and cause major shutdowns.

Insolation

The amount of direct and diffuse solar radiation reaching a unit horizontal area of the Earth is defined as insolation. A heat surplus in the low latitudes, between about 37° N and 37° S of the equator, is reversed in the high latitudes where outgoing radiation exceeds insolation thereby creating a heat deficit. Partial explanations for the non-uniform distribution of heat are given by:

- the oblate spherical shape of the Earth
- the elliptical orbit of the Earth around the Sun, which provides seasonal changes in orbital distance and axial tilt relative to the Sun
- the rotation of the Earth, which causes local changes in insolation throughout each day.

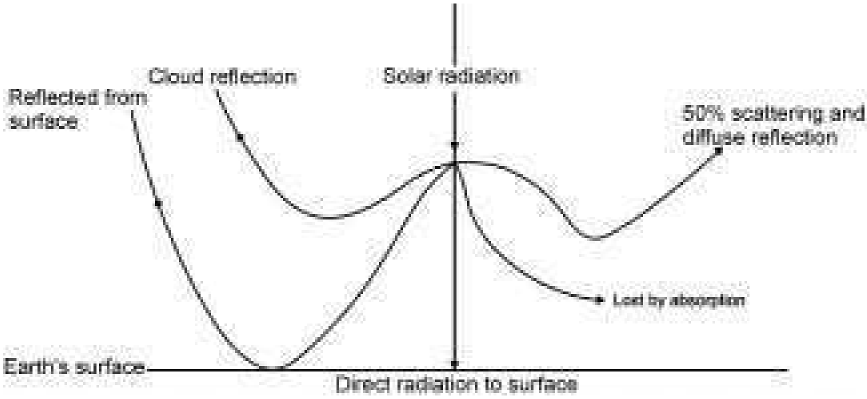
Incoming solar radiation is intercepted at different rates over different parts of the Earth in its passage through the atmosphere. Some 50% of the total annual incident radiation is lost due to scattering and diffuse reflection from solid pollutants in the atmosphere and from cloud reflection back into space. Of the 50% or so of energy received at the Earth's surface, the amount absorbed varies according to the nature of the surface and vegetal cover. The remaining heat is reflected outwards from the Earth's surface by mechanisms involving conduction, radiation and latent heat of evaporation, thereby providing an effective means of heating the air in lower levels of the atmosphere. An estimate of the absorption of solar energy by the atmosphere and at the Earth's surface is given in Table 3.3. According to this estimate, the total absorption (atmosphere and surface) is some 4.7 times greater at the equator than at the poles. The latitudinal variation of insolation (about 2.6 times greater at the equator than at the poles) provides energy differences that are vital elements of atmospheric circulation and terrestrial weathering. The Earth's radiation balance is also developed from other components of the total energy flow including energy absorbed by the Earth, energy transmitted in the form of reflection and long-wave radiation from the Earth into space. Figure 3.7 is a simplified diagram of the Sun's radiant energy.

Geographical and seasonal variations of climate are fundamentally supported by modification of solar radiant energy (insolation) intercepted in the

Table 3.3 Absorption of solar energy at Earth's surface (annual mean, Ly/day)* (after Fritz, 1987)

	0° N	30° N	60° N	90° N
Latitude				
Extraterrestrial	850	740	470	350
AE surface				
With clear sky	570	520	320	220
With normal cloud cover	410	440	200	150
Total absorption				
Atmosphere and surface	570	530	260	120

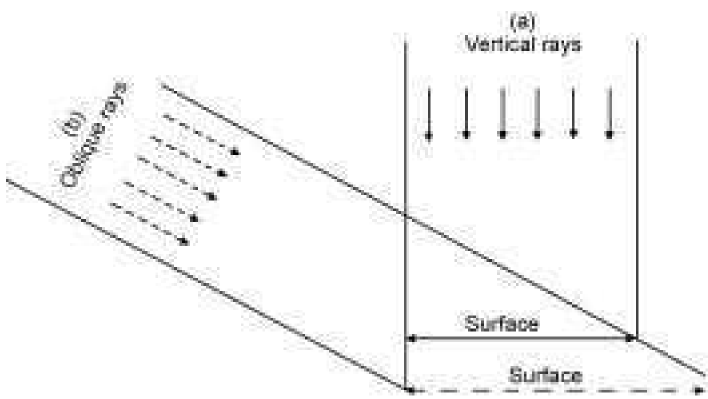
* Langleys per day (1 Ly = 1 cal/cm^2).



3.7 Simplified distribution of Sun's radiant energy.

atmosphere. The seasons are determined by the Earth's axis of rotation relative to the Sun. At present, the Earth rotates with its axis at about 23.5º in relation to the Sun, but over time there is evidence that its axis wobbles between 22.1º and 24.5º, which causes variations in the amount of sunlight received over the Earth's surface and modifies its seasons. According to the Milankovitch theory this phenomenon, which affects the amount of sunlight (insolation) received on the Earth's surface during summer and winter, cycles every 41,000 years.

In polar regions the Sun's rays have more atmospheres to penetrate than at lower latitudes and they strike the ground obliquely, thus spreading the rays over larger surface areas (Fig. 3.8). In permafrost regions, vegetation acts as a cover over the frozen ground keeping it away from the warming rays of the sun. Thus, while early miners in Alaska progressed from using fires, then steam and then water to thaw the frozen gravels underground, they eventually found that by clearing the surface vegetation, sufficient solar energy was absorbed to thaw the ground for mining (Woodsend, 1984). Insolation effects are also apparent in the



3.8 Intensity of insolation at surface from (a) direct and (b) oblique radiation.

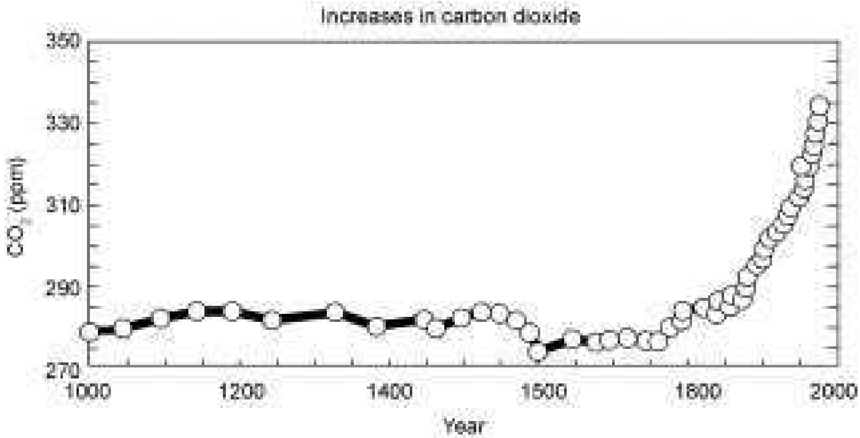
nature of weathering patterns developed on slopes facing in opposite directions in sub-polar and periglacial conditions. Slopes facing north in high northern latitudes remain snow-covered long after melting occurs on slopes facing south. This promotes increased chemical weathering and biological activity on the southern aspect, thereby encouraging animal grazing on the warmer side of the hill and adding to mass-wasting along that sector. North-facing hillsides are weathered mainly by frost action, slopes become steeper and more prone to landslides. The effects of insolation on slopes in southern latitudes are mirror images of those in the north.

The greenhouse effect

Gases in the troposphere include dry gases, which are fundamental to life, and others such as water and industrial gases. The natural 'greenhouse effect', which prevents temperatures dropping excessively when the sun goes down or in wintertime in middle and high latitudes, is largely due to the presence of concentrations of water vapour, carbon dioxide, methane, CFCs, halons and nitrous oxide in the lower levels of the atmosphere. Rapid condensation of water vapour provides precipitation in the form of rain, hail, snow and sleet. Fresh water is supplied for plant and animal life, and preserves the global water balance between that leaving the land and ocean surfaces by evaporation and that falling on the land and oceans by precipitation. Impurities cause environmental problems, however. Human activities result in the production of myriad microscopic dust particles from forest and grass fires, from industrial processes involving fuel combustion and from the vaporisation of meteors as they encounter the heat barrier in the upper atmosphere. Residues of minute crystals of salt also contribute to the troposphere as water lifted from the ocean dries out; more heat energy is retained in the troposphere and global temperatures rise.

According to scientists working on the Intergovernmental Panel on Climatic Change (IPCC), global warming over the 20th century has resulted in an average increase in air temperature of about 0.6 ºC and a corresponding mean sea level rise of 10–20 cm. Computer models suggest that global temperatures and sea levels will continue to rise during the 21st century although possibly at lower rates. These changes have occurred largely because of the additional amount of heat energy trapped by the absorption and emission of some of the infra-red radiation and warming of the lower atmosphere during the latter part of the 18th century. CO_2 appears to be the main contributor to the enhanced greenhouse levels as shown in Fig. 3.9, the CSIRO graph of the sharp increase of CO_2 levels in the atmosphere since AD 1000.

Another important influence on the enhancement of greenhouse levels is the presence of an ozone layer within the stratosphere, which provides a concentration of oxygen molecules at an altitude of about 15 km and extends upward to



3.9 CSIRO graph of increase of CO_2 levels in the atmosphere since AD 1000.

about 55 km. Ozone is a form of oxygen, (O_3), having three atoms to the molecule. It is produced by the action of solar radiation on oxygen, and is a powerful oxidising agent. The ozone layer guards the troposphere and the land surface from the effects of excessive amounts of ultraviolet radiation from the Sun's rays, thus playing an essential role in environmental protection. Synthetic compounds containing carbon, fluorine and chlorine atoms released into the atmosphere pose a serious threat of global warming by attacking molecules of ozone and converting them back to oxygen. So-called 'holes' in the ozone layer at the southern and northern polar-regions have been observed from studies based upon satellite data since the 1980s. The 'holes' relate to thinning of the ozone layer, the maximum thinning occurring at the time of minimum surface temperature. The intensity varies from time to time but a strong general trend towards greater seasonal thinning and global warming is of great concern.

3.2.2 Oceanic–atmospheric relations

The interface between atmosphere and land constitutes a region in which there is a continual interflow of matter and energy with atmospheric conditions exerting control on the land surface while the land surface exerts an influence on the properties of the adjacent atmosphere. A similar interface exists between the atmosphere and the surface of the oceans in contact with it. The oceans influence the atmosphere above it; the atmosphere exerts control over the sea beneath it.

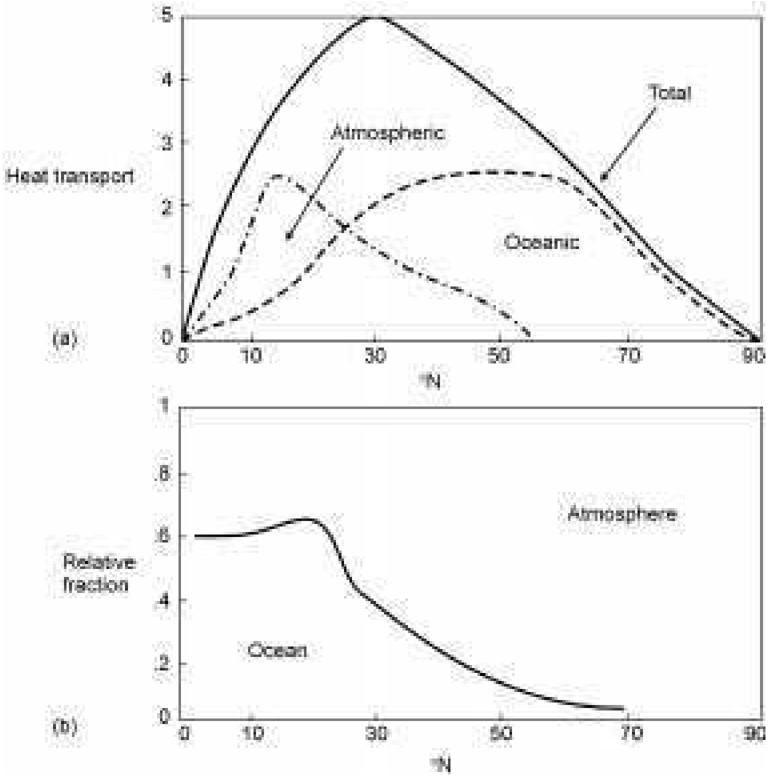
Different reactions of land and sea to insolation significantly affect both global climates and local weather patterns at land-sea boundaries. The oceans warm and cool at lower rates than the atmosphere but due to the greenhouse effect the oceans are slowly warming. This will have a delayed warming effect upon the Earth's atmosphere in the coming centuries, affecting the Earth's other natural systems. Rainfall patterns change as temperatures increase and future generations will experience an increase in the size of arid and semi-arid areas, leading to a greater risk of desertification. There will also be an increase in the incidence of disease, insect attack on existing forests, rising sea levels and coastal erosion.

Oceanic heat flow

The oceans cover about 71% of the total Earth's surface and play a major role in both local supply of heat energy to the atmosphere and regional transfer of heat energy from the tropics to polar-regions. The ocean contains an enormous amount of heat energy that is both gained and lost gradually. In addition to the energising effects of oceanic heat on the atmosphere, the oceans also act as moderating influences on air temperature, weather and climate. Minor surface currents and eddies along continental shorelines have important effects on local weather conditions. Continental weathering is closely linked to the interchange of heat between land and sea and with its effects upon a wide range of factors. Important amongst these are air composition and temperature, relative humidity, insolation, altitude, motion and atmospheric pollution, nature of the water cycle operating in the particular area, rock composition, vegetation, topography and time.

Global transport of surplus solar energy from the tropics to regions of deficient solar heat in the higher latitudes is now thought to be greater in the oceans than in the atmosphere. This is due to the enormous capacity of oceans to absorb and store heat during periods of surplus solar heat. Ehricke (1989) suggests that annually, some 28% of the total energy reaching the Earth is expended in evaporating water and lifting it into the atmosphere. It is estimated that of this amount, some 369,000 km^3 (85.5%) is evaporated from the oceans and 62,500 km^3 (14.5%) by evapotranspiration from the land. Estimates of total oceanic and atmospheric transports given in Fig. 3.10 demonstrate the dominance of atmospheric heat transport north of about 30° N but that oceanic transport is responsible for most of the heat transfer from the tropics.

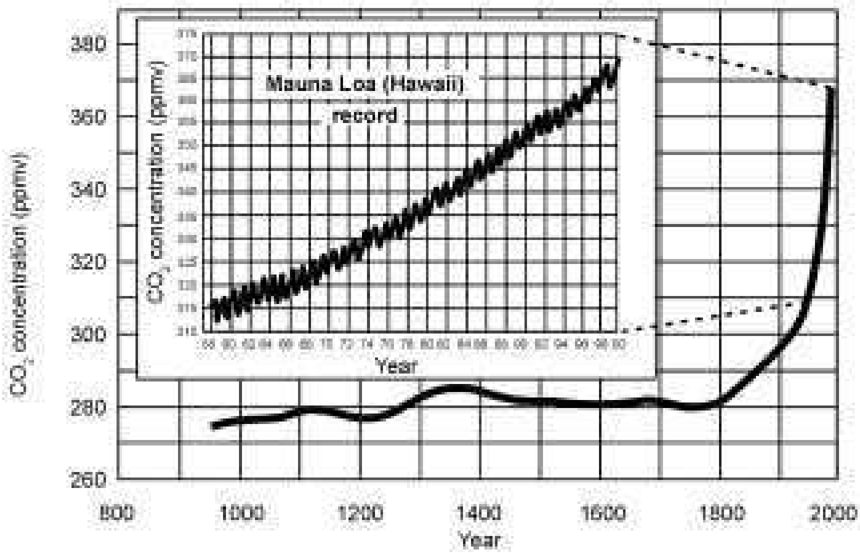
Globally, the general circulation of surface and deep-sea ocean currents depends upon such large-scale systems as the Gulf Stream, the Equatorial Counter-Current and East Australia Current. Wind is the major driving force for surface currents. Deep circulating currents result partly from wind stress and partly from the influence of Coriolis forces, density differences and seabed topography. The effects of the increased contribution of zero-temperature water from melting of ice caps and sea ice on deep sea ocean currents suggests that they may be a factor of vital importance affecting global warming. For example, the largest glacier in South America, the Upsala Glacier is now retreating at the rate of 200 m a year. Most scientists agree that sea level could rise as much as 70 mm if present levels of melting of the Greenland and Antarctic ice sheets and



3.10 Global heat transfer (a) magnitude of total oceanic and atmospheric heat transport in Northern Hemisphere, (b) fraction of total poleward transport accounted for by the atmosphere and ocean (after Vonder Haar and Oort, 1973).

thermal expansion of the oceans continues for the next 100 years. Of more immediate concern are the effects of warmer temperatures on the frequency and severity of violent storms along coastal shorelines and consequent high levels of marine incursions and erosion; and exacerbation of the worldwide problem of desertification.

However, the complex nature of the climate system means that changes cannot always be readily predicted, different parts of the world change in different ways. Predictions of ancient patterns of climate are based mainly upon present-day knowledge of atmospheric physics and the existing geological record based largely upon ages that are constrained by radiometric dating. Based upon ice core analysis and on direct measurements at Mauna Loa, Hawaii, since 1958, the change in the atmospheric concentration of carbon dioxide shows a slight upward trend for the years 800 to 1800, and a very steep upward trend during the past 50 years (Fig. 3.11). Figure 3.12 illustrates the global temperature anomalies during the period 1860–2000. Most of the earlier evidence is



3.11 Trend of CO_2 in the atmosphere (Mauna Loa) record.

fragmentary; it is only during the past 650 million years of Earth history that abundant fossil evidence has shown a relatively even fluctuation of climates between geologically short glacial-interglacial intervals and the long intervening periods of ice-free poles and warm equable climates (Hay, 1987).

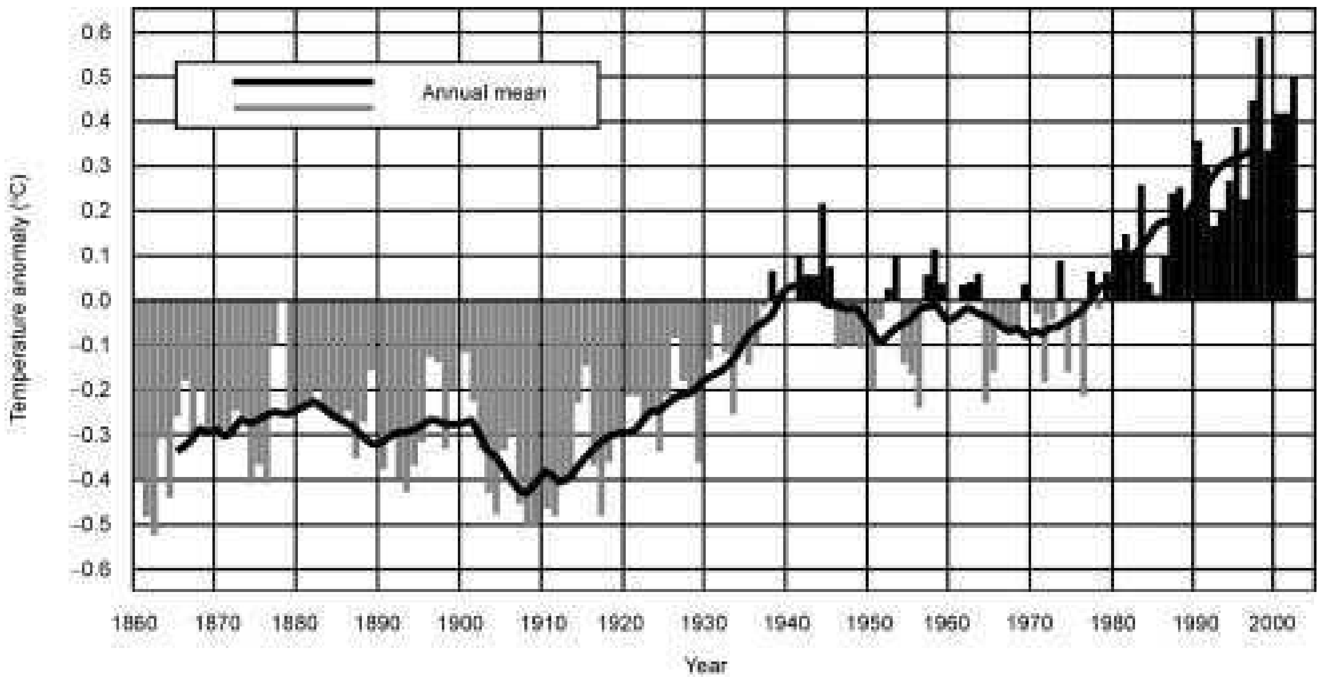
The following examples of global aspects of oceanic-climatic relations illustrate phenomena that have re-occurred throughout Phanerozoic times:

- A current ocean phenomenon involves a mass influx of fresh water into the usually salty North Atlantic, which reaches south from Greenland to the coast of Carolina in the USA.

This may impede the Gulf Stream and so reduce the transport of warm air to the northern latitudes in winter. Some researchers predict that if this trend continues, temperatures could fall abruptly around four degrees centigrade and Earth's climate could enter into a completely different mode of operation. Others suggest that the current trend could be a phase in a natural cycle, and that ice core evidence indicates that similar type events may have happened several times in the last 100,000 years.

- Warming of the Tasman Sea is a disturbing trend that has begun, and will probably continue, significantly to reshape the offshore Australian East Coast environment.

The cause appears to be the movement of the East Australian Current downward from Coral Sea along the Eastern Coast of Australia to the Tasman Sea in the South. The current flows generally at about 4 kpm but has been clocked at as



3.12 Global temperature change 1860–2000.

much as 7 kpm in shallow waters of the continental shelf before eddying out into the Tasman Sea. According to Wen ju Cai, a senior researcher at the CSIRO: 'A shift in wind systems is affecting circulation in the Pacific, Indian and Atlantic ocean basins and this is leading to a rate of warming in the South-west Tasman Sea that is fastest in the Southern Hemisphere.' The resulting strong seasonal pulse of warm water has increased water temperatures in the Tasman Sea by two degrees during the past 60 years whereas global temperatures have risen by only half a degree. The EAC is being pushed further towards Tasmania by a southward shift in the westerly weather systems.

Atmospheric circulation

The principal factors determining the pattern of global circulation are the greater retention of solar heat in low rather than high latitudes, and the rotation of the Earth. Equatorial heating induces comparatively shallow low-pressure systems in contrast to the formation of shallow high-pressure systems in polar regions. Flow patterns are produced as a result of these differences. Masses of hot air, displaced in the tropics and moving poleward, are displaced to the right in the Northern Hemisphere and to the left in the Southern Hemisphere, so that winds in the upper troposphere tend to be westerly. Air masses have a characteristic zoning of temperature and humidity for which values are closely constant at all points in a horizontal plane. Weather fronts are formed when westerlies of middle latitudes and masses of cold air from the poles collide.

Individual air masses reflect the temperature and humidity of the surfaces over which they pass. For each temperature there is a saturation point at which precipitation from water vapour to droplets of water must occur (Table 3.4). The air is in constant motion and clouds form by condensation of the water vapour when its relative humidity reaches 100% in its upwards rise. Hot winds blowing across oceans become increasingly humid until they reach saturation point for the conditions of flow. Precipitation occurs when winds laden with moisture are swept up over steep and high mountains. Rapid cooling gives rise to precipitation on the seaward side, thereby promoting favourable climatic conditions for

Table 3.4 Saturation point for precipitation (after Barlow and Newton, 1974)

Temperature (°C)	Maximum water vapour capacity (g/m^3)
0	4.8
10	9.1
20	17.3
30	30.0
40	50.9

placer formation. Arid and sometimes desert conditions, which result from the dehydration of air streams that continue into the interior, are not conducive to placer formation.

3.2.3 Climate

A statement of the climate of a designated region includes measurements of the above components, summarised over long periods of time. Account is taken of averages and departures of means of values obtained and of probabilities that extreme climatic conditions will recur at roughly predictable, e.g., 50- and 100-year intervals. Such a statement is an essential part of any evaluation exercise and requires careful analysis of both short- and long-term effects of these extremes. Climatic variables include air and ground temperatures, as measures of available heat energy; diurnal and seasonal transfer of temperature through local winds and oceanic circulation patterns; and types and rates of precipitation.

Based upon these three sets of variables, climates may be broadly classified within three morphogenic zones: glacial-periglacial, desert, and humid tropic. There are many sub-divisions of climate within these groupings. Many geographers favour the Koppen Geiger Kohn system, which classifies five principal and eight sub-groups on the basis of meteorological records; equivalents for the 13 climatic types are given in Table 3.5. Geologists usually prefer a simpler classification in which the term ‘selva’ is used to describe an equatorial rain-forest environment; maritime refers to continental margins, and moderate to temperate climates to conditions in regions between torrid and frigid zones. These climates are discussed specifically in this book in relation to the geomorphic history of regions of residual and placer gold deposits. Of particular importance are the effects of climatic change and extremes of change on stresses that produce intensive mass wasting and glaciation on the one hand, and fluvial transport, sorting and deposition on the other. Either directly or indirectly and regardless of age, these changes have all played significant roles in the formation, modification, reconstitution and/or dispersal of detrital gold accumulations. The nature and extent of adjustments after secular decay is examined in Chapter 4 in relation to both long-term and short-term tectonic and climatic change.

Climatic cyclicity

Predictions of ancient patterns of climate are based mainly upon present-day knowledge of atmospheric physics and the existing geological record is based largely upon ages that are constrained by radiometric dating. Most of the earlier evidence is fragmentary; it is only during the past 650 million years of Earth history that abundant fossil evidence has shown a relatively even fluctuation of climates between geologically short glacial-interglacial intervals and the long intervening periods of ice-free poles and warm equable climates (Hay, 1987).

Table 3.5 Classification of world climate (after Barlow and Newton, 1974)

Group	Strahler's name	Koppen's symbol	Descriptive name	Causal factors
Group I Low latitude climates controlled by equatorial and tropical air masses	1. Equatorial rain forest 10° N to 10° S	Af	Equatorial hot-wet	Convergence of moist air masses along zone of I.T.C.
	2. Trade wind littoral 10° to 25° N and S	Af (seasonal)	Tropical eastern maritime	Seasonal occurrence of moist Tm air masses
	3. Tropical desert and steepe 15° to 35° N and S	B climates	Tropical deserts	Subtropical highs and all-year presence of dry Tc air masses
	4. West coast desert 15° to 30° N and S	B climates	Tropical deserts	Stable subdising Tm air masses. Cold currents
	5. Tropical savanna and tropical monsoon	Aw Am	Tropical wet-dry and tropical monsoon	Moist Tm in summer. Dry Tc in winter. Moist Tm in summer
Group II Mid-latitude climates controlled both by tropical and polar air masses	6. Humid sub-tropical 25° to 35° N and S	Cfa	Warm temperate eastern maritime	Moist Tm, frequent invasions of Pc air in winter
	7. Marine west coast 40° to 60° N and S	Cfb	Cool temperate western maritime	Moist Pm all year, frequent cyclonic storms
	8. Mediterranean 30° to 40° N and S	Csa	Warm temperate western maritime	Pm in winter – wet Tc in summer – dry
	9. Humid continental 35° to 60° N	B and D climates	These climates grade from moist Df coastal types (Strahler's 'Humid continental') to arid desert and steppe climates of continental interiors (Strahler's 'Mid-latitude desert and steppe').	
	10. Mid-latitude desert and steppe 35° to 50° N and S	B claimates		
Group III High latitude climates controlled by polar and arctic air masses	11. Subarctic climates 55° to 70° N	D climates Dfc Dfd	Cold continental	Controlled by Pc air. Precipitation from frequent invasions of more moist Pm air
	12. Tundra north of 55° N and south of 50° S	E climates	Cold coastal	Moderating influence of oceans. Frequent storms along fronts between Pm and Pc and arctic air masses
	13. Icecap climates of Greenland and Antarctica	E climates	Perpetually frozen continental icesheets	Cold and dry arctic and antarctic air masses

Present understanding of the effects of climatic change on the burial, dispersal, recycling and re-working of pre-existing placers relates mainly to the erosion and recycling of Tertiary placers during rapid changes in the Pleistocene. The best-known gold placers in North America were formed in Eocene times when global climatic conditions were humid and very hot; tropic and sub-tropic rain-forests cloaked most of the northern hemisphere and prolonged periods of deep weathering and erosion led to the large-scale development of alluvial goldfields. Down cutting of streams due to tectonic uplift was extensive and pay streaks were formed characteristically in valley centre positions as lags at the base of gravels and other bedrock features. Volcanic activity along the edge of the Pacific volcano belts then produced a cover of rhyolitic tuffs, andesitic breccias and basalt lavas to depths of up to 500 m. Renewed uplift and tilting in late Pliocene–early Pleistocene times caused streams to cut deeply into the volcanic rocks. In response to the formation of ice sheets, rapid atmospheric cooling and ensuing cold climates increased mass-wasting on slopes and decreased fluvial transport in valleys, thus producing burial of many of the alluvial goldfields.

Palaeoclimatology

The older the rocks, the sparser their distribution and the more problems are involved in predicting the patterns of climate of their times. For example, the Precambrian Eon (+4.8–0.6 Ma) includes almost 90% of geological time, yet about 80% of Precambrian rocks have been removed by erosion, the remainder have either been extensively altered by metamorphic processes or lie buried beneath younger sediment or volcanics. It was probably not until about the end of the Archaean Eon that the Earth's climate first experienced the cooling effects of large continental masses. However, large cratons continued to evolve through to the Proterozoic Era and this may have been the approximate stage of division of the Earth's climate into both warm and cold periods. The extremes of climate are represented by ice ages on the one hand and by warm equable climates on the other. The range of average annual temperatures between ice ages and sustained interglacial intervals appears to be about 10–12 °C.

Investigation of ancient world climates is based upon evidence of geochemical changes in the properties of the atmosphere, spatial and temporal variations of latitude, altitude and location of land and sea, influence of oceanic-atmospheric circulation on the global energy balance and evolution of plant and animal life. The breakup of Pangaea, which commenced in the Mesozoic era, was accompanied by the creation of major mountain chains throughout most of the world. Initially, because of the distance of most of its terrain from an ocean, arid conditions prevailed over most of the land. Changes then occurred gradually as rifting divided the land and seawater inundated low-lying areas of individual continental masses. Rivers and lakes formed in previously arid areas and sea levels rose globally during the late Jurassic. Subduction of the oceanic plate beneath the margin of North America commenced during the Mesozoic era. The resulting orogeny, which is still active, has led to the continued accumulation of intrusive and extrusive igneous rocks of the Sierra Nevada Mountains and to the formation of chains of folded mountains along the Pacific coast of the Americas, from Alaska in the north to Chile in the south.

Surface temperatures were essentially warm and equable during the Cretaceous and Stanley (1986) suggests a possible connection between global temperature, water circulation and the presence or absence of black muds in the sedimentary record. He notes that these muds form at levels where bottom waters are depleted of oxygen. The widespread accumulation of black muds provides evidence of the particular warmth of mid-Cretaceous times; not even the deep-sea waters were cold. Cooling of the atmosphere in the late Cretaceous resulted in extinction of the dinosaurs and marked a major transition in Earth history.

Ice ages

It is known from the fossil record that since the development of an oxygenated atmosphere, global climates have passed through great many cyclic and non-cyclic changes. Long-term patterns of climatic change are represented by major glacial epochs (ice ages or glaciations) with mean annual global temperatures of 5–7 °C. Development of an ice age is characterised by the growth of massive ice sheets (glaciations), which advance and retreat over great expanses of the Earth's surface. A succession of alternating glacial and deglacial intervals interspersed by interglacial stages over time intervals of up to several millions of years constitute an ice age. Global temperatures return to favourable metabolic conditions during interglacials with average temperatures of around 25 ± 10–12 °C (Fairbridge, 1987). The Earth is currently within an interglaciation following a deglaciation that set in about 15,000 years ago.

However, while periodicity of ice ages is thought by most geologists to be mainly controlled by rotation of the galaxy, many other possible causes also strongly affect climate and its cyclicity. To name a few, these causes and effects include tectonically induced topographic relief, continental drift, the periodic arrival of continental crust at polar positions and changing patterns of the circulation of warm and cold oceanic currents. Ice sheets cover 13 million km^2 of the Antarctic landmass, which is presently centred on the South Pole. By contrast, no land exists over the North Pole where only sea ice accumulates. The Greenland ice sheet is centred some distance from the pole at latitude 75° N. Furthermore, evidence from glacial samples of atmospheric CO_2 trapped in Greenland ice appears to offer a close linkage between CO_2 levels in the atmosphere and periodicity of ice ages (Hay, 1987). Possible connections are also being made with major tectonic upheavals, which result in downwind aridity. Thus, while a general correspondence exists between orbital variations and the waxing and waning of glaciers, the mechanisms of climatic change remain elusive.

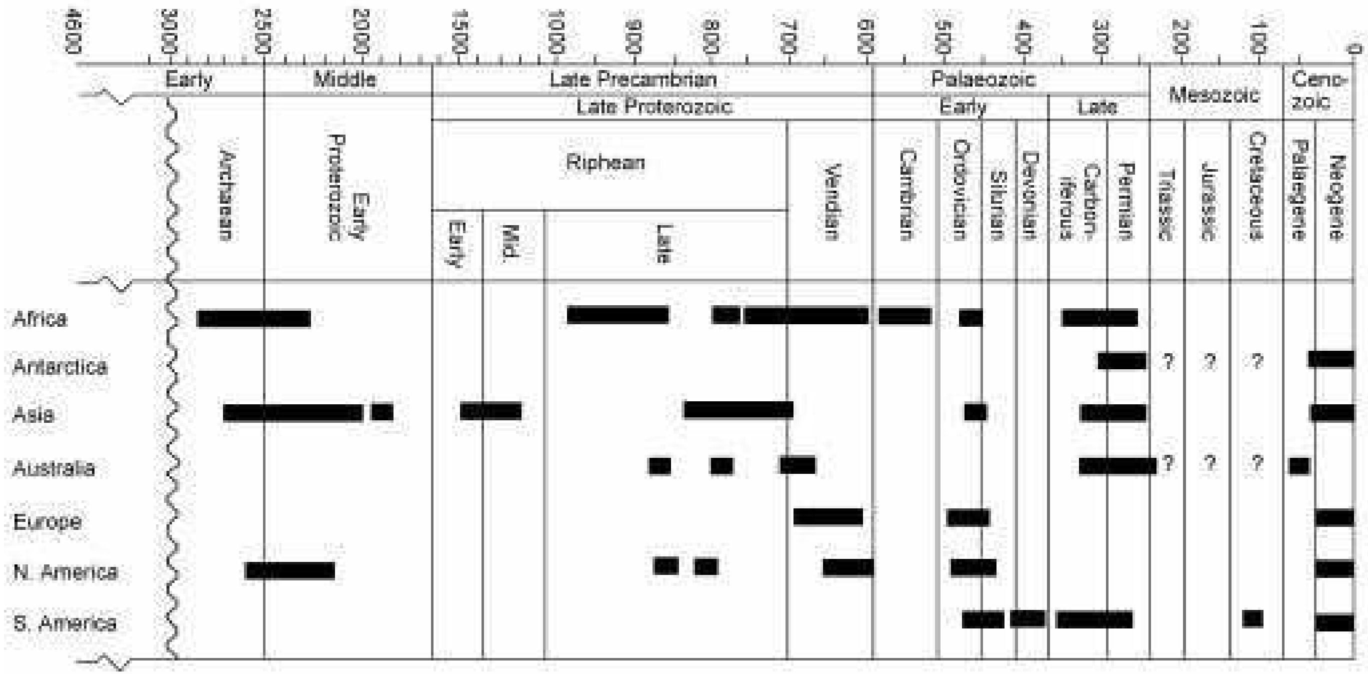
Short-term patterns of climatic change are associated with glacial and deglacial stages of waxing and waning of ice sheets and alpine type glaciers and, to a lesser extent, the warmer more equable climates of interglacials. For all such conditions local aberrations are due to a variety of possible causes, e.g. blotting out of sunlight by outpourings of volcanic ash, changes in albedo, changes in oceanic circulation and disturbances of the Earth's motion. Cretaceous mean global temperatures were probably similar in the tropics to those of today (around 32 ºC) but mean annual polar temperatures could not have been lower than the freezing point of fresh water, or more than 12 ºC higher than today (Hay, 1987). Glacial action could not have been significant except at high altitudes.

Successive episodes of glacial and fluvial reconstruction during Pleistocene ice ages and humid interglacial intervals profoundly affected both local and global weathering patterns. Events leading to Pleistocene ice ages were probably set in motion about 20 million years ago with the Alpine orogeny which created mountain ranges stretching from the Pyrenees and Alps in Europe through the Caucasus to the Himalayas in Asia, and the Rocky Mountains and Appalachians in North America. Glacial erosion resulted in the development of discontinuous valley margin pay streaks, which were either buried by renewed mass-wasting on slopes or dispersed and reconcentrated by shallow marine processes on beaches and platform areas. Holocene climatic fluctuations provided at least seven individual intervals of erosion representing separate episodes of placer reconstruction in both warm and cold climatic conditions.

Much less is known of the older ice ages. Extremes of climate and climatic change are represented by alternations of ice ages and interglacial intervals that are cyclical over a wide range of geological timescales. Ancient glaciations dating back to the Precambrian Huronian (2.0–2.5 billion years ago) indicate an extensive period of glaciation shortly after the transition from Archaean to Proterozoic time. Interglacial intervals appear to extend throughout the next 1–1.5 billion years although there may have been some glacial episodes during this period that have not yet been identified. Glaciations in the Late Proterozoic range in age from 1.0 to 0.6 billion years. The widespread distribution of till from these deposits suggests that Late Proterozoic global temperatures could have been quite low. Eyles and Eyles (1992) broadly demonstrate the intermittent recording of the Earth's glaciations in Fig. 3.13.

Depositional effects of climatic change

Possible effects of different processes acting upon alluvial gold deposition at ground surfaces during change from one set of environmental conditions to another are summarised as follows:



3.13 Earth's glacial record (after Eyles and Eyles, 1992).

- Change from arid to semi-arid conditions involves higher rates of precipitation and run-off, increased chemical weathering and more extensive fanning out of detritus from weathering.
- Change from semi-arid to temperate climatic conditions increases the extent of weathering and transport of detritus in source areas; it thus provides conditions more conducive to placer formation on a large scale than those that are related to the predominantly mechanical type breakdown of rocks in drier climates.
- Change from temperate to humid tropic conditions usually results in increased chemical weathering but lower rates of sediment production; exceptions may be found in higher ground where slopes are steep and gravitational forces can more easily overcome the frictional forces resisting movement.
- Change from humid tropic to periglacial conditions is followed by reduced chemical weathering and a dramatic increase in mechanical disintegration, particularly in rocks already weakened by previous chemical action; mechanical freeze-thaw processes are dominant in alpine regions where brief periods of sunshine in late spring and early summer bring melting of the snow and short-lived but violent fluvial activity.
- Change from periglacial to glacial conditions produces glaciation of valleys and recycling of any placer deposits not protected from glaciation by their location, e.g. in incised bedrock features. In some cases this involves a radial dispersion of pay streaks across the valley, in others the telescoping of auriferous gravels down valley into the boreal system and, for a few, glacial transportation of the gold-bearing materials into coastal settings as at Nome, Alaska.
- Change from glacial to humid tropic conditions is dominated by climatic change; usually this involves recycling Tertiary age placers through repeated cycles of Quaternary climatic change and glacio-fluvial transport of the gold-bearing material downstream into either a boreal environment or in low relief areas into a jungle setting.

3.3 Agents of weathering

In any area of gold mineralisation, the vertical entrenchment of valleys will establish a general lowering of land surfaces so that eventually, the gold source rocks will be unroofed and exposed to weathering. Primary deposits all have a different gold-bearing potential and in order to form alluvial gold concentrations, the rocks must then break down in such a way as to release the contained gold. Ultimately, this means rock disintegration down to the scale of individual grains. There must also be a means of removing the detritus. Fine materials are removed from the surface by sheet wash and deflation; soluble salts are carried away by percolating ground waters and rock waste shed from source rocks cropping out on hillsides eventually gravitates down slope to be carried away in flowing streams of water. Throughout all of these actions, gravity is the dominant force, with water as the main processing medium and air as an essential adjunct. Neither fluid acts entirely alone and water, as a solvent for the principal agents of chemical weathering (oxygen, carbon dioxide and other dissolved gases and impurities), is the means by which they are brought into contact with subsurface materials.

3.3.1 Gravity

Based upon Newton's law of gravitation every body in the universe attracts every other body with a force, the intensity of which varies directly as the product of their masses and inversely as the square of their distances apart. Thus, for two bodies of mass m_1 and m_2 separated by a distance d the gravitational force:

$$F = Gm_1m_2d^{-2} \qquad 3.1$$

where G is the gravitational constant.

Application of the universal law of gravitation in Newton's second law of motion relates force F, mass m and acceleration due to gravity g, as expressed in the formula:

$$F = mg \qquad 3.2$$

Substituting for F in eqn 3.2 the mass cancels out and

$$g = Gmd^{-2} \qquad 3.3$$

This equation states, in effect, that all objects in free fall in a vacuum continue to fall to the Earth with constant acceleration regardless of mass, place and time. In the context of Newtonian physics, the mass of a body is an intrinsic property that does not depend upon its chemical composition. Its weight, however, is affected by various forces such as the gravitational attraction of the Earth on the body and centrifugal forces due to the rotation of the Earth. The presently accepted value of G is 6.67×10^{-11} mkg units and the mass of the Earth 5.98×10^{27} g. The uncertainty of these values appears to be that of the order of one-half unit of the last place given.

Thus, while mass 'M' as the quantity of matter in a body does not change, the acceleration due to gravity 'g' varies with distance from the Earth's centre of mass hence bodies of equal mass vary in weight from place to place on the Earth's surface. The variations are small, but have important implications for geophysicists who are concerned with very slight differences in weights of bodies of equal mass at various locations and elevations. The value of g is greatest at sea level at the poles and least on high mountains in higher latitudes. Reasonable approximations can be made from the calculated values listed in

Table 3.6 Calculated values of coefficient g at different latitudes

Latitude	0°	10°	20°	30°	40°	50°	60°	70°	80°	90°
g(ms^{-2})	9.781	9.783	9.787	9.784	9.802	9.811	9.819	9.826	9.830	9.831

Table 3.6. For greater accuracy, using the base value $g = 9.781236$ m/s, the numerical value for g can be calculated at any desired location from the formula:

$$g = 9.781236(1 + 0.005243 \sin^2 L)(1 - 0.000000097e) \qquad 3.4$$

where L is the latitude north or south of the equator, and e is elevation in metres above sea level.

Most engineers, although recognising that geophysicists require extreme accuracy, are usually content to assume a standard value of 9.781 for g and hence constant body weights regardless of position. On sloping surfaces the downslope component of force F_d is given by:

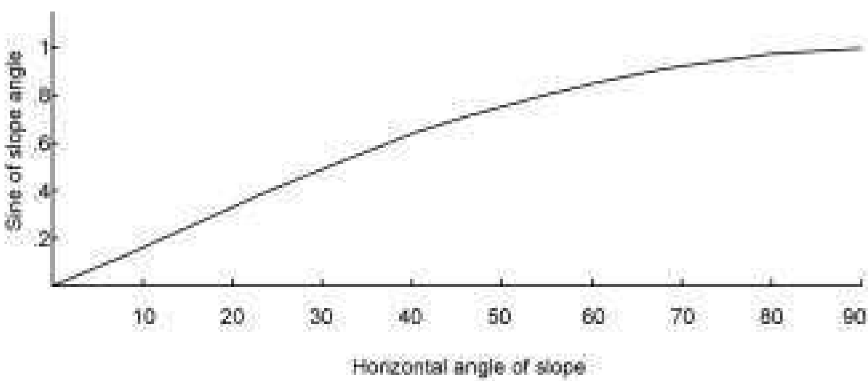
$$F_d = Mg \sin \alpha \qquad 3.5$$

where α is the slope angle.

Gravitational forces increase as the sine of the slope and can be calculated for any point on the sine curve, as illustrated in Fig. 3.14.

3.3.2 Water – the processing medium

As a processing medium, water plays an important role in most weathering and mass-wasting processes. Due to its high intermolecular attraction (melting temperature 0 ºC, boiling point 100 ºC) water exists in all three physical states at earth-surface temperatures: solid, liquid and gas. In its solid state, ice consists of water molecules joined together by a regular arrangement of hydrogen bonds, which apparently leaves empty spaces between the water molecules. The



3.14 The sine curve.

molecules of liquid water are also held together by hydrogen bonding, but some of these bonds are broken when ice melts and the remainder are too few to keep the water molecules in a regular arrangement. When heat is added to water at 0 ºC, the water contracts until it reaches 4 ºC after which it slowly expands until it reaches boiling point and turns to steam. In the gaseous state above 100 ºC, water vapour consists of water molecules moving independently of one another. Water is most destructive in a solid state and most reactive chemically when heated.

Water as ice

Ice occurs naturally in glaciers as masses of frozen water that move under the influence of gravity. They are composed of mixtures of snow, firn and ice and are fed in areas of snow accumulation above the mean snow line. Freshly fallen snow consists of snowflakes of average density 0.08 g/cc. Prolonged periods of high snowfall and increased snow depths result in the transformation of snow to firn (density 0.1–0.9) and then to glacial ice (density 0.92). The continued increase in density occurs by packing, melting and freezing during which air entrapped between ice particles is partly squeezed out and the final compression of ice takes place in deep sections of the glacier.

Water as a fluid

Water falls onto a land surface in the form of rain, hail or snow. Run-off at the ground surface takes the form of sheet flow, streams and rivers. In arid regions, where rainfall is only occasional and evaporation rates are high, infiltration is negligible and surface run-off may occur only once in several years. Infiltration and run-off are both seasonally heavy in monsoon regions, but flow rates fall away sharply for the rest of the year. Rivers that drain alpine regions receive little run-off from melt-waters during the winter months, but typically swell to flood proportions from the melting of snow and ice in spring and early summer. Only in climates such as those of the British Isles and Tasmania is rain abundant throughout most of the year and even these areas are subject to occasional flooding and periods of drought.

As already indicated, water as a weathering agent is a solvent for many reactive gases, particularly those which dissociate and form ions. Rainwater thus carries various constituents of the air into parts of the ground not in direct contact with the atmosphere. Weathering processes are becoming increasingly destructive with the growth of fossil fuel-dependent industrial applications. Close to cities, where industrial gases discharge directly into the atmosphere, rainwater acidities have been recorded as high as pH 4.4 (from Douglas, 1977). Conversely, in highly vegetated terrain, impurities are filtered out of the air and rainwater reaching the ground may be slightly alkaline, hence much less destructive.

Table 3.7 Ranges of porosity (n) and void ratios (e) of earth materials (modified from Dingman, 1984)

Material	n	e
Granular materials		
Gravel	0.25–0.40	0.33–0.67
Sand	0.25–0.50	0.33–1
Silt	0.35–0.50	0.54–1
Clay	0.40–0.70	0.67–2.3
Sandstone	0.05–0.30	0.05–0.43
Limestone and dolomite	0–0.20	0–0.25
Shale	0–0.10	0–0.11
Rocks with joints, fractures, or solution channels		
Fractured basalt	0.05–0.50	0.05–1
Fractured crystalline rock	0–0.10	0–0.11
Dense crystalline rock	0–0.05	0–0.05
Karst	0.05–0.50	0.05–1

$$\theta \equiv \frac{V_w}{V_T}$$

and the degree of saturation, s

$$s \equiv \frac{V_w}{V_y}$$

where V_w is the volume of water contained within a volume of material V_T. Clearly, for saturated conditions $s = 1$ and $\theta = n$; for unsaturated conditions $s < 1$ and $\theta < n$.

The motion of water through rocks in a weathering zone depends upon the porosity (n) and void ratio (e) of the soil and rock material. Typical ranges of porosity and void ratios for selected earth materials are given in Table 3.7. The variation is probably much greater and with a wider diversity of flow, in a weathered ore zone in which flow takes place through the locally irregular and interconnected openings of variously fractured rocks. Under such conditions, some of the water is absorbed hygroscopically by minerals during the weathering process known as hydration; some is returned to the surface in capillary pores and is either evaporated, or transpired back into the atmosphere. The downward percolation of water through the broken rock creates ground water aquifers that both provide the storage of ground water in significant quantities and permit the water to move laterally within the strata. Ultimately, a balance between recharge and discharge is struck at a level that rises and falls with changing rates of precipitation and infiltration. This level marks the boundary between an upper, unsaturated zone in which the pores contain both air and water; and an underlying, saturated zone that rests upon fresh bedrock. Flows in these two zones are called 'soil water' and 'ground water' flow respectively.

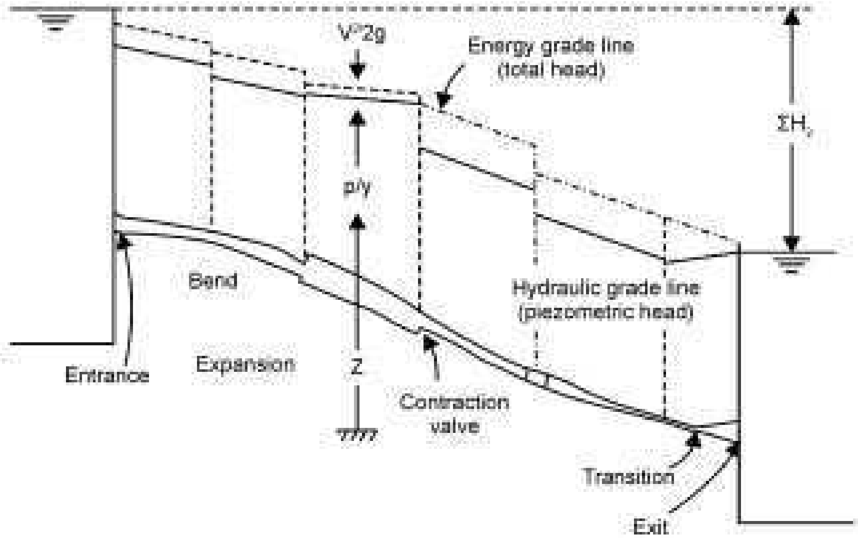
Soil water flow

Water rises towards the surface in soil conditions that provide intermolecular bonding of the water at a sufficient strength to promote capillary attraction. Flow takes place under tension, gauge pressures are negative and only the capillary fringe is saturated. Soluble salts and metals in solution are transported and precipitated within the porous material when the water is vaporised by plant transpiration or direct evaporation. Granular disintegration of rock surfaces in arid areas may occur by the swelling of salt crystals when saline solutions soak into the rock pores. Crystals of salt build up gradually and exert pressures that eventually break the rock apart. Similar processes can operate along shorelines under rapidly drying conditions.

Ground water flow

Under ground water flow conditions, motion is induced either by gravitational or pressure gradient forces. The gravity force is a function of the slope of the potential energy gradient and the piezometric head. The slope of the grade line is given by the ratio of the difference in elevation between the level of the water table and the point of discharge (piezometric head), and the horizontal distance between the two points. Pressure-gradient forces act where the thickness of the flow zone decreases in the direction of the flow. One of these forces may resist motion, depending upon the configuration of the flow zone (Dingman, 1984). Parameters of pipe line flow, viscous and other energy losses can usually be calculated closely for constructing an energy grade line (Chapter 7). For ground water flow, however, energy losses occur abruptly and are unpredictable; the various relationships are usually determined empirically as shown in Fig. 3.15.

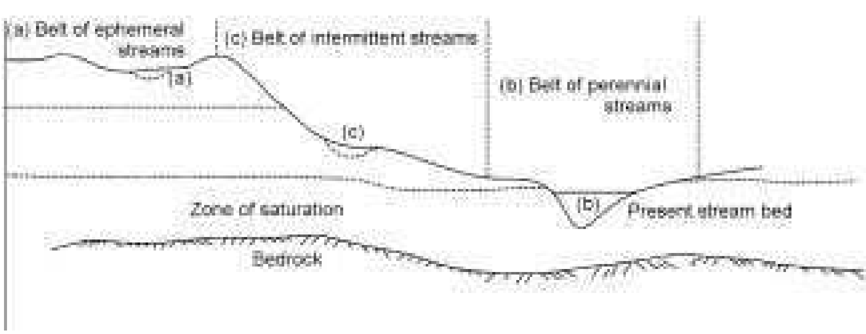
One important aspect of ground water storage is the relationship between ground water seepage into springs and stream flow at different levels of downcutting. The stream bed at level 'a' in Fig. 3.16 is located in rocks above the maximum water table; the stream is ephemeral in that it does not tap any of the ground water resources and flows only during periods of intense precipitation and run-off. At level 'b' the stream lies just above or below the maximum water table; flow is perennial for much of the time but may fall to a trickle or cease altogether during extended dry periods. At level 'c' the stream lies just above or below the minimum water table and variations in flow from surface run-off are smoothed out by the addition of water from the underground storage; this stream is perennial and flows strongly at all times.



3.15 Schematic representation – energy and hydraulic grade lines.

3.3.3 Air as an adjunct to water

The early Precambrian atmosphere was once thought to consist of a highly reducing mixture of methane, ammonia and molecular hydrogen. Recent studies suggest instead, that it consisted of molecular nitrogen, carbon dioxide and water vapour, with trace levels of molecular hydrogen and carbon monoxide originally trapped in the Earth during its accretionary stage (Levine, 1987). Iron had already migrated to the core and, during the second stage of development of the atmosphere the oxidisation-state of the volcanic gases released from a geologically differentiated Earth would probably have risen, more or less continuously, towards its present level. In this scenario, methane would have been converted to carbon dioxide and ammonia to nitrogen which would have become



3.16 Relationship of water table to stream flow.

the dominant component of the atmosphere at the end of this stage, with carbon dioxide and argon as its most common minor constituents (Holland, 1984).

Oxygen is principally generated by plant life, which has the ability, by photosynthesis to utilise sunlight to change water and atmospheric carbon dioxide into organic matter with the release of oxygen. It is generally believed that a moderate level of plant life was reached early in the Proterozoic, but that it only became sufficiently great about 1.8 billion years ago to become a major influence on the overall production of atmospheric oxygen. Since that time, oxygen pressure has risen to its present level, carbon dioxide would have decreased with its breakdown into oxygen and incorporation into carbonates, and molecular nitrogen would have increased with continued degassing.

The main constituents of the present atmosphere are nitrogen (78.084%), oxygen (20.946%) and argon (0.306%). Minor components are Ne, He, Kr, CH_4, H_2 and NO. The most reactive atmospheric gases in the weathering environment are carbon dioxide and water vapour. However, gases such as NH_4, HNO_3, H_2SO_4 and Cl, although present in trace quantities only, are so reactive over long periods of time that they, too, are most destructive. When removed from the atmosphere by water in the form of rain, hail and snow, all of these gases will interact strongly with other weathering agents involved in the chemical and biochemical breakdown of surface and near-surface rocks. The same reactive gases, together with dust and vapours from active volcanoes, salt from the oceans, waste industrial gases and other contaminants play a major role in the interception and distribution of solar energy when it reaches the Earth.

3.4 Weathering processes

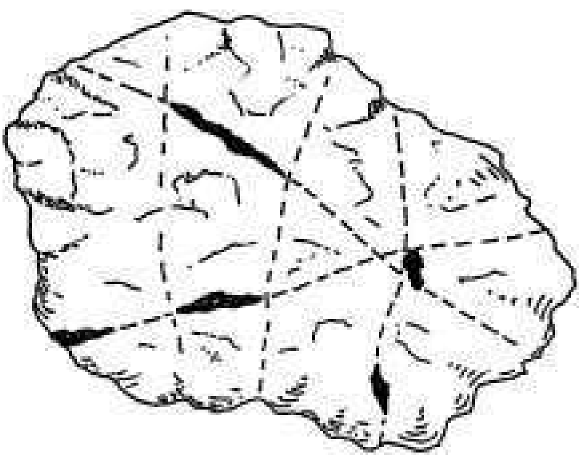
Weathering is the combination of processes whereby rock on the surface or near surface of the Earth is broken down physically and/or decayed chemically to form a regolith of waste overlying fresh bedrock. Regolith thus comprises all weathered rock materials including fragmented and chemically decayed basement rock, saprolites, soils, and sediment of all types both residual and transported, that accumulate in the soft surface layer. Properties of bulk density, porosity and permeability influence the movement of ground waters and *in-situ* weathering processes. The free gold content is determined by the extent that gold grains have been released and/or modified by weathering. The basic processes comprise a complex variety of physical and chemical reactions that differ from one another under different environmental conditions. Weathering may be divided broadly into two groups, mechanical and chemical. Biotic forces play an important role in both processes.

3.4.1 Mechanical weathering

Intrusive bodies fracture on a large scale during cooling when stresses set up by the force of the rising magma from below are transmitted to the immediate country rock. This causes the roof zone to yield and fractures rock formations over a large area. New textures and structures are developed within the rock. Being relatively plastic under conditions of high temperature and pressure, deeply buried sedimentary rock formations may respond to stress by extensive bending and folding over a wide area. Shale, which tends to deform rather than fracture under stress, may be metamorphosed to schist. Planes of parting in the schist provide two most important factors affecting the flow of hydrothermal fluids through the rocks: competence and structure.

Following these disturbances each intrusive body exists within its own uniform stress field and remains in equilibrium with its confining rocks until again disturbed, either by renewed tectonic movement, which may create further stresses, or by removal of the overlying rocks which gradually dissipates the internal stresses. As contraction gives way to expansion the rock develops patterns of expansion joints along which the rock may split. Massive rocks, such as granite, are more likely to be affected by this form of pressure release (dilatation) than rocks, which are initially close jointed.

Additionally, since each type of rock-forming mineral has a different modulus of elasticity, pressure release also leads to the development of myriad micro-fractures and loosening of individual grains. Differences in the physical parameters of the various minerals cause fracturing along mineral-mineral interfaces and rock disintegration may then be initiated on a grain to grain scale. Thus, when the rock crumbles, breakage along the plane between a flake of gold and the enclosing quartz may lead to release of the gold (Fig. 3.17). Some gold grains are plate-like, some are jagged and irregular in shape, others are rounded, but most have recognisable characteristics that will affect their subsequent hydraulic behaviour.



3.17 Illustrating how differences in modulus of elasticity and other physical characteristics can provide potential fracture directions loosening included gold grains.

Two most important agents of *in-situ* mechanical weathering are the crystal growth of ice and salts (freeze-thaw), and thermal expansion-contraction.

Freeze-thaw

Freeze-thaw involves repetitive growth and melting of ice crystals in cracks and pores in the rock. Frost action occurs at low elevations in high latitudes and in cool inland areas of lower latitudes where temperatures fluctuate below and above freezing point. In its frozen state, water expands by 9.2% in volume at atmospheric pressure. At various stages of confinement, freezing exerts pressures up to a theoretical 2,000 kg/sq.inch in joints, cracks, etc., well in excess of forces holding rocks together. The action of freeze-thaw is enhanced in near surface rocks that are opened up by dilatation, so allowing water, bacteria and plant roots to penetrate the openings and wedge rock fragments further apart.

Stresses created by the crystallisation of salts such as sodium chloride, calcite and gypsum may enter pores and other openings in rocks in dissolved form. On drying and crystallisation they expand and set up similar disruptive effects as those caused by ice during its stage of crystal growth. Crystallisation has been observed to occur against pressure as great as 47 bars, at least twice the tensile strength of many rocks (Bryant, 1976). The growth of salt crystals may create honeycombing of rocks that is sometimes thought to be due wind abrasion or chemical weathering.

Thermal expansion and contraction

The effects of diurnal and seasonal heating and cooling of surface rocks on their physical disintegration is the subject of many debates. Some believe that normal diurnal temperature changes cause rocks to exceed their thermal expansion limits thus making the outer skins of boulders peel away from their inner cores. Others suggest that the differential heat absorption of dark- and light-coloured rocks in a multi-coloured rock (e.g. granite) should provide sufficient stress to shatter such rocks. Excessive day temperatures in some desert areas appear to have this effect on some rocks. A possible contributing factor is the varied content of atmospheric water, some of which is always present. In most cases, however, a better case can be made for shattering by a combination of weathering forces, the dominant factor being the cumulative effect of innumerable repetitions of expansion and contraction in rocks already weakened by other weathering processes.

3.4.2 Chemical weathering

Many rock-forming minerals show incipient alteration characteristics prior to atmospheric weathering; feldspars are often slightly kaolinised, mafic silicates are slightly chloritised and so on. This slight alteration takes place in igneous rocks during the deuteric stage of crystallisation with chemical changes taking place along structure planes of the mineral, e.g. cleavage traces. The resulting clay-type minerals will cause slight expansion, thus aiding the subsequent weathering process.

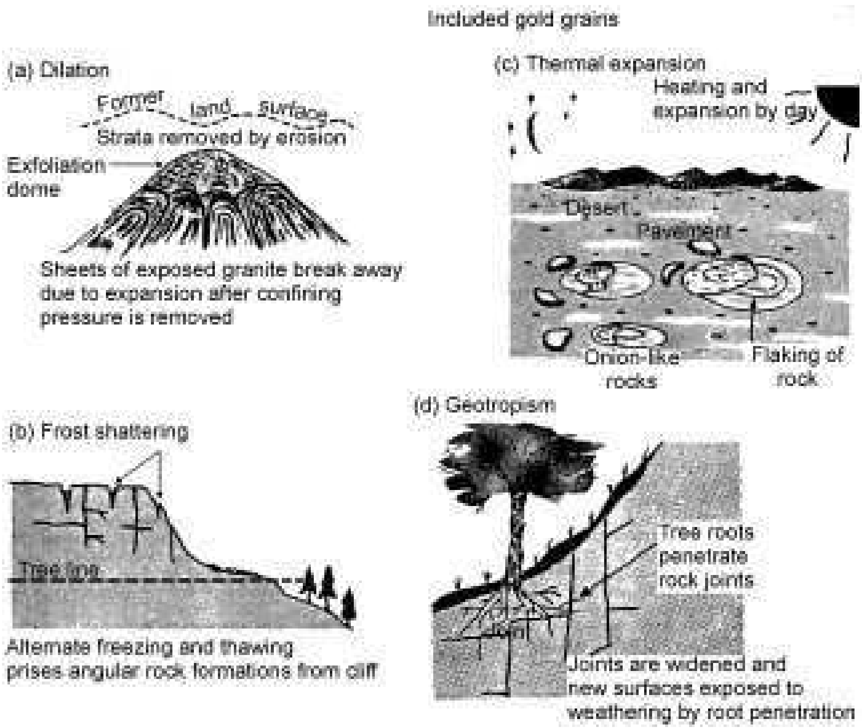
Rainwater contains dissolved gases capable of setting up a series of chemical reactions. Nitrogen and carbon dioxide are the more reactive of these gases. In addition there are traces of NH_4, HNO_3, H_2SO4, NaCl and Cl which, though present in minor amounts, are so reactive over a period of time to be considered important in rock decomposition. Humid conditions accelerate decomposition and areas of low relief allow the reactants a longer time for the effective reactions to proceed to completion. Rapid run-off of surface water is effective in removing the products of chemical change but militate against reaction time.

Chemical weathering processes involve the complete removal of some substances (e.g. limestone) and the creation of secondary, more stable minerals to replace constituents that are less stable in the weathering environment. The principal forces acting together to break rocks down to their decomposition products are oxidation, carbonation and hydrolysis. The presence of dissolved oxygen in water promotes oxidation of metallic elements (Mg, Fe, Al, Ca, etc.) that are abundant in silicate minerals. Carbon dioxide in solution forms weak carbonic acid, which assists in the decomposition of granite and gneiss, both of which contain an abundance of feldspar minerals. The water itself combines with certain rock minerals to form insoluble precipitates (clay minerals).

The various silicate minerals decompose at different rates, the order of increasing susceptibility to weathering being the same as the sequence of crystallisation from a melt, i.e., olivine–pyroxene–amphibole–plagioclase–biotite–K-feldspar–vermiculite–smectite–muscovite–kaolinite–quartz. Notwithstanding the susceptibility of minerals such as olivine, pyroxenes and amphiboles to chemical weathering they do occur in placers, often quite plentifully. In part, this may be accounted for by prior mechanical disintegration followed by rapid transport to the site of deposition with insufficient time for complete oxidation to take place.

The effects of chemical and biological weathering are most pronounced in the humid conditions of tropical rain-forest areas. Vegetation cover is at a maximum and a proliferation of plant roots enlarges openings in the surface layers, thereby enhancing water penetration and chemical reactiveness. Destruction of organic matter by decomposition results in the production of carbon dioxide and organic acids for chemical weathering and the rate of decay is further increased by the action of termites, bacteria and fungi. Surface attack by biological agents is less significant in temperate climates, where tree roots may penetrate to greater depths in their search for water, but cause much less lateral disturbance.

The role of micro-flora varies with changing ecological conditions (Douglas, 1977). The formation of living matter from natural organic compounds is largely



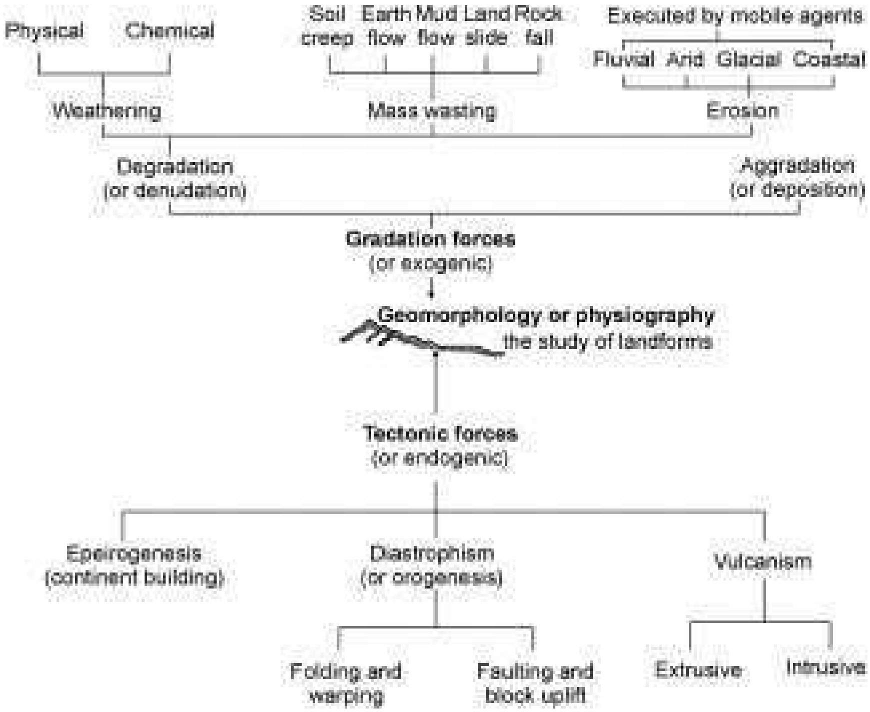
3.18 Stresses imposed by mechanical weathering forces (Barlow and Newton, 1974).

due to metabolism whereby food is built up in green plants as nutriment for animals. The nature of the vegetation and its decomposition products influences the types of animal attracted to it. Soil fauna (earthworms, ants and other burrowing animals) contribute to disintegration of the surface layers by forcing individual grains apart and breaking up and mixing the rock materials. Plants such as fungi and lichens break off tiny fragments of the rocks, absorbing them into their own tissue.

The disintegration of rocks by physical stresses imposed upon them at or near the Earth's surface involves mainly stresses set up by expansion and contraction as illustrated in Fig. 3.18.

3.5 Landscape denudation

Denudation is the sum total of the combined efforts of weathering, erosion and mass wasting involved in the lowering of the Earth's surface and transport of sediments to the sea. The denudation cycle commences when continental crust is elevated above sea level by tectonic processes associated with ocean floor spreading. Uplift is rapid during the most active phase of orogeny and occurs in



3.19 Balance between tectonic forces, which build up land, and erosive forces that wear it down (after Barlow and Newton, 1974).

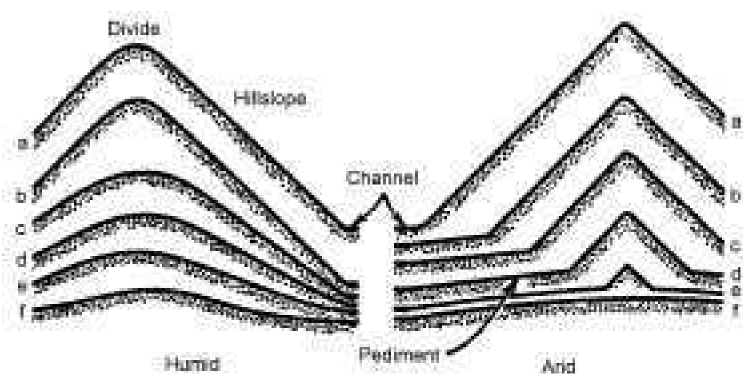
stages. The principal effect is one of aggregation, building up the surface topography against the base levels at which the erosional forces act (Cas and Wright, 1995). Only during extended periods of relative tectonic calm, e.g. in later stages of orogeny will erosion rates exceed those of upward movement. The balance over time between tectonic forces that create mountains and the simultaneous action of the contributing processes of denudation that act to wear them down is illustrated in Fig. 3.19.

The beginning of mobilisation of sediments by the various agents of erosion (glaciers, rainwater and wind) is marked by erosion of weathered surfaces. Spoil gouged out of the valley floor and walls by glacial erosion (plucking and abrasion) may be transported for considerable distances before being dumped at the foot of the glacier. Rainwater disturbs surface particles by impact when it strikes the ground and continued precipitation leads to sheet flow and rivulets, which wash the lighter particles away. Water seeping downwards along seepage planes fills the voids between particles and provides lubrication for the mass to move as a whole. Wind sweeping over the ground entrains small waste particles which are carried away from the surface by deflation processes involving traction, saltation and for dust sized particles, suspension. Large animals, grazing on hillsides are a major cause of slope instability.

Aeolean desert style denudation is characterised by extreme fluctuations in temperature and wind. Rainfall is sporadic and low with periods of flash flooding during which ephemeral streams are reactivated and both small-scale and major mass movements may occur. Individual fans are small and may pass downward into desert floor environments with internal drainage, including playa lake environments (Boggs, 1987). Fans in arid to semi-arid regions may join together laterally to form bajadas along mountain fronts. Aeolean desert style deposition is characterised by extreme fluctuations in temperature and wind. Rainfall is sporadic and low with periods of flash flooding during which ephemeral streams are reactivated and both small-scale and major mass movements may occur. Slope angle and length, the manner in which individual mass flows split or converge, and the differential fluidity that follows an uneven distribution of seepage and run-off direct the movement of materials along the lines of least resistance and govern sorting relationships on the fans.

Hillslope profiles show a great number of shapes of which two contrasting models, one for humid climates the other for arid climates are described schematically in Fig. 3.20. The weathering profile for the humid climate commences typically on a rounded summit on which the effects of rain-splash and soil creep are dominant. Barren material is removed both as sediment and as dissolved solids. With the passage of time the rounded summit will increase in area while the lower part of the profile flattens. Colluvium built up at the base of the slope will further reduce the relief; ultimately the topography will be lowered to a level of peneplanation.

The initial profile in the arid environment is assumed to be steep and straight. Throughout all stages of denudation the steep angle of the slope profile element remains constant but retreats toward the divide in successive parallel profiles. The pediment widens as the base of the steep element recedes until a pediplain



3.20 Hillslope evolution with decreasing relief in humid and arid climates (from Strahler and Strahler, 1992).

consisting of pediment surfaces, alluvial fans and playas is formed. A most important climatic change relates to the overprinting of parts of a deeply weathered regolith formed under humid warm to tropical conditions by later features related to weathering under semi-arid to arid climatic conditions.

3.5.1 Stability of slopes

Slope characteristics are natural responses to the interaction of ever-changing patterns of denudation systems inherited from past erosional and depositional cycles and are inherently complex and unstable. Variables affecting those characteristics are bedrock geology, climate, soil, vegetation and angle of slope. The stability of the regolith is a function of climate and the ability of the weathered rock material to withstand the stresses imposed upon it.

The conventional view of straight evenly graded hillsides over which the intensity of energy is equal across each horizontal plane is too simplistic. Hillside slopes are invariably marked by convexities, concavities and other irregularities, which direct the movement of materials along the lines of least resistance. Colluvial trains break up into separate flow paths and sorting relationships are governed by slope lengths, the manner in which individual mass flows split or converge, and the differential fluidity that follows an uneven distribution of seepage and run-off. The rate of movement is variable and sometimes abrupt depending upon the angle of slope, the intensity of precipitation and the nature of the slope materials. Spontaneous downward movement occurs when the internal strength of the regolith declines to a critical point at which the force of gravity cannot be resisted. As modified by presently acting processes, two opposing forces determine existing levels of slope stability, shear strength and shear stress.

'Shear strength', as the resistance of a material to shear, is a function of internal friction and cohesion (the electrostatic attraction among very small particles). 'Shear stress', as the applied force tending to cause failure, is a function of particle friction (which depends upon the friction angle) and gravitation forces. Soils of different particle size and mineral composition have different strength capabilities and respond differently to the application of stress. Just prior to failure shear stress is equal to shear strength, which then represents the maximum strength of the material. Failure is imminent only when shear stress exceeds shear strength.

For mixed soils, subject to changing pore water pressure, shear strength is generally represented by Coulomb's 'failure law', which may be expressed as:

$$S = \gamma f = C + \sigma \tan \psi \mu s \qquad 3.6$$

where S = shear strength, γf = shear stress at failure, C = cohesion, σ = pressure normal to the shear plane, and $\psi \mu s$ = static angle of plane sliding friction.

3.5.2 Movement on slopes

The course of erosion can be expressed in terms of a ratio of mechanical to chemical weathering. In terrains of high relief mechanical weathering is dominant because of the steepness of slopes and lack of vegetation, and the spontaneous downward movement of the regolith in response to the influence of gravity with or without the dynamic action of a moving fluid. Large masses of unsorted debris may be washed rapidly (up to 10–15 km/h) downslope, settling finally into accumulations of partly worked debris fans wherever the supply of weathered material is greater than the capacity of local streams to carry it away. Ground surfaces are significantly above the base level towards which the flow processes are working (e.g., sea level) and as the detritus is in constant downslope movement more fresh rock is continuously exposed to chemical and physical change. Vegetation is sparse and weathering relates only to the present environment. The rate of movement is variable and sometimes abrupt depending upon the angle of slope, the nature of the slope materials, intensity and type of precipitation and climate. No significant accumulation of heavy minerals including gold takes place in such settings.

However, as time passes and land surfaces are reduced in altitude, chemical weathering processes become more important and both the depth of weathering in the eluvial-colluvial ore zone and thickness of the regolith increase. The stability of the slope material increases in response to the decreasing amount and size of the material in transport and because more time is available for it to attain equilibrium with its environment. The rate of movement is directly related to the slope angle, the relationship between soil-water content and the applied pressure. The infiltration rate is a function of the properties of the soil and the rate of supply of water to the soil surface. Small amounts of water play a subordinate role by providing lubrication between grains and under most types of climate alternate wetting and drying of the soil will allow the detritus to move very slowly by soil creep in close grain to grain contact on very low slope gradients.

Gravity, as the fundamental force causing movement relates directly to the slope on which it acts. Natural surface slopes are about 35º in loose scree material but may exceed 45º if silty. Landslides tend to occur when cliff faces steepen to as much as 70º. Gravity acts at its maximum value on vertical faces and declines as the sine of the angle of slope to zero on the horizontal plane. However, although gravity is the ultimate cause of downslope movement and the rate of movement is directly related to the slope angle, the relationship between soil-water content and the applied pressure is also a major cause of mass movement. The development of positive pore pressure, with increasing levels of saturation, decreases cohesive resistance, which becomes negligible when the soil approaches complete saturation.

Small amounts of water play a subordinate role by providing lubrication between grains and under most types of climate this will allow the detritus to move very slowly by soil creep in close grain to grain contact on very low slope gradients. However, with increasing levels of saturation the development of positive pore pressure decreases cohesive resistance, which ultimately becomes negligible when the soil approaches complete saturation. Contrasts in permeability affect saturation and on a completely impervious (e.g., frozen) surface, run-off leaves the underlying material stable except for any movement that may be induced by earth tremors or local disturbances. In periglacial regions the surface thaws to depths of several metres during early spring and summer. Melting of the ice provides sufficient water to produce a highly saturated and mobile surface layer which slides downward along the plane between the surface layer and the underlying permafrost. Intermittent flow processes such as solifluction, slumping and sliding and the more rapid movement of mud/debris flow are not conducive to sorting in themselves; their importance lies in the supply of gold-bearing feed materials for fluvial processing.

Soil creep

Suitable conditions for soil creep are provided on gradients that allow the unconsolidated detritus to move very slowly downslope (about 1 m/y) in close grain to grain contact. The process is thought by some (e.g. Ruhe, 1975) to be a viscous mechanism in which the stresses are too small to cause shear but where nevertheless, internal and permanent deformation is associated with the movement. Others, including Douglas (1977), believe that soil creep is the result of the cumulative effect of intergranular forces, which displace individual particles relative to adjourning particles. There is no simple explanation. While the presence of soil water is essential for lubrication, movement by creep is also affected by, although not dependent upon many other processes. Important amongst these are rain splash, thermal expansion and contraction (e.g., frost heave and hydrostatic pressure in joints and cracks) and geotropism. Rain splash disturbs the surface particles by impact and by forming rivulets and sheet flow washes the lighter particles away. Frost heave lifts particles normal to the inclination of the ground and deposits them vertically downslope when the frost melts. Plant growth encourages infiltration and helps to initiate movement by modifying the effects of forces holding particles together. Surface creep is then a function of the cumulative effect of displacement by plant roots and swaying of trees and shrubs and other disturbances caused by animal grazing, tunnelling by termites, worms and other burrowing animal life, deflation by the wind itself and human activities.

Solifluction

Solifluction is most effective in periglacial climates, though its action is also possible in other climatic conditions, e.g. when more water is present. The nature of the movement and susceptibility of slope materials to the action of freeze-thaw processes appears to depend upon the soil grain size distribution, permeability of the soil, rate of freezing and depth of frost penetration and availability of water. With increasing levels of saturation the development of positive pore pressure decreases cohesive resistance, which ultimately becomes negligible when the soil approaches complete saturation.

Movement by solifluction is usually restricted to a few metres each year when seasonal thawing of the mantle in periglacial areas causes masses of material to break away, typically forming an eroded rock basin surrounding an armchair-like depression (cirque). Provided that movement first takes place on steeper slopes solifluction may occur on slopes as flat as two degrees or even less. The process is typical of periglacial weathering in arctic and alpine regions but may also occur in humid mountainous areas of the tropics. Andersson (1982) coined the term solifluction in his studies of the ‘mud glaciers’ of Bear Island in the North Atlantic and ‘stone rivers’ in the Falkland Islands. He described solifluction as slow flowing from higher to lower ground of soil or earth saturated with water, thereby distinguishing the process from the surface creep of unsaturated soils.

Mud/debris flow

Rapid mass-movement by mud/debris flow may occur on almost any slopes wherever the rainfall is seasonal or heavy. Periods of torrential rain cause weathered slope material to become saturated and behave more as a fluid than as a solid. This may take place in:

- humid tropic areas where slope materials typically comprise unconsolidated debris of glacial or volcanic origin
- frigid areas where the wedging action of freezing water disintegrates the near-surface rocks
- humid temperate to tropic regions where source rocks, prior to uplift, are subjected to long periods of deep chemical weathering.

3.6 Low-temperature aqueous geochemistry

In order to understand the various chemical transformations involved in low-temperature aqueous solutions, the particular properties of the matrix and of the chemical composition of the gold must be known. Two important solution properties are acidity (pH) and the oxidation potential Eh. Acidity is usually measured as pH (the negative logarithm of the H^+ concentration) and is controlled by the nature of the underlying geology, acidic or basic (see Section 1.1.2). Carbonates (CO_3^{2-}), protons (H^+) and bicarbonate (HCO_3^-) all exist in solution in controlled proportions. This control is defined in terms of equilibrium potential. A description is given by Hostettler (1984) of Eh theory, and of the problems involved in measurement and interpretation.

3.6.1 Solution processes

Factors such as pH, Eh and salinity have major effects on the speciation and solubility of many elements, and how they are controlled in natural systems (Gray, 1997a). Salinity can arise from various processes:

- rock weathering
- evaporation
- dissolution of previously deposited halite (NaCl)
- sea water incursion
- aerosol deposition of sea water.

One or more of these processes could be important at a particular site. For example, the ground waters in the Southern Yilgarn district of Western Australia are hypersaline, with salinities greater than seawater. This appears most likely to be due to long-term deposition of salt from aerosols, coupled with low rainfall, high temperature and flat topography, with evaporation as a major source of water removal in a virtually closed system.

Various salts precipitate as salinity increases with evaporation. The first major ion to be precipitated is calcium, either as calcite ($CaCO_3$) under neutral to alkaline conditions (i.e. where CO_3^{2-} is present) or as gypsum where there is an excess of Ca to CO_3^{2-} (Gray, 1997a). Halite precipitates are observed in saline playas under highly saline conditions (about 30% solids, or 300 mg/L). Saline playas are common across most of the Yilgarn Basin as the result of these processes (Mann, 1982). The pH of the ground waters is largely controlled by the underlying geology with acid lithologies giving rise to neutral to acid ground waters and basic lithologies giving rise to more alkaline waters. The upper limit to the pH of ground waters is about pH 10, due to carbonate deposition.

This lithological control however, is complicated by oxidation during weathering. Gray (1997a) describes the effects of the oxidation of pyrite and other sulphides. Pyrite oxidation occurs in two stages, both of which generate hydrogen ions (acidity). The generation of sulphate, which generally occurs at depth, is the first reaction:

$$\underset{\text{(Pyrite)}}{2FeS_2} + 7O_2 + 2H_2O \Leftrightarrow 2Fe^{2+} + 4SO_4^{2-} + 4H^+ \qquad 3.7$$

Diffusion of oxidising species such as O_2 limits this step and acid conditions may be neutralised by other reactions, such as carbonate or feldspar weathering.

Closer to the surface, the second process ‘Ferrolysis’, which also generates acidity results in the oxidation and hydrolysis of Fe (Brinkman, 1997).

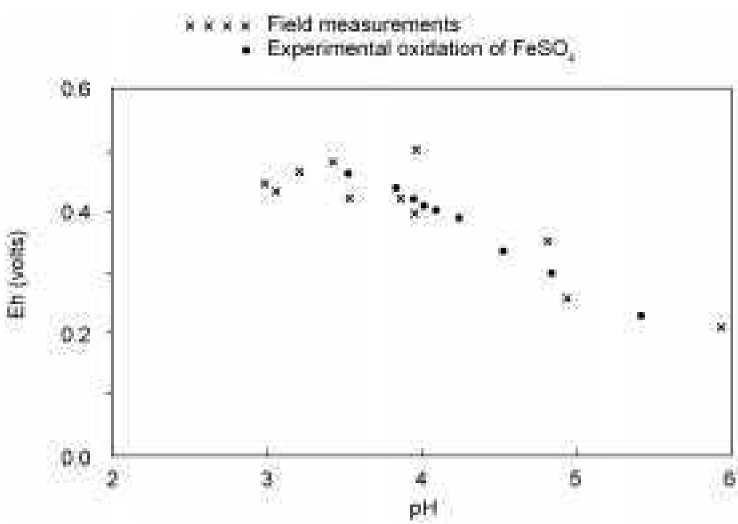
$$2Fe^{2+} + 0.5O_2 + 5H_2O \Leftrightarrow 2Fe(OH)_{3(s)} + 4H^+ \qquad 3.8$$

Decoupling of the two components of pyrite oxidation (Reactions 1 and 2) is commonly observed (Blowes and Jambor, 1980). As the upper weathered profile does not usually contain alkaline minerals such as carbonates, ferrolysis can result in shallow ground waters becoming highly acidic. Buffering of the waters by dissolution of clay minerals such as kaolinite, which precipitate as acid sulphate minerals such as alunite [$KAl_3(SO_4)_2(OH)_6$], then give an acidity limit of about pH 3.

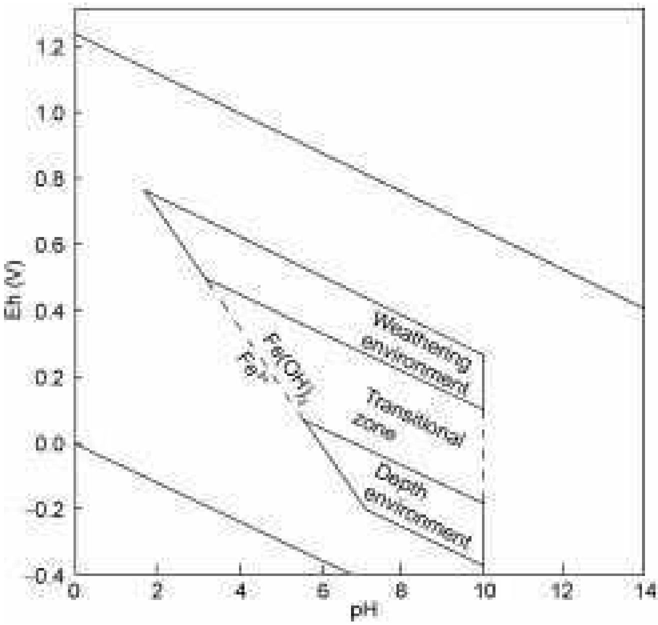
Ferrolysis (Reaction 8) provides one of the major pH/Eh controls on the ground waters and sets the lower limit on Eh in many waters. The result of oxidising a 1 M solution of Fe^+ is illustrated in Fig. 3.21. Eh and pH follow a particular trend from neutral/reducing to acid/oxidising. Actual field measurements correlate with this trend (Gray, 1997a). In the absence of dissolved Fe, Eh may be controlled by the O_2/H_2O_2 couple considerations of the slowness of the redox reaction and the poor stability of the solution led to Sato's summary of ground water pH/Eh controls, shown graphically in Fig. 3.22. Ground waters were divided into three environments:

- depth environment dominated by the chemistry of the magnetite/haematite and SH^-/SO_4^{2-}
- transitional zone
- upper weathering environment where ground water Eh is controlled by the O_2/H_2O couple.

Additionally, pH is maintained below 10, under control of carbonate precipitation, and in acid conditions pH and Eh are controlled jointly by the



3.21 Eh-pH measurements of salt seepage (after Blowes and Jambor, 1980).



3.22 Sato's summary of Eh-pH values for both the weathering environment and the deep environment.

$Fe^{2+}/Fe\,(OH)_3$ couple. Determinations by Baas Becking and Moore (1960) are in close agreement with this model. Note that as the ground water becomes further oxidised Mn^{2+} may also oxidise and hydrolyse.

$$Mn^{2+} + O_2 + H_2O \Leftrightarrow Mn_xO_y.H_2O + H^+ \qquad 3.9$$

This reaction is the equivalent of ferrolysis and may control the Eh and pH of a solution in a very similar manner.

3.6.2 Gold deposits in terrains of high relief

Eluvial deposits

'Eluvial' type deposition results from the weathering of source rocks *in situ*. The base of the weathering front (saprock) defines the lower boundary of weathering although Fe staining may penetrate to considerable depths along joints and shear planes. Chemical weathering is dominant and gold enrichment is related largely to the removal of soluble and colloidal waste material by percolating rainwater and, to a much lesser extent, by sheet flow and deflation. For the most part, source rock fabrics and spatial distributions of gold, silica and other chemically resistant species remain virtually unchanged. The gold-bearing systems are usually primary in nature but may be secondary where deep lateritic regolith has been dissected and eroded by weathering following epeirogenic uplift.

Alteration is confined mainly to the zone above the water table and decreases progressively with depth due partly to the less complete fragmentation of the rocks and partly to the decreasing dissolving power of the circulating water as more and more salts are taken into solution. Rainwater is most chemically active and vadose water can move freely through shattered surface rocks. In humid oxidising environments many elements hosted by sulphides (Cu, Ni, Zn and Mo) are subject to preferential leaching because of the instability of sulphur in oxidising conditions. Elements with high solubility (Na, K, Ca, Mg and Sr) are strongly leached. Feldspars are readily converted into soluble salts plus insoluble residues of clay, mica and other decomposition products.

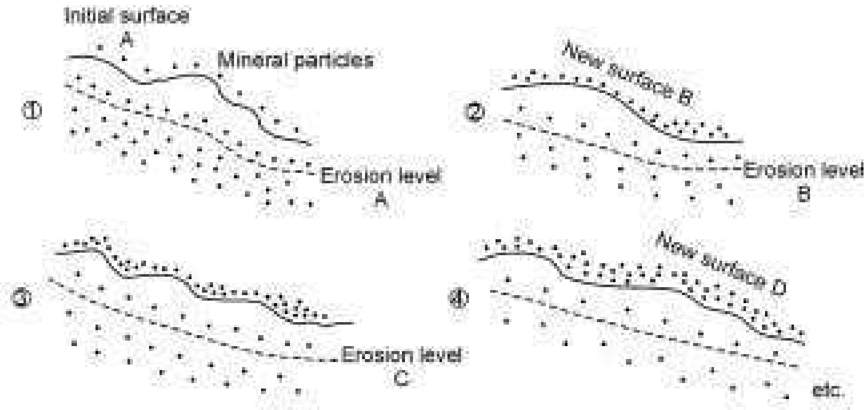
Colluvial concentrations

Products of weathering of lode formations on hillsides are subjected to various forms of transport, which determine the nature of their movement from source to site of deposition. Mechanical rather than chemical weathering processes are dominant; the tendency is for whole sections of partly weathered rock to break away and to merge with the loosely packed spoil moving slowly downslope. The colluvium thus comprises heterogeneous mixtures of clay, silt and sand with fragments and occasional boulders of the materials of which the source and country rocks are composed. Variables affecting the movement of colluvium on slopes are bedrock geology, climate, soil, vegetation and angle of slope.

Divisions of transported gold-bearing materials on slopes are deluvial and proluvial. Deluvial gold placers are formed by coarse particle deposition under conditions of gravitational sorting immediately downslope of the primary ore body. Slow intermittent mass movements of the loosely packed spoil are predominantly controlled by gravity. Mechanical enrichment is effected mainly by the winnowing action of gravitational forces and surface creep in the virtual absence of running water. Processes of interstitial settling cause large heavy particles to sink gradually towards bedrock through the slowly moving mass. Finely divided gold grains tend to rise to the surface with the faster moving, lighter gangue minerals. The exponential pattern of gold enrichment by deflation is described schematically in Fig. 3.23. The deluvial ore zone extends downslope from the primary source to the lowest limit of economic gold concentration. Proluvial detritus, comprising fragments of partly weathered colluvium containing quantities of gold-poor detritus mixed with barren material from the country rock, occurs from that point downward to the bottom of the slope.

3.6.3 Gold deposits in terrains of low relief

Lateritic and saprolitic deposit types form by chemical dispersion associated with the secondary mobilisation of gold in the regolith of loose broken rock overlying unweathered gold source rocks in terrains of low relief. The regolith

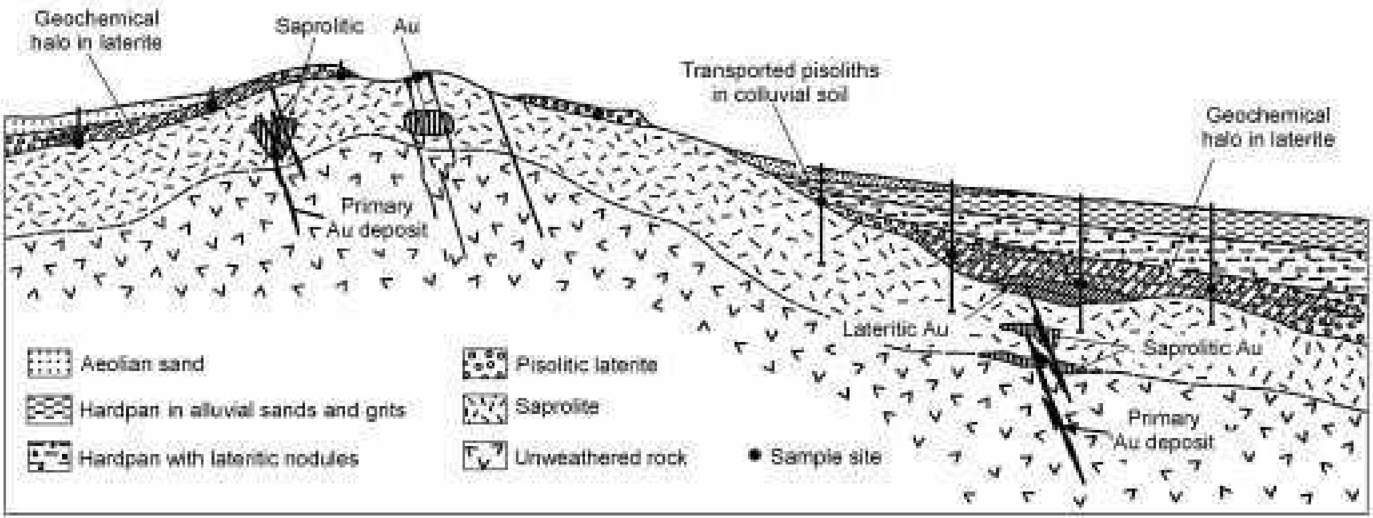


3.23 Exponential increase in gold values by deflation on slopes (after Macdonald, 1983a).

may be derived by alteration of bedrock directly below it (residual regolith) or by transportation in a fluid medium (water, glacial ice or air) from a more distant source on residual regolith or the bedrock itself (transported regolith). Regionally the regolith comprises the entire unconsolidated or secondarily re-cemented cover that overlies the solid bedrock. Locally, the combined influences of topography and climate determine the development of regolith and the distribution of gold within it. The influence of etchplain relief as a basis of comparison between different morphoclimatic zones is illustrated in a generalised section of some common Yilgarn regolith-landform situations is illustrated in Fig. 3.24. Erosion of chemical and cemented deposits in landforms of tropically weathered terrain results in (i) topsoil and duricrust of deep weathering profile (ii) upper zone of weathering profile (iii) colluvium and alluvium on a valley floor.

Regolith terminology

A deep-seated regolith typically includes varying amounts of fractured and weathered bedrock, saprolite, colluvium, organic matter, till, evaporites, and wind-blown sand in addition to soil layers that represent the uppermost biochemically weathered horizons of the regolith. The terminology and classification of the principal regolith units is summarised in Table 3.8. The profile grades upward from a narrow zone of partly weathered rock (saprock) overlying fresh basement rocks, through leached saprolite, a clay or sand rich zone, then a mottled zone in which iron oxide concretions mark a transition into a ferruginous zone. The pisolites become more numerous in the ferruginous zone (laterite) where they may be cemented together to form ironstone or duricrust.



3.24 Generalised section of some common Yilgarn regolith landform situations (as modified by Butt, 1997, from Smith and Anand, 1988).

Table 3.8 Regolith terminology (after Butt and Anand, 1997)

	Alternative terminologies				
Broad subdivision	General terms	Butt and Zeegers (1992)	Nahon and Tardy (1992)	French	CSIRO
Pedolith	Ferruginous soil [Latosol]	Soil	Soil	Sol ferrallitique	Soil
	Ferruginous zone [Laterite]	Lateritic gravel	Pebbly ferruginous layer		Lateritic gravel
	[Lateritic ironstone] [Plinthite]	Cuiresse (pisolitic, nodular, massive)	Indurated conglomeratic iron crust	Culrasse [Ferricrete]	Lateritic duricrust (pisolitic, nodular, massive, vermiform, fragmental)
			Soft nodular crust	Carapce nodulaire	
	Mottled zone	Mottled (clay) zone	Mottled (clay) zone	Argilos tachetées	Mottled zone
		Plasmic/arenose horizon			Plasmic/arenose horizon Collapsed and/or bracciated saprolite
Saprolith	Saprolite [Pallid zone]	Saprolite	Fine saprolite	Lithomarge Argiles bariolées	Ferruginous saprolite Clay saprolite Saprolite
		Saprock	Coarse saprolite	Altération pistache Arène/grus	Saprock
Protolith	Bedrock Unweathered rock	Unweathered/fresh bedrock	Bedrock	Roche mere	Fresh rock

Descriptive terms: (); Informal or equivocal terms: []

Deeply weathered profiles are products of long histories of weathering under widely varying climatic conditions and the full sequence of horizons is not developed until a considerable interval of time has passed. Climatic cyclicity is an important part of the genetic model; a basic requirement appears to be repeated wetting and drying and a fluctuating water table. Most lithologies are affected and regoliths exhibit great variation in fabric and origin even within the same profile or sequence (Butt and Anand, 1997). Factors determining the distribution of elements in regoliths are the stability of their primary host minerals, the presence or absence of secondary host minerals and the effects of changing climates on the chemical mobilities of the elements.

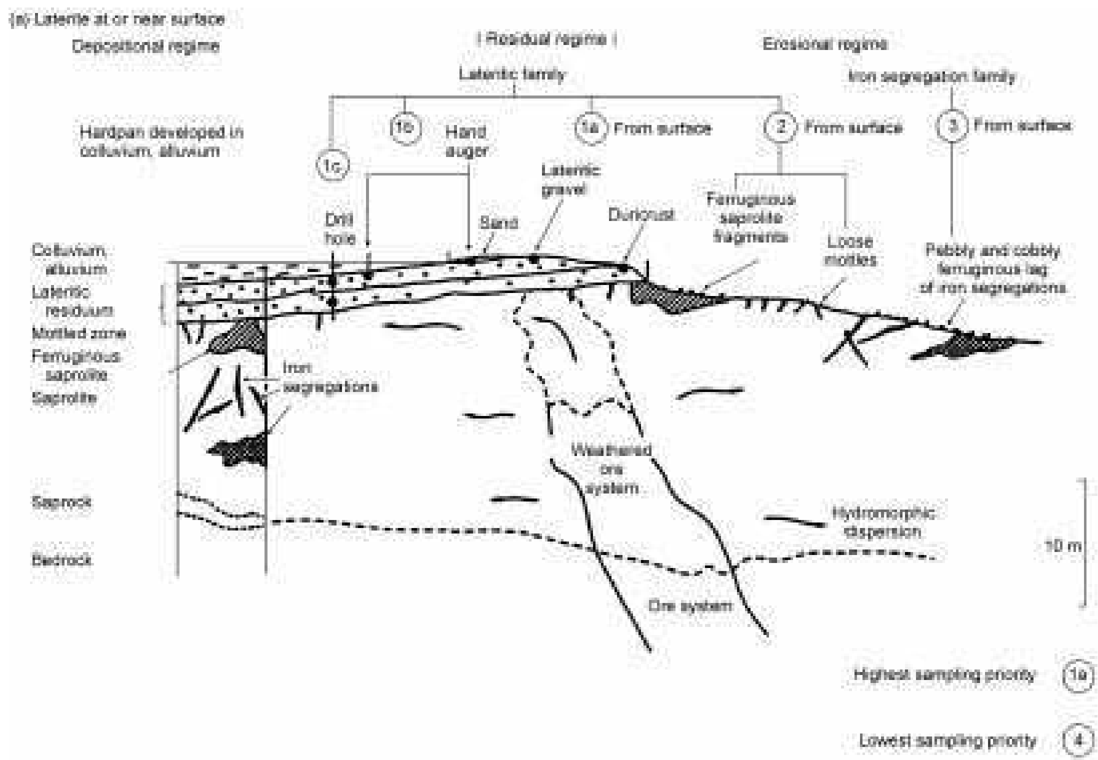
Lateritisation

Ferruginous zone. The ferruginous zone (laterite) is composed principally of secondary oxides and oxyhydroxides of iron (e.g., goethite, haematite) and hydroxides of aluminium (gibbsite). The term lateritic residuum is used by Anand *et al.* (1989) as a collective term embracing units such as loose gravels and duricrust that have a close genetic and/or compositional relationship with the substrate. Lateritic gold deposits contiguous with the ferruginous zone contain both Ag-rich primary gold and Au-rich secondary gold.

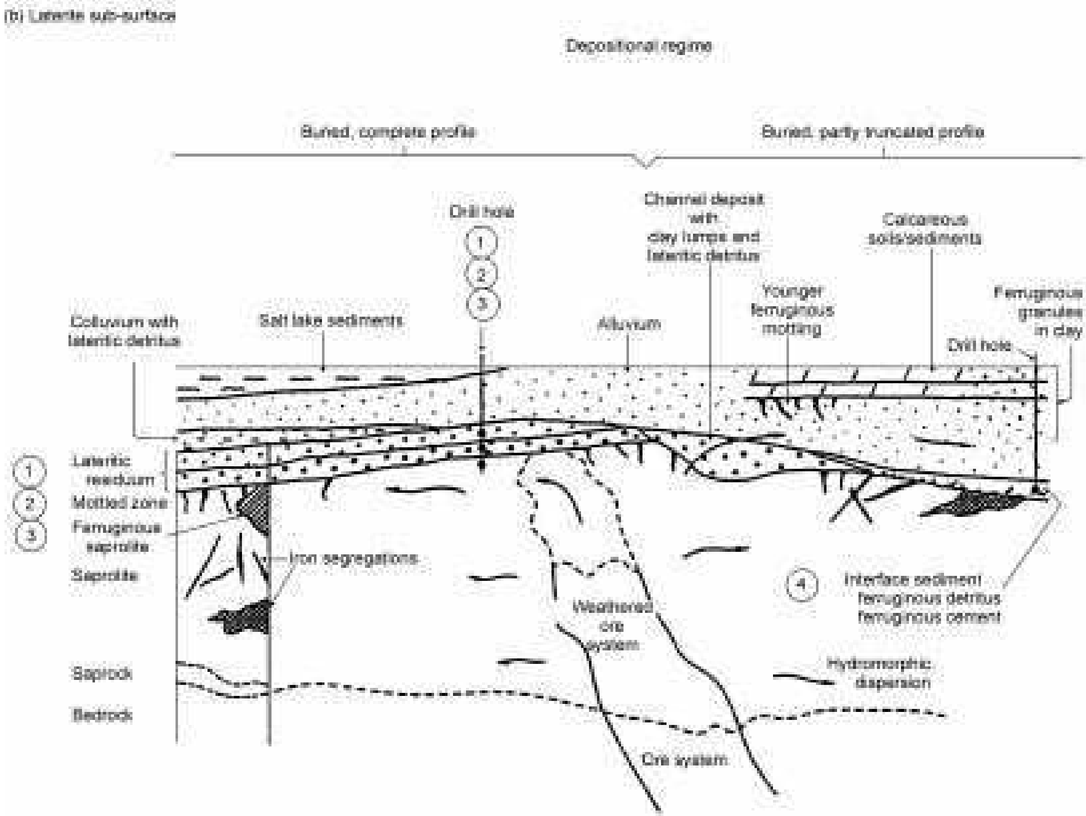
Thiosulphate ion, humic acid, and possibly cyanide ion are the most active ligands capable of complexing and dissolving the gold. A schematic representation (Fig. 3.25) is presented by Smith *et al.* (1999) of the conditions under which lateritic deposits may be formed (a) at or near the surface, and in (b) where laterite is likely to occur in the sub-surface. The principal units of the lateritic profile are ‘saprolith’ and ‘pedolith’.

Saprolith. The base of the saprolith marks the effective onset of weathering, though Fe oxide staining may occur along joints, shears and veins to considerable depths. Two saprolith horizons are recognised, saprock and saprolite. Saprock is a partly weathered rock with less than 20% of the weatherable minerals altered. The base of the saprock marks the onset of weathering (weathering front) which occurs along mineral boundaries and intra-mineral fissures, shears and fracture planes. Upper and lower boundaries are gradational and vary in depth and thickness over short distances. Weathering usually commences with the oxidation of primary sulphides, which are highly susceptible to weathering and only persist higher in the profile if enclosed within vein quartz. Carbonates are also chemically unstable and elements hosted by them, e.g. Ca, Mg, Sr, are similarly affected by leaching in the weathering zone. Gruz, a fragmental disintegration product of largely unweathered granitic rock is similar to saprock in many ways but is more friable.

During weathering, oxidation of the weathering front deep beneath the water table produces neutral to acid conditions, with lower pH favoured by felsic rocks



(a) Laterite at or near surface
Depositional regime
Residual regime
Erosional regime
Lateritic family
Iron segregation family
Hardpan developed in colluvium, alluvium
1c
1b
Hand auger
1a
From surface
2
From surface
3
From surface
Lateritic gravel
Drill hole
Sand
Duricrust
Ferruginous saprolite fragments
Loose mottles
Pebbly and cobbly ferruginous lag of iron segregations
Colluvium, alluvium
Lateritic residuum
Mottled zone
Ferruginous saprolite
Saprolite
Iron segregations
Weathered ore system
Hydromorphic dispersion
Saprock
Bedrock
10 m
Ore system
Highest sampling priority
1a
Lowest sampling priority
4



3.25 Choice of sample media, in situation (a), laterite at or near the surface, (b), laterite is sub-surface, based upon the orientation studies of Smith *et al.* (1999).

and high sulphide contents. Gold associated with tellurides or held in the lattice of the sulphides and other minerals may be released, but free gold remains immobile due to the absence of complexing ligands.

Saprolite is an upward extension of saprock, but one in which more than 20% of the weathered materials have been altered. Fine fabrics of weathered bedrock are retained but saprolitic materials tend to become more massive upwards as the proportion of clay increases. Cementation by secondary silica, carbonates, aluminosilicates and especially iron oxides is not uncommon. Gold deposits, formed either within the weathered ore zone or laterally dispersed into the wall rock consist primarily of secondary gold with residual primary gold confined to the lode system. Upgrading of a saprolite is similar to upgrading in an eluvial gold deposit in that original fabrics of the source rock are retained while mobile constituents of the rock are lost.

Pedolith

The four principal zones of the pedolith are the arenose zone, mottled zone, ferruginous zone (laterite) and soil. Although intensely weathered major structural features such as quartz veins may be preserved, usually with some change in orientation (dip). In the arenose or plasmic zone, massive clays or sandy clays are developed. The mottled zone is characterised by blotches of Fe oxides, which may develop with further mobilisation into nodules and other secondary structures containing tubular voids. These nodules increase in number upward and in the ferruginous zone may be cemented together to form a duricrust.

Post-lateritic modification

Modifications of the pre-existing lateritic regoliths may occur by partial truncation or by cementation as they adjust to complex sequences of environmental change (Butt and Anand, 1997). Gold grains are gradually liberated and/or modified until ultimately, two different styles of gold may be present in the weathering profile, primary and supergene. If gold-silver alloys are more soluble than gold, primary gold enrichment will be preserved unless extreme conditions prevail. A marked depletion of gold at depths of about 5 m to 15 m over most mineralised units and lodes follows the repeated strong leaching of the upper saprolite. This has occurred even when the profile has been truncated (Butt, 1988).

Partial truncation

Erosion typically follows episodes of climatic change or drainage rejuvenation following uplift. It results in removal of part or all of the upper layers of the profile and may expose the lower horizons and unweathered rock to surface weathering. These fresh surfaces may themselves become the parent material of newly formed soils or be overlaid by transported sediment. Aridity also promotes erosion although most of the debris is usually redeposited locally on lower ground.

Cementation

Secondary accumulation of alkali and alkaline earth elements in ground waters and their precipitation in the regolith commonly results in the precipitation of secondary sulphates and carbonates in the soils and upper horizons. Cementation is an important modification to the lateritic profile, the most important cementing agents being iron oxides (ferricrete), silica (hardpan), Ca and Mg carbonates aluminosilicates and gypsum (Butt and Anand, 1997). Modifications to lateritic regoliths due to changes in tectonic and climatic conditions are summarised in Table 3.9.

Table 3.9 Modification to lateritic regoliths due to changes in tectonic and climatic conditions (after Butt and Zeegers, 1992)

A. Tectonic activity

Uplift

- increased erosion
- lowering of the water table
- irreversible dehydration and hardening of ferruginous and siliceous horizons
- increased leaching of upper horizons under more oxidising conditions.

Downwarping

- decreased erosion, increased sedimentation in valleys
- waterlogging of lower parts of the landscape and imposition of reducing conditions.

B. Climatic change

To a more humid climate

- decreased erosion (due to thicker vegetation)
- increased leaching and deeper soil development.

To a less humid climate

- increased erosion (due to less vegetation)
- lowering of the water table
- irreversible dehydration and hardening of ferruginous and siliceous horizons
- decreased leaching.

To a semi-arid or arid climate

- increased erosion from uplands, with sedimentation on plains and in valleys
- lowering of the water table
- irreversible dehydration and hardening of ferruginous and siliceous horizons
- decreased leaching
- salinisation of ground water
- retention and precipitation of silica, alkaline earths and alkalis.

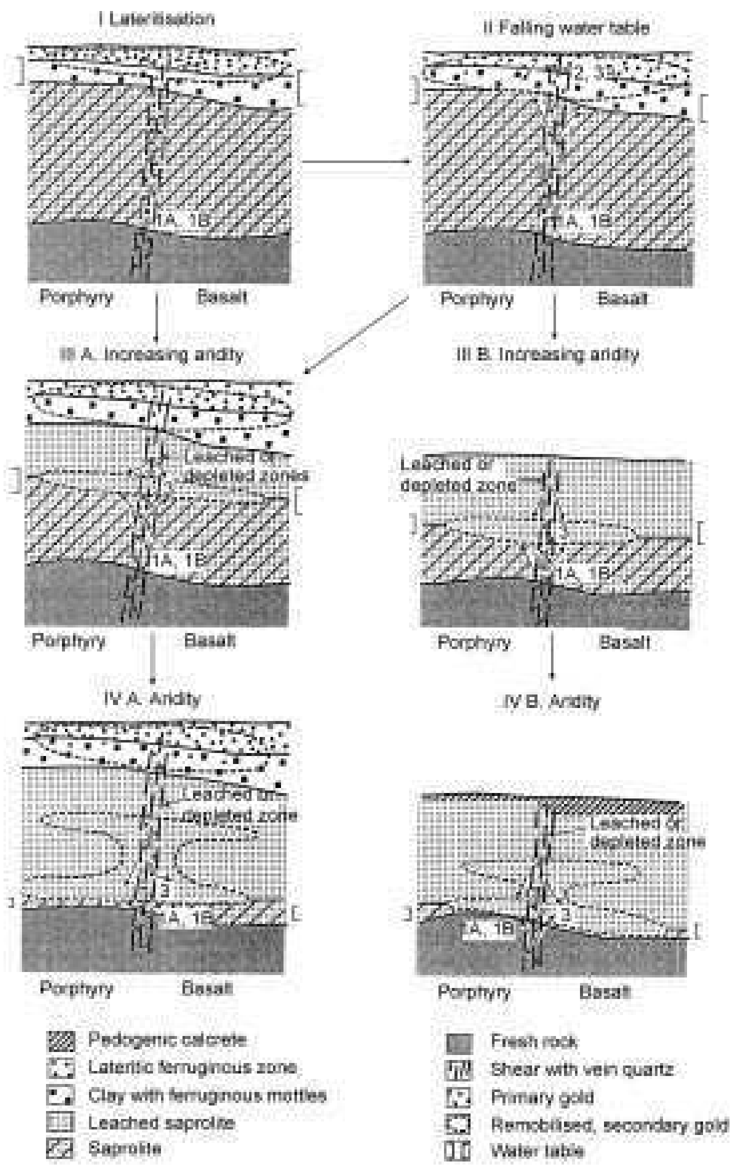
Cementation may be due in part to residual concentration and surface wash during surface erosion and partly to mobility in solution, or as colloids complexed by humic acids produced by rapid degradation of organic matter in soil. Some gold may also have been contributed directly to the soil after intake by plants. Reduction of the complexes by oxidation of ferrous iron results in the incorporation of fine-grained gold with low silver content in iron oxides in the ferruginous and mottled zones. Lateritic profiles are commonly modified by cementation in response to change from a humid to an arid climate. In the Yilgarn Block of Western Australia, the first protracted period of deep chemical weathering took place in the humid, warm to sub-tropical climates of the Cretacious to mid-Miocene, probably similar to the wetter savanna climate of the present (Butt, 1997). High water tables and generally acid conditions during this regime gave rise to extensive deep lateritic weathering and to profiles containing great variations in mineralogical and chemical composition.

A post-Miocene arid to semi-arid climatic regime coupled with minor uplift on the continental margin resulted in a general lowering of water tables and changes to alkaline chemical weathering reactions (Butt, 1997). Partial truncation of the regolith occurred due to climatic change (e.g., warm tropic to arid climatic conditions and/or drainage rejuvenation). The failure of weathering rates to keep pace with increased erosion rates and erosion of the upper horizon led to exposure of the lower horizon including bedrock, further reducing the already low relief.

Dispersion of gold and associated elements

Lateritic deposits with their mixture of high-fineness and low-fineness gold form widespread blankets over relatively narrowly weathered mineralised zones. Generally flat-lying, lateritic supergene deposits are developed contiguously with the ferruginous horizon and underlying mottled layers of the weathering profile. These deposits contain very fine-grained, low-fineness (>2% Ag) primary gold with occasional nuggets (Wilson, 1984) and Ag-poor (<0.5% Ag) secondary gold as fine-grained particles with some euhedral crystals typically associated with iron oxides.

The behaviour of gold under lateritic conditions is uncertain due to the overprint of remobilisation under subsequent arid conditions. Butt (1997) suggests that some indications can be gained by analogy with studies (Freyssinet *et al.*, 1989; Zeegers and Lecomte, 1992) of regolith geochemistry and the dispersion of gold and associated minerals in tropical and sub-tropical terrains. These studies propose that in humid lateritic environments gold is essentially immobile throughout the saprolite but is physically dispersed in the lateritic horizons and chemically associated with Fe oxides, to give widespread surficial haloes. Hence, although the Au-Fe oxide association is dominant in humid tropic areas, there may be a strong Au-carbonate association in arid areas due to subsequent remobilisation. It is generally accepted, however, that primary gold grains relict in the ferruginous horizons and the extent of secondary haloes in the Yilgarn Block of Western Australia are probably inherited from dispersion during lateritisation (Butt, 1997).



3.26 Model illustrating developments of supergene gold deposits by modification of a lateritic profile following uplift and change to an arid climate (after Butt, 1997).

Gold mobility in arid environments

During the gradual change from a wet savanna climate in mid-Miocene to an arid climate, general changes to and slowing of chemical reactions in the drier (post-Miocene) climates resulted from lowering water tables coupled with minor uplift as shown in Fig. 3.26. Several reversals to more humid climatic conditions favouring deep weathering probably occurred in many places. This would have caused fluctuations in the water table with temporary rises and stillstands. In the Yilgarn Block of Western Australia, erosion, partly from the loss of protective vegetal cover, reduced the already low relief so that the upper horizons became exposed to highly oxidising conditions.

Concomitant with such climatic and environmental changes, the principal mechanisms of gold dispersion may also have changed, in particular increased salinity will have encouraged the formation of soluble gold and silver halide complexes. Lawrence (1984) suggests that because of the solubility and reaction rates of gold in saline solutions any earlier formed enrichment, whether lateritic or saprolitic would be redissolved during later events. Salinity can arise from various processes, e.g. rock weathering, evaporation, dissolution of previously deposited halite, seawater incursion and aerosol deposition of seawater. Saline playas are common features in lateritic gold placer settings across most of the southern Yilgarn Block (Mann, 1984b).

In redeveloped humid conditions of the Holocene in the Darling Ranges of the Yilgarn Block, leaching of precipitated salts by increased rainfall and recreation of redox conditions suitable for ferrolysis would have produced acid, saline and oxidising ground waters capable of dissolving gold (Mann, 1984a). Although of very short duration in geological terms, chemical reactions during these humid periods would have been very rapid so that a significant redistribution of gold is possible. In their study of the reduction of gold halides by organic matter, Gatellier and Disnar (1988) calculated rate constants of the order of $10^{-5}\,s^{-1}$ in the temperature range 40–100 ºC. During these relatively short humid periods existing gold grains could be dissolved and mobilised as chloride complexes and reprecipitated either by organic matter or following reduction by ferrous iron. A wide variety of organic/biologically based complexes include cyanide complexes, organic complexes and colloidal gold where stabilised by organic matter (Gray, 1997b).

Similar type lateritic concentrations are known from deeply weathered regoliths in South Africa, Brazil and the Guyanas of South America. In South Africa the possible hydrothermal origin of Witwatersrand gold is still being discussed after more than 100 years of mining (see Section 5.4.4). The fact that the great Rand deposits gave rise to neither extensive eluvial nor alluvial deposits is difficult to explain (Boyle, 1979).

4
Sedimentation and detrital gold

Processes of entrainment, transport, sorting and deposition in natural stream channels depend as much on the physical properties of the sediment as they do on the hydraulic characteristics of the flow. Hydraulic properties are typical of the individual particle, of particle distribution and of the sediment in bulk. Individual particles are irregularly shaped and of diverse size and distribution. Fluvial drainage systems are described briefly in respect of the effects of various shapes and patterns of stream channels on sediment transport and deposition. Discussions of bed-load and suspended-load as the two most distinct modes of transport serve as an introduction to practical aspects of fluvial gold placer formation. The initial development of gold paystreaks takes place in stream channels during a single stage of downcutting and before tectonic and/or base level change can produce entrenchment, recycling and reconcentration. The effects of Quaternary tectonism and associated climatic adjustments are reflected in changes in the base level of erosion, hence of the consistency of rate of erosion of valleys. Quaternary adjusted placers of economic significance are preserved as deep leads under outpourings of basaltic lava or are reconstituted in other forms in fluvial, fluvio-glacial, fluvio-aeolean and shallow marine settings.

4.1 Sediment characteristics

Sediment comprises solid particles and grains of rock material that have been eroded from their parent bodies in a depositional environment. Quartz and other silicate minerals, as the most common and durable constituents of sediment at all stages of transport may survive throughout several erosional cycles. Less stable minerals such as calcite, pyroxene and sulphide minerals (pyrite, arsenopyrite, chalcopyrite, etc.) weather rapidly and their presence in an alluvial train usually denotes a very close source. The wide variability of gold shape and density is a major factor in predicting the behaviour of very small quantities of gold grains in transport with very large quantities of stream sediments.

Physical properties of sedimentary particles usually reflect many of the parent qualities of grain size, shape and density, modified according to the

Table 4.1 General order of resistance to mechanical wear of some common placer minerals

Economic minerals	Non-economic minerals
Gold	Pyrite
Monazite	Plagioclase
Zircon	Orthoclase
Rutile	Muscovite
Ilmenite	Quartz

intensity and duration of the forces tending to break them down. The less resistant minerals break down quickly to form clays and silts or, if soluble, are taken into solution in ground waters. More durable rocks and minerals survive longer, but are still subject to physical disintegration and chemical decay and disappear progressively with distance of travel and time. The general order of increasing resistance to wear of some common gold placer minerals is shown in Table 4.1.

4.1.1 Size

Of the various sediment properties, size is the most important parameter determining the hydraulic behaviour of sediments in solids-fluid flow. It is also the most readily measured and other physical properties of sediments such as shape and density tend to vary with size in a roughly predictable fashion. Cobbles, gravel and sand comprise the main constituents of streambeds with cobbles and gravels represented preferentially in the lower layers. Division between sand and silt occurs at around 62 microns. This approximates the upper size limit of a quartz sphere settling in still water in accordance with Stokes Law but does not describe the behaviour of such particles in turbulent flow, as discussed in Section 4.4.2.

Sieves probably emerged as a means of sizing when man first commenced to deal with commodities in bulk. References were made to sieving by the Greeks and Romans around 150 BC in written descriptions of sieves constructed from planks, hides punched full of holes, and screens woven from horsehair, reeds or human hair. Sieving was an established procedure in the Middle Ages, although still in a relatively crude form as illustrated in a sketch by Agricola (1556) (Fig. 4.1).

A number of sediment size classifications have been proposed, of which the system developed by Udden (1898) and modified by Wentworth (1922) is the most generally accepted. The Wentworth scale (Table 4.2) envisages five main size groupings represented by boulders, gravel, sand, silt and clay. It forms a geometrical progression with 1 mm as the base (1/8, 1/4, 1/2, 1, 2, 4, 8, etc.) which makes it convenient for plotting and subsequent mathematical treatment. Free settling particles can usually be sized into fractions, which exhibit

4.1 Sieving in the Middle Ages (from Agricola, 1556).

lognormal characteristics; if plotted on lognormal paper, the graphs will assume normal proportions.

The need to cope with very small measurements is only partly satisfied by micrometre measurements, which are unwieldy in the finer sizings (e.g. 1 μm = 0.001 mm). Krumbein (1941) found it more convenient to express each division of the Wentworth scale in terms of the negative logarithm to the base 2 of the diameter d of the particle in millimetres.

$$\varphi = -\log^2 d \qquad 4.1$$

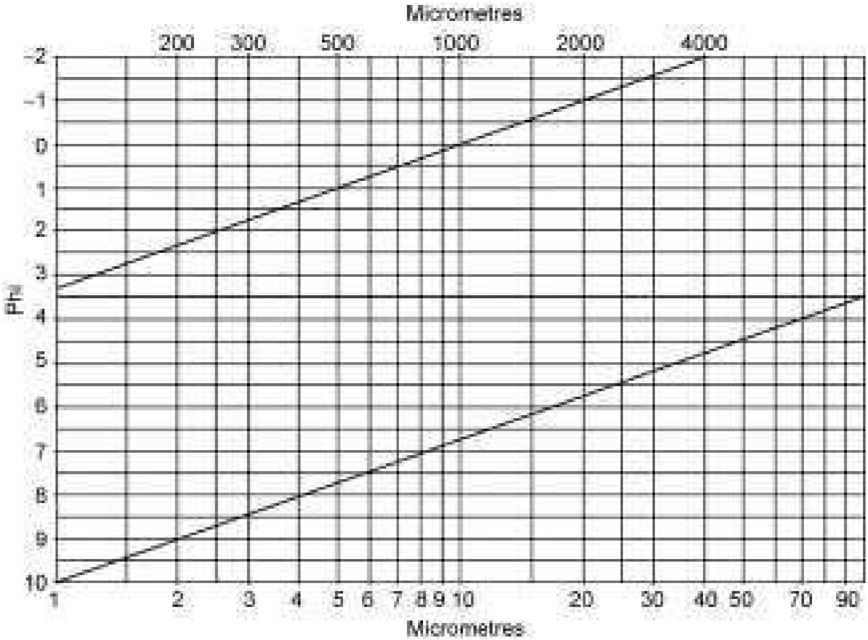
Krumbein introduced the phi (φ) unit of measurement, expressing each division of the Wentworth scale as one φ unit, while retaining its essential features in a simplified form. Whole φ numbers are substituted for the small Wentworth fractions and because the negative logarithm is used, the φ sizings increase

Table 4.2 Wentworth scale of sediment measurement

Phi size (φ)	Millimetres (mm)	Micrometres (μm)	Wentworth grade
−6.0	64	64 000	Cobbles 60.0 mm
−5.5	44.8	44 800	
−5.0	32	32 000	Coarse gravel
−4.5	22.4	22 400	
−4.0	16	16 000	20.0 mm
−3.5	11.2	11 200	Medium gravel
−3.0	8	8000	
−2.5	5.6	5600	6.0 mm
−2.0	4	4000	
−1.5	2.8	2800	Fine gravel
−1.0	2	2000	2.0 mm
−0.5	1.4	1400	
0.0	1	1000	Coarse sand
0.5	0.71	710	
1.0	0.5	500	0.6 mm
1.5	0.355	355	
2.0	0.25	250	Medium sand
2.5	0.18	180	0.2 mm
3.0	0.125	125	
3.5	0.090	90	Fine sand
4.0	0.063	63	0.06 mm
4.5	0.045	45	
5.0	0.032	32	Coarse silts
5.5	0.023	23	0.02 mm
6.0	0.016	16	
6.5	0.011	11.0	Medium silt
7.0	0.008	8.0	
7.5	0.0055	5.5	0.006 mm
8.0	0.004	4.0	
8.5	0.00275	2.75	Fine silt
9.0	0.002	2.0	0.002 mm
9.5	0.00138	1.38	Clay
10.0	0.001	1.0	

inversely with the particle size. As shown in Fig. 4.2, the relationship between micrometre and phi scale measurements is, in ascending particle size:

- Clay minerals, which comprise platy layers of alumino-silicates are about 1 nanometer (10^{-9} m) diameter and can be photographed only with the aid of an electron microscope.
- Silt composition tends towards relatively inert silica in the size range 4 to 62 microns (10 φ to 4 φ).
- Sands range in size from 62 to 2,000 microns (4 φ to -1 φ) and are almost pure silica or quartz.



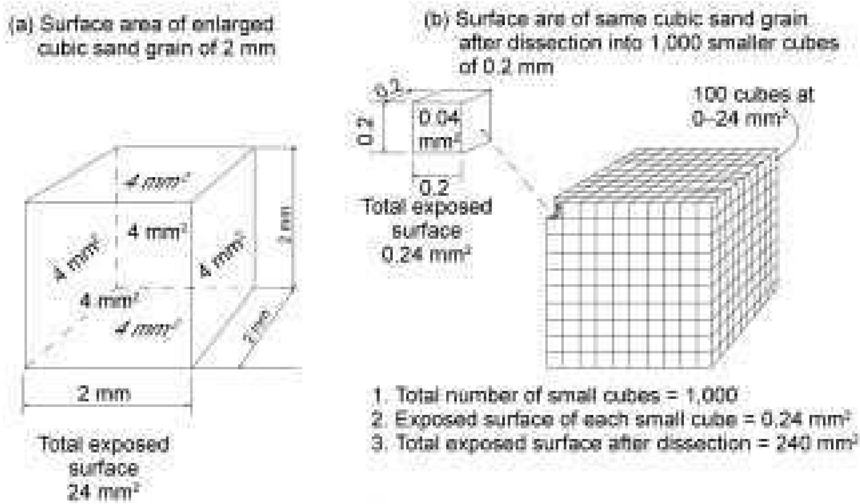
4.2 Phi–micrometre conversion chart. The phi scale converts data that are non-normal when measured on a simple arithmetic scale to a normal distribution (after Briggs, 1977).

- Gravels, 2 to 64 mm diameters ($-1\,\varphi$ to $-6\,\varphi$) include fragments of the parent rock.
- Boulders up to several metres diameter may contain all components of the original rock.

The sand-gravel class is typical of most gold placers and individual measurements need only be approximated in the field to determine how much oversize material can be screened out ahead of the gravity concentration plant. Flowsheet design depends upon measurement of associated silts and clays. While fractions of these materials rarely contain economically recoverable gold, they cause both handling and recovery problems at all stages of mining and treatment.

Particle size reduction

Particle size reduction is accompanied by increasing specific surface area to volume relationships. This may be demonstrated by successively reducing a cube of 1 cm/side into smaller cubes of 0.1, 0.01, 0.001 and 0.0001 cm sides respectively (Fig. 4.3). Whereas the ratio of volume to surface area for the 1.0 cm/side cube is 1:6, the ratio of volume to surface area of the same cube,



4.3 Increasing specific surface area to volume relationships.

when split up into one million 0.01 cm/side cubes is 1:600. For 0.0001 cm/side clay particles, the ratio is 1:60 000 and is enormously large for very minute particles (colloids). Suspensions of particles whose settling properties are significantly affected by the slow settling of finely divided particles are loosely defined as slimes in placer technology. The particle size at which this occurs appears to be less than about 100 μm for quartz density sediments.

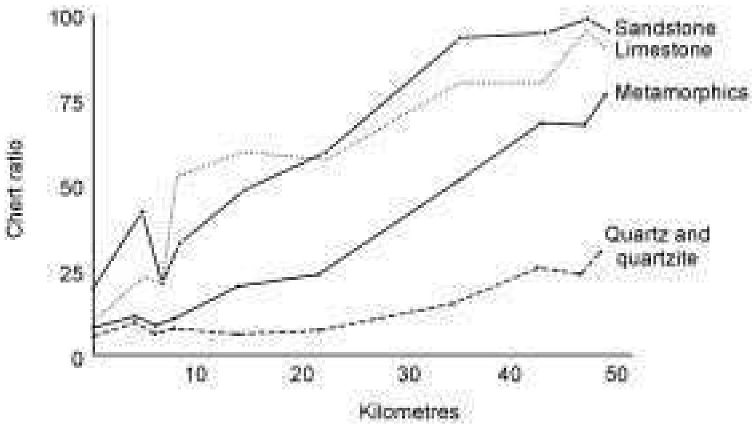
Local sorting is a function of distance of transport and Plumley (1948) demonstrated the variable rate of degradation of different sediment types in natural stream settings by recording the downstream size changes of sediment in 600 samples of terrace gravels from Battle Creek in the Black Hills of South Dakota. Figure 4.4 plots the results of these measurements. Chert, as the most resistant of the minerals present in the gravel, was used as the standard of reference in assessing the degrees of lithological change. The chert ratio R_c is defined as:

$$R_c = C/X + C \qquad 4.2$$

where X is the percentage of other rocks and C is the percentage of chert in the sample. The higher the chert content the more complete is the removal of other rock types.

Gold grain-size modification

As shown in Chapter 1, corrosion (chemical weathering) of gold grains in an alluvial setting may sometimes increase the size of gold particles, either geochemically by supergene enrichment or electro-chemically by the overprinting



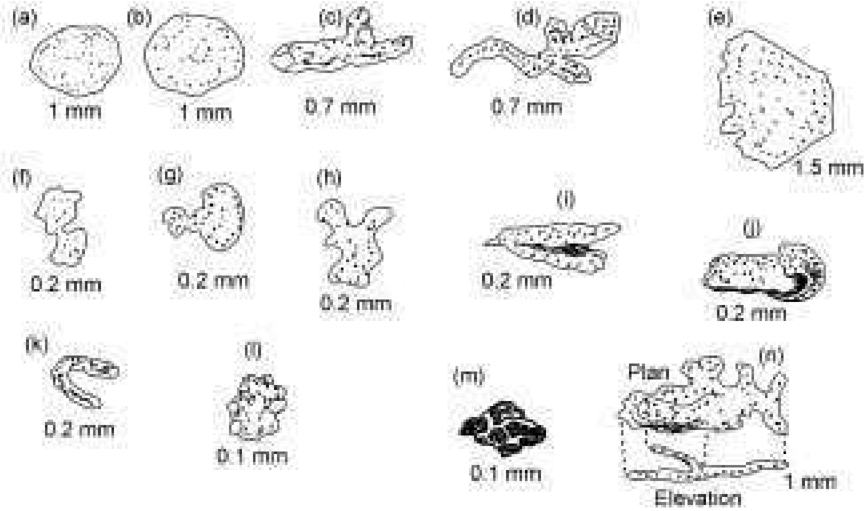
4.4 Changes in sediment lithology in the terrace gravels of Battle Creek, Black Hills, South Dakota (after Plumley, 1948).

of secondary films of gold on grain surfaces. Size increases may also occur mechanically by impact and melding together of gold grains during transport or by cementation of new grains by new gold. Samples must be sufficiently large at every significant stage of deposition to ensure the recovery of representative samples from all of the material to be evaluated.

Usually however, gold grain modification takes place by largely destructive processes, e.g. physical deformation and surficial wear. Detrital gold grains are degraded differently in different geomorphic settings (regolith, glacial, fluvial and aeolean) depending largely upon the habit of the original grain. Any prediction of rate of wear with distance of travel is highly speculative without adequate knowledge of hinterland geology. Physical deformation includes cracking and rounding of equant grains, compacting, pinching and folding of flat grains, and folding and rolling of wire gold to form cigar-shaped particles. Figure 4.5 describes some repetitious gold grain shapes from Kasongan, Kalimantan, Indonesia. Shapes (a) and (b) make up 70% of the suite, and (e), (f), (g), (h) 20%; the others 10%.

While the type of gross deformation and surficial wear of a particle of gold is generally predetermined by grain morphology, the nature and energy of the environment should also be considered:

- Fluvial regimes grade from high energy boulder-pool action with high impact forces in the upper reaches of streams, to low energy-low gradient action and mainly abrasive forces acting in lower channel sections.
- Under aeolean transport conditions deformation occurs by both impact and abrasion under high-energy conditions.
- In a glacial setting comminution ranges from mainly abrasive due to high-pressure contact with wall rock and basement, to mainly polishing during low-pressure englacial transport; sub-glacial transport takes place in channels under



4.5 Degradation of gold grains in samples repetitious from placer deposits in Kasongan, Kalimantan, Indonesia.

the ice under high-pressure flow conditions; the effects upon wear of such processes produce fluvio-glacial modifications that are not well understood.
- In shallow marine environments comminution may result from either high or low energy conditions, depending upon exposure to the open sea and the geological history of the shoreline.

Model experiments have done little so far to do much more than confirm facts already indicated qualitatively by experience in the field. None has yet provided quantitative results due to the impossibility of scaling down distributions of small particles uniformly without significantly changing the rheological patterns of their behaviour.

4.1.2 Shape

Shape, i.e., the gross morphology of a surface, is difficult to quantify because of the variability of its overall form. Briggs (1977) proposes four major influences on particle shape:

1. the original particle shape as inherited from previous cycles of erosion
2. lithological properties (e.g., mineralogy and structure) that may affect the three-dimensional shape of material supplied by weathering
3. duration and type of weathering during transport
4. duration and type of weathering after deposition.

Indices used to describe the different shapes taken by sedimentary particles refer either to two-dimensional shapes (roundness, angularity, sphericity), or three-dimensional shapes (e.g., sphericity and roundness). Sphericity is defined as the ratio of the surface area of a sphere having the same volume as the particle to the surface area of the particle. Roundness is the ratio between the radius of curvature of the particle and that of an inscribed circle and is controlled largely by the type and extent of weathering, thus being an indication of wear. Angularity is essentially due to fragmentation. Since spherical particles have the lowest resistance to transport, sphericity is adopted as the standard against which irregularly shaped particles can be compared.

Most sizing analyses in gravity concentration plant are classified by sieve aperture size and not by projected area diameter. British Standard BS3406 (Part 4, 1963) suggests multiplying sieve aperture size by a factor of 1.40 to obtain projected area diameter, but this factor should be applied with caution if the particles are of very irregular shape. Two other shape factors that have gained limited practical acceptance are the Corey and Heywood Shape factors. Of these, the Heywood factor is used mainly in mineral processing applications and is dealt with in Chapter 8.

Corey shape factor

The Corey factor S_f regards each particle as being represented by an ellipsoid of the same general proportions as the particle and is defined as:

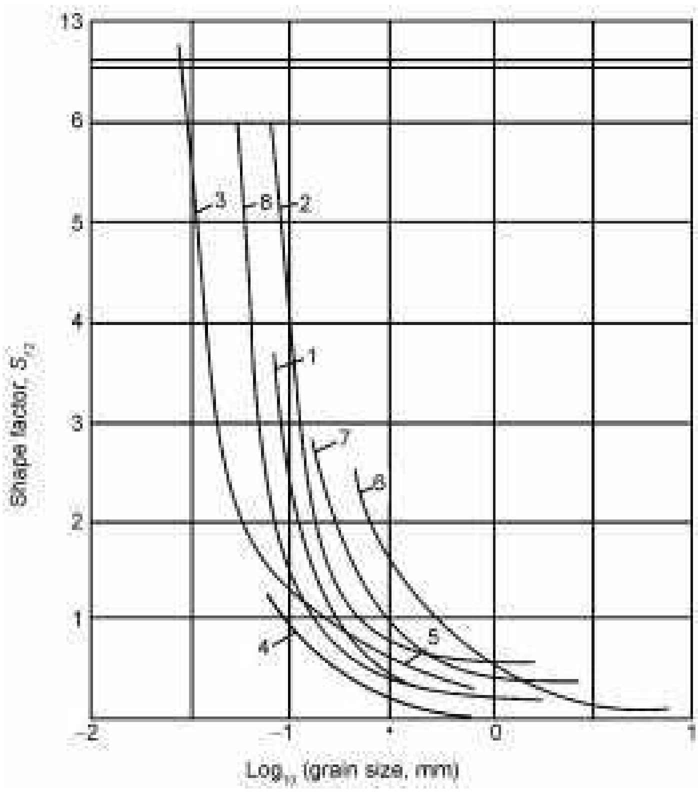
$$S_f = T/(LB)^{0.5} \qquad 4.3$$

where T is the thickness of the particle, L is the length, and B the breadth. This factor is determined by direct measurement of the three principal axes. The sphere ($L = T = B$), which has a factor of unity, is adopted as the standard for settling.

By definition, all other shaped particles have factors less than unity with settling rates decreasing according to their degrees of departure from unity. In practice, the Corey factor works reasonably well for river gravels and sand grains that are relatively equant in shape, and for coarse nuggety gold. The method suffers a number of constraints in the finer sizings, particularly when related to the settling of fine and flaky gold. Surface area:volume ratios increase with decreasing size, and viscous drag rather than density becomes the dominant factor influencing settling below about 100 microns. Borehole sample measurement is extremely labour intensive and the task of testing a statistical population of gold grains from most test programmes by the Corey method would probably not be economically feasible.

Weight-size factor

A different type of shape factor used in parts of Asia is related to average sieve sizes of gold grains. Figure 4.6 shows the relationship between this shape factor



4.6 Relationship between S_{f2} and average sieve sizes of gold grains from different placer ores in the USSR (from Zamyatin, 1975).

S_{f2} and average sizes of gold grains from eight different placer deposits in the Soviet Union. The ordinate in this figure S_{f2} is defined as:

$$S_{f2} = W/w_s \tag{4.4}$$

where W is the average weight of 50–100 particles from a sieve fraction, w_s is the weight of a gold sphere equivalent to the average diameter of the sieve fraction.

4.1.3 Density

Density is the quantity of mass per unit of volume; stated in g/cc, mass is defined as the quantity of matter in a body and its mass density (ρ) is the mass per unit volume. The mass density of water ($\rho = 1.0$) is the standard against which the densities of all other substances are compared. In the Centigram Second (c.g.s.) system of units a gram is defined as the mass of 1 cubic centimetre of pure water at its temperature of maximum density (0.99987 g/cc) at 3.98 ºC.

Weight density

Whereas weight is the force exerted on a mass by gravitational attraction, weight density γ is the weight per unit volume of the substance. The force per unit volume due to gravity is equal to the weight of the substance per unit volume. The properties ρ and γ can be related via Newton's second law of motion ($F = ma$) to give $\gamma = \rho g$ where ρ is the density of the solid and g is the acceleration due to gravity, $9.81\,\mathrm{ms}^{-2}$.

Individual particles of different weight density settle at different rates in a fluid according to the resistance to movement imposed on the particles by the fluid properties of density and viscosity. Consider two spherical particles of gold totally immersed in water of density ($\rho = 1$). One particle is of high-grade gold, density 19.0; the other is a lower grade particle (electrum) of density 16.75. The effective specific weight of the first particle is $(19.0 - 1) \times 9.81 = 176.58$ that of the second particle $(16.75 - 1) \times 9.81 = 154.51$. The rate of settling of the lower grade particle, having a lower density and hence a higher surface area to volume ratio than the high-grade particle, will be slower because of higher drag forces.

4.2 Fluvial hydrology

The flow of water in stream channels is governed by the interaction of two opposing forces: gravity and friction. Gravitational forces act to pull the water downslope and exert pressure on the confining channel walls. Resistance to flow is provided by a combination of:

- viscous shearing between the fixed channel boundaries and the moving water
- turbulence and eddying within the fluid
- expenditure of fluid energy when impact and viscous forces build up sufficiently to overcome the inertia of particles at rest.

These forces, which involve velocity and acceleration possess both magnitude and direction and hence are vector quantities. Pressure, temperature, length, area and volume, as scalar quantities, have magnitude alone and can achieve directional status and show a gradient only when mapped spatially.

4.2.1 Dimensions and units

Measurements of physical quantities are expressed both in terms of a numerical magnitude and as a unit of measurement. The concept of length is fundamental and is applied to measurements of depth, width, length, height and diameter. Three other fundamental dimensions are mass (force), time and temperature. Units of measurement are grouped dimensionally into three main categories geometric, kinematic and dynamic:

1. Geometric dimensions are described in terms of length (L), area (L^2) and volume (L^3).
2. Kinematic dimensions are described as time (T), velocity (LT^{-1}), acceleration (LT^{-2}) and discharge (L^3T^{-1}).
3. Dynamic dimensions are described as mass (M), force (MLT^{-2}), pressure intensity ($ML^{-1}T^{-2}$), impulse and momentum (MLT^{-1}), energy and work (ML^2T^{-2}) and power (ML^2T^{-3}).

Thermodynamic measurements are expressed as some combination of length, mass of (force), time and temperature. Important physical quantities in which the dimensions cancel out are the dimensionless Froude number 'F' and the Reynolds' number 'Re'.

4.2.2 Fundamentals of physical equations

To gain an understanding of hydrological and fluvial morphology it is necessary to first understand the essential characteristics of equations that describe the various parts of the hydrological cycle (Dingman, 1984). Some properties, such as the physical characteristics of water, can be considered as constants in the matters discussed in this chapter. Other properties of importance to hydrology such as rates of flow in streams, water depths and precipitation, are extremely variable in space and time.

It should be noted that most empirical formulae describe sedimentary processes in terms of theoretical equations that are dimensionally homogeneous, i.e., analytically correct, and descriptions of specific features of their particular dimensional systems are provided in qualitative terms. This does not mean that they are necessarily correct. Experimental results, although faithfully reproduced, are often derived from data that are either inadequate or incorrect. The equations so modified may deviate materially from mathematical correctness and lead to entirely wrong conclusions. For example, although Heywood's shape factor (refer to Chapter 8, eqn 8.8) is dimensionally homogeneous, it will be physically correct only if the correct value of the dimensionless coefficient k is used. Approximate value becomes increasingly uncertain when applied to the more extreme shapes of gold.

Conservation of energy

Conservation problems relate to the fundamental statements that matter, energy and momentum cannot be created or destroyed in any normal physical process. As an expression of the conservation of energy for a steady flow of fluid, eqn 4.5 defines the different forms of energy at every point in a stream. Since water is virtually incompressible, fluid density remains constant throughout the region of flow. The equation of motion (Bernoulli's theorem) is a constant represented by the sum of the potential head, the pressure head and the velocity head at every point along a stream line:

$$\text{Energy } (E) = 0.5\ pv^2 + P + pgZ = \text{a constant} \qquad 4.5$$

where Z is the elevation of a point relative to some arbitrary datum, e.g. sea level; pgZ is the energy of position, i.e. potential energy; P is the pressure at that point; and $0.5\,pv^2$ the energy of motion, i.e. kinetic energy.

The proportionality constant (often referred to as C or K) expresses the frictional and other losses in the system due to the boundaries over which flow is taking place and the physical nature of the flow. An example is Darcy's law:

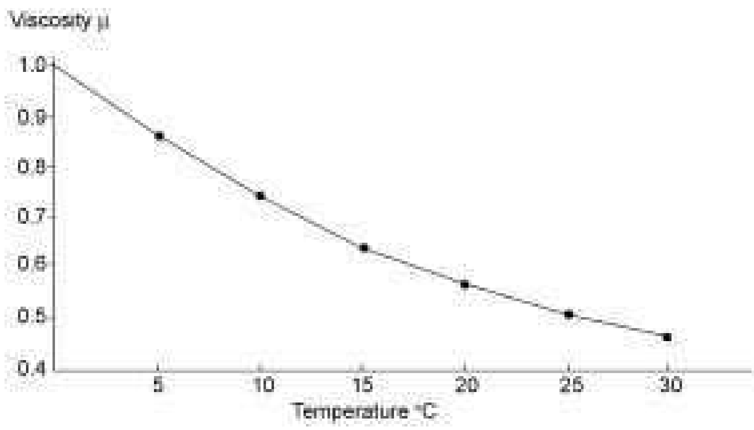
$$K_D = K_I(\gamma/\mu) \qquad 4.6$$

where K_D is the hydraulic proportionality constant (LT^{-1}), which is a system parameter that depends upon the intrinsic permeability K_I of the medium (L^2), and on the weight density $\gamma(FL^{-3})$, and dynamic viscosity $\mu(FTL^{-2})$ of the water. Note that a rapid decrease in dynamic viscosity takes place with increasing temperature. The steep gradient of the curve between 0 ºC and 20 ºC (Fig. 4.7) suggests that some aspects of solids-fluid flow in sub-arctic conditions may be significantly different from those in the tropics. Table 4.3 lists the various values of water properties as functions of temperature.

Diffusion equations

Diffusion equations describe the movement of matter, momentum and energy through a medium in response to a gradient of matter, momentum and energy respectively (see 'Geochemical dispersion', Chapter 5). The general dimensions of diffusion are (L^2T^1). Since flow is always away from a region of high concentration to one of lower concentration:

$$Q_S = -D_S\ ds_{/dx} \qquad 4.7$$



4.7 Plot of dynamic viscosity μ vs. temperature °C.

Table 4.3 Relative values of water properties as functions of temperature (after Dingman, 1984)

Temperature °C	$\rho\gamma$	μ	Υ	σ	C_p	$\lambda\nu$	K_σ
0	1.0000	1.0000	1.0000	1.0000	1.0000	1.0000	1.000
5	1.0001	0.8500	0.8500	0.9907	0.9963	0.9953	1.016
10	0.9986	0.7314	0.7315	0.9815	0.9940	0.9904	1.031
15	0.99926	0.6374	0.6379	0.9932	0.9934	0.9857	1.046
20	0.99836	0.5607	0.5616	0.9630	0.9915	0.9810	1.062
25	0.99720	0.4983	0.4997	0.9524	0.9910	0.9763	1.077
30	0.99580	0.4463	0.4482	0.9418	0.9907	0.9715	1.093

ρ = mass density; γ = weight density; μ = dynamic viscosity; Υ = kinematic viscosity; σ = surface tension; C_p = heat capacity; $\lambda\nu$ = latent heat of evaporation; K_σ = molecular thermal conductivity.

where Q_S is the rate of movement of matter, momentum, or energy through a unit area normal to the direction of gradient of mass, momentum, or energy. $ds_{/dx}$ represents the gradient of mass, momentum, or energy in the x direction. D_S is a diffusion coefficient, or diffusivity for mass, momentum, or energy in the medium. The term Q is generally called a flux density (flow per unit area per unit of time). Equations for the specific case of matter are sometimes called mass-transfer equations. For momentum, the rate of momentum transfer is proportionate to the viscosity and to the velocity gradient. The gradient of heat energy depends upon the diffusivity of heat energy in the medium, the heat capacity of the medium and its temperature.

4.2.3 Gravitational forces in open channel flow

Gravitational forces (including hydrostatic pressure) may be derived in magnitude and pressure for fluids at rest and in motion. Due to these two forces, the elevation of an element of fluid above a horizontal datum will represent its gravitational potential energy. Expressions derived for the magnitude of potential energy at any point in a stationary fluid allow gradients of mechanical potential energy to be computed that will induce flow in open channels. The relative magnitudes of these and other forces that come into play once flow commences tend to resist or change the direction of motion. The most important forces that affect the nature of flow in natural stream channels are due to the relative proximity of boundary conditions

Hydrostatic pressure

The static water pressure P_W exerted against a plane area of surface A under a fluid of height h is given by the expression:

$$P_W = \gamma hA/A = \gamma h \qquad 4.8$$

The weight of the water column is γhA, where γ is the weight density of the water. At sea level with air as the fluid, P becomes P_A the atmospheric pressure of an element of matter at sea level. The total pressure at the plane is then:

$$P = P_W + P_A = \gamma h + P_A \qquad 4.9$$

Pressure is one component of potential energy and as already noted a gradient of gravitational potential energy is needed between two points for flow to take place. The direction of the movement is in the direction of the lowest pressure. The rate of motion is proportional to the spatial rate of change (gradient) of potential energy between the two points.

Mechanical potential energy

As applied to natural flow conditions, an element of water moving from rest in the headwaters of a stream contains mostly potential energy, i.e., the product of its density ρ, acceleration due to gravity g and its elevation above sea level z. At any point downstream, losses of potential energy with decreasing z are compensated for by gains in kinetic energy. Some of this energy is expended in eddying, turbulence and changes in momentum, particularly at the foot of rapids and waterfalls. A further interchange of energy takes place when the stream changes direction. Flowing around a bend, the velocity increases on the outside and is retarded along the inside of the bend. Elements of water in a horizontal line across the bend thus have equal quantities of potential energy and total energy but different quantities of kinetic energy and pressure energy. At sea level, any remaining energy is dissipated in turbulence and intermolecular friction.

4.2.4 Forces acting on fluids

Forces acting on fluids and on solids and fluids in relative motion react differently according to differences in the physical characteristics of the solids and the relative magnitude and direction of hydraulic forces, which act both to induce and resist movement thereby determining the nature of the flow. The net rate of entrainment, transport and deposition of bed-load materials depend upon the degree of balance between the individual phases of such exchange. The forces involved in the reactions are termed either 'body' forces or 'surface' forces.

Body forces

Body forces act from a distance upon the whole bulk of a fluid element or solid immersed in a fluid. Typical body forces comprise:

- fields of force exerted by magnetic and electrostatically charged devices in processing plant
- forces of electrostatic attraction, repulsion and ionisation in cohesive soils
- magnetic forces due to the Earth's magnetic field
- centrifugal forces arising from rotation of the Earth-Moon planetary system
- gravitational forces exerted by the Sun, Moon and Earth.

Prime consideration is given to gravitational body forces in this chapter and throughout the book generally.

Surface forces

Surface forces act by direct contact between the surface of a fluid element or submerged body and its surroundings. A force that acts perpendicular to the surface of action is a normal force with pressure intensity. Shear stress is pressure intensity acting tangentially to the surface of action.

Dynamic pressure

Solids and fluids can exert dynamic pressures and impact forces only when they result from changes in momentum. By definition, momentum or impact I is the product of mass times velocity $(I = MV)$ and the force F changing the momentum is its time derivative:

$$F = Mdv/dt \qquad 4.10$$

As applied to sluicing operations where a high velocity jet of water is directed against a bank of alluvium, $dv/dt = 0$ at impact and the momentum of the jet is totally destroyed in the direction of flow. For a vertical bank, since force equals the rate of change of momentum, the pressure exerted by the wall on the jet equals pAV^2 in the horizontal plane, where A is the cross-sectional area of the jet at the point of impact. As applied to centrifugal pumping equipment, when a free body of fluid in steady motion undergoes a change in angular motion, the resultant of the external forces acting on the body is a torque equal to the time rate of change of the angular motion.

Shear stress

A force that acts tangentially to a surface (shear stress) expends part of its kinetic energy in overcoming viscous forces. Shear forces are developed both from intermolecular friction between adjacent fluid elements and from fluid drag along surfaces in contact with and in relative motion with the fluid. Shear stress and rates of deformation remain constant for any given pressure and temperature, regardless of the duration of the action.

4.2.5 Background to sedimentation

Theoretical aspects of solids-fluid flow are mainly concerned with the ideal conception of frictionless, incompressible fluids. In practice the rheology of many of the fluids encountered changes according to variations in drag forces due to viscosity. The mass density of water in proportion to the mass density of the solids and their concentration by weight is increased by addition of dissolved or suspended solids. Due to its content of dissolved salts seawater has a higher mass density than fresh water.

The three principal fluids in alluvial gold settings and their approximate densities are:

- fresh water $\rho_w = 1.00\,\mathrm{gcm}^{-3}$
- seawater $\rho_{sa} = 1.25\,\mathrm{gcm}^{-3}$
- air $\rho_a = 1.25\,\mathrm{kgcm}^{-3}$ (dry air at sea level).

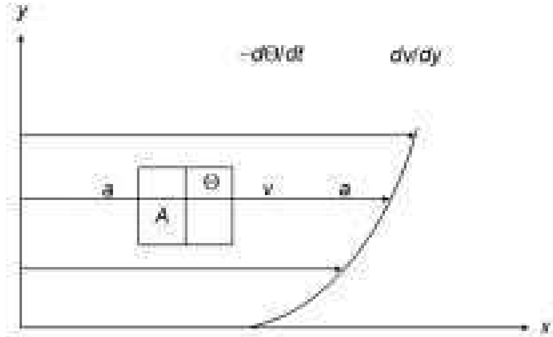
Air, broadly speaking, has an average unconfined pressure at the surface of the Earth of about $1\,\mathrm{kg/cm}^2$. However, because it is a mixture of gases, it has the attribute of filling any container and can readily be compressed into a smaller volume. The air pressure within the container then rises above that of the surrounding air. Otherwise it has certain similarities to water by exhibiting a definite viscosity.

The analytical framework and equations of fluid motion are based upon investigation of one-dimensional flow in an 'ideal' hence frictionless fluid. In such conditions, stream flow is said to be either 'steady' or 'unsteady' at any point in the fluid, depending upon whether the velocity vector changes or does not change in either magnitude or direction with time. 'Uniformity' of flow refers to the lack of variation of the velocity vector with distance along a streamline. Non-uniform flow is such that conditions involving the velocity vector vary from place to place at any instant. In all cases the condition of steady, uniform flow, even in laboratory scale apparatus, can be regarded only in the statistical sense.

Flow in natural stream channels takes place in a 'real' fluid (water) which, being essentially turbulent in nature, is both unsteady and non-uniform. The flow is invariably 'unsteady' because the magnitude or direction of velocity or both varies with time. It is 'non-uniform' because the velocity, measured from one point to another in the direction of flow, changes with every change in boundary geometry. At flood time, the stages of flow change instantly as the waves and surges pass by and the rate of sediment transport past any one section varies accordingly.

Viscous flow

Viscosity is the property of a fluid that gives rise to an internal shear stress opposing change in the shape or arrangement of the elements of the fluid during flow, and is the degree to which this property exists in a particular fluid. In



4.8 Shearing stress on element *a* travelling at mean velocity *v* in direction *x*. It will be deformed at angular rate equal to d*v*/d*y*.

simple terms the dynamic viscosity μ of a fluid describes the relationship between the stress intensity and the accompanying rate of fluid deformation. The elementary form of a shear stress produced by the layers between adjacent elements of a fluid slipping over one another is illustrated in Fig. 4.8. This figure shows that when element a travels at a mean velocity v in the direction x it will be deformed at an angular rate of dv/dy and the intensity of shear along a-a will be as first expressed by Newton:

$$\tau = \rho\, dv/dy \qquad 4.11$$

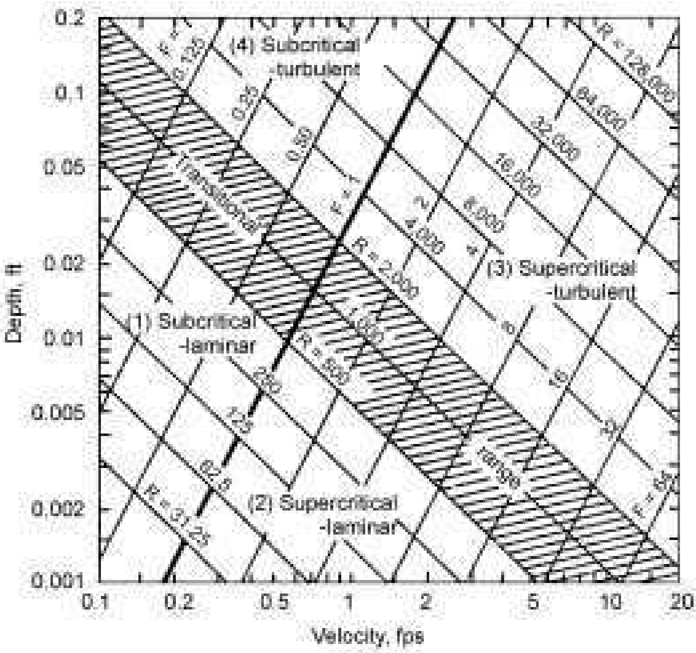
The shear stress τ is related to the rate of angular deformation dv/dy through a proportionality factor ρ representing the viscosity of the fluid.

Kinematic viscosity is measured in 'Stokes' as the absolute viscosity of a fluid divided by its density. Kinematics is that branch of mathematics, which treats of pure motion without reference to mass or cause. ν is the ratio of dynamic viscosity μ to mass density ρ:

$$\nu = \mu/\rho \qquad 4.12$$

The behaviour of a fluid in motion is governed basically by the effects of viscosity and gravity relative to the inertial forces of the flow (Chow, 1959). Depending upon the relationship of viscous to inertial effects, three intergrading flow lines are developed laminar, turbulent and transitional:

1. Flow will be laminar if viscous forces are sufficiently dominant and streamlines remain virtually separate from one another over a defined length of flow.
2. Flow will be turbulent if inertial forces are dominant; particles will then move in highly irregular paths and streamlines will become hopelessly confused.
3. A mixed or transitional state of flow exists between the laminar and turbulent states.

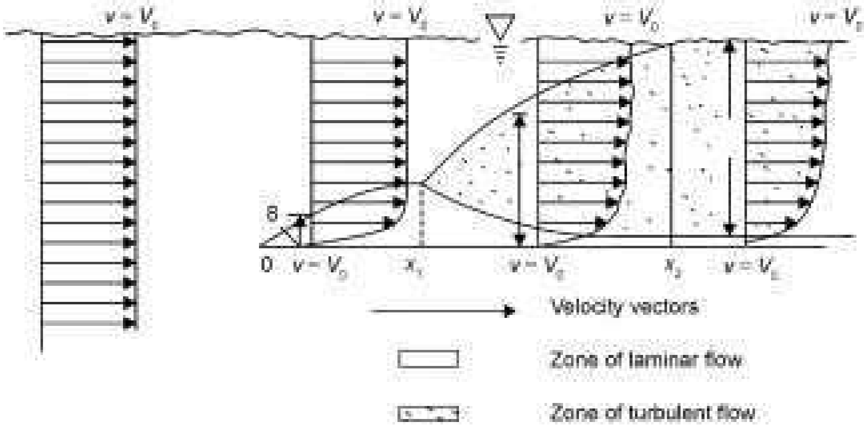


4.9 Depth–velocity relationships for four regimes of channel flow (after Robertson and Rouse, 1941).

Since open channel flow is inherently turbulent, the only practical regimes of flow in natural stream channels are sub-critical turbulent and super-critical turbulent in that order of importance. Sub-critical laminar and super-critical laminar regimes of flow may occur only where there is a very thin depth of fluid. The rate of momentum transfer (shear stress τ) is proportional to the viscosity and to the velocity gradient; such conditions are of particular significance in the operation of gravity concentration devices such as shaking tables, vanners and other thin film separating plant (see Chapter 8). Depth–velocity relationships covering the fundamental regimes of flow are illustrated in Fig. 4.9.

Boundary layer concept

The discovery that 'momentum is transferred from a flowing stream to a solid surface through a surrounding thin layer of water' was probably recognised first by Prandtl (1942) in 1931 from articles on the motion of fluids and gases by Gustav Fischer published in 1913. Prandtl found that within this narrow region of stress, the velocity is zero at the interface and increases parabolically to reach its maximum value at some distance into the stream where it approaches stream velocity. The zone in which this vertical velocity gradient exists is called the



4.10 Development of boundary layer in open channel with ideal entrance conditions (Dingman, 1984, modified from Chow, 1959).

boundary layer. Within the boundary layer, flow possesses a velocity gradient that enables it to transmit stress. Viscous forces are negligible in flow outside of the boundary layer and hence cannot exert stress on that boundary.

A simplified two-dimensional profile of a boundary layer with accompanying downward transfer of momentum across a wide open channel section is demonstrated in Fig. 4.10. The absence of a vertical velocity gradient denotes the absence of friction in the flow entering the horizontal boundary which thus has a common velocity of $v = V_0$. Friction at the boundary retards the flow inducing a downward transfer of momentum and creating a boundary layer of thickness δ. A laminar boundary layer is developed between 0 and x_1 where turbulence arises. Thickness of the boundary layer increases and turbulent flow is fully developed at x_2 with a thin zone of laminar flow near the bottom. This condition is typical of most streams.

Significance of Reynolds number

The parameter known as the Reynolds number 'Re' provides a relationship between inertial forces and viscous forces for all types of fluid motion. Re has the dimensions:

$$\mathrm{Re} = VL/v = L^2T^{-1}/L^2T^{-1} = 0 \qquad 4.13$$

In open channel (stream) flow the characteristic length is taken as the hydraulic radius $R = A/P$, where A is the cross-section of flow and P is the wetted perimeter. Equation 4.13 can be re-written:

$$\mathrm{Re} = vR/\nu \qquad 4.14$$

Reynolds numbers less than 500 signify dominantly viscous (laminar) flow. When Re is greater than 2,000 viscous forces are insignificant. Reynolds found that in pipeline flow, flow becomes fully turbulent at Re 12,000 regardless of pipe diameter and fluid viscosity. On returning to the laminar-state the inertial effects persisted until Re again drops to 2,000. In practice however, the actual magnitude of Re varies widely with the boundary geometry in open stream channels because of the arbitrary nature of the characteristic length L and inherent differences in the pattern of flow.

Froude number

The effect of gravity on the state of flow is represented by the ratio of inertial forces to gravity forces. This ratio velocity^2/flow depth x acceleration is defined as:

$$V^2/dg = L^2T^{-2}/L.L^{T-2} = 0 \qquad 4.15$$

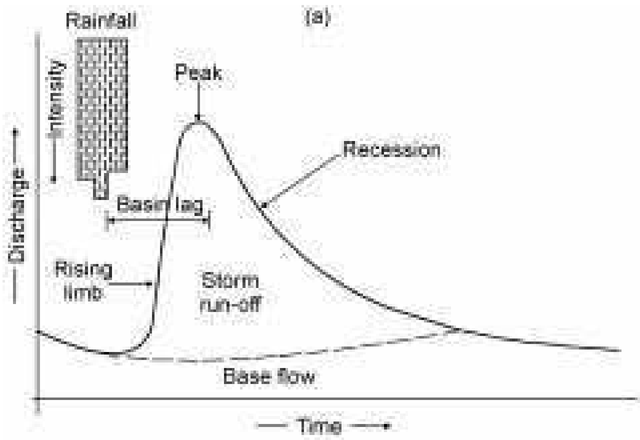
The dimensionless quantity V^2/gL is called the Froude number F, where V is the mean velocity of flow and L is a characteristic length. The Froude number is computed by depth rather than by hydraulic radius. In open channel flow where the boundaries are irregular, the mean depth or hydraulic depth represents L, i.e., the area of flow normal to the mean velocity divided by the width of the free surface. Based upon the Froude number F and neglecting other forces, criteria for flow classification are as follows:

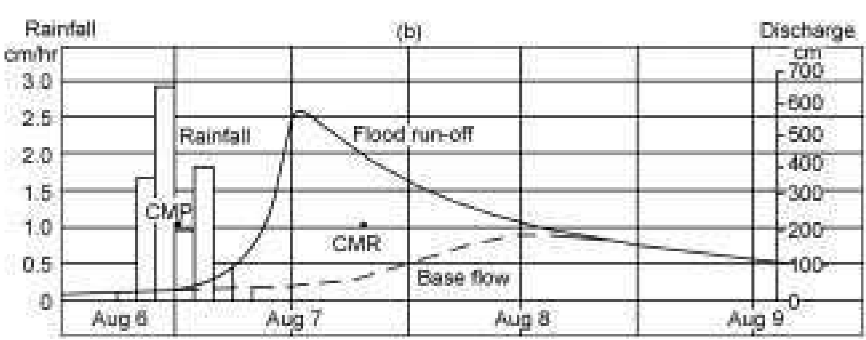
- When F is less than unity, flow is sub-critical (tranquil); wave velocity exceeds flow velocity so that a wave caused by an obstruction in the flow can travel upstream.
- When F is greater than unity, flow is super-critical (shooting) and waves cannot be propagated upstream.
- When F equals unity, the flow is said to be critical; it can be identified by the celerity of small currents that occur in shallow water in response to instantaneous changes in the local water depth.

4.3 Drainage systems

The boundary separating weathering from erosion marks the beginning of a network of channels that provide conduits along which sediments from various parts of the drainage area can come together in increasingly higher-order streams. The drainage pattern in headwaters evolves from sheet flow and development of fingertip channels in which surface run-off is restricted to periodic flow events by the small size of individual catchments. Seepage from the interfluve is negligible and stream flow is dependent upon surface run-off during periods of intense rainfall or thaw. Sub-surface components of the drainage system may only achieve significant proportions, if they are derived from a large area of ground water storage further down in the drainage basin. Such streams may become self-supporting in the valley system on a seasonal basis.

Components of a typical hydrograph are illustrated in Fig. 4.11 (a) and (b) as it relates to the results of four days of flow of Sugar Creek, Ohio. This figure illustrates the ongoing effects of sustained heavy rainfall on base level flow levels. The bar chart records precipitation in a drainage basin of about 800 km^2 located in an area of Ohio in a region of moist continental climate. Some water was lost by evaporation. Infiltration into the soil provided soil and ground water above the water table, which largely passed down Sugar Creek three days later. The unbroken line of the hydrograph shows the time lag between peak rainfall and peak discharge and the rate of decline of discharge after the peak has passed. The dashed line indicates the rise and fall of the amount of discharge contributed by base flow.



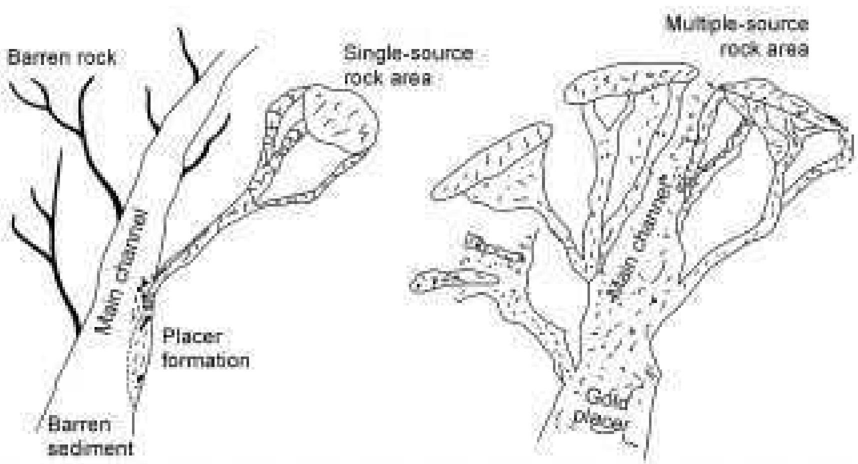


4.11 Components of a hydrograph (after Strahler and Strahler, 1992).

4.3.1 Channel networks

The basic pattern of drainage is established and channel networks develop most rapidly during early stages of denudation. Subsequent changes are due largely to headward erosion and expansion of tributaries. Effectively, the networks are open systems in which the inputs are water, energy and sediment. The outputs are losses of energy expended in friction, erosion and transport; sediment deposition in flood plains and deltas; and water discharged into lakes and oceans. Channels aggrade as a result of net sediment deposition when changes in channel geometry are inadequate to cope with increased sediment loadings. The same channel beds erode and downcutting occurs when the capacity of the stream to transport sediments is greater than needed to deal with the actual sediment input. Network efficiency is determined by how quickly the channels adjust to any changing flow conditions.

Individual channels are descriptive of the nature of the underlying rocks and may be interpreted to assess the texture of a landscape and to identify features pointing to differences in rock type and structure associated with possible metallogenic settings. The most productive channel systems (in terms of placer concentration) are developed where exposed primary gold-bearing veins are controlled structurally in relation to basin development. Figure 4.12 illustrates the effect of multi-source vs. single-source provenances on the possible extent of placer deposits formed in main trunk streams. Major alluvial goldfields such as those in California and Australia are all associated with drainage systems having extensive and closely spaced networks of tributaries in individually confined basin settings. Parameters relating to drainage density and stream frequency are interrelated features; the density of a drainage network is the total length of



4.12 Effects of single-source and multi-source gold provenances on patterns of paystreak development.

channel per unit area. Stream frequency is the number of channels making up the total length.

Drainage density

The manner of dissection of a drainage basin is quantified by the expression:

$$Dd = \Sigma L/A \tag{4.16}$$

where Dd is the drainage density, ΣL is the total length of streams in the drainage net and A is the area of the basin. The rate of dissection is strongly affected by climate and the intensity of individual rainstorms may be more significant than annual rates of precipitation. Run-off from violent rainfall is more erosive and can support more channels per unit area than run-off from persistent light rain that never reaches a peak. Altitude affects the rate of accumulation of water equivalent (snow and ice) and its seasonal rate of discharge. In dry climates run-off is ephemeral and drainage density is determined principally by the physical properties of the surface material.

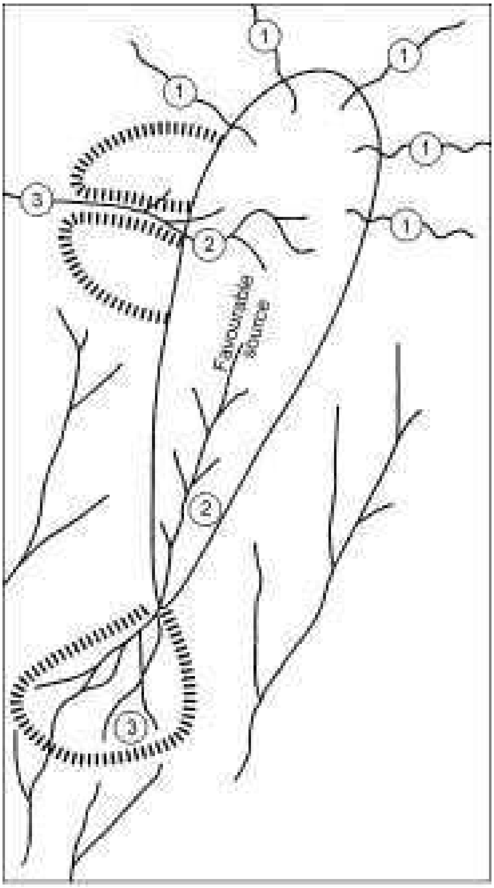
Stream frequency

Quantitative assessment of individual components of a network of streams forming a single pattern involves their classification within a graded system of ordering. At least two streams of a given order are needed to form a stream of the next highest order. Streams having no tributaries (e.g., fingertip channels) are classified as first-order streams; two or more first-order streams join together to form second-order streams; these in turn unite to form third-order streams and so on until the main trunk stream reaches the valley outlet.

Figure 4.13 is a conceptual diagram of fluvial distribution systems numbered according to this system of ordering. Features outlined by dashes are depositional features that could be alluvial fans; in this case the feature shown at the bottom of the diagram is a modern fan within the detritus of a favourable source; the other two fans are dissected ancient fans. Two systems of drainage are shown which separately contribute gold-bearing and barren sediment to a fourth-order main trunk stream gravel.

4.3.2 Channel styles

As represented by the plan view of a river, channel trace patterns are traditionally described as straight, braided, meandering or anastomosing. Miall (1985) suggests separating channels into 'fixed', 'mobile' and 'sheet-like' types (according to width/depth ratios) as modified from Etheridge (1985). In this concept, fixed channels are narrow with ratios less than 15:1; mobile channels are broad and shallow with ratios between 15:1 and 100:1 and sheet-like

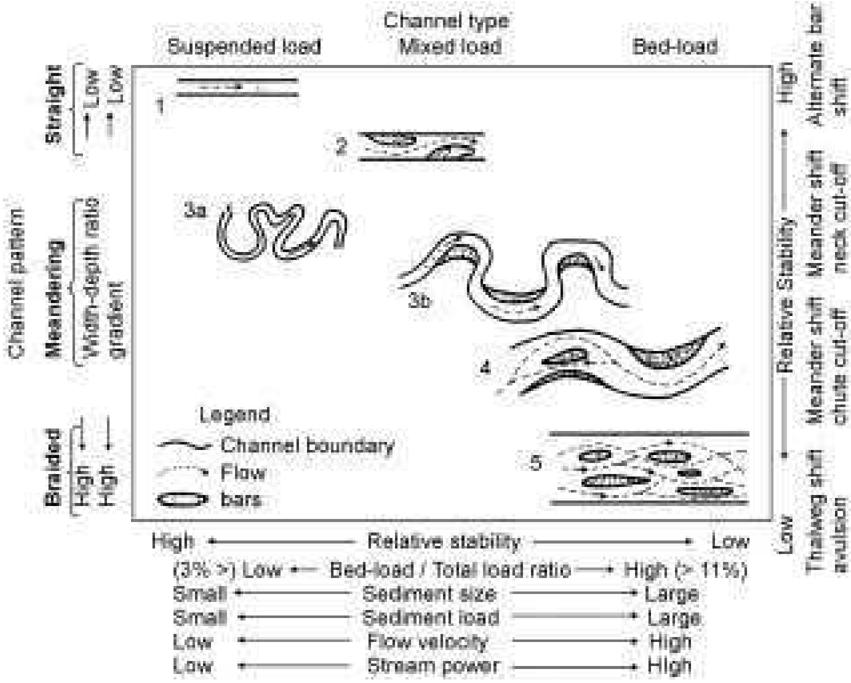


4.13 Numbered stream distribution (after Force, 1991).

channels, with ratios greater than 100:1 are essentially un-channelled. There are many graduations between them. As described in a detailed review of the historical development of fluvial sedimentology by Miall (1985), the extreme complexity of the relationships that exist between different channel styles results from the great variety of independent controls associated with their development. Channels are usually classified according to their patterns and corresponding sediment types (bed-load, mixed load and suspended load) in Fig. 4.14. Rust (1978) defines four basic types of channels according to high (greater than 1.5) or low (less than 1.5) sinuosity as shown in Table 4.4.

4.3.3 Channel shape

Definitions of geometric elements of fundamental importance to studies of stream channel flow can be illustrated from the dimensions shown in Fig. 4.15.

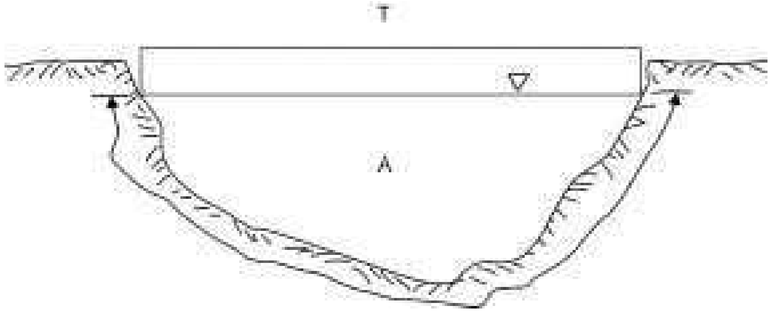


4.14 Channel classification based upon patterns and styles of sediment load (from Miall, 1985).

- **'Stage'**. The heights of free surface of flow above a given datum in a river channel, usually the lowest point of the bed.
- **'Top width'**. Width of the channel section at the free surface.
- **'Water area A'**. Cross-sectional area of flow normal to the direction of flow.
- **'Wetted perimeter P'**. Length normal to the direction of flow along the line of intersection of the channel-wetted surface.
- **'Hydraulic radius R'** is given by $R = A/P$.
- **'Hydraulic depth D'** is given by $D = A/T$.
- **'Section factor Z'** for critical flow computation is the product of the water depth and the square root of the hydraulic depth given by $Z = AD^{-2}$.

Table 4.4 Channel classification (after Rust, 1978)

Sinuosity	Single channel B.P. < 1	Multiple channel B.P. > 1
Low < 1.5	Straight	Braided
High > 1.5	Meandering	Anastomosing



4.15 Geometric elements of stream channels.

Although natural channels cannot be measured as precisely as can artificial channels, the application of hydraulic theory to artificial channels will usually provide approximate values that are reasonably consistent with actual observations and experience. Channel shape is influenced strongly by the interaction of gravitational forces pulling the water downslope, frictional resistance, and the volume of water available for discharge. It is no coincidence that the approximate shape developed by narrow stream channels is semi-circular, since this profile offers the smallest possible shear surface for a given cross-sectional area in both artificial and natural stream channels.

Longitudinal profile

The plan view of the longitudinal section of a river exhibits individual channels and channel sections of varying configuration in terms of straightness, anastomosing, braiding and meandering. The longitudinal profile is described in terms of the continued fall in elevation of the streambed and the horizontal distance between source and mouth. It is characterised by steep gradients in the upper reaches, which become flatter in the lower, wider valley sections where the river flows in its own alluvium. Gravitation forces put pressure on the confining channel walls as they pull the water downslope; frictional forces oppose its movement along the channel bed and walls. Sediment loading and the varying degrees of susceptibility of the bed and the banks to erosion determine both the cross-sectional shape and the slope of the stream. If the sediments are fine-grained and the banks more resistant to erosion than the bed, the cross-section will be narrow and deep. If the sediments are coarse and the banks more resistant than the bed, the stream will tend to widen and shoal.

4.3.4 Stream power

The effective power of a stream is a function of the amount of sediment it transports from one place to another and the rate at which it does this work.

Channel velocities vary throughout the water body due to the interrelationship of such factors as the presence of a free stream surface, sediment loadings, channel geometry, and friction along the channel bed and walls. The maximum velocity appears to occur below the free surface at a distance of about 0.05 to 0.50 of its depth, the closer the banks the deeper the maximum (Chow, 1959).

Velocity distribution

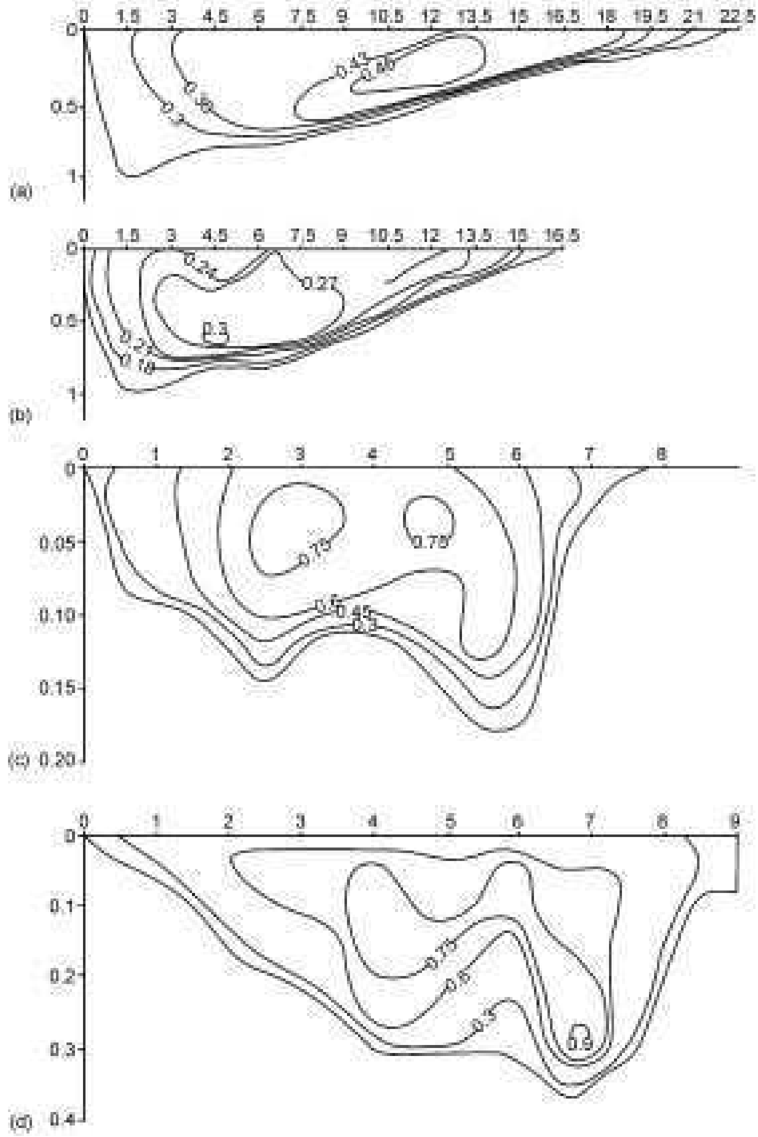
The actual velocity distribution depends in each case upon the configuration of the channel bed and the shape of the channel upstream of the velocity measurement site. Examples of the effects of changing sectional shape upon the distribution of velocity are given in Fig. 4.16. This figure shows a series of velocity curves constructed from measurements of velocity taken at various stages of flow and discharges in Behana Creek, at Aloomba, North Queensland, Australia.

An infinite variety of flow rates and sediment loadings impose constantly changing physical constraints on streams flowing through locally different geological structures. Progressive increases in discharge and decreases in the size of particles in transport are normal responses to downstream changes in width, depth and roughness. Shape differences depend largely upon the extent to which flow is retarded by frictional resistance along its boundaries. Roughness has a marked affect on energy losses due to friction. If the sediments are fine grained and the banks are more resistant to erosion than the bed, the cross-section will be narrow and deep. If the sediments are coarse, armouring of the bed will protect it against erosion and the stream section will tend towards lateral expansion and shoaling.

Grading concept

Theoretically, if flow conditions remain unchanged through time to a final state of equilibrium the longitudinal profile will achieve a smooth concave upward curve. In practice, grading is a complicated process and many factors intervene to prevent this happening and compromises are sought. In 1948, Machin defined a graded stream as one in which, over a period of years; channel form and slope are delicately adjusted to provide with available discharge, just the velocity required to transport the incoming load. He classified the Shoshone River, Wyoming with an average gradient of 10 m/km as graded, the Illinois River with a gradient of less than 1 m/km, ungraded. The distinction was made on the basis of differences in the nature of the materials in transport. The Shoshone River must handle material up to 25 mm to 250 mm diameter; bed-load in the Illinois River is mainly silt and clay.

Modern views of grading hold that any stream may be called graded if, by adjusting its geometry to achieve an average state of operation, it achieves a



4.16 Distribution of velocity at various discharges in Behana Creek (after Douglas, 1977).

reasonable balance between aggradation and degradation on a seasonal basis. Contrary to earlier views, such a balance may be achieved by any river at any stage of development and not simply in its later stages. Typical curves of equal velocity in various channel sections illustrate the pattern of evolution of a graded profile from original succession of falls rapids and lakes. The ability of a stream channel to reach such a state of equilibrium is governed by its capacity to adjust rapidly to any significant change in conditions of flow. Flow conditions vary with every change in level of discharge and in the nature of the sediments in transport. Flow regime thus adapts both to changes in channel form and velocity, and friction on the bed surface and channel walls; and variations in sediment grain size, sediment loading, river stage, stream type and disturbances caused by depositional units such as alluvial bars and lag gravels. Given an appropriate channel depth, most fluvial land-forms are developed in the low flow regime following active sediment erosion and transport at high river stage

Time rate of energy expenditure

The concept of the time rate of energy expenditure is fundamental to all studies of sediment transport and settling. Stream power is directly associated with flow conditions and for every change in flow conditions, there is a generally predictable reaction:

- Discharge variations result in scour during flooding and aggradation at low water.
- Narrowing of the channel decreases the transport capacity of the stream and promotes scouring.
- Changing the depth of a channel, without modification of either discharge or width, requires a change in shape.
- Channel shortening increases the slope and transport capacity of a channel, thus leading to local scour.
- Increasing channel roughness requires either the depth or slope to increase, or both.
- Local changes in the nature of the bed material lead to local changes in transport capacity.
- Selective sorting along one stream section eventually stabilises the channel in that locality; however the flow will then be loaded below its capacity when it enters into a zone of finer material, thereby causing scour.

The 'law of least time rate of energy expenditure' states that during its evolution towards an equilibrium condition, a natural stream chooses its course of flow in such a manner that the time rate of energy expenditure per unit mass of water along this course is a minimum. One consequence of the law is a requirement for the channel slope to decrease in the downslope direction so that the time rate of energy expenditure per unit weight of water is zero where the stream reaches its ultimate base level. This requirement explains why the longitudinal profile is usually concave (Stall and Yang, 1972). In applying the principle of stream power to predicting channel forms, it may be concluded that the tendency to minimise the expenditure of stream energy under the constraints of discharge and sediment load imposed by conditions in the drainage basin determines the river form.

4.4 Entrainment, transport and sorting

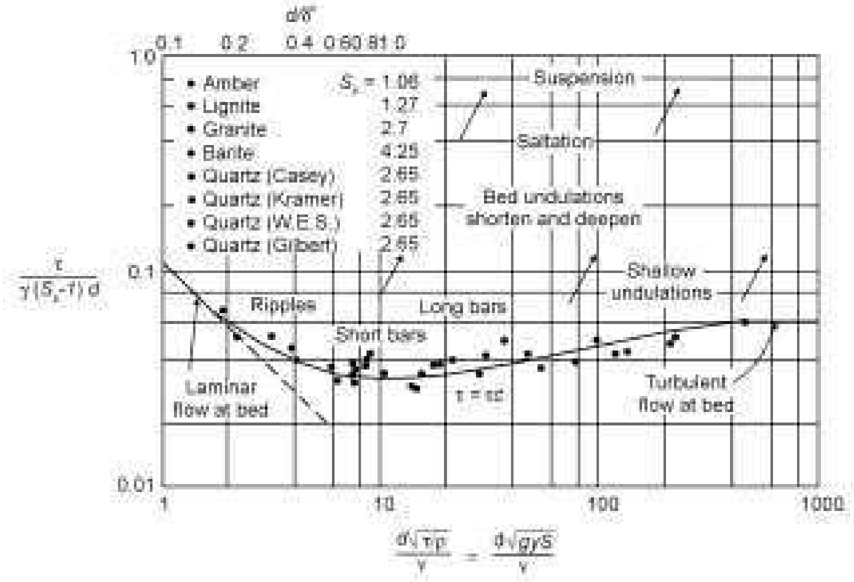
Weathering on slopes marks the beginning of mobilisation of sediments by the various agents of erosion (glaciers, rainwater and wind). Spoil gouged out of the valley floor and walls by glacial erosion (plucking and abrasion) may be transported for considerable distances before being dumped at the foot of the glacier. Rainwater disturbs surface particles by impact when it strikes the ground and continued precipitation leads to sheet flow and rivulets, which wash the lighter particles away. Water seeping downwards along seepage planes fills the voids between particles and provides lubrication for the mass to move as a whole. Wind sweeping over the ground entrains small waste particles, which are carried away from the surface by deflation processes involving traction, saltation and for dust-sized particles, suspension.

4.4.1 Mechanism of entrainment

Mathematical explanations of entrainment conditions at the threshold of movement are typically restricted to analytical expressions of the resultant force acting on a spherical particle free to move on a horizontal planar bed. A number of general models related to the transportation of sediment in gravity treatment plant are discussed in Chapter 8, particularly those modified from empirical relationships proposed by Bagnold (1966) for conditions of steady, uniform, simple shear flow of neutrally buoyant spherical particles. Application of this approach to bed-load movement as a whole however, necessitates describing various aspects of the phenomenon by certain functions of unknown form. In 1936, the approach by Shields was to make certain gross assumptions and then to confirm and supplement the analysis experimentally. His analysis of the entrainment function (Fig. 4.17) foreshadowed other works including Rigby and Hamblin (1972). Variation in character between sandy (cohesionless) and clayey (cohesive) material is due to properties such as structure and chemical composition.

Cohesive sediment

The most common 'cohesive' materials are beds composed of very fine particles. Particles smaller than 0.06 mm diameter (e.g., clays and fine silts) have very large surface area/mass ratios and are bonded together mechanically or by electrostatic attraction or by both. Chemical reactions occur mostly at the surface of contact between water and mineral particles, particularly in the presence of colloidal organic materials. In clayey soils the process of cation exchange of mineral elements between colloid molecules and the soil solution rearranges the molecules so that positively charged and negatively charged ions will ultimately be attracted to and held onto their surfaces. The very large surface area of clay particles provides a great water-holding capacity.



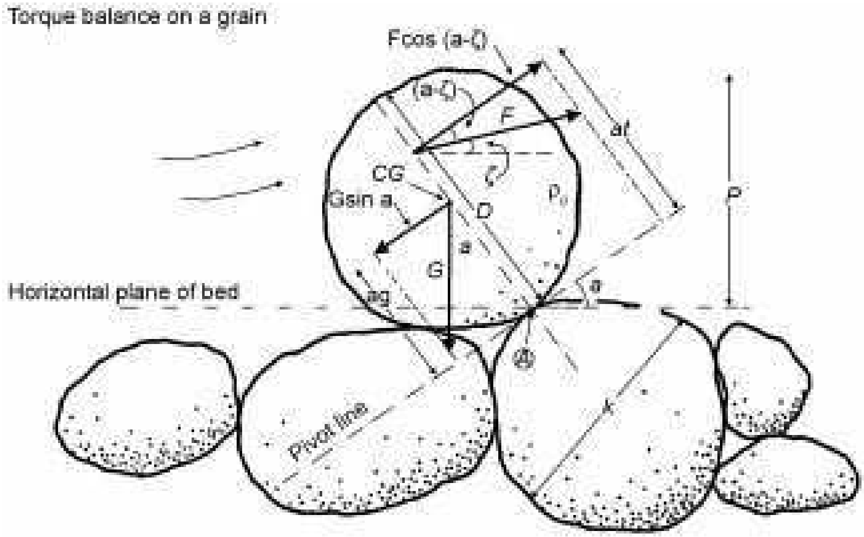
4.17 Shield's analysis of entrainment function.

Cohesiveness may also occur when small grains fill the voids between larger outstanding grains, thus producing a cohesive action, which bonds the grains together both mechanically and electrostatically. The ensuing reduction in surface roughness results in reduced turbulence at the bed and smooths the surfaces over which flow takes place. The lifting action of the flow is reduced and increased velocities are needed for the entrainment of sediment, which is then torn from the bed in clusters that vary in size according to the local cohesiveness of the bed materials and the velocity of the flow.

Non-cohesive sediment

Air fills the spaces between purely 'cohesionless' particles thus promoting the ready passage of water through the soil and relative motion between individual grains. For such deposits the theoretical critical stress can be arrived at analytically as a function of particle size and weight and the dimensions of cross-sectional areas exposed to the flow.

Very extensive sediment transport experimentation has since been devoted to the development of completely general models, which relate the bulk flow rate of sediment to prevailing hydrodynamic conditions for solids that are generally assumed to be of quartz density. It is now generally agreed that the theoretical prediction of critical conditions proceeds from a torque balance on a grain in the bed (e.g. Everts, 1993; Yalin, 1977; Slingerland, 1977). The sediment is



4.18 Definition diagram for calculating the initiation of grain motion on a horizontal bed (after Slingerland and Smith, 1986). (Reprinted, with permission, from the *Annual Review of Earth and Planetary Sciences*, Volume 14, © 1986 by Annual Reviews www.annualreviews.org).

assumed to be cohesionless, all particles in the bed are equant in size and are at rest. For a grain to rotate about a pivot point A, a moment balance shows that the ratio of the fluid forces F to the gravity forces G is:

$$F/G \geq a_G \sin \alpha / a_f \cos (\alpha - \zeta) \qquad 4.18$$

where a_G and a_f are the moment arms about A, ζ is the angle between the fluid force vector and the horizontal, and the point of application is assumed to be along the normal to the pivot line.

Figure 4.18 is a definition diagram for calculating the initiation of grain motion on a horizontal bed. A sub-spherical particle of diameter D and density ρ_p, protruding above the bed a distance P, must pivot about point A on a downward flow grain of diameter K. The resultant of the lift and drag forces (in magnitude and direction) is a fluid force vector F, which acts at a distance a_f away from a pivot line through A and at an angle ζ, from the horizontal to rotate the grain about a pivot point P. A grain weight vector G acts through the grain centre of gravity CG at a distance a_g away from the pivot line. At the moment of entrainment, the fluid torque must be greater than the resisting torque.

4.4.2 Transport

Fundamental principles governing the nature of stream channels and the shapes of streams in cross-section map view and longitudinal profile have been explored in the previous section. Predictions of sediment transport rates, and prediction of bed form and thus of flow resistance are two additional aspects of sediment transport of particular interest to placer engineers and geologists.

Bed-load

The bed-load is transported by both traction and saltation under conditions of shear flow (refer to Chapter 8). Particles in traction move by sliding and rolling in the direction of flow without losing contact with the bed. 'Saltation', from the Latin 'saltare' (to jump), describes the motion of any particle that is too heavy to remain in suspension in prevailing stream conditions but by virtue of its size and shape, may be re-entrained as soon as it reaches the bed. Movement takes place in a zone of viscous shearing within which particle concentration and hence the apparent density of the fluid determines the collision conditions between individual particles. Large particles protruding from the bed are subject to greater lift and drag forces than particles with smaller cross-sections.

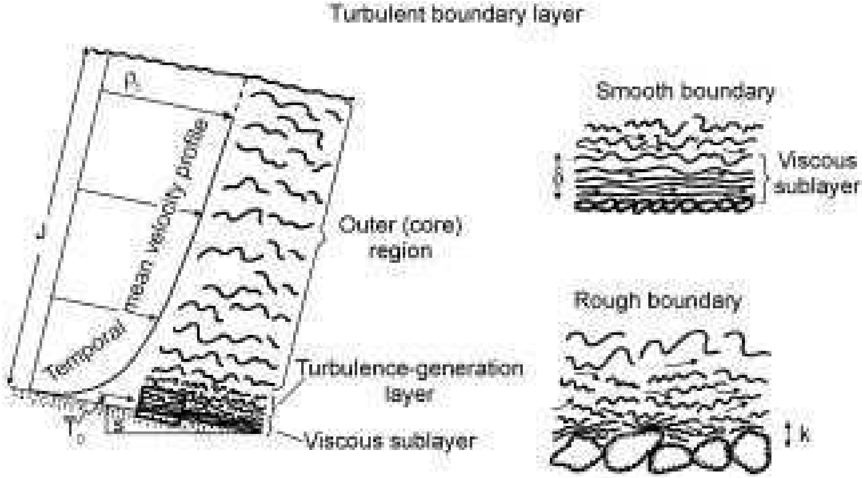
Differences in the gross character of flow over rough and smooth surfaces provide plausible explanations for differences in the nature of flow zones. They do not, however, explain in quantitative terms how the free settling of grains is modified by fluid turbulence in a polydispersed concentration of grains. Slingerland and Smith (1986) note that predictions cannot be made of actual transport rates or of the ultimate fates of size density fractions in a mixture undergoing unsteady, non-uniform flow and therefore under aggrading or degrading bed conditions at the current state of knowledge. Instead, they visualise the gross character of the flow by considering a vertical cross-section in the downstream direction with a planar channel bottom (Fig. 4.19). A straight channel segment is assumed for this exercise in which neither depth nor average velocity changes occur downstream thus providing a state of steady uniform flow. The frictional resistance of the boundary balances the force exerted by the gravitational flow on the channel bottom and sides because the flow rate is constant. The temporal mean tractive or boundary shear stress τ_o is given by:

$$\tau_o = \rho_f gRS \qquad 4.19$$

where $\tau_o = \rho$ is the fluid density, R the hydraulic radius and S the slope of the streambed and water surface, both of which are equal in uniform flow. For natural stream channels where width greatly exceed the depth, if follows that $R \sim J$ (the flow depth) and thus that:

$$\tau_o = \rho_f gJS \qquad 4.20$$

Grass (1983) suggests if a coefficient of variation equal to 0.4 is adapted for the instantaneous shear stress at the bed, the grains could experience shear stresses at least twice as great as the temporal mean given in eqns 4.16 and 4.17.



4.19 Internal structure of natural turbulent flow, where J is the flow depth, ρt the fluid density, τ_o the temporal mean velocity, S the bed slope surface, δ the thickness of the viscous sublayer, and K the height of roughness elements (after Slingerland and Smith, 1986). (Reprinted, with permission, from the *Annual Review of Earth and Planetary Sciences*, Volume 14, 1986 by Annual Reviews *www.annualreviews.org*).

Because it cannot be measured directly, the average shear velocity U_* need only be considered as a surrogate or convenience variable for tractive shear stress since this is always much smaller. A parameter used in certain problems in fluid mechanics is defined as:

$$U_* = \sqrt{\tau_o}/\rho_f = gJS \qquad 4.21$$

For smooth boundaries, the thickness of the viscous sublayer, δ, depends upon shear velocity and viscosity:

$$\delta = cv/U_* \qquad 4.22$$

where c is a constant. For a ratio of bottom roughness size to sublayer thickness, K/δ, substituting from eqn 4.19 gives:

$$k/\delta = U_*\,k/cv = \text{constant} = \mathrm{R}_* \qquad 4.23$$

The dimensionless quantity is the boundary Reynolds number R_*. The value of $\mathrm{R}_* = 5$ is commonly accepted today as a reflection of the degree to which the distribution of fluid forces acting on grains can be expected to be a reasonable function of that value. A fully rough boundary has a value of $\mathrm{R}_* \approx 70$.

Suspended load

The suspended load may be subdivided into:

- **'Wash load'**, i.e., those particles held in suspension by fluid momentum transfer alone, i.e., by random eddy currents of turbulence having velocity current normal to the bed greater than the terminal velocity of the particles relative to the surrounding fluid.
- **'True suspensions'** of particles (e.g., clays) that remain in suspension from surface to bedrock and only settle in stilled water over long periods of time; such particles are distinguished from particles that are held in suspension in direct proportion to the energy of the stream at each point in the flow.

The wash load makes up the greater part of the total stream load and in active stream channels its constituent particles are typically absent from the coarse bed-load sediments. The presence of clay-sized particles in pay gravels at the base of a fluvial sequence is thought to be due to clay-bearing ground waters seeping through the gravels and eventually filling the spaces between individual clasts. Lag gravels, though originally laid down in turbulent conditions in the virtual absence of very fine sediments, may thus present major desliming or lithification problems when mined.

4.4.3 Sorting and deposition

Fundamentals of the transport mechanism relate to sedimentary processes involving the development of paystreaks in natural stream channels. Sorting and deposition are time related and placer formation occurs at various stages along the sedimentary train where local flow conditions provide suitable conditions and time for gold and other heavy minerals to settle out preferentially to the lighter particles. At normal flow rates only the upper sediments are disturbed and the pebbles act as riffles to trap and hold back particles of gold. As flood stage approaches, the stream velocity increases and the lower parts of the bed-load are disturbed thus allowing the coarser gold grains to settle towards the bed. Where flow velocities exceed the critical velocities for incipient cavitation absolute pressures approach or equal the vapour pressure of the fluid. Cavitation is a special condition that may occur in the upper reaches of streams at flood stage. Varying levels of flooding result in small-scale features that are enriched superficially at several positions on bar surfaces by the deposition of skim, or flood gold. Skim bars are transient features and tend to be removed by scouring during each fresh flooding.

Sorting

Selective removal of some particles occurs at and above a certain critical velocity below which no movement can take place while gold, being of higher density and smaller in size than the particles with which it is associated, tends to collect within spaces between particles exposed to the flow. Entrapment of gold grains takes place by these means when movement occurs by rolling and sliding at low velocities. Grain size is a dominant feature and large sediment particles are subject to preferential removal because of their projection into the flow. Voids separating smaller particles will provide a large measure of protection against movement of the gold until velocities rise sufficiently to disturb all of the particles in the upper layers.

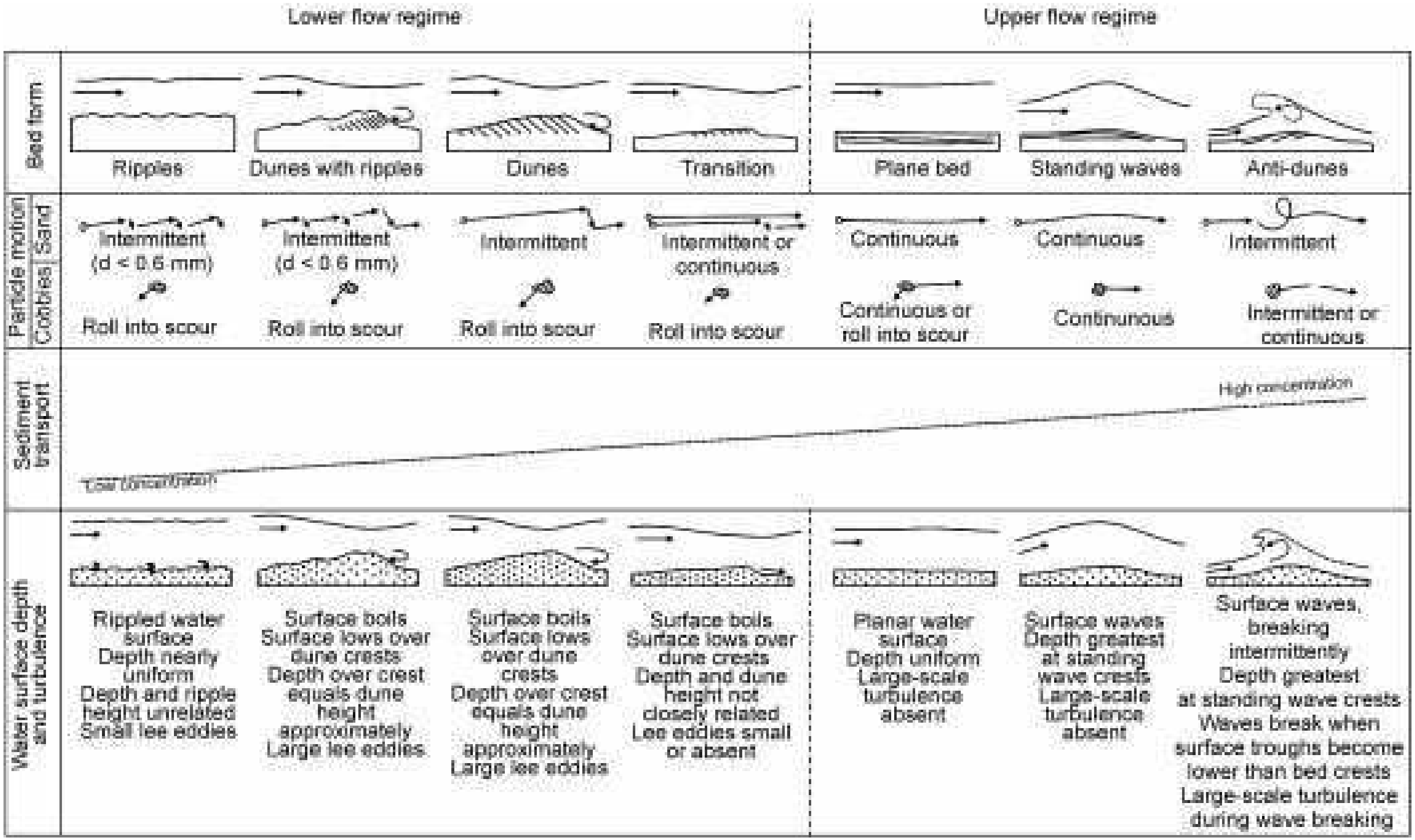
A typical sequence of bed forms develops with change in discharge. Classified broadly as either low or high flow regimes these bed forms are related to processes of erosion that occur in accordance with grain size and shape, sediment loading, river stage and stream type when local critical entrainment velocities are reached. Most interpretations are based on Fig. 4.20. Gold entrapment occurs at the surface of the bed only in the low flow regime, which includes all stages up to the formation of dunes. The upper flow regime is associated with relatively low flow resistances and high gradient braiding, and is represented typically in streams emerging from the confines of channel flow. The division between low and high regimes in the above figure occurs during transition from braiding to meandering.

The development of a rippled bed form and cross-bedded structures leads to 'dispersive' sorting and to rippled dunes as the main forms of transport, with ripples tending to climb over the dunes. 'Suspension' sorting is established when the stream reaches its peak at velocities high enough to create anti-dunes, i.e., natural sand waves which travel upstream against the current. This occurs in the upper reaches of streams when periodic flooding during periods of intense precipitation and run-off disturb the bed load as a whole. Because of inertial effects, the larger and denser particles lag behind the other sediments and settle down into the bed to form concentrations in the lower gravel layers. The lighter, flakier particles are hindered in their settling by rising inter-grain currents and are carried further downstream.

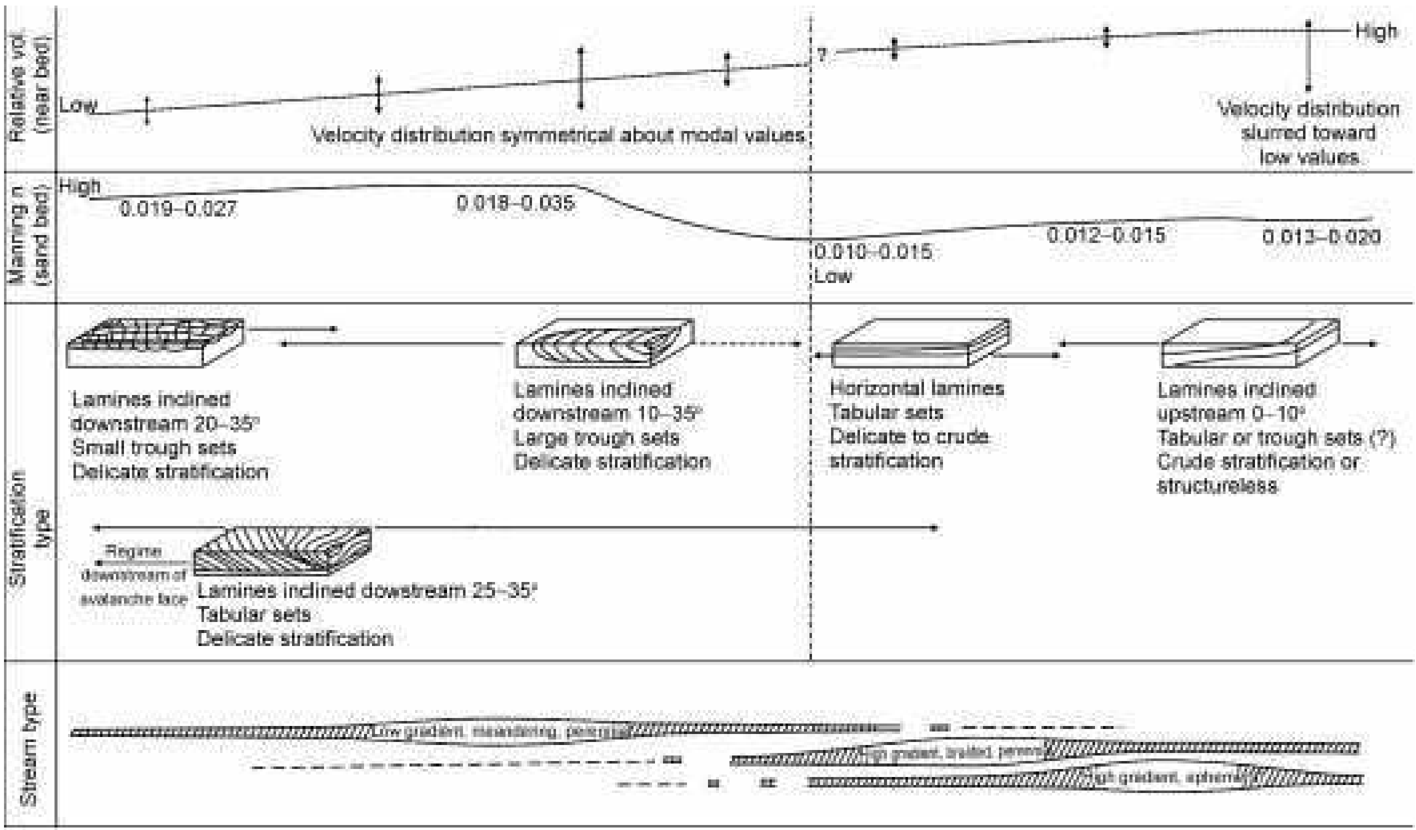
In natural stream channels, the sorting mechanism relies upon differences in the hydraulic behaviour of suspensions of a range of different particle sizes, shapes and densities under conditions of unsteady and non-uniform flows that vary both instantaneously and with time. Coarse gold is initially dispersed into cracks and fissures in bedrock structures. Smaller and more transportable particles are carried away in suspension following each base level adjustment. Accumulations of winnowed gold accrue in depositional sites in bars and in channels within the stratigraphic section downstream. Individual relationships are categorised in terms of hydraulic equivalence, entrainment equivalence, transport equivalence, and granular dispersion in shear flow.

Deposition

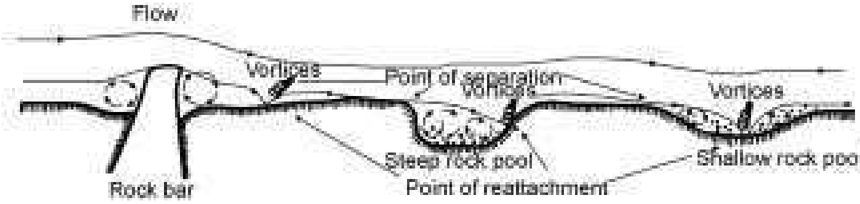
The preferential deposition of gold grains and other heavy minerals is usually represented in illustrations of separated flow as a two-dimensional flow field. By



Lower flow regime
Upper flow regime
Bed form
Ripples
Dunes with ripples
Dunes
Transition
Plane bed
Standing waves
Anti-dunes
Particle motion
Sand
Cobbles
Intermittent (d < 0.6 mm)
Intermittent (d < 0.6 mm)
Intermittent
Intermittent or continuous
Continuous
Continuous
Intermittent
Roll into scour
Roll into scour
Roll into scour
Roll into scour
Continuous or roll into scour
Continuous
Intermittent or continuous
Sediment transport
High concentration
Low concentration
Water surface depth and turbulence
Rippled water surface Depth nearly uniform Depth and ripple height unrelated Small lee eddies
Surface boils Surface lows over dune crests Depth over crest equals dune height approximately Large lee eddies
Surface boils Surface lows over dune crests Depth over crest equals dune height approximately Large lee eddies
Surface boils Surface lows over dune crests Depth and dune height not closely related Lee eddies small or absent
Planar water surface Depth uniform Large-scale turbulence absent
Surface waves Depth greatest at standing wave crests Large-scale turbulence absent
Surface waves, breaking intermittently Depth greatest at standing wave crests Waves break when surface troughs become lower than bed crests Large-scale turbulence during wave breaking



4.20 Sediment movement by fluid flow (Rigby and Hamblin, 1972).



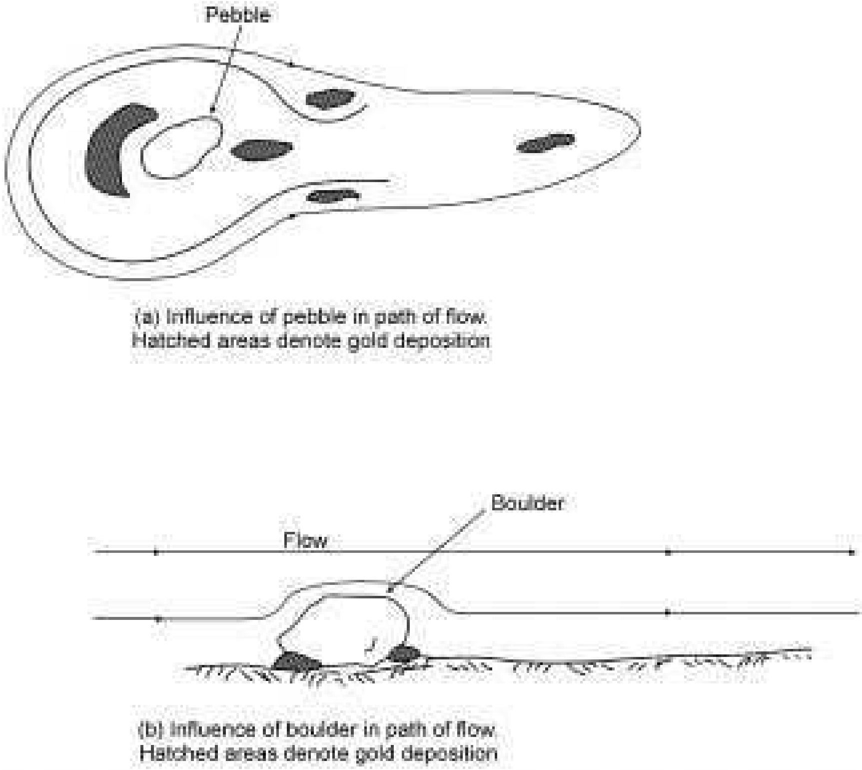
4.21 Flow separation across rock pools and bars on a streambed; formation of vortices and scour.

these means, the concept of streamlines can be used to depict the division between recirculating flow and external flow where a boundary layer separates from a solid surface and enters the flow as a free shear layer. Re-attachment of separation streamlines may occur in the body of the flow or at some point on the solid surface downstream.

Practical examples of the preferential settling of gold particles across pools and bars on a streambed are shown in Fig. 4.21. In this figure, a rock bar cutting transversely across a stream illustrates the effects on separation of local pressure gradients higher than the general gradient for the stream. The flow is not disturbed upstream of the bar but increased pressure during an active river stage leads to re-circulation of the flow and a slow moving vortex is developed as a closed loop in the zone of separation. In the lee of the bar, the abrupt change in flow velocity provides enhanced conditions for backward flow over a wider range of flow rates. An eddy is formed where flow passes over a sharp edge and the sudden fall in velocity causes the flow to re-circulate in a roller or closed loop. The phenomenon of separation for low velocity flow passing across a transverse slot or hollow in the surface of the bar is illustrated in the same figure. Deposition of gold and other heavy minerals occur predominantly in the most active zones of flow separation and paystreaks may occur on both sides of the bar.

Depositional sites described in Fig. 4.22 (a) and (b) show the influence on the preferential settling of gold of different sized particles cropping out on a streambed. In (a) the disturbance is small around a small pebble and in practice the effects are similar to the settling of heavy minerals under aeolean flow conditions where the sites of deposition are in flow shadows surrounding the obstruction. In (b) the swirling of waters around boulders and other large objects sets up eddies and velocity fluctuations affecting settling and entrapment; the larger gold particles are caught under the edges of the boulders, the finer particles are swept away to be deposited in less turbulent stream conditions downstream.

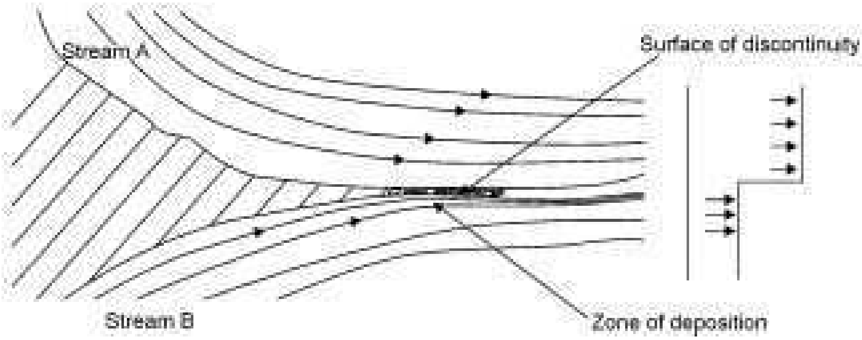
Flow separation occurs at the confluence of two streams. Since no equality of pressure exists along the surface of discontinuity separating the two streams, the velocity of flow must differ on the two sides. The direction of flow is also different on the two sides and between them these features result in an abrupt longitudinal discontinuity in the velocity and one in the transverse section (Fig.



4.22 Influence of different sized streambed obstructions on the free settling of gold.

4.23). The surface of discontinuity breaks down into a large number of irregular vortices, and fluid in regions of excess pressure will tend to move towards adjacent regions of reduced pressure. When the bed-load of one or both of the streams is auriferous, paystreaks will be developed along the line of discontinuity. Owing to fluctuations in the flow and a fine upward grading of the sediment load, eddies will be distributed irregularly. The final stage of deposition is the development of an irregular, but limited medley of paystreaks along the line of discontinuity.

Sand and gravel associated with braiding and meandering comprise the coarsest materials of the bed-load and accumulate the most gold. Braided stream sediments are the first particles to settle and are correspondingly coarser than meandering stream sediments, particularly in flood plains where fine sands and muds periodically cover the flooded areas and fill any abandoned channels and low lying ground. In each case, the resulting mixture of gravel, sand and fines is characteristic of the local balance between viscous and gravitational forces, local differences in stream turbulence and depth and the settling properties of the



4.23 Surface of discontinuity and zone of deposition at confluence of two streams.

solids. Textural variations result from changing velocities and stream competency over the surfaces of the bars. High velocities and a coarse sediment load in the deepest part of the channel give rise to bedded gravel horizons and cross beds, which may, as meandering continues, create an underlying gold-bearing lag zone. Large and medium sized cross beds in the mid-zone of a bar may also carry anomalous gold values but are unlikely to be as rich in gold as basal lags although there are exceptions. Individual layers are frequently separated by clay 'false bottoms' that represent overbank deposits, or deposits from ephemeral bodies of standing water in lakes formed by a single base level drop.

Lag gravels

The highest gold values in any stream section are associated with lag gravels, so called because they are the slowest moving constituents of the bed-load as it moves downstream. Lag gravels may occur in any stream section that represents a local region in which a temporal lag occurs between a change in flow and a corresponding change in bed form. They are most extensive and continuous in well-developed drainage channels in which the gravels are well graded and where a general movement of much of the bed-load takes place at flood stage without excessive scour. The nature of the facies produced by bar accretion is particularly relevant to their gold content and its distribution. High-grade zones (paystreaks) develop in lag gravels by the entrapment of gold in bedrock structures along the valley floors.

4.5 Fluvial gold deposition

A normal approach to the study and exploitation of fluvial gold deposition lies in the formulation of models establishing the relationships between accumulations of heavy minerals, the dynamic conditions of transport and the direction of sediment transport. Texture, and sedimentary structures, geometry and facies

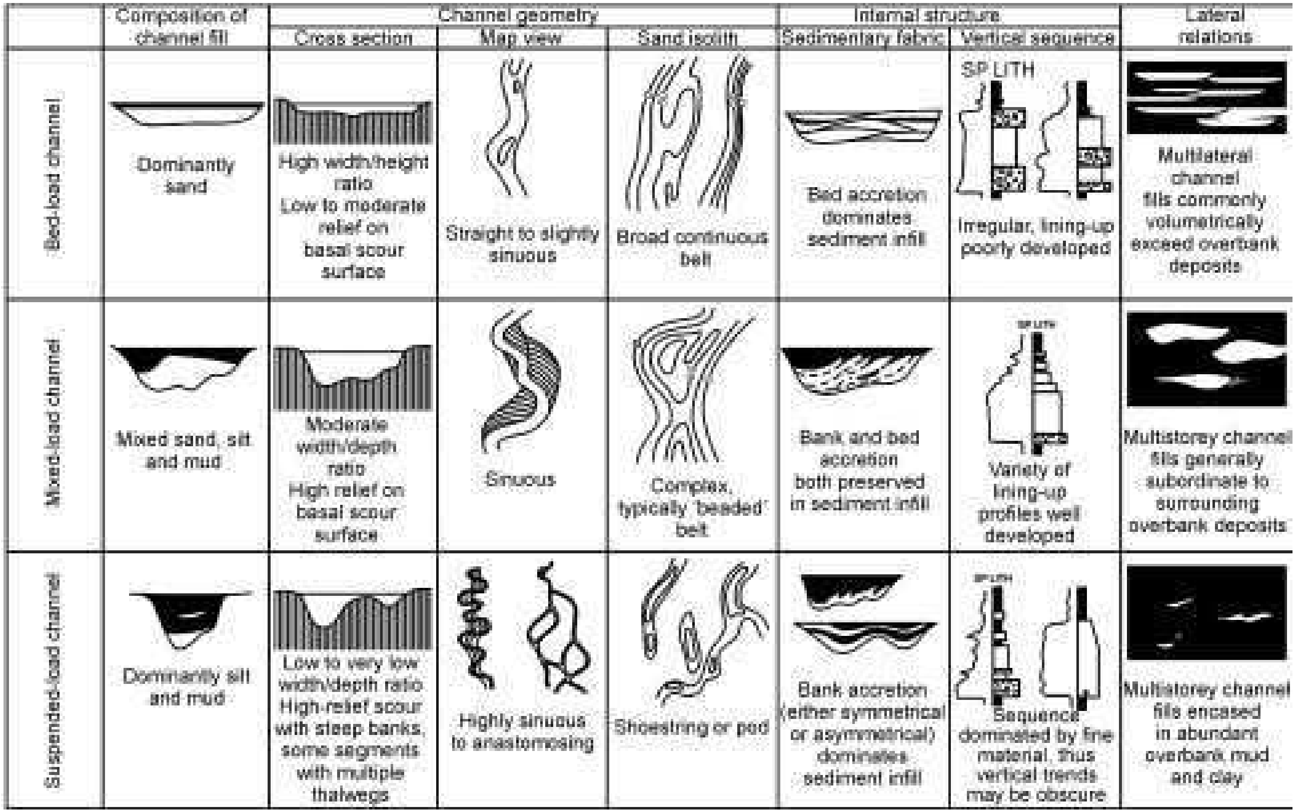
(Dyson, 1990) reflect the environment of deposition. Attempts to reconstruct the morphology and flow characteristics of ancient systems rely heavily upon the application of empirical relationships derived from modern streams. Galloway (1985) illustrates the geomorphic and sedimentary characteristics of bed-load, mixed-load and suspended-load channel segments and their deposits in Fig. 4.24. Table 4.5 is a classification of alluvial channels by the same author.

Evolution of topography as produced by tectonic uplift and volcanism at convergent plate boundaries is controlled by valley entrenchment and extension in the headwaters, and by the rejuvenation of streams and migration of weathering fronts through valley systems to the headwaters. The most active stages of orogeny produce steep irregular slopes in the headwaters of streams and the rapid downslope movement of large quantities of partly weathered and unsorted rock material. No significant development of gold placers takes place on slopes at this time. Only during protracted periods of sediment transport and sorting, are the effects of tectonic adjustments reflected in changes in the base level of erosion of streams and hence in the consistency of rate of erosion of valleys. Conditions favourable for the progressive liberation of gold from source rocks and its concentration in sites of preferred accumulation at the base of sedimentary sequences are portrayed schematically in Fig. 4.25. The most productive streams (in terms of placer concentration) are developed where the gold-bearing veins are distributed over the whole of the catchment area.

4.5.1 Paystreak development in unglaciated terrain

The concept of a model for the formation of paystreaks in unglaciated terrain provides a genetic scheme for the development of gold placer settings in sequence to upper valleys and middle and lower stream settings. Sediment transport is a function of topography and is thereby time related. In terrains of high relief transportation down very steep valleys involves intervals of deposition in settings of optimum concentration potential. The sites of paystreaks for unglaciated terrain are basinal intervals controlled by recessive lithologies in the valley reaches. Downstream flattening provides progressively longer periods of time for sedimentation and sorting at each basinal interval and for the concentration of progressively more finely sized gold. This system envisages:

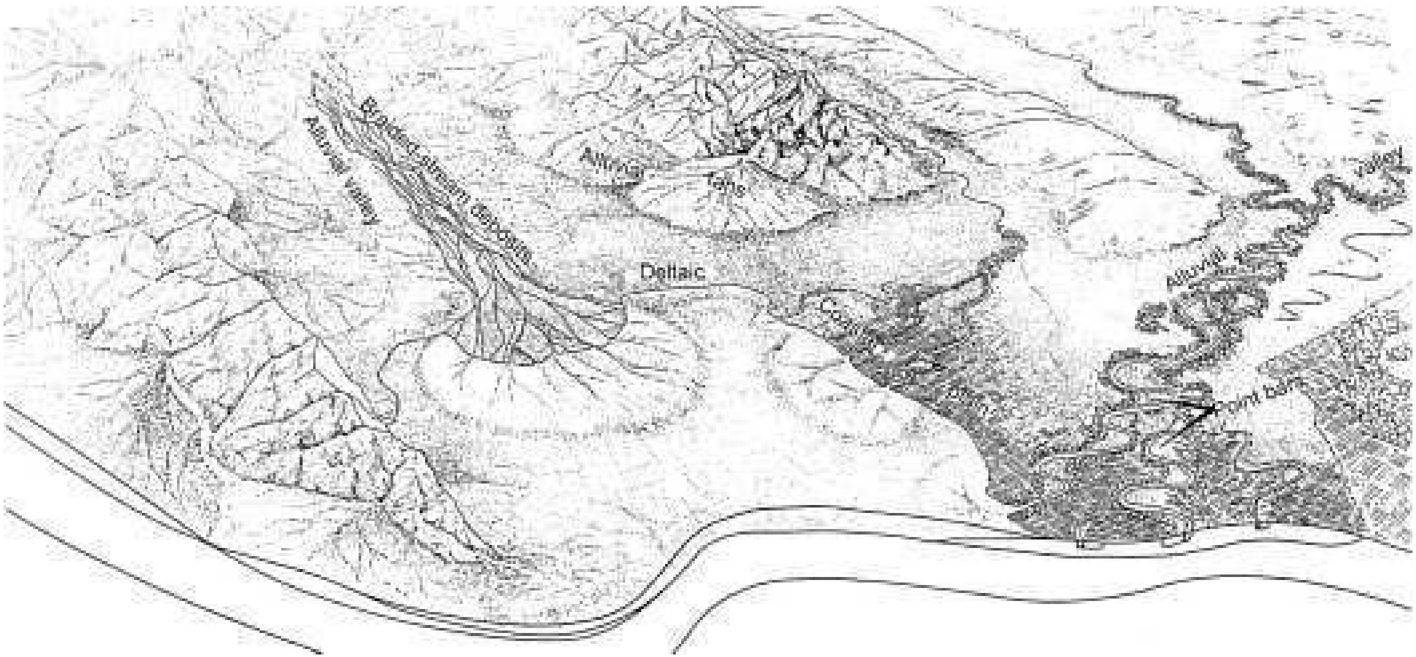
- a single stage of downcutting with minor crustal compensation, but without abrupt changes in either base level or climate
- development of sites of gold concentration in basal gravel and bedrock structures during intervals of stillstand
- a virtual state of equilibrium between the inflow and outflow of sediment across each section of the deposit
- gold concentrations that typically become centrally located along the valley floor as the valley widens.



4.24 Geomorphic and sedimentary characteristics of bed-load, mixed-load and suspended-load channel segments and their deposits (Galloway, 1989).

Table 4.5 Classification of alluvial channels (after Galloway, 1989)

Mode of sediment transport and type of channel	% silt and clay deposited in channel perimeter (m)	Bed-load (percentage of of total load)	Channel stability Stable (graded stream)	 Depositing (excess load)	 Eroding (deficiency of load)
Suspended-load	>20	<3	Stable suspended-load channel. Width-depth ratio > 10; sinuosity usually > 2.0; gradient relatively gentle	Depositing suspended-load channel. Major deposition on banks causes narrowing of channel; initial streambed deposition minor	Eroding suspended-load channel. Streambed erosion predominant; initial channel widening minor
Mixed-load	5–20	3–11	Stable mixed-load channel. Width-depth ratio > 10, < 40; > 1.3; gradient moderate	Depositing mixed-load channel. Initial major deposition on banks followed by streambed deposition	Eroding mixed-load channel. Initial streambed erosion followed by channel widening
Bed-load	<5	>11	Stable bed-load channel. Width-depth ratio > 40; sinuosity usually < 1.3; gradient relatively steep	Depositing bed-load channel. Streambed deposition and island formation	Eroding bed-load channel. Little streambed erosion; channel widening predominant



4.25 Potential sites of fluvial placer formation.

Headwater tracts

Valley forms as produced by slope and channel processes operating on the various substrates are dependent upon the local lithology and structures. Headwater tracts are regions of erosion in which detritus is swept downstream at relatively high velocity during periods of intense precipitation and run-off. Drainage patterns reflect the manner of dissection of the surface rocks and the ability of the environment to cope efficiently with intermittent run off and surges of rock waste into the channel system. Rivers cut deep narrow gorges through bedrock structures and ground surfaces that are typically irregular in profile and variable in their resistance to wear. Marked changes in gradient, which may be as steep as 1:5 and even more are represented by the development of waterfalls, ponds and rapids where streams flow over rock formations of varying resistance to wear. Waterfalls, which create sharp breaks in the longitudinal profile, are characterised by the development of deep plunge pools below the falls. The mechanism of undercutting and erosion of the riverbed beneath the plunging stream of turbulent water undermines the falls, gradually transforming them into rapids as nick-points advance up the valley. The flowing stream abrades the rapids and degradation gives way to aggradation as the profile gradually flattens.

The stream load comprises clusters of partly weathered gold-bearing detritus and slope materials that are mobilised and fall into the channel during periods of intense precipitation and thaw. Particles in transport tend to settle selectively out of the flow according to size. Bed-load movement is climatically controlled and any significant movement of large masses of rock debris in the upper reaches of stream channels occurs only at high flood stage. Boulders and other large fragments tend to vibrate in place at high flow velocities without forward movement, and are only gradually reduced in size by natural attrition to more transportable proportions. The coarsest gold grains tend to become trapped behind or under rock bars and boulders or lodge in cracks and potholes in the channel floor. Grooves and any ridging in the bedrock collect gold just as efficiently when alligned in the direction of flow as when the flow is at right-angles, e.g. bedding of the country rock. Very finely divided gold grains are carried out of the system in turbulent suspension along with the clay and silts. The remaining gold is caught up for a time in patches of sand and gravel that gather into unstable and loosely sorted fractions on flatter sections of the bed or behind rock bars and other transient features which disturb the pattern of flow. Materials deposited during falling-stream stage are re-entrained at rising-flood stage.

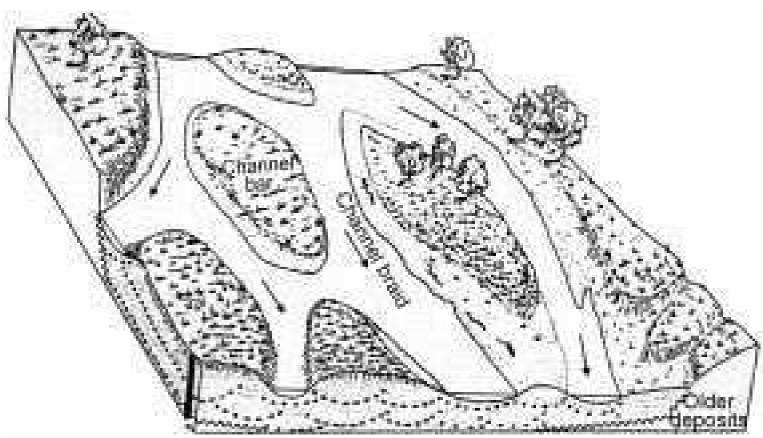
Prior to tectonic and climatic change deposits in the headwaters of streams are typically thin and discontinuous because of the ephemeral nature of the flow and the steep and irregular gradients over which flow takes place. Such deposits, though often quite rich and transient in nature, may not be of immediate commercial importance except as possibly bonanza-type discoveries for small prospecting groups. They do, however, represent the first stages of placer gold concentration and can provide valuable geochemical information of their source, and of the possible size and value of larger concentrations further downstream.

Middle and lower stream settings

In contrast to conditions that promote net erosion in the upper reaches of streams, the middle and lower tracts are regions of net deposition in which the stream widens and flattens and high-energy flow and degradation gives way to low-energy flow and aggregation. During periods of high discharge additional energy is directed against the banks and the valley continues to widen as more material is eroded and added to the bed-load. The abundance of sediment and the high and sporadic nature of discharge cause the channel to be rapidly choked with sediment. This results in lowering of the relative water level as sediment builds up across the valley floor. The channel gradient is thereby decreased and multiple connected anastomosing channels (braids) create a net-like formation, with small islands (braid bars) located centrally within the net (Fig. 4.26).

Braiding

Interrelated parameters of a stream are its width, depth, velocity, slope and transporting power. Individual changes occur and alluvial fans develop as a function of discharge (total volume of water flowing through the channel in a given space of time) and the quantity and nature (e.g., size) of sediment in transport. Alluvial fans are typical features of stream channel sediments where braided rivers debauch out violently from narrowly incised channels into a wide valley or plain. 'Sieve deposits' may appear as coarse gravel lobes on the fans where the source supply contains relatively little sand, silt or mud. In arid



4.26 Main features of braided stream deposits.

climates, detritus in fault-bounded areas is moved by infrequent flooding and accumulates in fan-like structures, which coalesce at the base of mountain ranges to form extensive sloping plains or bajadas (Spearing, 1974).

Intermittent torrential rainfall in humid climates gives rise to sequential mud-debris flows in intermontain basins and coastal areas. Mud/debris fans are developed comprising mud layers inter-bedded with layers of sand and gravel with occasional very large boulders. Traction current activity during periods of relative tectonic calm establishes a partial upgrading of gold in streams that traverse the fan surface. Depositional processes are typical of braided stream sediment facies in which steadily decreasing levels of stream competence are reflected in a transition from mainly coarse to mainly fine sediment sizing away from the apex of the fan. As the fans grow, the larger channels divide into networks of distributaries, and sites of deposition change from one side of the fan to the other.

Braid bars and islands are built up by lateral and vertical accretion and are predominantly lenticular in shape although otherwise of varying dimensions. With continual working and reworking in changing flow conditions, longitudinal bars migrate in the direction of least pressure by eroding sediment from the upstream ends of bar the and depositing it in the lee of the bars downstream. New bars are created as others disappear but ultimately, all of the material is moved downstream. A steady reduction in particle size makes for better sorting and gradually decreasing stream velocity results in the grading of each sedimentary unit from coarse to fine upward. Although occasionally bars become stable for a time when silt deposited during flooding is covered by vegetation the structures are essentially transient.

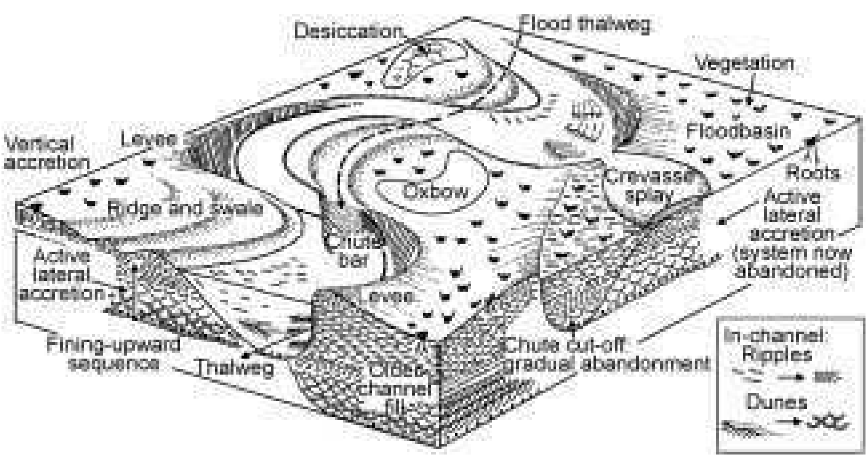
Meandering

A not very well understood feature of fluvial channels is alternation between braiding and meandering and conditions that influence the accretion of braid and point bars and their style of mineralisation. In general, braided rivers differ from meandering rivers by having steeper gradients and a coarser sediment load. However, frequent alternations from braiding to meandering occur in streams traversing alluvial fans where gradients change rapidly; in some cases a steepening gradient may lead to braiding in an otherwise meandering stream section. Gradation from braiding to meandering may also occur locally in a valley that flattens and widens sufficiently for braided streams to meander freely. Such streams are typically low gradient with moderate sediment loading of mixed size range and moderate fluctuations in discharge.

Historically, it was thought that meanders were initiated by such factors as rotation of the Earth, obstructions in streambeds and the Coriolis effect. Although these factors may contribute to the development of meandering channels the control of meandering appears to be related mathematically to the spacing of pools and riffles, meander wavelengths and average bank-full/bank-width relationships. According to Leopold *et al.* (1964) each channel meander wavelength contains two, pool-riffle sequences each being separated by six to seven channel widths. Pools are located on the bends, riffles at the inflection points and point bars form in successive stages on the inside bends of the meanders in conformity with the shape of the meander, stream depth and erodibility of the banks.

Low gradient streams typically assume a meandering pattern in areas of moderate rainfall and moderate discharge. Current velocities are greatest along the thalweg, which swings from one side of the channel to the other, even in straight-sided channels. Material eroded from the banks by impingement of the swifter current against the channel side is deposited in slacker water on the opposite side. The end result is the development of point bars on the inside of meanders, successive stages of growth comprising basal lag gravel overlaid by sandy upward fining point bar deposits which, themselves are overlaid by silty and muddy overbank deposits. The morphology of a meander system is illustrated in Fig. 4.27.

Deeper streams and thicker deposits have larger meander traverses, and incipient sidebars are common along the flanks of relatively straight sections. Alluvial landforms in meandering channels extend downstream from the point of maximum curvature of the meander belt. Meandering channel paystreaks, which follow the courses of lag gravels in successive migrating meander, may be reworked during intermittent pulses of uplift and erosion produced by minor crustal adjustments. Meander scrolls migrate across the placer following each adjustment and tend to disperse the concentrations of coarser gold according to size into basal gravel and bedrock structures. Ultimately, the limits of economic concentration are reached downstream when the gold grains are so reduced in



4.27 Morphological elements of a meandering river system.

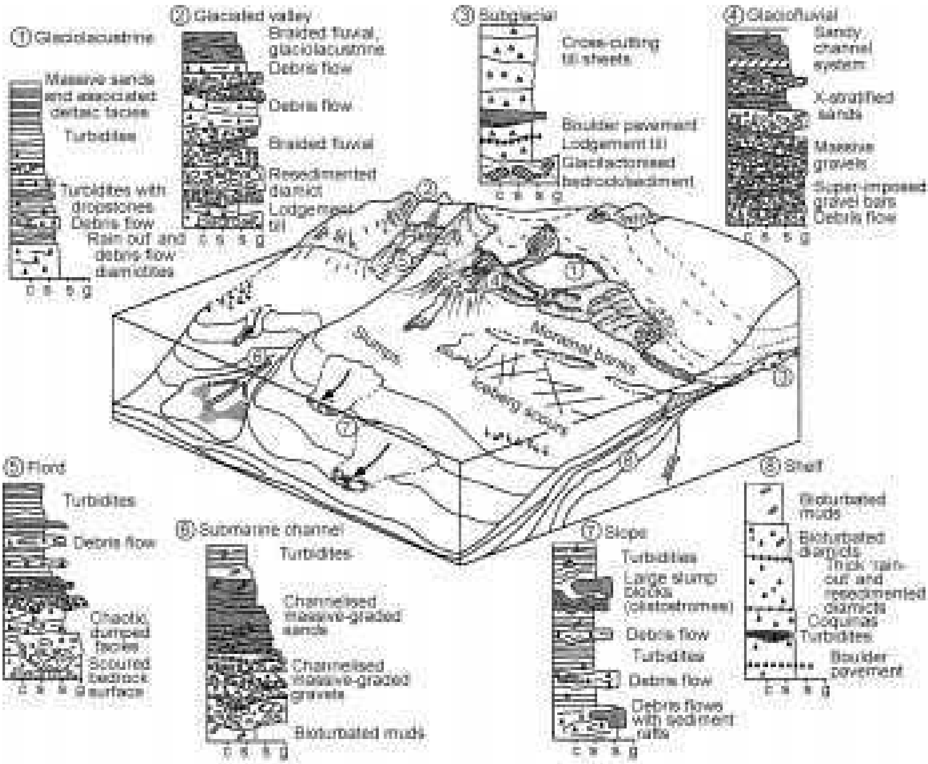
size that they can no longer settle faster than the sediment with which they are associated. Deposition tends to be unpredictable because of local changes in flow resistance and stream energy.

Because of the variable nature of stream sediments and processes, it is clear that no single facies model can be used to describe sedimentation in a fluvial setting. Other local conditions that may change the character of streams include bed roughness, variations in discharge, obstructions caused by falling trees along river banks and differences in channel geometry due to changing bank or bedrock lithologies. In this respect, Dyson (1990) has extensively reviewed literature describing fluvial facies models and sites of gold placer deposition and warns against adopting too rigid an assignment of one or other of the models to any fluvial placer deposit.

4.5.2 Glacial deposition

As discussed in Chapter 3, short-term patterns of climatic change are associated with glacial and deglacial stages of waxing and waning of ice sheets and alpine type glaciers and, to a lesser extent, the warmer more equable climates of interglacials. In response to the formation of ice sheets rapid atmospheric cooling and ensuing cold climates increased mass wasting on slopes but decreased fluvial transport in valleys, thus producing burial of many of the Tertiary placers. For many Cainozoic placers glacial erosion then resulted in the development of discontinuous valley margin paystreaks, which were either buried by renewed mass wasting on slopes or dispersed and reconcentrated in other settings, e.g., by shallow marine processes on beaches and platform areas. Existing channel sediments derived from downcutting the rivers contained reworked gravels in which the gold was typically redistributed in a much-diluted form. Only remnants of palaeo-drainages now remain as terraces around valley walls.

In tracing the evolving pattern of secondary placer development over time, data from primitive placer environments can be introduced into the basic model in order to build up and finally elucidate the geological history of a promising area and hence its resource potential. Glaciation and the high rate of sediment formation by freeze-thaw processing follows long periods of deep chemical weathering in tropic and sub-tropic environments. Processes of erosion, transport and deposition are reactivated in direct response to renewed tectonism and climatic change. A pause in uplift or tilting of the strata brings a variety of changes such as the generation of elevated paystreaks within an aggrading fluvial system and a tendency for the superposition of drainage, and stranding of pre-uplift rivers and flood plains on uplifted plateaux. Each environmental system thus produces some unique features; every environmental change in some way modifies the existing forms. Changes include the glacial transportation or telescoping of pre-existing placers in very steep valley segments, and secondary reconcentration in low gradient intervals during cyclical periglacial



4.28 Depositional environments and typical vertical profiles of facies deposited during a single phase of glacial advance and retreat in various glacio-terrestrial and glacio-marine environments (after Eyles and Eyles, 1992).

weathering in a boreal weathering system. The basic problem is to recognise those elements of a landscape that were adjusted to the base level at the time of their formation and to fix their elevation relative to the present time. Analysing these environments and the sequence of geological events is essential to the study of glacial gold placer development.

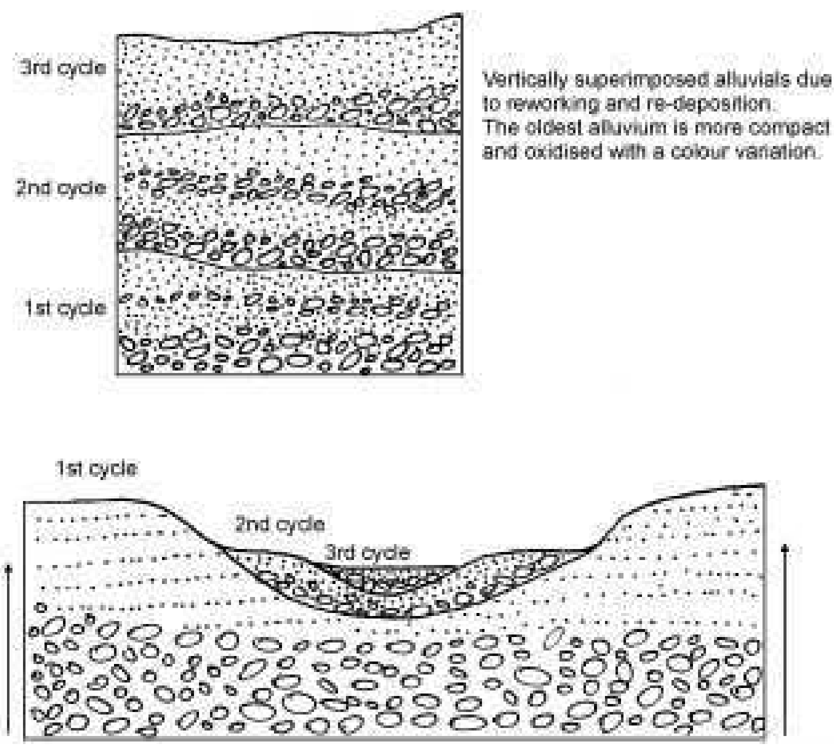
The spatial and chronological linkage of depositional environments in Fig. 4.28 identifies two distinctive system tracts during a single phase of glacier advance and retreat in various glacio-terrestrial and glacio-marine environments. Cyclicity of these glaciations is proposed for the high rate of colluvium derived by chemical weathering in humid intervals and concomitant colluvial encroachment in valleys of those times.

Pleistocene glaciations

Glacial erosion of primary gold placers in the Pleistocene contributed to the evolution of secondary placers in several ways:

- Discontinuous valley margin paystreaks were produced by a radial dispersion of primary gold paystreaks away from centre valley positions.
- Some paystreaks were dispersed and reconcentrated at the base of the glacial system while others were telescoped to lower levels.
- Glaciers with outlets into the ocean deposited englacial material at the shoreline, to be reconcentrated by shallow marine processes on beaches and deltas.
- Further entrenchment then occurred as a result of interglacial warming and renewed sedimentation in the valleys.

This situation is demonstrated in a schematic cross-section of the Upper Turon River, NSW (Fig. 4.29). The youngest alluvials lying topographically below the older gravels following the reworking of alluvials during three periods of uplift. Stratigraphically the third cycle is the present stream and the cycle runs late Pliocene, Pleistocene, and recent. Volcanic action accompanying orogenic upheavals protected sections of existing placers from further erosion under a cover of ash and lava to depths of up to 500 m and more. Renewed uplift and tilting in late Pliocene-early Pleistocene times caused streams to cut deeply into the volcanic rocks. Successive episodes of glacial and humid interglacial



4.29 Superposition of drainage in the Upper Turon River, NSW.

intervals in the Quaternary then profoundly affected both local and global weathering conditions.

Glacial till

Spoil deposited by glaciers is termed drift or till. Tills that are brought together directly by the ice, i.e. without fluvial transport, are deposited in the form of lateral moraines along the sides of the glacier and as terminal moraines during glacial retreat. A terminal moraine appears at the furthest advance of the glacier as it recedes and successive recessional moraines are deposited at intervals of stillstand in its retreat. Medial moraines occur where lateral moraines of intersecting glaciers join together centrally in the ice flow. Drumlins of clayey till form groups of oval shaped hills tapering in the direction of the ice flow. Long sinuous ridges of sands and gravels (eskers) of fluvio-glacial origin, mark the sites of melt water streams flowing in crevasses and tunnels within or at the base of the ice. Eskers, capable of transporting large volumes of englacial sediments at high velocities are formed under considerable hydrostatic pressure.

Exposure of till to glacially induced flow at the base of a glacial system creates ‘lodgement’ till, which is to a greater or lesser extent both stratified and sorted. Eyles and Kocsis (1989) note the common enrichment of the basal portions of lodgement till as a result of the sluicing action of sub-glacial melt waters. The presence of intraformational gravels within lodgement till sequences record erosion and deposition by sub-glacial rivers. A glacially reworked lodgement till is often covered by ‘ablation’ till when the ice melts in a stagnant marginal zone. Lodgement till and ablation till are termed ‘sub-glacial’ tills and are classified on the basis of the processes involved in their formation and location. Other till classification applies to ‘sublimation’ tills and ‘melt-out’ tills, as products of glacial reworking.

Sub-glacial placers

Glacier-related placers have been discussed widely in the literature but until recently, gold-bearing glacial debris has been regarded as of little economic importance except where it has been upgraded by post-glacial stream processes. Wells (1969) quotes Blackwelder (1932): ‘Since it is the habit of a glacier to scrape off loose debris and soil but not to sort it at all, ice is wholly ineffective as an agency of metals concentration.’ And – ‘If a glacier advances down a valley which already contains gold-bearing gravel, it is apt to gouge out the entire mass, mix it with much other debris and deposit it later as useless till. Under some circumstances however, it merely slides over the gravel and buries it without distributing it.’ On the other hand, Boyle (1979) recognised the importance of ‘auriferous glacial outwash gravels’ and ‘post-glacial stream gravels’ in placers of the Cariboo (Barker) Mining District, British Columbia, Canada.

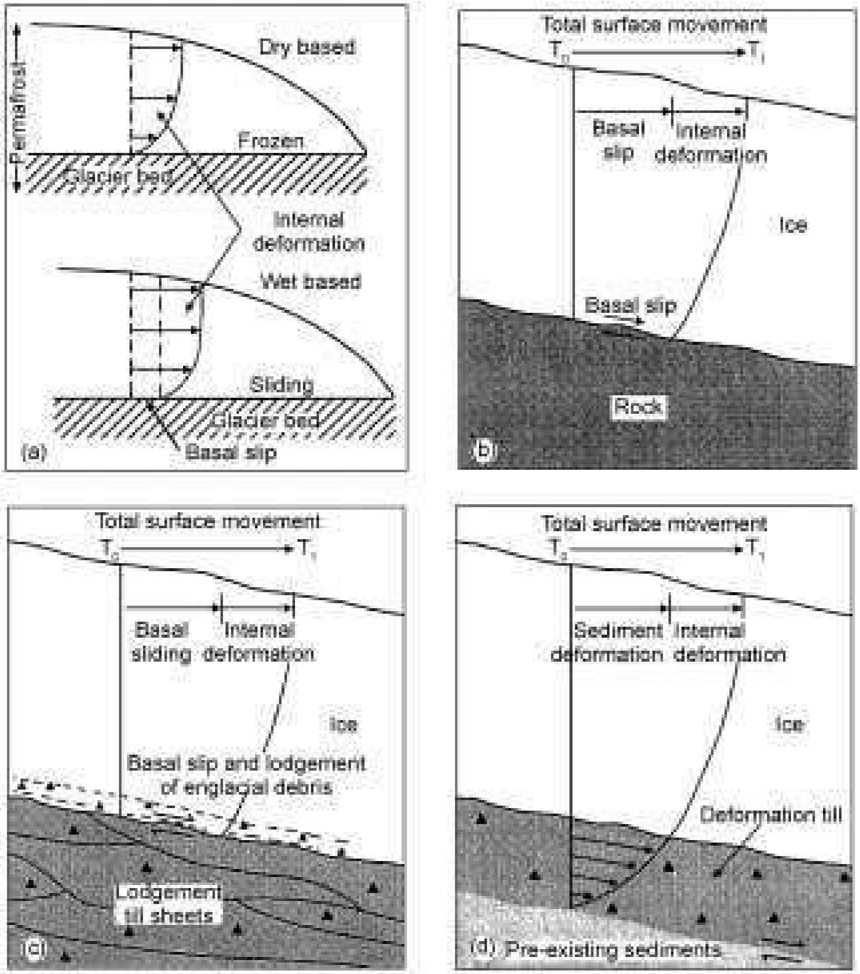
Boulton (1982) and Drewry (1986) were amongst the first to recognise the effects on placer formation of wet-based ice flow smearing of englacial debris over the underlying substrate. Working in the same general area of British Columbia, they all drew attention to the highly dynamic nature of sub-glacial flow in both the high-pressure transport of fine-grained sediment at the base of the ice and the channelled fluvial transport of coarse and fine-grained sediment. The process by which englacial till is released from the base of the ice as the glacier moves over the underlying bed involves frictional resistance and pressure melting. Debris within the basal layer of the glacier is lodged against the substrate when ice velocities are less than 50 m/y; at higher velocities, the bed is swept clear and erosion becomes dominant. Rates of lodgement till deposition in modern sub-glacial settings are reported to be around 2 cm/y. Eyles and Eyles (1992) illustrate the widely varying response of conditions at the base of large ice sheets to different ice temperatures and velocities in Fig. 4.30 (a), (b), (c) and (d).

Eyles and Kocsis (1989) describe the genesis and overall characteristics of economic glacial placers within lodgement till complexes of the Cariboo Mining District. The principal gold pay zones associated with these complexes are shown in Fig. 4.31. The authors in Fig. 4.32 present a schematic representation of Pleistocene stratigraphy and associated placer mines in north central British Columbia. The very coarse, nuggety character of the Cariboo gold placers is thought to have resulted from the incorporation of pre-glacial Tertiary paystreaks in basal tills along low parts of the valley floors.

Glacial outwash fans

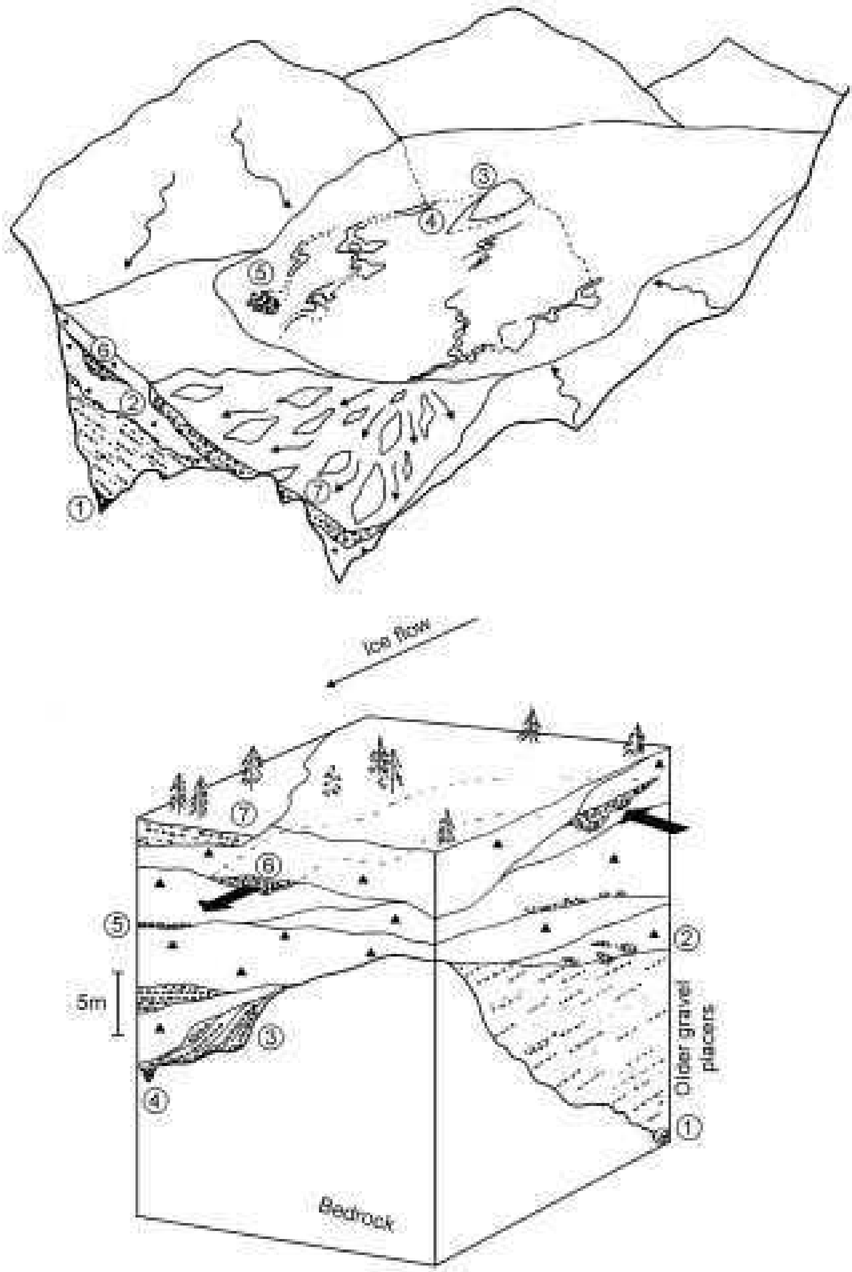
Outwash fans are built up by melt-water erosion of glacial debris on gravel plains. Over time, some of them may extend up to several kilometres in length and hundreds of metres in width at their terminal end. Many carry economic gold and are quite productive both for themselves and as secondary provenances for ongoing fluvial processes. As early as the Inca period, glacial outwash fan deposits were worked on both the southwestern (altiplano) and Cordilleran slopes in the Andes of Peru and Bolivia (Fornari *et al.*, 1982). In Papua New Guinea, auriferous conglomerates deposited under periglacial conditions in the Lakekamu Embayment, extend for about 40 km from their source. Small-scale mining ventures in the outwash fan have recovered more than 70,000 oz. of gold from shallow alluvial operations. Preliminary testing of channels at lower stratigraphic levels has indicated a much greater potential for the area as a whole.

Geomorphological control of gold evolution and distribution in glacial and fluvio-glacial placers of the Ancocala-Ananea Basin, Southeastern Andes of Peru has been studied by Herail *et al.* (1989). The moraines of the two glaciations Ancocala and Chaquiminas (middle and upper Pleistocene) provide

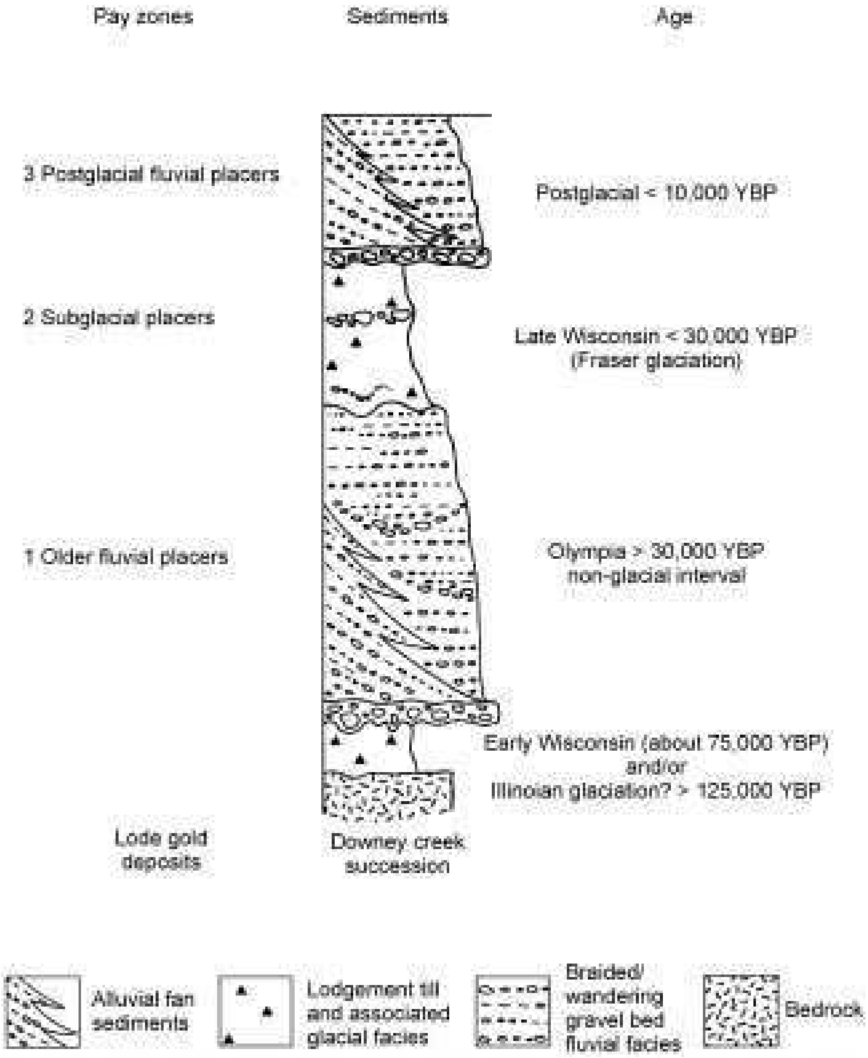


4.30 (a) Movement of dry-base (polar) glacier by internal corrosion. Glacier is frozen to the bed: bottom, in contrast wet-based glaciers move by internal deformation and basal sliding; (b) movement of wet-based glacier on bedrock substrate; (c) 'stiff-bed' model for accretion of till sheets below wet-based ice; (d) 'soft-bed' model where till is produced below wet-based ice by sub-glacial shearing of overridden sediments (Eyles and Eyles, 1992).

economically significant glacial and fluvio-glacial placers. The deposits occur where a glacier has cut through a primary mineralised zone comprising gold-bearing quartz veins related to arseno-sulphide deposits in the lower palaeozoic (Ananea) formation. Transition from glacial (moraine) to fluvio-glacial processing is accompanied by the gradual appearance of particles exhibiting fluviatile type morphology (high degree of flatness, bending and folding).



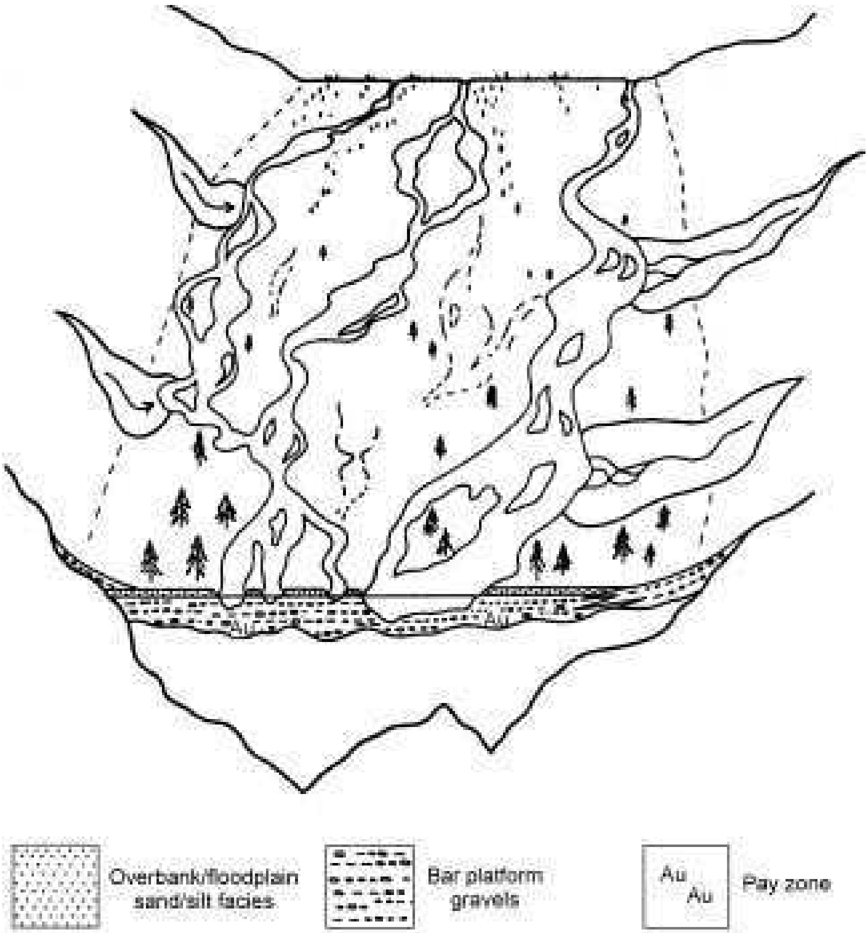
4.31 Depositional model portraying pay zones in lodgement till complexes. 1 – Bedrock gutter; 2 – glacio-tectonic structure and incorporation of gold-rich 'older' gravels; 3 – bouldery lee-side deposits; 4 – bedrock notches and vertical shafts; 5 – boulder pavements; 6 – intraformational channel fills; 7 – proximal braided river facies (modified from Eyles and Kocsis, 1989).



4.32 Schematic representation of Pleistocene stratigraphy and associated gold placer mines in north central British Columbia.

Wandering gravel-bed placers

Wandering gravel-bed rivers have repeatedly scoured and reworked older placers in the Cariboo Mining District of British Columbia (Eyles and Kocsis, 1989). Pay zones in these deposits are dominated by two distinct fluvial styles: braided rivers and 'wandering gravel-bed streams' (Church, 1983). The braided river systems were formed under cold boreal climatic conditions with sparse vegetation, increased mass movement on slopes and decreased fluvial transport



4.33 Wandering gravel-bed placers (adapted from Desloges and Church, 1987).

in valleys. Wandering gravel-bed streams evolved under conditions typical of present day temperate climatic conditions with densely forested drainage basins and valley floors (Fig. 4.33).

A wandering gravel-bed stream is commonly sinuous, with large medial bars and occasional braided reaches in which gravels are interdigitated with fine-grained over bank and flood plain deposits (Desloges and Church, 1987). Repeated lateral migration of wandering bed channels and the multiple reworking of the bar platform gravels results in the development of an extensive gold-rich basal horizon across the whole of the valley. The reworking of the Bella Coola River, typical of the model portrayed in the above figure appears to have occurred every 150 years since the end of the last great ice age some 10,000 years ago.

4.5.3 Quaternary adjusted deposits

Quaternary adjustments of river systems resulting from changes in annual precipitation and episodes of glaciation and deglaciation have widely modified the characteristics of most present-day river systems from those of their parent river systems. Tertiary climates were predominantly hotter, more humid and precipitation was much higher than today hence, in most cases the ancestral streams were much larger than are those of today. Douglas (1977) cites the Murray-Darling basin of southeastern Australia as a natural laboratory for testing the principles of hydrologic geometry. The modern Murrumbidgee River, which now transports very little sand, was preceded by ancestral channels that were much larger and straighter and of steeper gradient than the modern channels because of greater annual run-off and higher flood stages (Table 4.6).

Changing weather patterns also lead to the development and/or modification of a wide variety of placer types. Braid bars and point bars are successively modified and change position with each fresh cycle of flooding. Cold ice age climates provide increased mass wasting on slopes as a result of periglacial weathering and the transport of thick blankets of frost-riven detritus off slopes and onto valley floors. Removal of this sediment contributes to the exhumation and redistribution of pre-existing placers. It is generally recognised in this regard, that many terrace features in the upper reaches of streams may be due to changes in the load-water discharge ratio, rather than in changes in base level.

Depending largely upon the stage of development of a channel system, depositional units in a valley are built up by lateral accretion, vertical accretion or by a combination of the two. The spatial relationship between these units and

Table 4.6 Morphology of riverine plains channels (after Schumm, 1968)

Location	Murrum-bidgee River	Palaeo-channel 1	Paleo-channel 2
Channel width (m)	67	140	183
Channel depth (m)	6.4	10.7	2.7
Width-depth ratio (F)	10	13	67
Sinuosity (S)	2.0	1.7	1.1
Gradient, S (m/km)	0.13	0.15	0.38
Meander wavelength (m)	853	2134	5490
Median grain size (mm)	0.57	–	0.55
Channel silt-clay, M (%)	25	16	1.6
Bed-load, Q_s (%)	2.2	3.4	34
Bankful discharge ($m^3 s^{-1}$)	594	1443[a]	651[a]
Sand discharge at bankful (t/day)[b]	2,000	19,000	49,000

[a] Calculated by use of Manning equation and channel area.
[b] Calculated by Colby's technique.

the distribution of values at any one time may be exceedingly complex. Differences in sedimentary behaviour stem from changing base levels and variations in stream power. The cyclical inflow of sediments may have been different for different reasons and may not have been derived proportionately from the same sources or at the same rate. During epeirogenic uplift the same valley may be filled and scoured many times. In a typical sequence:

- Tributaries are rejuvenated as the weathering front moves up the valley.
- Surges of sediment brought down by the tributaries choke up the main channel, which becomes braided.
- Paystreaks are developed in the channel lag.
- Renewed deep scouring takes place at the valley outlet and the sequence is repeated until equilibrium is reached at the reduced base level.

River terraces

Terrace deposits are remnants of alluvial valley fill that now exist as bench-like landforms in which streams have incised their way into the underlying rocks. The deposits follow the course of the streams, approximating their gradients and containing remnants of earlier placers deposited on the valley floors during periods of tectonic stillstand characteristic of a temporary stay in downcutting. Prior to a further strong tectonic uplift, terraces at each erosional level tend to reflect a particular stage of stratification and sorting of gold grains within the stratigraphic section.

In most areas affected by pulsatory epeirogenic uplift in Eastern Australia and in the Pacific and Southeast Asia, the depositional sequence of placer gold deposits is generally as follows:

- Recent deposits are found within presently active streams.
- Pleistocene gravels occur in terraces rising some 4 m to 10 m above present stream levels.
- Pliocene gravels are represented in terraces at levels of 10 m to 60 m or so above present stream levels.

Pliocene gravels and boulder beds are usually thicker and vary more widely in depth than Pleistocene gravels. This denotes a more extensive range of pulsatory uplift during the Pliocene, a longer period of weathering and a correspondingly greater supply of both sediment and gold.

The Turon model

In the Turon Valley, NSW, Australia, pre-Miocene river gravel carried gold derived from Pleistocene times. A post-Miocene uplift left these rivers and outwash gravels stranded high in the Miocene peneplain. Some gold from these gravels was fed into the ensuing Pliocene gravels below. A further uplift at the end of the Pliocene repeated the process with some gold being passed into the Pleistocene gravels still further below and flanking the Turon River some 3–5 m above river levels. Erosion of these Pleistocene gravel beds fed gold into the present active river which, itself, has two levels; the slightly older one through which the presently active river flows is only active at flood time (refer back to Fig. 4.29).

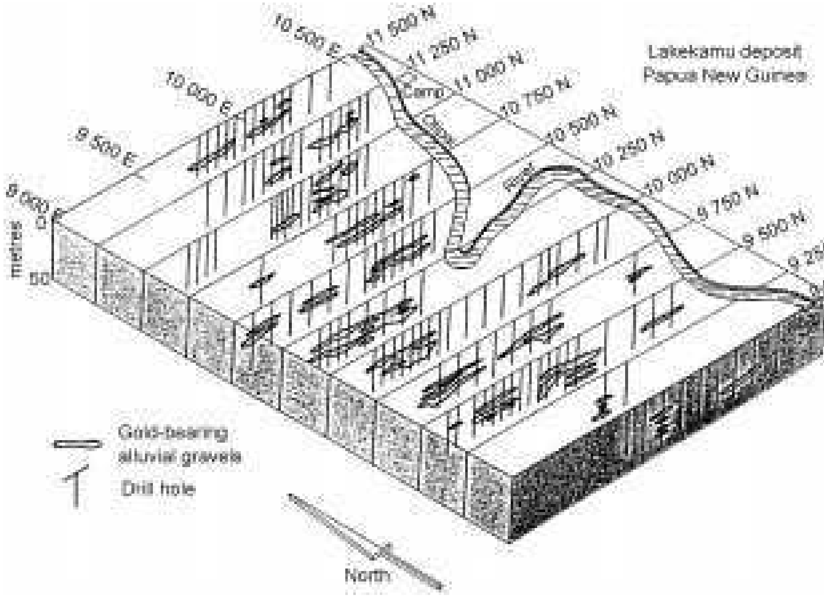
The Lakekamu model

Elevated terraces with lateritised clays and outwash boulder beds or fanglomerates make higher ground above the flood plain occur along the Olipai River and elsewhere in the Lakekamu Embayment of Papua New Guinea. The terraces, possibly of epi-Pliocene age, rest with low-angle unconformity on bedded mudstone, conglomerates and fine tuffs of Pliocene age, which form the basement for both the Olipai River and its palaeochannel and of the terraced fanglomerates. Placer gold occurs firstly in the fanglomerates or palaeochannels therein and secondly in the flood plain with concentrations in palaeochannels. Examination of the gold of these two environments suggests that there are more points of similarity than there are differences. It is probable that the two had a common primary origin in the late Mesozoic Owen Stanley Metamorphic series though a somewhat different geomorphic history.

The Olipai palaeochannel is stepped or terraced with four levels, including its base, indicating pulsatory periods of minor uplift followed by subsidence which has buried the channel and covered the flood plain with silty, sandy and gravelly sediment. A discontinuous obstruction to the flow had the effect of accumulating a barrier of gravels and cobbles that abruptly changed the course of the main 350 m wide palaeochannel into a narrow (less than 50 m wide), scoured channel. This corresponds roughly with the course of the present Olipai River, which is diverted for about 500 m to the west before once more opening out and flowing again in its original southerly direction (Fig. 4.34). The original cause of the obstruction is not known, but flow in the narrowly confined sector was almost certainly super-critical and chaotic; only traces of gold were identified in bedrock drill samples.

Tertiary deep leads

High level sub-basaltic placers are common in Victoria and New South Wales, Australia and in California, USA and are distinguished from ordinary deep leads by having a basaltic lava covering. Figure 4.35 (a), (b), (c), (d) illustrates how remnants of such earlier fluvial deposits may occur as deep leads. A pre-Tertiary surface (a) with a fluvial placer occupies the lowest portion of the valley. An outpouring of basalt has mantled the area in (b) and stress fractures develop due to stretching along the higher portions. This initiates new erosion along the lines



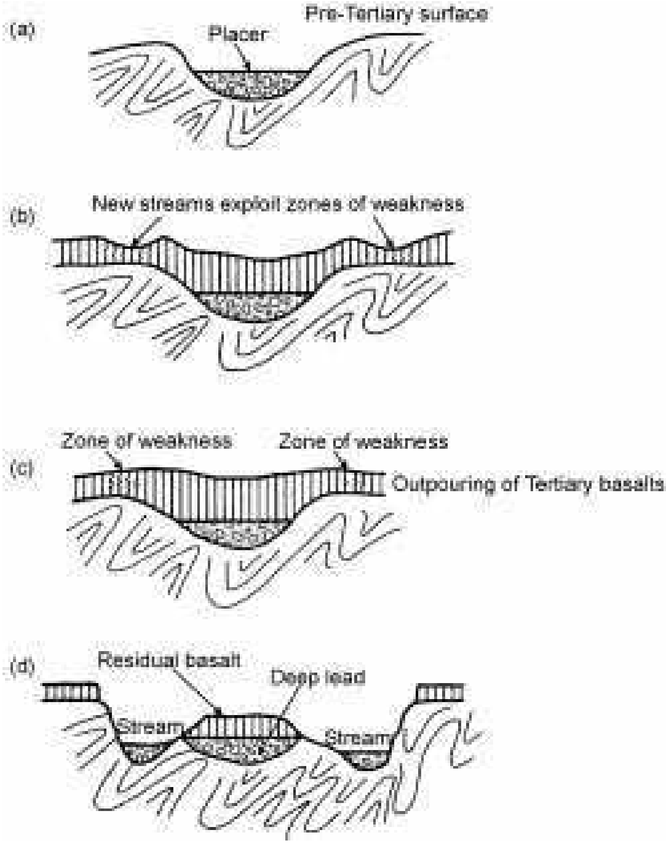
4.34 Course change due to obstruction in the original Olipai River, Lakekamu Embayment, Papua New Guinea.

of weakness as shown in (c). Finally the former channel deposits remain as high-level gravels protected by a capping of basalt high above present stream levels (d). Initially the outpouring of Tertiary basalts preserved a fluvial gold placer over pre-Tertiary alluvial landforms. Ultimately the Tertiary basalt was almost completely removed by the erosive forces leaving remnants of fluvial placers such as the one depicted in the above illustration on the tops of hills protected by the basaltic capping.

Fluvio-Aeolean placers

Deserts cover about 30% of the continental surface and vary from small areas covered by bare rock undergoing erosion, to vast areas covered by dunes that are in constant motion. The great tropical deserts of the world occur along the tropic of Cancer at latitudes 15–35 ºN and the tropic of Capricorn at latitudes 15–35º S. These deserts lie under virtually stationary sub-tropical cells of high pressure characterised by a subsiding air mass that is adiabatically cooled and dried as it sinks. Precipitation is largely convectional and unreliable in tropical deserts, typically less than 25 cm annually and sometimes less than 5 cm. The principal tropical desert regions are the Kalahari and Sahara (Africa) and the Thar Parka (India and Pakistan).

Major desert regions in the middle latitudes, 35–50º N and 35–50º S, occur in central Asia, Australia and the Great Basin and Mojave Desert areas of the



4.35 Typical changes in development of a Tertiary deep lead (after Macdonald, 1983a).

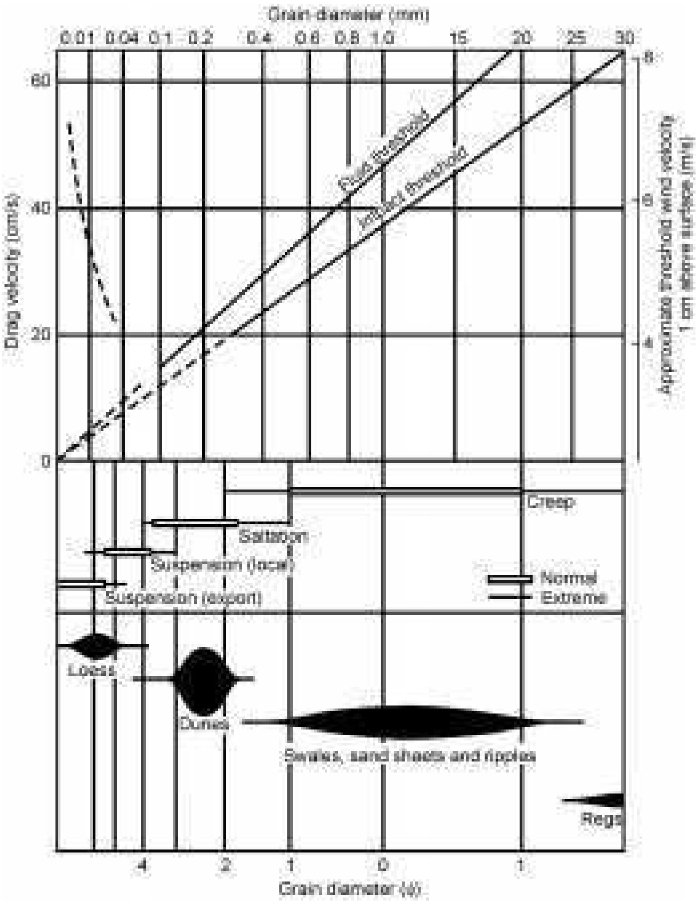
western USA. Dryness in these regions is due to location, either distance from the ocean or in a rain shadow on the sheltered sides of mountain ranges. Desert placers are characteristic of basin and range topography with small watersheds supplying sediment to alluvial fans. Annual precipitation ranges from 10–50 cm.

Aeolean processes

Wind blowing over the surface of the ground gives rise to forces and stresses at the ground/air interface that are generally similar in nature to those produced by a river flowing along its bed, but with differences in scale. Air ($\rho_a = 1.25\,\text{kgm}^{-3}$ at sea level) has a much lower mass/unit volume than water ($\rho_w = 1.00\,\text{g/cc}^{-3}$). Dynamic viscosities are also much lower for air ($\mu_a = 3.62 \times 10^{-7}$ mPas at 40 ºF) than for water ($\mu_a = 3.24 \times 10^{-4}$ mPas at 40 ºF). Because of these differences, impact and viscous forces are correspondingly smaller for air than for water as also are the respective buoyancy effects of the two fluids. Bagnold

(1941) has shown that particles of about 0.1 mm are the most easily moved by airflow and those both larger and smaller particles require higher velocities for entrainment. Figure 4.36 shows the relationship between grainsize, fluid and impact wind velocity thresholds, and characteristic modes of aeolean transport and resulting size grading of aeolean sand.

Similarly as for fluvial transport, particles are moved from rest when the combined turbulence and forward motion of the fluid lifts them from their beds. The stress varies as the square of the velocity but is also affected by the roughness of the surface and the size of the particles. Surface roughness induces turbulence in the boundary layer thus promoting lift; the size of the particles affects their mobility. The larger particles roll along the bed (traction); hit against and dislodge other particles that bounce into the air and are carried along in a flat trajectory (saltation) before falling to the ground to strike and dislodge other particles. The process continues until the wind velocity falls below the



4.36 Aeolean transport features (after Folk, 1980).

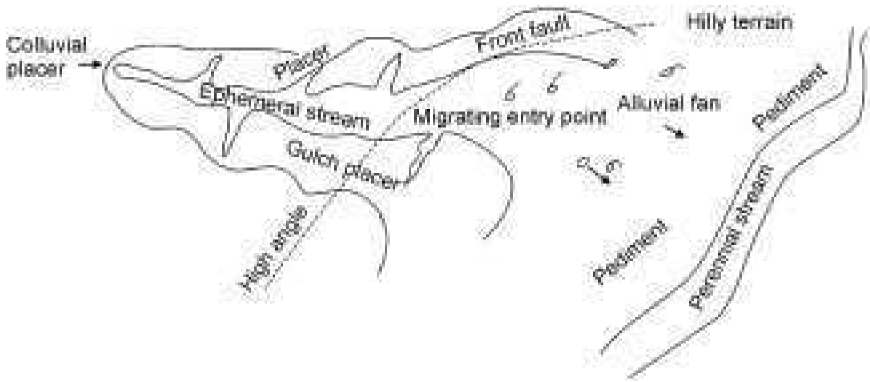
critical entrainment velocity. Particles much smaller than 0.01 mm, once entrained, do not settle freely. Dust-sized particles are swept up to very great heights and may be transported for hundreds or thousands of kilometres before being washed out of the air by raindrops, perhaps to be deposited as beds of loess. King (1966) notes that the mean particle diameter of thick loess deposits as in China, is about 0.05 mm, i.e. coarse silt in the Wentworth scale of sediment size classification.

The depth limitation of the weathering profile of recently exposed source rock in hot dry conditions is a few metres at most. The volume of gold-bearing detritus is typically small and its value rests mainly as a pointer to the possible size and value of the primary orebody and/or to the possible presence and whereabouts of palaeo-placer deposits formed previously under more humid climatic conditions. For example, although evidence of gold mineralisation is widespread in Saudi Arabia, a regolith of shifting sands that covers most of the landscape has a masking effect on the geochemical indication of primary gold and fluvio-aeolean palaeochannel development on a regional scale. Possibly because of this, only small-scale gold mining activities appear to have been carried out in much of the western part of the Kingdom in pre-Islamic days. In the Murayjib area, gold placer workings in Wadi Haradah can be traced for a distance of about 7 km from source to larger workings at Efshaigh adjacent to Wadi Kohr but this type of occurrence is rare. Recent checking has shown that the ancients were very thorough in their treatment of the surface materials and shallow channels, but a great deal more remains to be done in locating major alluvial and primary gold mineralisation.

Fluvio-Aeolean settings

Fluvial-aeolean gold placers are formed in semi-arid environments as the result of heavily concentrated, though ephemeral, stream flow over short and intermittent periods of time. Although rainfall rates are low and sporadic in desert regions, running water is essential for the accumulation of commercially viable gold placer concentrations. The general absence of plant cover over exposed rock formations in deserts provides for high rates of run-off over the stony desert surface and fans grow in stages as sites of deposition change from one side of the fan to the other. Sedimentation is cyclic and climatically controlled and placers exhibit definite sediment patterns of sorting, rounding and particle distribution according to weight, rate of flow and channel gradient.

An elevated area of source rocks intermittently eroded by short ephemeral streams is envisaged by Prudden (1990) in the construction of a conceptual geological model of fluvio-aeolean placer formation (Fig. 4.37) from the gradual release of gold from the weathering of exposed source rock. Tributaries draining down from these deposits join together at lower levels to form a larger integrated channel system into which the intermittently flowing streams discharge their



4.37 Desert placer gold model.

loads. Some of the gold is deposited in 'gulch' placers the remainder of the auriferous debris is carried on into an alluvial fan system in an open valley setting at the base of the slope. The streams do most of their transporting and sorting during flood times and deposit most of their spoil when the floods recede. The duration of the active placer-forming processes is a critical factor in the development of an economically viable deposit.

Deltaic placers

The name 'delta' was given to the outpourings of the Nile by a Greek philosopher Herodotus (484–424 BC) who recognised the similarity of the shape to the fourth letter of the Greek alphabet. The delta classification is now based upon formation rather than shape and the term is applied generally to all tracts of alluvial ground between diverging branches of rivers where they empty into large bodies of standing water. Deltas occur as freshwater deposits when the streams discharge into lakes and as marine deposits when they flow into shallow waters of the open sea.

Deltas formed under lacustrine conditions comprise simple arrangements of topset, foreset and bottomset beds which increase in size and complexity with differences in sediment supply and changes in the action of waves, tides and currents. Changing rates of discharge and the depth of water at the river mouth influences rate of growth. The Witwatersrand conglomerate deposits of South Africa were laid down along the southwestern shoreline of an extensive inland lake in the Witwatersrand Basin during the Archaean-Proterozoic transition 2.8–2.2 billion years ago (see Chapter 2).

Bulolo lacustrine deposits

The Bulolo gold placer in Papua New Guinea is an example of Cainozoic gold placers of deltaic origin. Prior to faulting in the Bulolo Valley, Papua New

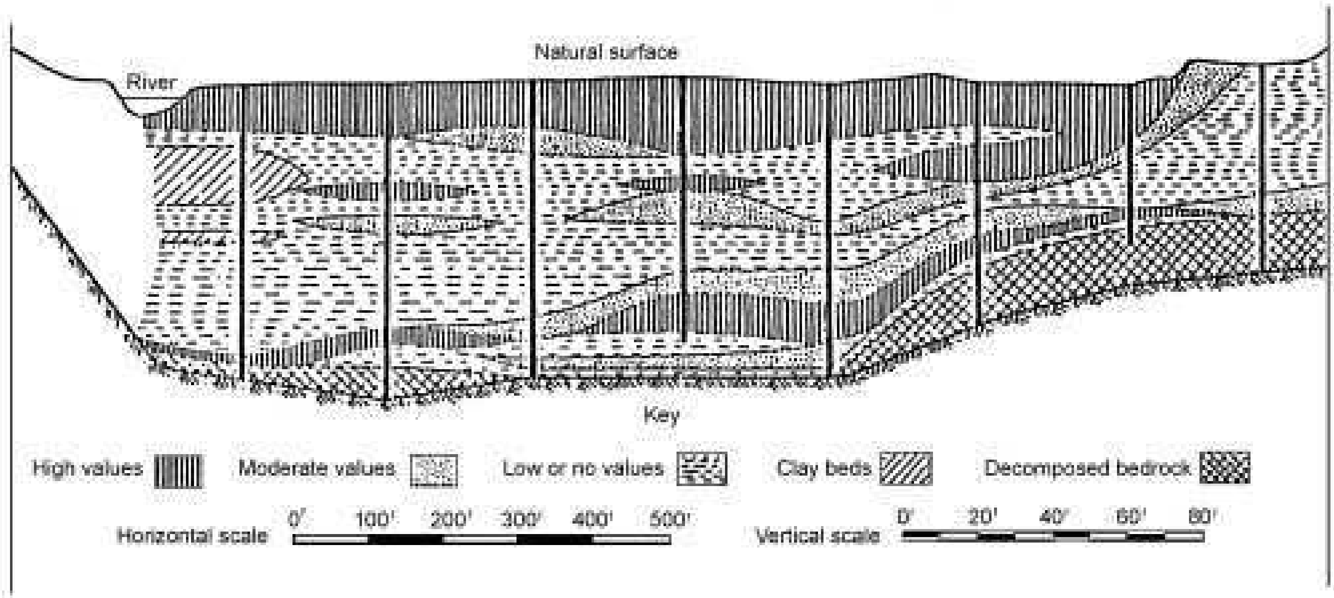
Guinea and damming of the Bulolo Valley, gold-bearing tributaries of the Bulolo River included Big Wau Creek, Koranga and Namie Creeks and Edie Creek. The formation of a lake created lacustrine deltaic conditions, which led to formation of the Bulolo placer. Dredgeable ground in this deposit extended 6.4 km downstream to the junction of the Bulolo River with the Watut River and 4.3 km along the Watut. Production statistics show a recovery of 2.13 million oz. of fine gold from some 207 million m^3 of gravel between 1931 and 1967. A typical cross-section of one of the shallower valley sections is shown in Fig. 4.38. Although good sample values were obtained in drill samples down to 90 m in depth (Fisher, 1935), dredging was constrained to about half that depth because of limited dredger capabilities. The presence of the water table near to the surface precluded dry stripping.

Shallow marine placers

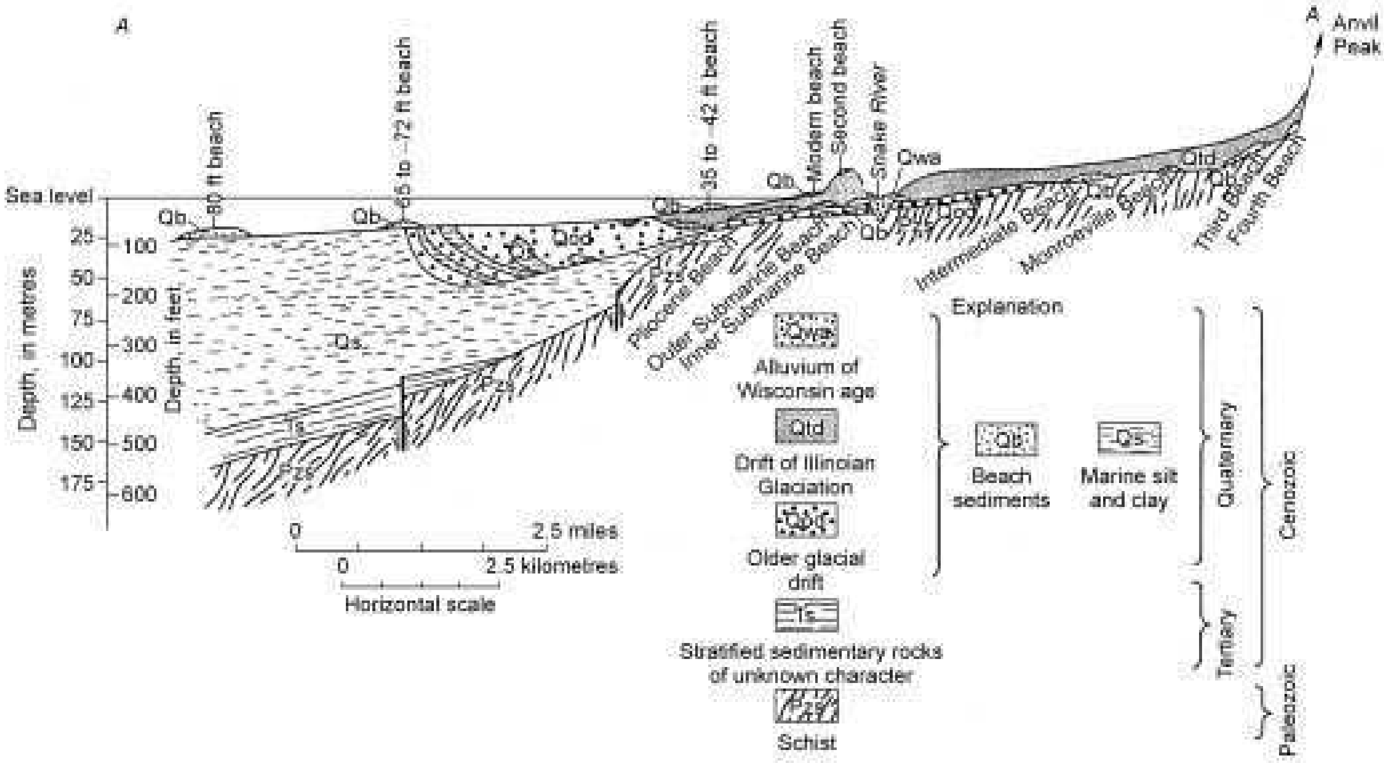
The sediments of shallow marine placers are derived from source rocks either on the land or below present sea levels along continental shelves. Gold-bearing gravels derived from on-shore provenances may reach the sea front for further sorting only where the source rocks occur near the coast and are drained by steep gradient streams, or where transportation is by ice flow or glacial telescoping. Gold grains deposited glacially at shorelines comprise all sizes from coarse to fine. The larger particles remain close to the shore but are less well sorted than gold in stream placers and rely upon wave action for further upgrading. The very fine particles are carried out to sea by wave action. Initial gold grades need not be high, but grain size is important. Small isolated occurrences of gold found on the beaches at Yamba in NSW, Australia between 1870 and 1885 could not be recovered economically but they led to the mining of much larger concentrations of heavy minerals (rutile, zircon, ilmenite, etc.) with gold as a by-product.

Beach placer concentrations are formed at the base of frontal dunes on open beaches and in natural traps as provided by headlands and other barriers to the flow of longshore currents. The movement of the sea gradually sorts the beach sands, directing the finer particles into deep water and the coarser materials towards the shore. Because of its high density, gold becomes concentrated along with the other heavy minerals and coarser sediments. The final distribution of values is influenced by the differential sedimentation rates of the particles and by the strength and direction of the wind, waves and ocean currents.

Present evidence suggests that shelf areas were exposed to atmospheric weathering for only brief periods of time during Pleistocene interglacial intervals. The most recent exposure may have occupied less than 25,000 years and earlier interglacial intervals were probably of similar short duration. During the course of the Holocene Marine Transgression, which commenced about 10,000 years ago, sea level rose in a series of oscillations from a low of minus 130–160 m up to its present level.



4.38 Typical cross-section – Bulolo gold placer, Papua New Guinea.

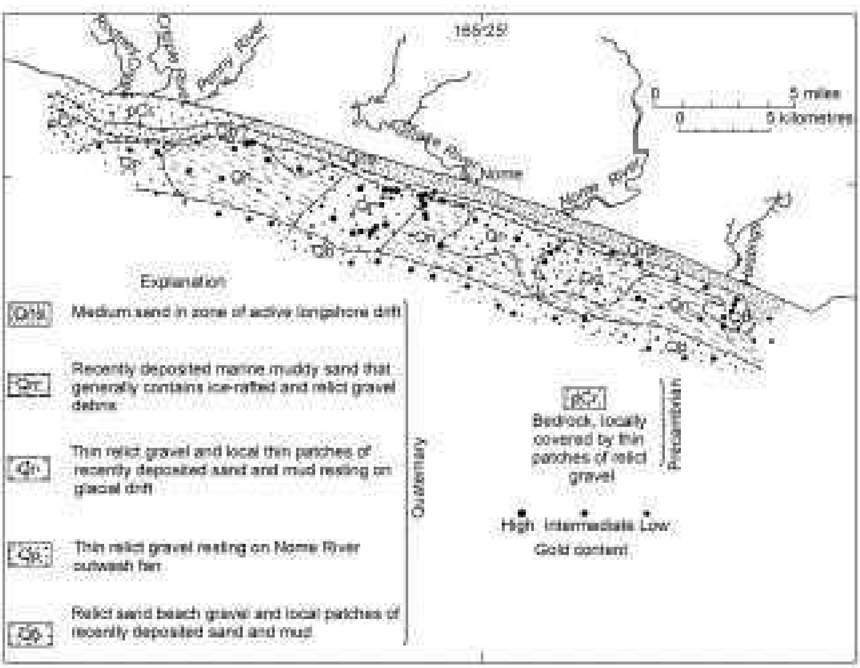


4.39 Strandline deposition on and offshore, Nome, Alaska (after Nelson and Hopkins, 1972).

Because of the relatively short time of exposure and the low gradient topography, the development of gold placers derived from now submerged source rocks is possible but unlikely. Few primary gold source rocks are known to occur on the shelves themselves; any streams that may have serviced them during periods of emergence were probably small. Furthermore, while some areas such as the Sahul shelf, northwestern Australia and the coastal shelves of Canada and northeastern USA are extensions of metallogenic belts onshore, the generally thick cover of marine sediments and excessive water depths puts them beyond the reach of present-day exploration and mining techniques. Down warping has submerged other shelf areas to presently unmineable depths.

Beach placers at Nome, Alaska

The best-known examples of drowned strandline gold placers of glacial origin are in California and Nome, Alaska. The Nome deposits were laid down on a flat alluvial plain over which twelve or more beaches were developed successively during the Quaternary and earlier periods of fluctuating sea levels. They are classified separately as offshore, modern, submarine, second, intermediate, Monroeville, third and fourth placers (Fig. 4.39).



4.40 General geological map showing trends of gold content in surface sediment in Nome near shore area (after Nelson and Hopkins, 1972).

Drowned strandline deposits have been identified offshore at various distances beyond the present shoreline of Seward Bay and nearly to the centre of Chirikov Basin from the Siberian Chukotka Peninsular. The Nome drowned strandline deposits are located at depths below sea level of about 11, 21 and 25 metres that probably represented stillstands at the time of their formation. A general geological map of the Nome near-shore area shows the distribution of gold in the surface sediments (Fig. 4.40).

The 'modern' beaches were worked at various levels for distances up to 9 km inland from the present shoreline during the gold rush days but became uneconomic early in the twentieth century. The raised beaches were mined until about 1963. An unsuccessful attempt was made to dredge shelf deposits in the late 1980s, but after a promising start the operation was brought to a halt by machinery failure. Recovered grades were lower than expected and although losses were probably high, there appeared to be little hope of improvement. The dredger (BIMA) used in this operation was ferried to Alaskan waters after being shut down at the close of an Indonesian tin-dredging operation in 1985.

The widespread disposition of the Nome placers appears to have been due to the role of glaciers in the dispersal and redistribution of low-grade auriferous tills derived from provenances in mountains some distance north of Nome. Both source rocks and segments of older placers in the coastal plain were sequentially eroded and telescoped by the glaciers. When deposited on beaches, the weakly auriferous tills were successively upgraded by wave action during each interval of stillstand following uplift. Marginal gold accumulations are still worked sporadically on Nome beaches when cliffs of glacial debris are eroded by violent storms.

5
Gold exploration

Exploration is a search for the unknown, and challenging aspects of gold exploration are to find new deposits in hitherto unexplored areas and to re-examine previously explored areas that may have been rejected because of inefficient technology, faulty or inadequate geological investigations, or geographical constraints. Because some prospective areas have been looked at and rejected by others does not necessarily preclude them from further consideration. Radical new ideas and improved technologies have emerged during the past few decades, and more will emerge in the future. Theories of plate tectonics have had an enormous impact upon such traditional problems as the origin of folded mountains, effects of certain types of igneous intrusion on the creation of gold source rocks, and tectonic and climatic controls on the functioning of the fluvial system. Improved age dating techniques have been developed for placing the geological events of Earth history in sequence according to age. Control of deposit size and mineral content has been established as a function of the geological setting. Some mountain ranges are still in the process of formation, e.g. the Andes of South America and Rocky Mountains of the USA. Older systems, exposed to weathering and erosion for long periods of geological time, now exist only as remnants of palaeo-zones, represented by specific rock types, e.g. intermediate to acid intrusives and andesite volcanoes.

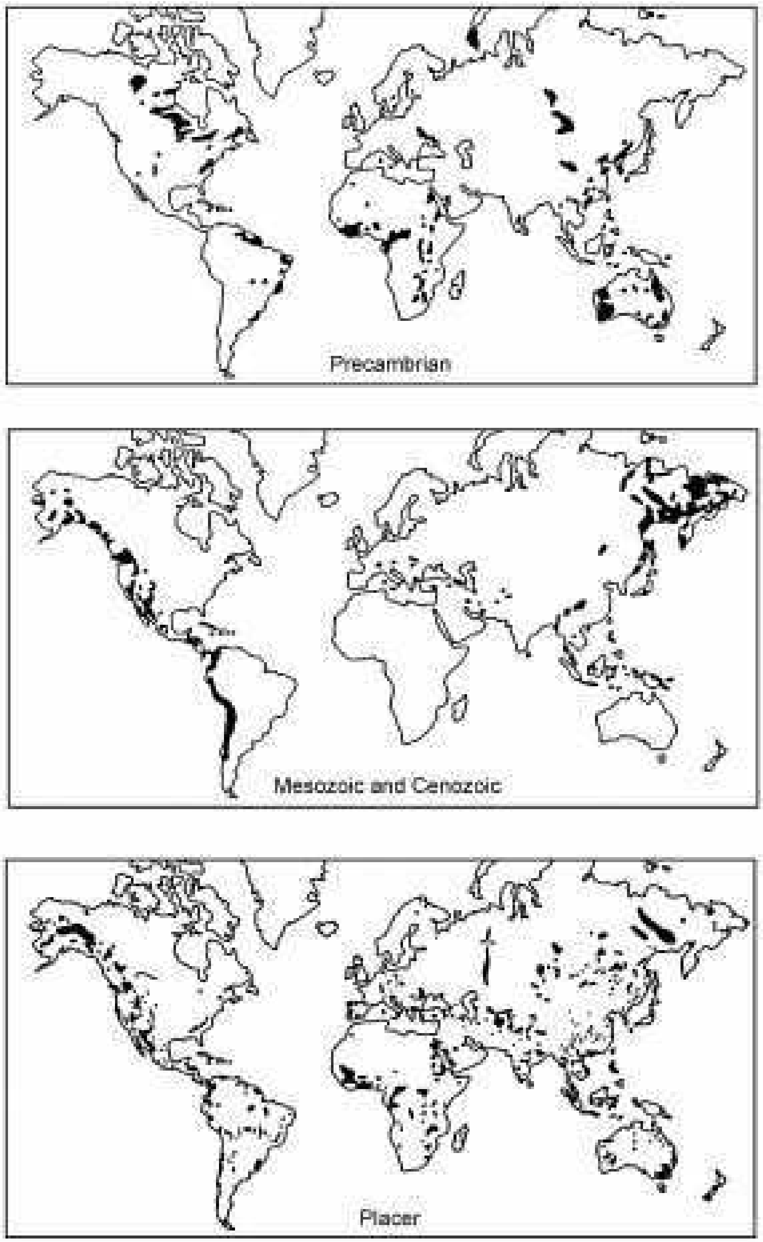
The scope of modern exploration programmes has been expanded by the deployment of remote-control systems such as thermal I.R., radar and high-resolution photography on spacecraft, and increasingly high-resolution (spectral and spatial) satellite imagery. Thanks to modern technology, exotic remote sensing and geophysical methods can indicate the potential of deposits that are now masked by profuse vegetation, or by a cover of later sediments or volcanics. With the general acceptance of seafloor tectonic and hydrothermal processes as being either analogous to or integral parts of geological processes operating in the continental crust, proposals postulating a seafloor origin for many land-based metallic deposits seem plausible. Giant mapping projects are being carried out into previously uncharted waters, piecing together previously unknown geological features in an under-sea landscape that has been developed by plate tectonic processes over millions of years.

5.1 Geological investigations

Observations of patterns and conditions of volcanism associated with different tectonic settings provide a reasonably clear understanding of the modern global tectonic framework; but uncertainties of the geothermal gradient and scale of tectonic systems attached to the events of ancient times have yet to be resolved. There is a general lack of evidence in Precambrian times of the state of the Earth's surface in relation to the distribution and nature of climatically controlled morphogenic regions or to the effects of ancient base level and/or climatic change. Few of the older mountain chains remain, and the association of residual gold ores is mainly with Phanerozoic mountain ranges, sometimes as the result of patterns and conditions of volcanism associated with variable tectonic movements and periods of erosion in which the sediment has been reworked and the gold reconstituted. Archaean (>2500 Ma) high-grade gneissic regions host only a few mineral deposits; most known occurrences are in the volcanic and volcano-sedimentary sequences called 'greenstone belts'. There may have been deep-seated fractures and rifts in the earlier cratonised rocks of Archaean times but few of the older mountain chains remain. The Archaean greenstone belt development was still in an embryonic state when it continued into the Proterozoic. As shown in Rodinian reconstructions of the Earth's crust, the earliest known rift with plateau basalt is only 1200–1100 Ma old.

Proterozoic Eon (600–2500 Ma) greenstones are characterised by intensive folding and faulting and large axial and marginal faults with a more diversified metallogeny. Basins were developed with thick sedimentary accumulations. The source of the Witwatersrand and similar type gold deposits of South Africa appear to have been the ultramafic and mafic volcanics of the Kaapvaal craton. This craton appears to have been richer in gold than its younger Proterozoic Yilgarn craton counterparts in Western Australia and in the Superior province of eastern Canada.

Residual gold ores in the Americas are associated mainly with Phanerozoic mountain ranges as the result of patterns and conditions of volcanism associated with variable tectonic movements, and periods of erosion during which the sediments were reworked and the gold reconcentrated. Yeend (1974) has suggested that more than 40 km depth of sediment was removed from the Sierra Nevada in California prior to the stabilisation of the drainage and the formation of the Eocene placers along the Yuba and associated rivers. New sources of elemental gold are still derived largely from magma at mid-ocean ridges, but most of the gold mineralisation in Phanerozoic ores probably derives from the re-cycling of gold and weathered auriferous material of all ages. Table 5.1 lists the geological criteria for selecting areas for gold exploration in belts dominated by volcanic rocks. The distribution of primary sources of gold in Precambrian and Palaeozoic rocks favourable to the distribution of residual gold is shown in Fig. 5.1.



5.1 Distribution of primary gold ores of Precambrian and Phanerozoic age and residual gold deposits derived from these sources (modified from Sutherland, 1985).

Table 5.1 Geological criteria for selecting gold exploration areas (after Hodgson and Troop, 1988)

Feature	Genetic significance
Rocks	
Low- to medium-metamorphic-grade, deformed supracrustal sequence consisting of fault-bounded belts of volcanic and sedimentary rocks intruded by major syn- to post-tectonic granitoid batholiths	Prograding arc-trench complex
Mafic to ultramafic volcanic rocks	Ultimate source of gold (?)
(in contact with)	
Sedimentary rock belt	Indicates fundamental fault in an accretionary geotectonic environment
(especially if)	
Molasse- or Timiskaming-type sedimentary rocks	Indicates fault developed after the assembly of rocks by accretion; on a smaller scale, sedimentary rocks occur along the part of this fault that (i) has maximal throw, and therefore vertical dimensions, to best tap deep fluid sources and (or) (ii) has dilated as a consequence of trans-tensional oblique-slip movement and so is favourably orientated to focus fluid flow
Major TTG or calc-alkaline batholiths	Arc magmatism that has prograded over accretionary complex, providing extra input of heat to drive devolatilisation of subcreted oceanic crustal rocks
Alkalic or trondhjemitic felsic porphyry intrusions	Like molasse, indicates a fault developed late in the geotectonic development of the accretionary belt, a time favourable for maximum fluid generation. On a smaller scale, intrusions are localised at structural sites on faults characterised by high permeability to magmas and ore-forming fluids

Structures	
Fundamental faults or 'breaks'	Magma and fluid conduits connecting lower crustal site of ore-fluid generation with upper crustal site of ore deposition; sites of seismogenic 'fault-valve' behaviour necessary for the development of supra-lithostatic fluid pressure to drive fluids into local gold-rich source rocks and allow cyclic fluid-pressure-release mechanism for gold deposition
Local faults and folds	Dilational environment allowing fluid degeneration by decompression or 'throttling' and (or) CO_2 effervescence, leading to mineralisation and alteration
Mineralisation and alterations	
Carbonatisation along fault zones	Indicates that deeply derived, CO_2-rich gold-mobilising fluids have been circulating through fault zone
Silicification, pyritisation, K- or Na-metasomatism, quartz-carbonate-sulphide veins	Gold-mobilising fluids have interacted with a gold source rock with the mineralogy of a felsic igneous rock (a felsic igneous rock or a clastic sediment)
Syn-volcanic peraluminous alteration, VMS or Ni-Cu mineralisation, pre-tectonic porphyry-type Cu-Au mineralisation	Local gold-rich source rocks (?)

5.1.1 Ancient vs. modern geologic settings

Processes associated with the aggregation and break up of continental masses favour specific variations in metallogeny, which were not recognised until the advent of plate tectonic theory in the late 1960s. This new and revolutionary approach to mineral exploration was ushered in with the identification of palaeomagnetic reversals across the seafloor and discoveries of hot metal-rich springs and poly-metallic sulphide deposits containing gold at seafloor spreading centres. Discovery of visible primary gold grains in white smoker chimneys at active vents in the Lau back arc suggested a close similarity between seafloor hydrothermal processes and processes associated with the development of some epithermal gold ores in volcanic terrains. Many potential metal deposits containing gold, silver copper, lead and zinc have also been identified in volcanic front and back-arc rifts in sea areas outside Japan (e.g. Izu and Ogasawara Arcs). Within these rifts, hydrothermally active submarine calderas exist, which have typically concave features. Seawater sinking through cracks in the seafloor is heated in zones around the magma chamber, rising again to the seafloor as a hot fluid solution containing heavy metals. This phenomenon results in the production of a kuroko-type sulphide deposit abundant in the above heavy metals.

It is thus essential to understand the age relations between mineral deposits and their host rocks to determine if the ores comprise some parts of orogenic belts that may have been transported for hundreds or thousands of kilometres from their place of origin. Key factors in establishing the criteria for finding such deposits are regional exploration programmes adapted to all different geological conditions pertaining to Precambrian Shield versus Phanerozoic orogenic belt, and to the various climatomorphogenic zones; jungle, desert, boreal and glacial (Goossens, 1983). A convenient division may be made on the basis of age and recognition of similarities and differences.

Points of similarity

Points of similarity between Precambrian and Phanerozoic geological conditions include:

- host rocks that provide a zoned pattern of hydrothermal gold-ore formation
- alteration assemblages and mineral suites of the same general type in epithermal deposits of both ancient and modern systems
- mesothermal orebodies of both age groups that are sufficiently alike to be considered as part of a single age-independent class
- features of both Phanerozoic and Precambrian geological environments that associate gold deposits with massive base metal sulphide
- the continental margins of all previous super-continents appear to have been subject to similar type morphogenic controls as are apparent around Pacific Ocean margins today.

Important differences

Important differences between epithermal and deeper-seated source rocks include:

- epithermal style orebodies found mainly in igneous and volcanic-hosted rocks of Palaeozoic and Mesozoic age but appear to be largely missing from rocks older than about 200 my
- generally low preservation potential of epithermal orebodies due to greater exposure to erosion at shallow depths of emplacement in the Earth's crust
- much greater heterogeneity of deposit types within the Phanerozoic (Nesbitt and Muchlenbachs, 1989)
- greater density of lode gold deposits per unit area in the Archaean relative to the Phanerozoic
- less extensive structural pattern of orogenic activity in Archaean times i.e., crustal development on a thinner continental crust without major drift and collision and from the recycling of sediments at continental margins.

Clearly, while many of the basic geological principles apply to both ancient and modern volcanic activity, the larger scale tectonic controls, processes and settings may have been to some extent different. Large areas of the Earth's surface have not yet been properly explored and at the present state of knowledge, most interpretation is necessarily based upon the geological history of the Cainozoic Era and more particularly, on the erosion and recycling of Tertiary placers during rapid climatic changes in the Pleistocene.

5.1.2 Geomorphic reconstruction

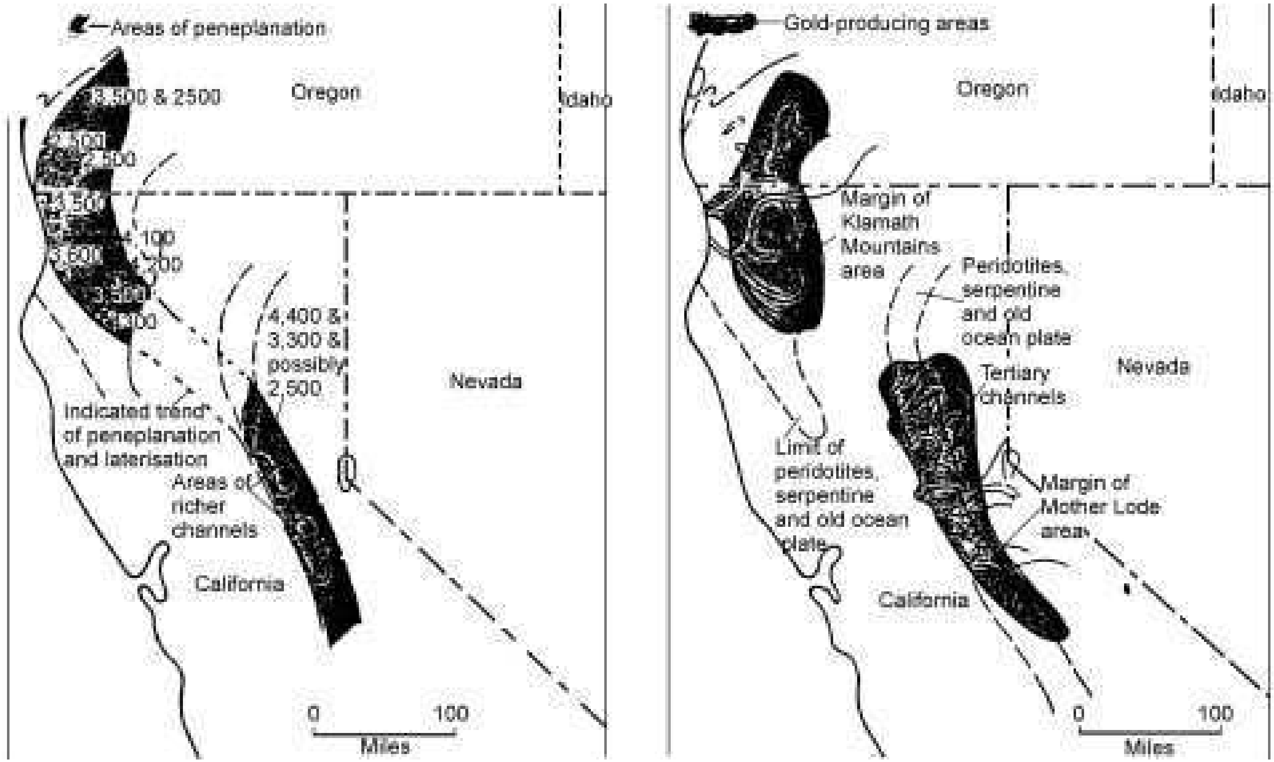
The study of geomorphic reconstruction involves tracing the denudation history of an area or region back to its beginnings. Its purpose is to investigate the nature of past events and thus throw light upon the possible prior existence of geologic conditions in which significant ore deposition may have taken place. Evidence to support the reconstruction will come partly from surface mapping but must finally be supported by drilling and sub-surface mapping. Diagnostic features include evidence of old erosion surfaces that were adjusted to the base levels of their time and aspects of lithology that could relate to periods of active down-cutting or conversely to channel fill. Present channel sediments usually contain reworked gravels in which gold, derived from downcutting of the river into its bedrock is redistributed in a diluted form. Most of the older deposits will have undergone sequences of pulsatory tectonic uplift, superposition of drainage and climatic extremes; often, the paystreak will have been partly or wholly remobilised and redeposited in other forms and in other settings. For many Cainozoic gold settings only remnants of palaeo-drainage systems may still remain as terraces around valley walls.

Palaeoerosion surfaces resulting from peneplanation are represented by strongly persistent unconformity in the geologic column. The peneplane, which represents an erosional surface at the end of an earlier depositional phase, characterises what has occurred across the landscape before the beginning of a new major depositional phase. In favourable circumstances any such well-sculptured surface might be a logical setting for all heavy mineral types, particularly gold. Evans (1981) notes the apparent relevance of peneplanation to large-scale placer formation in the California Mother Lode country and west slope of Sierra Nevada. He suggests that the Klamath Mountains/Coast Ranges and Sierra Nevada foothills both have future potential but only where, as with known deposits, they are coincident with peneplanation (Fig. 5.2).

Elucidating the size and nature of Cainozoic palaeo-channel systems requires a good understanding of climatic as well as tectonic variations at various stages of geomorphic evolution. Douglas (1977) cites the Murray-Darling basin of southeastern Australia as a 'natural laboratory for testing the principles of hydrologic geometry'. Generally, the ancestral streams were much larger than are those of the present; they were also of steeper gradient. Adjustments of many Quaternary river systems resulted from changes in annual precipitation and episodes of glaciation and deglaciation. The Murrumbidgee River, which now transports very little sand was preceded by ancestral channels that were much larger and straighter than the modern channels, and of greater annual run-off and higher flood flows.

Solving such geologic and geographical problems in the field relies for success upon the skills and experience of the exploration team, and its ability to gather and interpret data from both exotic and normal field conditions. The importance of adopting optimum procedures in all phases of an exploration programme cannot be overestimated. Eliminating large areas quickly on the basis of results from indirect methods such as remote sensing and geophysics, in order to concentrate on the efforts of small but promising target areas, may ignore the need to field-check non-anomalous and rejected areas at a later date. Goossens (1983) warns of 'the unfortunate but progressive abandonment of recording direct observations by field geologists – perspiring in the jungle or dehydrating in desert areas', and its replacement by the modern use of sophisticated but more indirect methods such as geophysics, remote sensing and geochemistry. It is true, nevertheless, that while the more exotic methods of exploration cannot take the place of fundamental ground studies in the field, they do provide important additional data for consideration by the field geologist. The efforts of integrated exploration techniques (geological, geochemical and geophysical) offer the best chance of rapidly making new discoveries in areas selected by geological reasoning.

All available literature on the region to be prospected is reviewed, including studies of any previous mining activities and collection of all relevant topographic maps and remote sensing data. At this stage, the geologist will



5.2 Connecting areas of peneplanation with gold-producing areas. Sierra Nevada, California (from Evans, 1981).

familiarise himself with the geology and geography of the environment within which he will be working. Operational sequences can then be defined and linked together in a logical time frame to ensure that priorities are identified and that efficient use is made of the available resources of manpower, money and equipment.

With the indication of a favourable source rock environment the next stage is to identify and map the geological features of deposit size targets by their geophysical and geochemical expression and/or by remote sensing. The individual ore zones will then be examined in a ground geological survey including geological mapping, geochemical sampling, pitting and trenching. The boundaries of potential areas, where a reasonable probability exists of economic development, can then be defined for more detailed exploration. Hodgson (1986) proposes geological criteria for selecting areas of gold mineralisation in belts dominated by volcanic rocks in Table 5.1.

Account must also be taken of the variable nature of past climates and the wide range of timescales within which individual changes may have occurred. The problem is to recognise those elements of the landscape that were adjusted to the base level at the time of their formation and to fix their elevation relative to the present base level. In a valley-fill sequence, separate deposits may exist at different levels and any apparently significant features should always be observed at a number of different points to help avoid mistaking local for regional features. The separate mapping of a sequence of erosion surfaces may be prone to error if based upon stratigraphic succession alone. It is not uncommon for older deposits to undergo a wide variety of pulsatory tectonic uplift, superposition of drainage and climatic extremes, and sometimes the paystreak will be reworked and relocated several times. In a pulsatory but net aggradational succession the lower erosion surface is least likely to have been disturbed by subsequent events and should be mapped first.

Denudation chronology

Investigating the denudation chronology of a residual gold setting entails differentiating between individual elements of palaeo-drainage systems and plotting the sequence of events from the first stages of weathering to the present. The process of fitting individual features into the geological framework requires sub-dividing the rock record into intervals representing small units of time. Historical records are sparse and generally, the older the geological setting the greater the gaps in knowledge and the less easily is its geomorphic record read. Earth probably formed about 4.5 billion years ago and its geology has been changing ever since. It is only possible to relate geological time units in the oldest, Precambrian settings to major episodes of igneous intrusion, metamorphism and mountain building with possible errors as large as 60 million years, close to the entire age of the Cainozoic Era. For the younger Mesozoic orogenic

Table 5.2 Precambrian time units, principal orogenies and major events (modified from Goossens, 1983)

		Orogeny and other major events
Proterozoic Eon		
era I	600–800 mya	Grenville, in America, Kalangan in Africa, followed by the Pan-African-Thermal event
era II	900–1600 mya	Hudsonian, in America (also recorded in Europe, USSR, S. Asia, Australia and Africa)
era III	1600–2500 mya	Kenoran, in America; Ebumean in NW Africa and elsewhere
Archaean Eon		
era I	2500–2900 mya	
era II	2900–3500 mya	
era III	3500–3800 mya	3800 Ma oldest rocks recorded by isotopic age dating
Origin of Earth and Moon	4300–4500 mya	Growth from accumulation of planetisimosis (meteorites and asteroids) after the Big Bang?

belt an overall picture of orogenic mountain building episodes may be developed from the identification of assemblages such as opiolites, blue schist series and melange characteristic of former plate margins or suture zones between two plates. In the youngest, most recent settings an interval of only a few thousand years could be sufficient for the complete reworking, dissection or exhumation of much older deposits. The Precambrian time units, principal orogenies and major events proposed by the International Sciences Subcommission on Precambrian Stratigraphy (1979) (as modified from Anhaeusser, 1981) place the main events from Precambrian to the present including the ‘Origin of Earth and Moon’ in broad historical sequence in Table 5.2.

Phanerozoic geological history dates from about 570 mya at the start of the Palaeozoic Era and extends through to present times. The Palaeozoic was characterised by reduced physical changes and abundance of sedimentary rocks; geological periods were separated on the basis of intervals of continental uplift followed by submergence and encroachment of oceans onto the land. The Caledonian Orogeny reached its peak during the Silurian Period (395–435 mya) and fusion of the continents into a single mass (Pangaea). A gradual fragmentation took place during the Mesozoic Era (235–280 mya) and the continents drifted apart, opening the way for new oceans. The Mesozoic featured extensive sub-tropical weathering and favoured the formation of placers. The Urals were weathered during the Triassic and the first phases of Alpine and Andean folding introduced the great mountain ranges of modern times at about 100 mya.

Age dating

Improved age dating techniques which have been developed for placing the geological events of Earth history in sequence according to age and denudation chronology has become an important aspect of placer exploration. Most known gold resources were already in the continental crust at the end of the Precambrian; and Phanerozoic ores appear to derive largely from the recycling of Precambrian gold and its weathered auriferous material. On a local scale, age dating is used to differentiate surviving remnants of palaeo-erosion surfaces from those of similar type material of a different age. Using this technique, marker beds and other features of palaeo-erosion surfaces can be correlated across the whole of an alluvial goldfield by plotting them to a common datum. Measurements are derived from the decay of radioactive substances in the rocks, fossil magnetism and orogenic evolution.

Radiometric dating

The age of a substance can be measured radiometrically by comparing the ratios of certain radioactive isotopes contained in the specimen. The function of radioactivity results from the instability of the atomic nucleus in certain atoms whereby the nucleus experiences a measurably delayed nuclear transition or transformation with the resulting emission of radiation. The rate of decay is usually stated in terms of half-life, i.e., the time required for the radioactivity of an isotope of radioactive material to decay to half of its original value. Isotopes used for dating the older rocks are listed in Table 5.3.

Radio-carbon dating

The carbon ^{14}C analysis is the best-known technique for dating rocks up to about 70,000 years old, although its accuracy falls away beyond 25,000 years. Its short half-life (5,730 years) makes it particularly applicable to events of the latter part of the Pleistocene. The method can be used on a range of materials including wood, charcoal, shell, bone and various carbonate deposits.

Table 5.3 Radioactive isotopes for dating the older rocks

Isotope	Half-life (billion years)
Rubidium (87)	48.6
Thorium (232)	14.0
Potassium (40)	8.4
Uranium (238)	4.5
Uranium (235)	0.7

Radiochemical dating

Certain aspects of geochemistry involve radiochemical-dating studies, which exploit the high sensitivity of isotope tracer techniques. In this field, the techniques measure minute quantities of matter such as fluid inclusions in rocks and isotope abundance and decay. Spontaneous decay of radioactive atoms in the rocks is a dependable means of timing Earth processes; the rate of disintegration is fixed and does not change in other chemical or physical conditions.

Fossil magnetism

Age dating of rock formations across the seafloor is vividly portrayed by the alternation of fossil magnetism (normal and reversed) in strips of the spreading seafloor. Each strip (or stripe) dates a particular segment of floor and is duplicated on both sides of the mid-ocean rift from which the oceanic plates evolve. Plates spread at an average rate of about 5 cm/y. The magnetism of alternate strips reverses at widely different intervals which may be as short as 50,000 years, but are sometimes as long as a million years. Occasionally, as in Permian times, the magnetic reversal may not take place for as long as 20 million years.

Observation of rock magnetism on land has produced evidence of recent continental drift from remnant magnetism in some of the younger rocks. However, the method has been less effective for dating the older rocks because there is usually some blurring of the geological record by later events. On the seabed itself, the stripes tend to be less magnetic over the older ocean areas, possibly because of chemical action or decay with time.

Organic fossil dating

This method of age determination is applicable to both old and recent rock formations. In certain conditions, the method may be more precise than the radiometric method. The life cycles of the various fossil types and their environments are fairly well known and the relationship between organic evolution and geologic time can be used to correlate widely separated beds of sedimentary rock. Pollen analysis is particularly useful in helping to decipher palaeo-environments affected by frequent climatic change. Being very resistant to destruction, pollen can be identified and counted in such weathered rocks as peat, organic soils and muds.

5.1.3 Techniques and targets for field investigation

Following an overview of geologic, geographic and geochemical data, hard facts are needed to support and where necessary supplement the preliminary geomorphic findings. Techniques include geological mapping supported by stream and soil sediment sampling. Conventional prospecting methods apply to placer gold exploration in regions where source rocks are exposed. Supplementary geophysical and remote sensing procedures may be needed to reveal the potential of areas masked by a cover of later sediments or lava flows.

Potential exploration targets

Some of the more important geomorphic settings that provide potential targets for exploration are summarised below.

Palaeo-erosional surfaces

The schematic portrayal of palaeo-erosion surfaces at various levels of stillstand is a useful tool for predicting the present location and geometry of fluvial gold deposits now buried and hitherto unsuspected. Outwash gravels afford scope for extensive placer development, in both active and buried river channels that flowed through earlier flood deposits. It is not uncommon for older deposits to undergo a wide variety of pulsatory tectonic uplift, superposition of drainage, climatic cyclicity and extremes, and sometimes the paystreak will be reworked and relocated several times. Significantly, the configuration of precursors to younger presently active river channels may not coincide with any subsequent stream or river. As King (1966) points out, 'the problem of the elucidation of the development of a river valley that has been influenced by both a rising and a falling base level is considerably more complex and requires further information'.

In a pulsatory but net aggradational succession fluvial placer deposits that have undergone epeirogenic uplift may occur at various levels above the present stream. This is not uncommon in eastern Australia where older gravels from palaeoplacers may be found on the tops of plateaux, along the margins of valleys that have been incised during uplift, and along marginal to present streams. Tertiary deep leads occur in regions that have undergone epeirogenic uplift along with outpourings of flood basalts covering old river gravels as well as the surrounding terrain. Generally, the basaltic material is 10 to 20 metres thick where it has covered the drainage of the region. In some cases these sub-basaltic placers are worked by underground mining methods. Gold placers may also become submerged in drowned river systems as the result of a rise in base level or a fall in the land surface. The drowned placer system of offshore Nome, Alaska is a classic example (refer to Fig. 4.40).

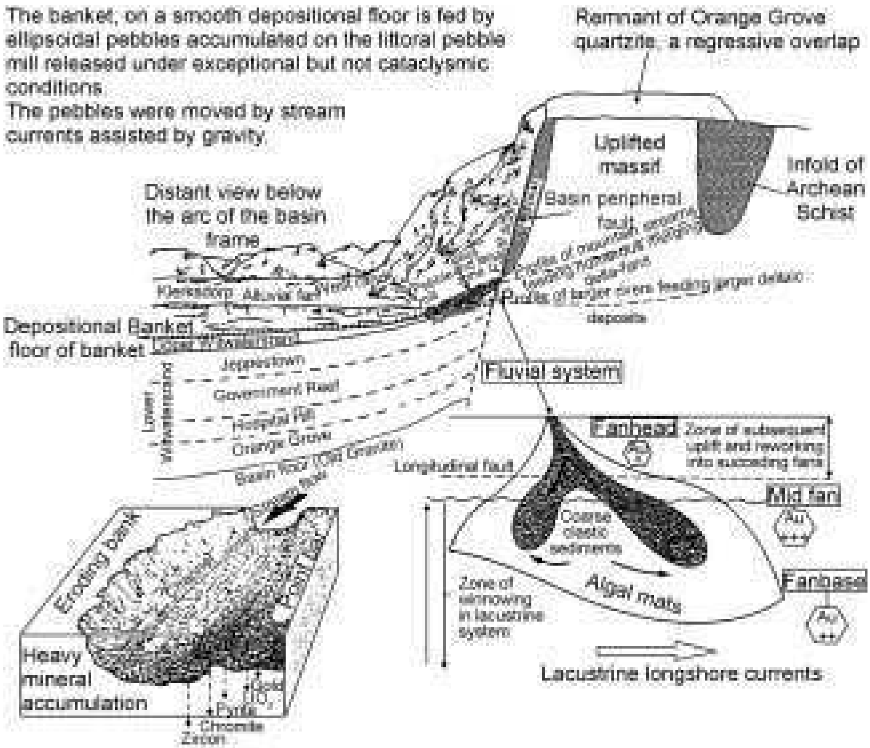
Depletion of most major alluvial gold deposits in pre-existing glacial mining areas such as in northern British Columbia, the Yukon, Alaska and eastern Canada has called for a better understanding of the geology, origin and likely three-dimensional distribution of Quaternary deposits (Eyles and Kocsis, 1989). Work identifying the principal sedimentary controls on the distribution of pay-zones in a late Pleistocene placer, Cariboo Mining District, Canada may form the basis for further exploration in other areas of Quaternary and pre-Quaternary glaciation.

Witwatersrand type depositional sites

The genesis of the Archaean gold deposits of the Witwatersrand Basin of South Africa poses a special problem with far-reaching possibilities for future global exploration. It was apparently developed as a foreland basin during the collision of two late Archaean cratons between 3.1 and 2.7 Ga. Gold enrichment occurred in conglomerate beds that were deposited around the shoreline of an intra-cratonic lake or shallow inland sea at or near to entry points of sediment into the basin. The bulk of the conglomerates are made up of quartz or quartzitic pebbles silicified by pyritic silicified cement. The events of ancient times however, are subject to great uncertainty and the source of the placer gold (detrital or supergene) is a much-disputed issue. The two most popular genetic models are a modified placer model and a hydrothermal model of which, a new type advocated by Phillips and Law (1997) views gold precipitation as being controlled by host rock chemistry, which in turn is controlled by sedimentary features.

Proponents of the detrital theory hold that the gold-bearing sediment is derived from the weathering of quartz lodes and veins in the mafic and ultramafic volcanic rocks of adjacent greenstone belts. Deposition of these materials in distributaries of alluvial fans is thus believed to be a natural consequence of fluvial transportation and sorting. Figure 5.3 is an early conceptual model of a Witwatersrand placer gold field based upon a basin edge concept of the land. This model proposes that debris brought down from the source area underwent sorting on alluvial fans in a closed fault-bounded continental basin that became increasingly unstable with time. A common view of those who favour this theory is that most of the heavy minerals including gold and uraninite were dispersed through fan deposits as crude suspension equivalents of other clastic grains (after Force, 1991). In this concept, the grain size of the gold was too small for it to settle in the fan head and optimum conditions for settling would have been in the mid-fan facies. It is suggested that algae, which grow preferentially in the non-turbulent conditions of flow along the margins and base of the fan, precipitated gold in the distal fan zones (Pretorius, 1976). Precipitation mechanisms involving Fe and hydrocarbons are well known and hydrocarbons in the Witwatersrand reefs were interpreted as algae mats preserved in their original state with minor sedimentary reworking that generated nodular carbon (e.g., Halbauer, 1986).

On the other hand, the widespread alteration of Witwatersrand sediments as previously noted by Emmons (1940) and the possible secondary nature of the carbon support concepts of a hydrothermal origin for Witwatersrand gold. Studies by Parnell (1996) and others propose a secondary origin for the carbon



5.3 Conceptual model of Witwatersrand alluvial gold field, South Africa: top, diagrammatic cross-section (derived from Pretorius, 1976).

in the form of liquid hydrocarbons introduced into the sediments at some time after burial. It is suggested that wall rock sulphidation would have been encouraged by the presence of Fe in these units, thus destabilising the gold-sulphur complexes causing gold to precipitate. Since iron is essentially a sedimentary feature, weathering under an oxidising atmosphere may have generated pisolites and other Fe-rich nodules in the soil profile. Carbon-rich units could then have caused fluid reduction and gold precipitation.

The model advocated by Phillips and Myers (1989) explains the development of high-grade mineralisation in the sediments as a consequence of the introduction of auriferous metamorphic fluid into the Witwatersrand basin. It is suggested that this fluid is similar to the fluid type indicated for greenstone and some slate deposits, and that iron and carbon reached their present-day sites by sedimentary and diagenic processes prior to peak metamorphism and gold mineralisation. Additionally, while previous analyses of gold from these ores were thought to indicate the primary nature of the gold grains, almost all petrographic studies of polished sections point to the gold as being of secondary origin. In this respect, Phillips and Law (1997) suggest that the method used for grain liberation, which involves centrifuging and stirring should be examined more closely. The possibility of grain rolling and abrasion during sample preparation would invalidate any exercise confirming a primary source for the gold grains by shape alone. Barnicoat *et al.* (1997) examined the gold in Witwatersrand reefs subsequently in a major study. Their reports stated that 'none of over 40,000 gold grains examined in this extensive study occurs in textural sites at the time of sedimentation'.

The assumption of a hydrothermal origin of the Witwatersrand gold also gains support from a theory put forward by Enderbee (2002) that the origin of ground waters in the Great Artesian Basin of Australia may have originated as part of the original constitution of the Earth. Enderbee suggests that these waters may have a similar source to the steam that explodes from volcanoes and the hot acid waters that gush from deep ocean vents. The Great Artesian Basin adjoins a long and wide zone of recent volcanism extending from Torres Strait to Tasmania. The concentration of nearby metalliferous mines at Broken Hill, Cobar, Mt Isa and Mt Morgan appears to be due to the deposition of metal sulphide deposits from hot sulphurous solutions deep in the Earth. In Enderbee's view, rising volcanic waters over millions of years were blocked under the impervious layer of sediments in the Great Artesian Basin during a period of volcanism extending over a period of 50 million years. His thesis that the ground waters of the basin were relict of the period of volcanism that extended over the past 50 million years opens up the similar possibility that steam from exploding volcanoes and hot sulphurous waters may have provided a hydrothermal source for the Witwatersrand gold. Refer to Table 2.6 in respect of the nature of the White Island hydrothermal fluid system.

Mapping the regolith

The required first step in geochemical mapping is to map the regolith-landform relationships and develop a regolith model for the area. This model should provide information on the nature, distribution and genesis of the regolith units in the area under review. Its purpose is to represent three-dimensional regolith-landform relationships that may be largely predictive in pointing to genetic interpretations. These relationships are used to design sampling strategies and are commonly presented as cross-sections and block diagrams (Anand, 1996).

Regolith geochemistry deals with the chemical and physical processes of weathering, soil formation and landscape development. It thus relies upon the determination of weathering history and recognition of the possible effects of environmental change on residual gold ores deposited under one set of climatic conditions and dispersed and reconcentrated under another. Here again, the number and types of variables and their interchangeability is practically limitless. Time periods during which changes occur may total many millions of years. Older gold-bearing sediments, already with a long history of geomorphic

Table 5.4 Principal groupings of geochemical exploration models (after Butt, 1997)

Glaciated terrains		
Dispersion influenced by effects of Pleistocene glaciation		
Present climate	*Past climates*	*Morphoclimatic zones*
Glacial	Glacial	Polar, periglacial
Temperate	Glacial	Periglacial, boreal
Mediterranean	Temperate, glacial	Boreal
Tropically weathered terrains		
Dispersion influenced by effects of deep tropical weathering		
Present climate	*Past climates*	*Morphoclimatic zones*
Mediterranean/sub-tropical	Savanna	Sub-tropical
Warm arid	Savanna	Warm arid
Savanna	Savanna	Peritropical
	Arid, savanna	Peritropical
	Rainforest, savanna	Peritropical
Rainforest	Savanna	Inner tropical
	Rainforest	Inner tropical
Erosional and youthful terrains		
Dispersion influenced by effects of present climate		
Models classified according to climatic zone and relief		

change, are frequently subjected to fresh cycles of tectonic uplift, thereby exposing them to erosion then further deposition.

Table 5.4 envisages three particular regions, which have broadly similar histories of weathering and landform development in terms of the effects of active geomorphic processes on pre-existing regoliths and relief elements. Glacial and tropical deep weathering are recognised as the most important agents of geochemical dispersion, particularly in areas of low relief. Erosional and youthful terrains and areas of high relief are of lesser importance. Included are pre-existing environments in which regoliths are absent, and dispersion is related only to the present environment. Two climatic regimes are interpreted to be of particular importance:

- a humid, probably temperate, warm to tropical climate similar to climates prevailing in the wetter savannas of today
- a second, more recent climatic regime associated with some minor uplift that produced an arid to semi-arid climate.

Soil profiles represent a vertical arrangement of soil horizons down to the unweathered bedrock in regolith dominated landscapes. The main horizons provide evidence of the physical, biological and chemical processes involved in their formation. The effectiveness of lateritic residuum as a sample medium is due to the relative stability of many of its geochemical characteristics. Samples taken from the pisolitic upper part of the residuum at the reconnaissance stage of exploration can take advantage of enhanced mechanical and hydrochemical dispersion and homogenisation (Anand *et al.*, 1989). Follow-up sampling is then directed at the basal part of the residuum to locate the source of the mineralisation.

Delineating the source mineralisation within a lateritic geochemical anomaly provides a better understanding of source rock geology and helps to pinpoint favourable locations for drilling where surface indications are lacking. Both costs and difficulties of intersecting and delineating buried geochemical haloes can be minimised by drilling only though the most prospective areas of transported sedimentary cover. Lateritic residuum (lateritic gravels, duricrust) is an important geochemical medium and its presence or absence must be established. Sampling of laterite at or near the surface is generally straightforward; the deeply buried situations require skill in recognising the different units of the regolith stratigraphy.

It is also important to delineate areas of substantial sediment cover:

- The relict regime is characterised by widespread preservation of a lateritic residuum consisting of relicts of an ancient erosional surface; it should be sampled in preference to soils because geochemical anomalies in the lateritic residuum are relatively larger and more consistent in relation to the primary ore deposit.
- The erosional regime is characterised by removal of the lateritic residuum to a level at which the mottled zone, clay zone, saprolite or fresh bedrock is either exposed or thinly covered beneath soil or locally derived sediments; ferruginous saprolite or soil horizons should be sampled in erosional regimes.
- The depositional regime is characterised by widespread sediments that may be several metres thick; broad Au anomalies and anomalous concentrations of ore-related elements in buried lateritic residuum can be effective indicators of primary and supergene mineralisation in depositional regimes.

5.2 Exploration geochemistry

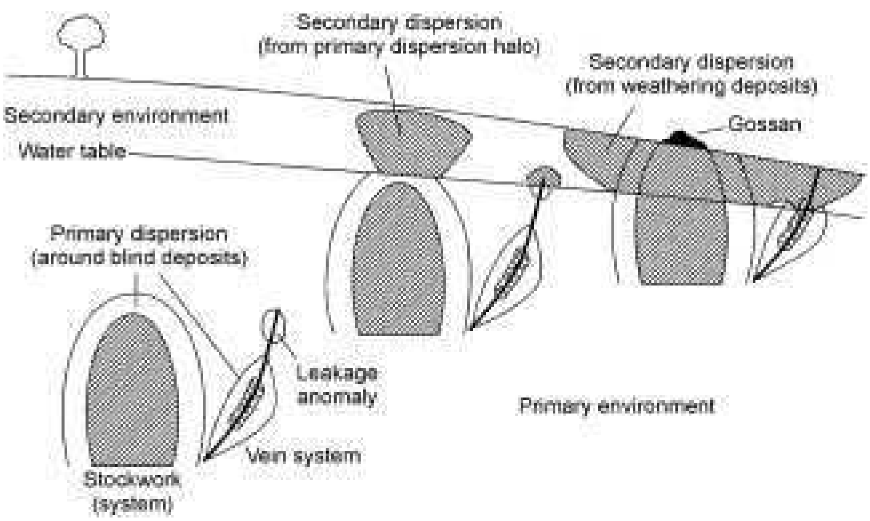
Geochemical exploration studies comprise investigations into the chemical mobility and distribution of gold and associated elements in both surface and near surface soils, and in deeply weathered regolith. The initial studies involve geochemical mapping supplemented by stream and soil sediment sampling. Control samples are taken of the various different types of material in the prospective area to determine the levels of background geochemical responses from zones of potential value and those which are likely to be barren. Increasing attention is being given to the geochemical analysis of ground waters, vegetation and of any naturally occurring material that may be present.

5.2.1 Geochemical dispersion

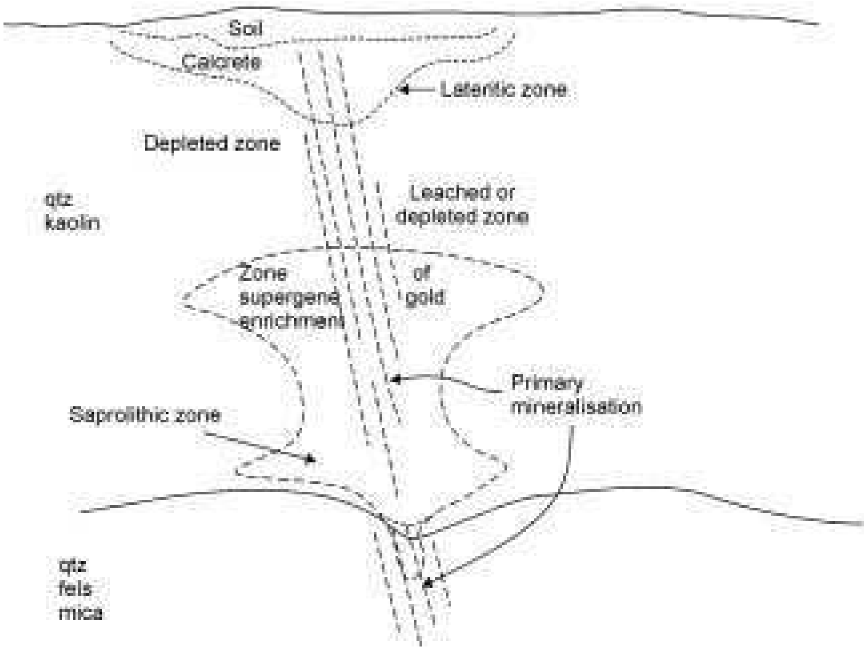
The nature and extent of weathering and secondary dispersion associated with gold mineralisation depends upon the type of deposit and the environment in which it occurs. Dispersion occurs as an outward spread of elements from both primary and secondary sources. The resulting geochemical haloes contain concentrations of elements that are intermediate in grade between the primary source rocks and those of the enclosing rock systems. Intensive research activities include regolith geochemistry (Butt, 1988); geochemical dispersion (McQueen, 1997); hydrogeochemical exploration (Gray, 1997c) and regolith-landform relationships, and regolith stratigraphy (Anand, 1996). One successful case has been the identification of a supergene halo at 60 to 70m depths from the ground surface in the Kanowna Belle Au deposit, northeast of Kalgoorlie Western Australia (Anand *et al.*, 1989). Figure 5.4 is a schematic illustration of the generalised nature of the primary and secondary dispersion pattern around an orebody; a typical example of primary and secondary gold enrichment in deeply weathered lateritic is sketched broadly in Fig. 5.5.

Primary dispersion

The geochemical dispersion of gold and other elements away from their primary host rocks results from the passage of ore fluids and diffusion of components along fracture systems in unweathered rocks of the primary environment (see Chapter 2, Section 2.3). It thus precedes mechanical disintegration in igneous



5.4 Generalised nature of primary and secondary dispersions around ore deposits (McQueen, 1997).



5.5 Primary and secondary gold enrichment in deeply weathered regolith.

rocks during the deuteric stage of crystallisation and is referred to by McQueen (1997) as syn-depositional dispersion. The extent, shape and intensity of the primary dispersion haloes depend mainly upon the ore-forming processes involved. For example, anomalous primary geochemical dispersion in the bedrock would occur over a much more extensive area in the case of a typical porphyry-copper-gold deposit than if associated with gold-only porphyry.

Patterns of primary gold dispersion along vein systems commonly result from the passage of ore fluids and diffusion of components along fractures or porosity in the host rocks. The discharge of ore fluids onto the seafloor can result in differential precipitation of the ore components laterally away from the vent in response to changing temperature, mixing with seawater and oxidation (McQueen, 1997). Large scale Ni-Cu deposits with subordinate Au commonly develop a halo of disseminated sulphides around massive or semi-massive ore, particularly in the hanging wall in cases involving gravity settling. Extent, shape and intensity of primary dispersion patterns depend largely upon the nature of the host rocks and type of associated structures. McQueen lists the following major ore-forming processes:

- magmatic concentration
- contact metamorphism
- hydrothermal deposition and replacement

- volcanic and hot spring exhalation with sedimentation
- sedimentary deposition
- residual concentration during weathering
- low temperature dissolution and deposition
- mechanical concentration
- metamorphic recrystallisation and remobilisation.

Primary dispersion patterns associated with hydrothermal deposits are divided into:

- deposits in which solutions deposit ore minerals in hydrothermal vein cavities and stockwork structures
- deposits related to the movement of hydrothermal fluids through the host rocks with replacement of silicates and carbonates by new gangue and ore minerals.

In both types of deposit the primary dispersion patterns are related to the movement of hydrothermal fluids through the host rocks. Individual responses are due mainly to the distribution of particular element suites along the flow path, which reflect differences in temperature and pressure relationships, progressive reaction with wall rocks and reactions with meteoric fluids of different pH, Eh, composition and temperature.

The style of hydrothermal mineralisation influences the nature and extent of the dispersion patterns. In large-scale stockwork systems in which ore minerals have been deposited in structures and pore-spaces the movement of hydrothermal fluids can develop primary dispersion on a major scale. Wall rock alteration patterns have been extensively studied and described for porphyry copper and epithermal gold-silver systems. For elements such as Cu, these zones of alteration may sometimes represent exploration targets greater by order of magnitude than the primary orebody.

Epithermal high sulphidation and low sulphidation gold systems are illustrated schematically in Chapter 2. Many of the rock-forming minerals associated with epithermal high sulphidation and low sulphidation gold systems will show incipient alteration with chemical changes taking place along structure planes of the mineral. Feldspars are often partly kaolinised; mafic silicates are slightly chloritised and so on. Replacement minerals formed by chemical action are larger than the original minerals thereby causing slight expansion and thus aiding later weathering processes and dispersion in the secondary environment.

Secondary dispersion

Secondary geochemical dispersion occurs as the result of weathering, erosion and hydromorphic processes in the near surface environment above and around the water table (McQueen, 1997). Secondary exploration techniques have evolved from the recognition of geochemical signatures of mineralisation in leached outcrops and residual soils towards targets that have been masked by transported overburden or barren weathered zones.

Typical examples of secondary geochemical dispersion mechanisms are given by Cohen *et al.* (1996) in his review of exploration techniques used extensively for gold and base metals in the Cobar Region, NSW, Australia. The Cobar region is part of an extensive palaeoplain (the Cobar Peneplain), which evolved while Australia was part of Gondwanaland. This feature has been preserved as part of the western margin of the Canoblis Divide and is bounded to the northwest, north and east by the late Cainozoic Darling Riverina Plain. Local zones of the more resistant lithologies (e.g., sandstone and conglomerate) rise to about 50 m above the gently undulating landscape. Fluvial quartz-rich remnants are associated with north-flowing palaeo-drainage lines of probable Cretaceous age. Weathering that may extend below 100 m has typically produced a quartz-kaolinite saprolite assemblage. Palynological evidence indicates that humid conditions extended until the mid-Miocene when a change to more arid conditions occurred during the late Miocene and mid-late Miocene with further adjustments during the Pliocene and Pleistocene. Robertson and Tayler (1987) have described depletion haloes in fresh rock around primary zones of mineralisation.

Cohen *et al.* (1996) group Cainozoic secondary dispersion processes into three main periods: pre-mid-Miocene, mid-Miocene–late Pleistocene and post-Pleistocene. Pre-mid-Miocene climates were principally humid with a probable high and fluctuating water table. Under these conditions predominantly hydromorphic geochemical dispersion would have resulted in the formation of thick zones of gossan containing a wide range of oxide minerals in oxidising orebodies and adjacent wall rocks. Large pyrite-rich orebodies with wall rocks displaying high fracture permeability typically give rise to large secondary dispersion patterns in the weathering profile down hydraulic gradient, with relatively immobile elements such as disseminated pyrite, galena and sphalerite preserved at the surface.

Mid-Miocene–late Pleistocene climates were less humid and marked by episodes of aridity during the Pleistocene glacial advances. Climatic change, together with spasms of uplift caused by minor crustal adjustments led to partial stripping of the thick and indurated weathering profiles. Surface lags were formed and mechanical dispersion of gossan and mottled zones occurred. Leaching of the gossan followed lowering of the water table and increasing salinity of the ground waters (Chaffee and Scott, 1995). In gossans this led to deep-seated supergene enrichment. Residual gossans were depleted of metals to a degree depending upon the amount retained after initial oxidation and the mobility of elements in the prevailing environment. In desert areas residual geochemical patterns in surface soils were significantly diluted by aeolean components.

Post-Pleistocene climates were generally humid. Continued silting of the drainage system resulted in dilution of the mechanical and hydromorphic patterns, although limited lateral hydromorphic dispersion may have taken place near the intersection between the water table and oxidising ore bodies. An upward movement of metals occurred to an undetermined extent as the result of evapotranspiration. Lags have been re-exposed by deflation in erosional landforms associated with recent pastoral activities.

5.2.2 Stream sediment sampling

Stream sediment sampling is the oldest and most successful technique for gold exploration and is still the principal technique for ground reconnaissance. Once a simple procedure, prospectors with shovel and pan traced the gold upstream until the colours cut out, and then worked their way upwards along the valley sides in search of the primary source. Early placer finds led to the discovery of major gold lode systems such as the Mother Lode of California, USA and the Bendigo-Ballarat deposits of Victoria, Australia. More recent major discoveries include the C.R.A. Kelian deposit, Kalimantan, Indonesia; and OK Tedi and Porgera, Papua New Guinea.

Today, with new deposits becoming harder to find, exploration is now more geologically orientated and the adoption of improved methods of analysis and interpretation has led to increased specialisation in the fields of geochemistry, sedimentology and geomorphology. New discoveries require more detailed lithological and mineralogical information to elucidate relevant features of the geology of source rocks and surrounding areas. Many of the recent residual gold discoveries would have been made much earlier if there had been an understanding of the meaning of surface features of alteration, oxidation and lateritisation. Since provenances of placer deposits comprise the host rocks of their source areas, all past and present processes that relate to the modification and release of gold grains in the weathering zones of orebodies are critical to investigations. A detailed knowledge of the surface and near-surface geology of a source area (petrology, structure and geological history) is thus of inestimable value to both primary and placer gold explorationists. Neither primary nor alluvial gold surface features can be studied in isolation without neglecting possibly vital evidence from the other.

The applicability of the method stems from the ability of the gold to survive in its elemental state under conditions in which many of the associated minerals are either altered or destroyed. By investigating any areas of anomalous gold content in stream sediments adjacent to the main area of interest the technique thus provides a rapid means of determining the various points of entry of detrital gold into a drainage system. This will usually allow major surface and near surface gold features to be identified at various levels above a valley floor. Additionally, it may also help to determine the areal extent of the primary anomalous zone, particularly in marginally economic areas. In terms of scale of operations and life expectancy, any submission made to investors will be of much greater interest if the exploration costs can be spread over several deposits.

Gold-rock paragenesis

Many placer minerals including gold have generally predictable gold-rock paragenesis, and regardless of age certain indicator rock-forming minerals act as pointers to particular rock types that may represent the source or origin of the gold. Although gold is its own best geochemical pathfinder and field estimates are predominantly based upon visible gold, a representative suite of both bedrock and detrital samples should be included for mineralogical study. Other common element associations may assist in deposit detection at the more detailed primary ore deposit scale. Pathfinder elements for gold include As, Ba, Bi, Cu, Ag, Te and W. Arsenic is the best-known indicator for gold because of its common association with arsenopyrite or arsenic-bearing pyrite.

The above paragenetic relationships epitomise the concept of provenance representing as they may do the source or place of origin of minerals found in a sedimentary setting including both economic and non-economic constituents. The unstable varieties disappear quickly downstream, although some may be carried on in clasts both from within and outside of the source area. Most, however, are found only in close proximity to the source, where rock fragments have sharp edges and particle size is at its maximum level. Away from the source, detrital minerals found with gold in the regolith are mainly non-economic silicates or silica (quartz). Associated mineral grains of possible economic value include platinum group elements, rutile, zircon, monazite and gemstones.

Quartz is dominant in most mineralised zones. Other important minerals are Ca, Fe, Mg carbonates; and sulphides (pyrite, arsenopyrite and chalcopyrite, and less commonly galena and sphalerite). Such minerals as graphite, scheelite, pyrrhotite, tourmaline and tetrahedrite usually appear in trace quantities only. Gold in mesothermal orebodies is paragenetically late (Nesbitt and Muchlenbachs, 1989) and is commonly associated with quartz, carbonate, galena and sphalerite, plus or minus tellurides in fractures in early sulphide orebodies. Table 1.2 (Chapter 1) lists the chemical composition and physical characteristics of the chief heavy minerals found in gold placer deposits (Information Circular #6786 United States Bureau of Mines, Table 3).

The panning method

Methods of screening and hand panning rely initially upon thorough puddling of the material by hand and an oscillatory movement to disturb the slurry in the pan. A swirling motion, with the pan held under water, continuously washes the top layer of light particles away until only heavier particles remain. Conditions of thin film separation prevail and very fine gold particles remain wholly within the boundary layer wherever they settle across the base of the pan. They are dislodged only when impacted upon by other particles swirling around the bottom of the pan and can then be made to segregate and form a 'tail' of progressively finer grains. Hand panning is a tedious process requiring continuous concentration by the panner and close supervision if the operation is to be carried out as efficiently at the end of a shift as it is at the start. There are many constraints to the process and individual panners are unlikely to be equally skilled or similarly conscientious in recovering fine gold.

Globally, gold pans comprise variously sized and shaped units made of metal, plastic and wood in various communities (Fig. 5.6). Different types of pan having different abilities to recover finely divided gold are used in different countries. The standard (California type) metal pan is flat bottomed and circular with sides tapering outwards at about 30º from the base. Sizes range up to about 35 cm diameter at the base and 15 cm in depth. Originally fabricated from light gauge iron the metal has been largely replaced by plastic, moulded to the Californian shape. Plastic pans produced in colours that highlight the gold are said to be quite as effective as the metal pans if not more so. The visual identification of fine particles of gold is enhanced in some light conditions by the colour contrasts offered. The basic pan dimensions have remained fairly constant although experimentation has resulted in a number of different patterns of grooves pressed or moulded into the sides. The grooves are supposed to catch any very fine gold that might otherwise pass out with the sand, but are of doubtful value.

Much greater size and shape differences are found in wooden pans used by local miners in the more primitive regions of the world. The following design differences probably stem from the different characteristics of the native trees from which the pans were carved. In Indonesia, the pan 'dulang' is gently curved; with the pan held horizontal, a swirling motion is imparted to the slurry to bring the lights to the top. In Malaysia the 'dulang' is more conical in shape and the motion is oscillatory. In India the wooden pans are elongated and, as with the Australian aborigine yandi described by Macdonald (1983b), the motion is reciprocating, with an element of forward as well as sideward movement. In South America the 'batea' is generally similar to the conical pans used in parts of South East Asia but is larger, having the dimensions 500 mm × 70 mm deep, capacity 4.3 litres (approx 240 pans/m^3). Because of these differences, the panning techniques also vary although the same principles apply to their use. Typically, because a great deal of attention must be paid to recovering the last vestige of gold for economic survival in some remote areas, these pans appear to recover finely divided gold more effectively than do the California types (see Fig. 5.6(d)).

Site selection

Although the practice of taking stream samples from obviously favourable locations may appear to violate certain well-established rules relating to randomness in sampling it is preferable to obtain gold-bearing material from intentionally biased sampling than to reject a prospect prematurely, simply because some

(a) Madya Pradesh, India

(b) Nilambur Valley, Kerala, India

5.6 Diversity of pan types.

(c) Melawi Valley, Kalimantan, Indonesia

(d) Lower Waria River, Papua New Guinea, California-type pan

5.6 Continued

randomly spaced samples were barren. Reconnaissance sampling based upon a predetermined geometrical pattern may often be misleading and geological judgement and experience is needed to obtain samples from sites where the right type of material is present rather than at some fixed location.

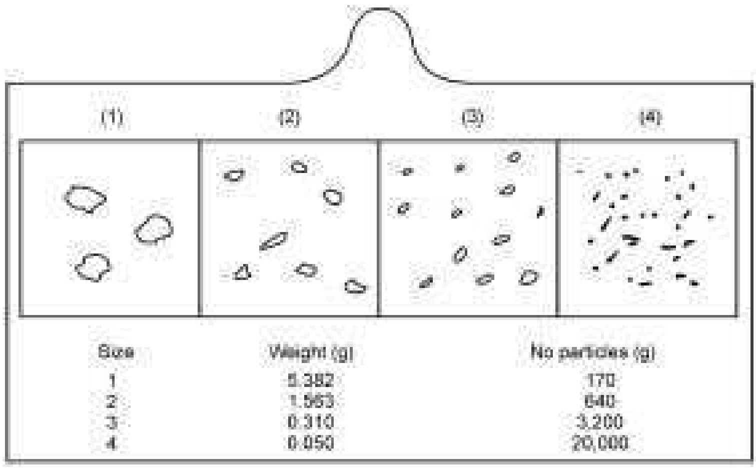
Favourable sites for sampling are found in high-energy zones (as evidenced by accumulations of coarse sediment) and in locations where rock bars or other natural traps for heavy minerals have interrupted the flow. Gold occurs in very small quantities and the finer silts and sediments rarely display visible gold even though adjacent gravel beds may yield colours from every dish. Nichol (1980) demonstrated justification for careful sample site collection at Tran Kruk, Suriname. Panned concentrates from gravel zones in one location displayed up to 12 colours of gold/dish, whereas heavy mineral concentrates from adjacent sand samples lacked any visible gold.

There is no common standard of site selection or sample density except that in general terms the sample interval should be geologically related. For example, the interval should be reduced at all intersections of tributaries with the main stream. Increasing the sample density in the main stream above and below each confluence will make it possible to gauge the effects of an influx of sediments from the smaller stream. Tributaries carrying gold will enrich the sediment just below the confluence of the two streams and should be sampled progressively upstream to identify the source of the mineralisation. Rapidly falling grades downstream of the confluence will usually be indicative of a barren tributary. Nevertheless, a few samples of sediments in any barren tributaries should be taken to obtain an overview of the geology of the basin.

An early appreciation of the likely grade of significant sections of the deposit is useful. A suitable standard for comparing the yield from each sample can be prepared by sizing a sufficient number of gold particles (say 400–500) into four separate size fractions using a microscope with lenses fitted with a selected range of graticules. The weight of each fraction is determined and divided by the number of particles to give the average weight per particle for each size distribution. Specimens are pasted upon a plastic protected card as shown in Fig. 5.7. Comparing the size and number of particles recovered from a composite sample against their counterparts in the standard will provide a visual estimate of the sample grade. Note that the standard must be prepared from gold taken locally from rocks associated with the same pulse of mineralisation in the same area. Gold from an outside source could have significantly different physical and chemical properties and lead to false economic interpretations.

Sample size

The minimum allowable sample size in any particular setting is a function of gold abundance and particle size distribution. Larger samples are required for coarse gold than for fine. For large particles, only one sample of ten equal



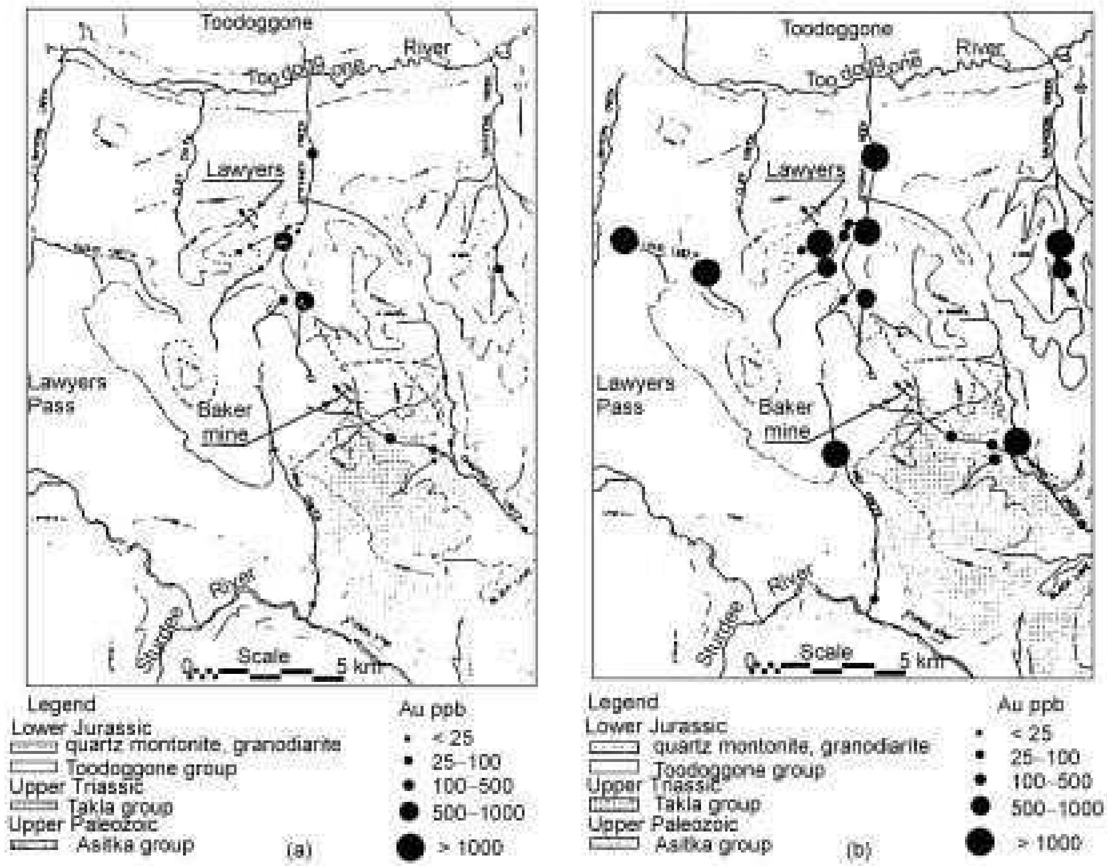
5.7 Gold particle standard for visual comparison.

samples could be expected to contain gold and that fraction would report a value of ten times the average value of the sample. An equal weight of fine-grained gold would probably report in each fraction at about a common level.

In practice, loose sample volumes of about 0.01 m^3 usually yield a measurable quantity of gold in stream sediment samples, although samples taken from zones of a streambed containing anomalous trash should be always be check-sampled to confirm their validity. The improved response in terms of length and strength of anomalous trash in panned concentrates relative to stream sediments is shown in Fig. 5.8. If there is still any doubt, a large enough sample size should be chosen to provide a constant heavy mineral fraction.

Sample spacing

Achieving a satisfactory level of sample representivity is largely a matter of good site selection and adequate sample size for the local conditions. Having regard to the size of the target and other practical considerations, requirements of sample distribution vary at different stratigraphic levels and with distance from source. For high-energy stream sections where fine particles are swept away, and only the coarser gold grains settle out, failure to find gold in the first dish should not discourage further sampling. It may be necessary to test the sediments in a number of different locations before deciding whether or not to continue sampling upstream. Field requirements based primarily upon determining the presence of sufficient gold to justify continued exploration are less exacting than those used later to calculate the ore reserves (Chapters 6 and 9). Any attempt to attribute absolute gold values from reconnaissance sampling is meaningless in the broad context of evaluation. Overall, reconnaissance pan samples are considered sufficiently representative if they indicate the relative



5.8 Comparison of gold dispersion in (a) stream sediment, and (b) in panned concentrates (after Barakso and Tegart, 1982).

abundance of gold in terms of low, medium and high. The main criterion of representivity is achieved by a spread of samples in which each group of samples fairly represents the material of upstream derivation in an unbroken sequence up to the source. Representivity should apply both to the gold content of the sediments and to the constituent parts of the sediments. Each anomalous dispersal train in the drainage should be evaluated separately.

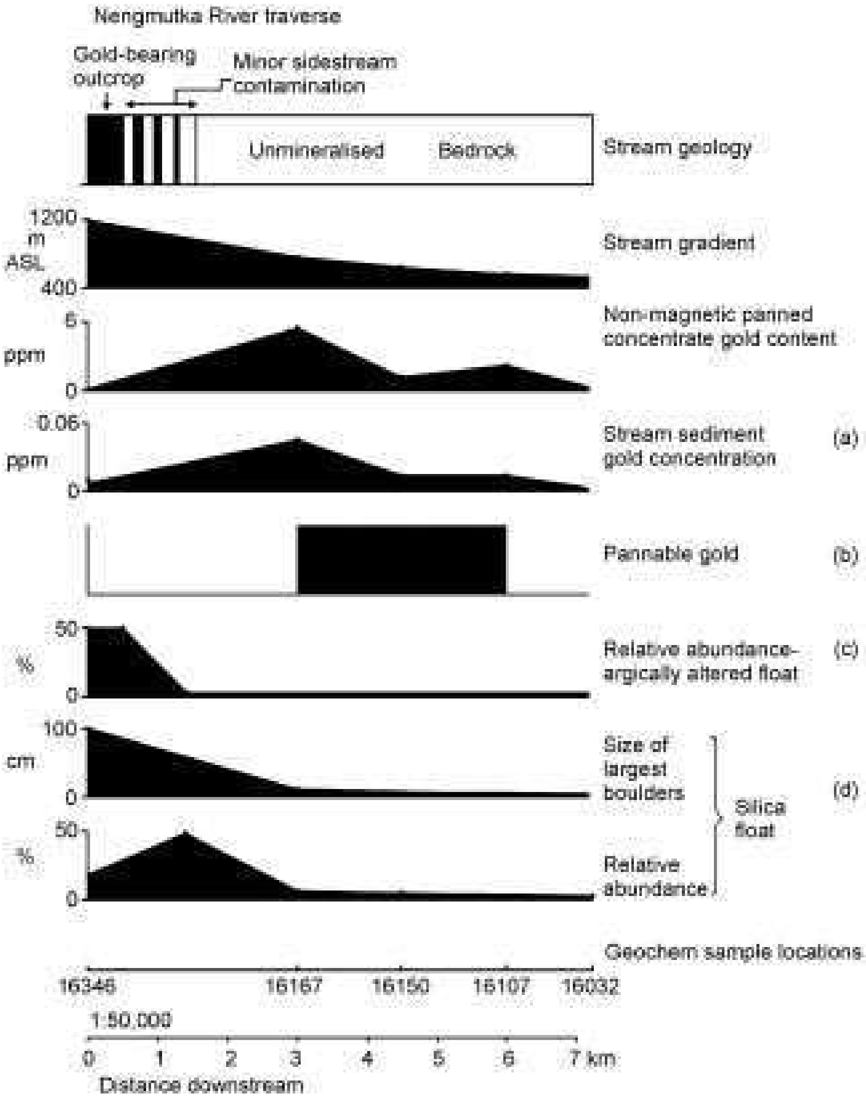
Case history – Nengmutka River area

Confusing features of gold/mineral relationships may require more detailed examination of the float than is usually undertaken. This was demonstrated by Lindley (1983) in one section of the Nengmutka River Area, New Britain, Papua New Guinea where there appears to be a direct relationship between the mechanical breakdown of silica boulders and the first appearance of visible gold. First indications of the Nengmutka River system were observed when reconnaissance sampling indicated anomalies in all tributaries covering an area of at least 30 km^2 of the Nengmutka River.

Panning was carried out in the 'Wild Dog' area of the river for the presence or absence of gold; the concentrate was split into magnetic and non-magnetic fractions. A favourable 'float association' had previously been made 20 km downstream of the area. Panning provided visible, although fine gold in the Nengmutka River alluvials and detected geochemical gold anomalies in the concentrate (122 ppm Cu, 05 ppm As, 2.24 ppm Au) and silt; detection limit and threshold for concentrate results was 0.005 ppm and 0.10 ppm respectively. The attrition rate of the brittle silica and soft argillic float was such that very little anomalous material remained in the bed-load after 3 km of downstream transport. After a distance of only 1.5 km the proportion of silica-kaolin float reduced from about 50% to less than 5% of bed-load. Silica boulder size also decreased rapidly downstream.

Two parallel zones of silicification were defined in the source area, the largest being 8 km long and 800–1000 m wide. Both zones are associated with argillically altered cap zones comprising fine silica, kaolin and alunite with lateral and underlying silica veins with widths to 20–30 m. The veins are auriferous and the gold appears with pyrite in fine hairline fracture coatings. Since the primary gold is very fine grained in the 30 m wide 'Wild Dog' quartz vein and is locked up in the pyrite structure, there seemed to be a need for at least 3 km of transport before aggregation is sufficient to produce visible gold grains. One sample with only one grain of visible gold returned an assay of 5.41 pm Au, thus indicating a significant proportion of invisible (to the eye) gold.

Figure 5.9 (a), (b) and (c) summarises the data for the Nengmutka River area, showing the usefulness and limits of the available reconnaissance techniques in



5.9 Summary of sample data from Nengmutka River, Papua New Guinea (from Lindley, 1983).

the 1 to 6 km downstream intervals. The Nengmutka River traverse shows the downstream gold content of samples by (a) geochemical analysis and (b) by panning alone. The downstream variation of relative percentages of argillaceous and silicious float (c) making up the total stream load is shown in the same figure.

5.2.3 Geochemical soil anomalies

The reverse procedure to stream sediment sampling involves predicting the possible presence and whereabouts of buried gold placers in earlier drainages and of secondary deposits of saprolitic and lateritic gold ores resulting from intense tropical weathering of surface exposures of appropriate rock types that have been deeply eroded. The cycle of erosion of a lateritic gold orebody begins with the formation of eluvial and colluvial gold concentrations on slopes in steep rugged terrain. Erosional forces are dominant and at first there is only time for a recent soil to develop over a transient and shallow zone of weathering. With continued erosion and decreasing relief however, both the depth of weathering and thickness of the regolith increase. Chemical weathering becomes the dominant erosive force and over long periods of geological time processes that refer to past regimes of long duration, to extreme climates or recent climates will establish many of the geochemical characteristics of the regolith. Recent advances in lateritic geochemistry have made possible the determination of weathering history over long periods of geological time (10 to 50 million year time periods) and of the changing climates of those times. As a result, lateritic residuum is now considered a highly effective sample medium for exploration geochemistry for concealed mineral deposits (Anand and Smith, 1997), and on a regional scale may result in the delineation of geochemical provinces and chalcophile corridors of strategic importance (Smith *et al.*, 1999).

Research has shown the merit of wide-spaced reconnaissance sampling of remnants of ferruginous saprolite, ferruginous mottles and iron segregations in areas where lateritic residuum has been removed by erosion. In Western Australia, the humid, sub-tropical conditions during the Cretaceous to mid-Miocene gave rise to extensive deep lateritic weathering. The lateritic profile was modified by more recent arid to semi-arid climates with some minor uplift. Wearing down of the landscape leads to peneplanation, and ultimately sediments of all types overlie the saprolite (Butt, 1997).

On a local scale, reconnaissance testing of lateritic residuum is typically directed at the pisolitic upper part of the residuum in order to take advantage of enhanced mechanical and hydrochemical dispersion and geochemical characteristics (Anand and Smith, 1997). Sampling to locate the source of the mineralisation within any lateritic geochemical anomaly may then be extended to the basal part of the residuum, which may be lateritic duricrust. Buried geochemical haloes, some of ore-grade can enlarge target size perhaps five to ten times in size and offer important targets for testing areas covered by transported overburden within both known mineralised districts and in grass roots exploration. Drilling requires a close control of regolith stratigraphy, accurate logging of drill holes, and correct sample classification.

The gold-carbonate relationship is developed typically in semi-arid climates of Western Australia as recent additions to deeply weathered regolith formed under humid tropical conditions. The strong association between Au and pedogenic carbonates is a significant feature of regolith geochemistry and an important consideration for delineating regional Au anomalies. Samples from the carbonate horizon appear to give surface expression to the mineralisation through as much as 40 m of leached saprolite and barren sediments (Lintern and Scott, 1990). The depth and thickness of the carbonate horizon may vary but it is generally within the top 1–2 m of the soil and is readily identified as it effervesces with dilute. Carbonate sampling according to these authors may be a preferred sampling technique either instead of or in addition to the sampling of ferruginous material for Au exploration in the semi-arid environment.

Ground water anomalies

Ground water anomalies, which in some cases are broader and more regular than the mineralisation and secondary dispersion of gold in the regolith, may enhance the exploration signature. The hydrogeochemical method has been extensively investigated for gold and other metals across the Yilgarn Craton and adjourning areas, but so far with only limited success. Dissolved Au provides the clearest indication of Au mineralisation but interpretation is complicated by there being two mechanisms of transport of Au in ground water (thiosulphate and halide complexing), with a third mechanism (organic complexation) specific to soils and possibly to lignitic horizons (refer to Chapter 1, Section 1.1.2). Most authors recommend restricting the use of ground water for exploration to shallow samples in Kalgoorlie and Central ground waters although depth appears to be less critical in the Northern Region of the Yilgarn Basin. Their conclusion is that waters should be in contact, or at least within a few metres of *in-situ* material. Gray (1997c) suggests that hydrochemical signatures may be more distinct in areas of high reactivity (e.g. faults and shear zones) where petrographic study is difficult.

Investigation of ground waters associated with known Au mineralisation of different types in the Black Flag Region of the Yilgarn Block reflects both the wide spread of Au mineralisation and the high solubility of gold in saline waters, with the most anomalous samples closest to bedrock deposits. These studies suggest that even widely spaced sampling (10 km $\times$ 10 km) would have a high probability of detecting the regional ground water anomaly for Au, which characterises the Black Flag Region.

Organic anomalies

The amount of organic matter is important because of the mobility of gold in soil profiles. A wide variety of organic/biologically based complexes include cyanide complexes, organic complexes and colloidal gold where stabilised by organic matter (Gray, 1997b). Precipitates in drainage channels have the facility for accumulating gold particularly ferrous and manganese oxides and calcium carbonate.

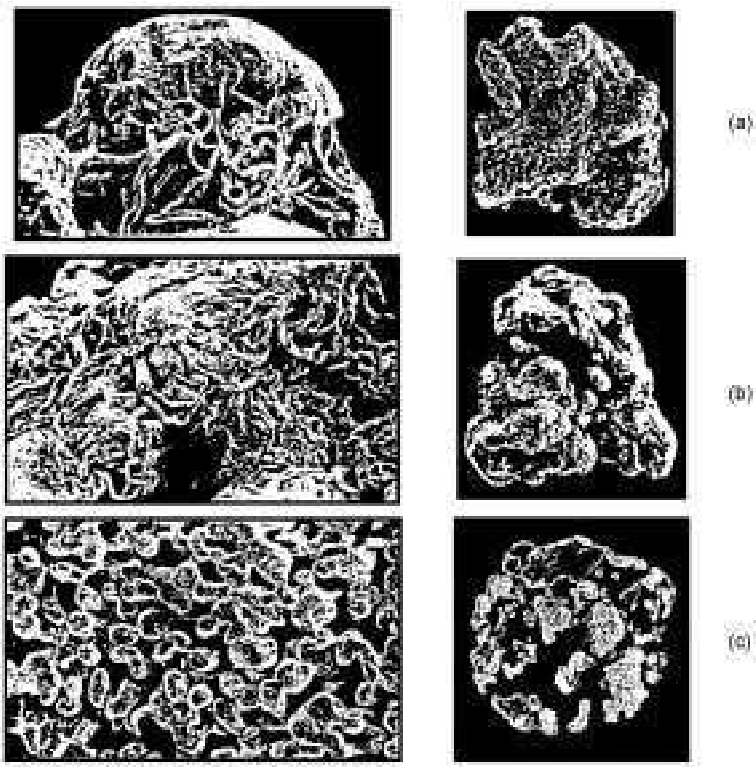
Despite studies that vegetation can produce stronger geochemical responses to mineralisation buried by transported overburden than more traditional surface sampling media, biochemistry has received scant consideration as a follow up technique in arid terrains of Australia (Cohen *et al.*, 1996). Plants selectively extract metals from soils and ground water that come into contact with their root systems, either redistributing the metals into various organs or exuding them via transpiration. In Table 5.5 a comparison is made between Au, As, La and Sb concentrations in cypress pine organs collected from mineralised and background test sites in different seasons.

In the Quebrada Grande Mining Zone of Venezuela gold specimens from alluvial strata consist of granules 20–90 μm in diameter, many showing their original structure, i.e., numerous hollow undulating, filamental appendages with thin gold walls that radiate from a denser central region rich in interstitial gold (Bischoff *et al.*, 1991). They are interpreted as precipitates of gold on a new bacterium belonging to the group of 'appendaged and/or budding bacteria' referred to by the authors as Mycoplasmatalesarium. Apparently in similar fashion to precipitates of metallogenium (an aerobic, organotrophic, micoplasma-type organism that oxidises manganese), they undergo a sequence of morphological changes with increasing amounts of gold. Three recognisable basic morphotypes exhibited a sequence of morphological changes bridged by transitional forms (Fig. 5.10). The authors describe them as filamentous types with long, freely extending appendages of constant diameter; the lobate type or lobate distal ends that fuse together to form a three-dimensional network; and

Table 5.5 Comparison between median Au, As, La and Sb concentrations in cypress pine organs, collected from mineralised and background test sites in different seasons (after Cohen *et al.*, 1996)

Organ	Site	As (ppm)		Au (ppb)		La (ppm)		Sb (ppm)	
		Jul	Dec	Jul	Dec	Jul	Dec	Jul	Dec
Needles	Mineralised	0.36	0.51	4.20	6.20	0.68	0.80	0.28	0.40
	Background	0.07	0.09	<0.30	0.90	0.24	0.28	0.01	0.06
Cones	Mineralised	0.33	0.51	2.80	5.65	0.73	0.77	0.20	0.38
	Background	0.12	0.15	<0.30	0.80	0.70	0.79	0.03	0.06
Bark	Mineralised	0.19	0.14	1.20	1.15	0.74	0.44	0.09	0.08
	Background	0.29	0.22	<0.30	0.40	0.81	1.45	0.05	0.04

Accumulation trends	**Element**
Needles > Cones > Bark	As, Au, Br, K and Sb
Cones > Bark or Needles	Ba, Cr and Hf
Bark > Cones or Needles	Ce, Co, Cs, Eu, Fe, La, Na, Rb, Sm, Se, Th, W and Yb



5.10 Association of gold with bacteria: (a) filimentous morphotype; (b) lobate morphotype; (c) advanced lobate morphotype (after Bischoff *et al.*, 1991).

the final massive globular type where interstitial gold fills all remaining gaps between the appendages.

The gold-precipitating bacteria appear to have existed in or on the surface of low-energy fresh water, probably associated with limnic, aqueous sediments, but also perhaps with near surface, ground influenced terrestrial sediments. Mercury was found in all five of the specimens polished and analysed, in one case reaching 19.8 weight percent. No major differences were found in percentages of Hg in cores and rims of any one specimen except in specimen 3 in which Hg was found in the rim but not in the core. This raises the uncertainty of whether the Hg amalgamated the gold either contemporaneously with its precipitation or after completion of the gold deposition on the rim. As source for the gold, Bischoff *et al.* (1991) considered solutions of inorganic rock salts, short-lived Au-organometallic compounds, or gold soils stabilised by organic acids. In their view, the Venezuela example establishes the case for the possible formation of metalliferous deposits of some economic importance by the natural accumulation of gold by appendaged bacteria.

Site selection

Establishing the nature of the geochemical dispersion of the gold will help to ensure the adoption of optimum procedures for sampling, analysis and interpretation. A wide range of ore deposit types result from the activities of hydrothermal fluids from different sources, of different composition and with different alteration effects (see Chapter 2). Some orebodies form within large mineralising systems where different deposits develop in different parts of the system at different stages of its development. This is illustrated schematically in Chapter 2 (Fig. 2.24) in relation to Fort Knox, Alaska where gold-only porphyries are at higher levels than Cu-Au porphyries that are themselves at higher levels than Cu-Mo porphyritic intrusives.

Sample size

Samples of stream silts and soils taken for geochemical analysis generally approximate 200 g of minus 80# material. Laboratory studies are carried out on sieved splits of from 3 g to 100 g. Some form of acid digestion/solvent extraction/atomic absorption analysis is used for the analytical determinations, with or without a fire assay start. Basically, the larger the sample, the more likely it will be representative of the material being sampled, but the higher will be the cost of analysis. Current analytical procedures (capable of detection limits of 5 ppb Au) allow even small differences in the gold content of the soils and adjacent rocks to be identified.

Sample spacing

How to space the samples will depend upon features such the size and shape of potential anomalies, the likely shape and attitude of expected ore deposits, dip of the bedrock sequence and whether this is the first survey of its kind or a follow-up survey. Initially, in a geochemical soil sampling exercise the regolith will be sampled on a large reconnaissance grid. Spacings of about 1 km may be appropriate for gold in a large reconnaissance survey but follow-up sampling is usually based on a grid located on lines reduced to 800 m and 400 m closing to 200 m apart with a sample spacing of 25–75 m. Lateritic residuum is sampled more closely at 100 m to 50 m spacing to delineate the source in the substrate in the strongest part of anomalies.

Having defined the lateral extent of anomalous gold distribution in the regolith, local conditions such as the depth and nature of the regolith and the weathered saprock will be determined by drilling and trenching. Hard horizons of outcropping saprolite are difficult to sample because usually only the most resistant parts are exposed; if these are chip sampled the samples will almost certainly be biased. Augering is unsuitable and usually a more powerful rotary air blast or reverse circulation drills are necessary. Samples are normally taken at set intervals or by lithology. The extent of this work will depend upon the area of the anomalous zone revealed by the regolith geochemistry. The major zones of primary and secondary dispersion will usually be located at reasonably shallow depths. Mechanical rotary drilling might be necessary to investigate mineralisation at depth.

Case history

The Salaman Gold Deposit in Spain is located on the Leon fault close to small stocks and dykes of intermediate to basic rocks to which the mineralisation is related. Epithermal type mineralisation was developed in two successive stages, the earliest stage being dominant and extensive. This stage comprised very fine crystalline gold-bearing sulphides, mainly pyrite, arsenopyrite and arsenic-bearing pyrite in a mass of quartz-chalcedony and dolomite. A later stage was in pockets and veins and replacements of the early stage. A stage of supergene mineralisation followed as a result of the oxidant action of meteoric waters over the previous minerals. An area of 2,000 m $\times$ 1,000 m was covered by a 100 $\times$ 25 m grid. This spacing was reduced to 50 m $\times$ 25 m and then to 25 m $\times$ 12.5 m in anomalous sections (Crespo *et al.*, 2000). The samples were analysed for Au (aqua regia digest/atomic absorption; fire assay fusion/direct coupled plasma) and multi-elements by aqua regia digest/induced coupled plasma. Au and As anomalies were particularly striking; other anomalous associations were Ag, Sb, Hg, Cu, Zn and Pb.

5.3 Remote sensing

Remote sensing control systems are used extensively in resource exploration to map regional lineaments and zones of local fracturing often with evidence of hydrothermal alteration of rocks associated with ore processing. The techniques are operated from moving platforms, e.g. aircraft, shuttles and satellites using heat, light, radio waves and other forms of electromagnetic energy. Unlike geophysical surveys, remote sensing techniques measure force fields (e.g., magnetic and gravity) away from the physical constraints of contact with the ground.

The method plays an important role in regional exploration by helping to differentiate between prospective and non-prospective ground. Mineralised belts or zones are selected from lineal interpretation and analysis of fracture patterns and hydrothermal alteration. The geometry of the depositional settings can then be mapped using techniques that are suited to the conditions and purpose of the survey. Aerial photographs are used for most topographic surveys, but radar imagery is best suited for mapping geological structures in forest areas. IR thermal images may sometimes be used to map buried fluvial channels; hydrothermally altered rocks exhibit different reflectance spectral signatures from those of the unaltered rock, and so on. None of these techniques on its own is a complete exploration tool, but if used in conjunction with ground reconnaissance methods may provide an important step along the trail of information leading to new discoveries.

In many mineral provinces, individually prospective areas occur along linear trends that may stretch for tens and hundreds of kilometres in length. Areas with concentrations of fracture intersections and circular features are good primary-gold exploration targets, being conduits for ore-forming fluids. The altered country rocks may contain distinctive assemblages of secondary or hydrothermally altered minerals, which exist as haloes around the parent orebodies.

5.3.1 Aerial photography

Aerial photography was pioneered for geological purposes during the First World War. It received a stimulus from advances made during the Second World War and has now become an essential tool for geological exploration. The method is useful for both topographic mapping and identification of specific morphological features such as changes in vegetation and vegetal cover related to different lithologies and buried channels. Techniques used in the laboratory to control and measure distortions of the changing geometry operate closely within a tolerance of plus or minus 1 to 2 microns. The measurements are applied as correction factors to the photogrammic restitution of images during the mapping process.

The advantages of the photographic method are:

- enormous amplification in the developing process
- high resolving power
- versatility
- low cost and ease of operation
- capacity of the film to store large amounts of information.

The main disadvantages are:

- the operation requires good lighting conditions and is restricted by weather and atmospheric affects
- atmospheric scattering reduces the contrast ratio and resolving power of aerial photographs in the shorter wavelength regions
- variations in reflectance are recorded in non-digital format that may preclude quantitative interpretation by computer processing.

High altitude aerial photography provides standard topographical data at 1:50,000 to 1:100,000 and each frame thus covers a relatively small area. Increasing the frame coverage by increasing the elevation from which photographs are taken increases the possibility of errors of parallax in stereoscopic pairs. The speed of the aircraft, number of frames per second, height of the aircraft at each photographic station and the focal length of the camera all influence parallax errors. The precision of mapping depends upon the precision with which parallax is measured.

Aerial photographs also provide a possible data source for satellite remote sensing. Spatial resolution is good, cost is low and films are available that provide a sensitivity range from the UV spectral vision through the visible and into the reflected IR region. Stereo photographs may also assist many types of geological interpretation. Knowledge of the techniques for interpreting aerial photographs also provides essential background for interpreting other remote sensing images (Sabins, 1986).

5.3.2 Landsat satellite remote sensing

Whereas aerial photo mosaics are built up from individual runs at different times and in different conditions, the synoptic view from satellite altitudes has the advantage of almost constant illumination over the whole area of investigation. It is also more economical, one spacecraft being the equivalent of a fleet of aircraft in its output. The basic attributes of Landsat multi-spectral scanner (MSS) and thematic mapper (TM) images are useful in the following geoscientific disciplines:

- geology, soils, mineral prospecting
- forestry, agriculture, land use
- urban and regional planning.

Landsat, formerly called ERTS (Earth resources technology satellite) operates on an ‘open sky’ policy, which allows images to be acquired on a continental scale without the prior consent of any government body. The images provide a worldwide database and are available for purchase at uniform prices by users anywhere in the world. The data are available in digital format suitable for computer processing. Procedures range from simple colour composites of several spectral bands and combinations of Landsat with other data such as SLAR and geophysical maps to facilitate recognition. Digital manipulations include contrast enhancement by mathematical transformation using computer matching of characteristic spectral radiance. Landsat images are complementary, not competing sources to aerial photographs and should always be conducted with ground truth programmes. Landsat, Spot, AVHRR and AVRIS are classified as passive remote control systems because they use EMR (electromagnetic radiance) from the sun, reflected from the Earth’s surface or EMR emitted from the Earth.

A typical radar system comprises:

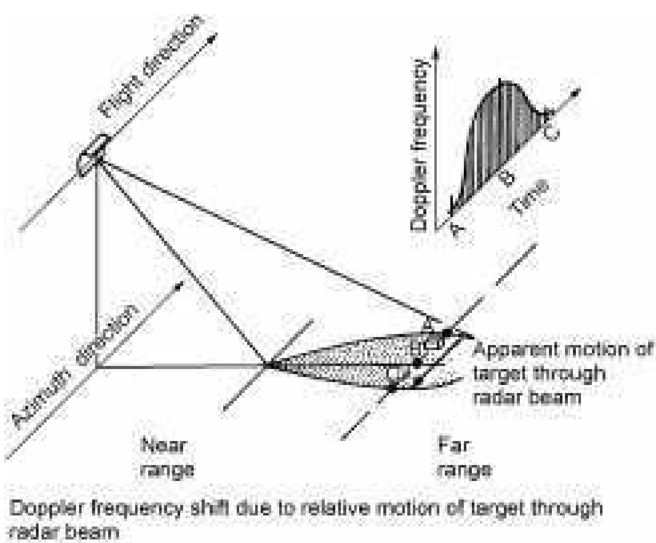
- pulse generator that discharges timed pulses of microwave/radio energy
- transmitter

- duplexer
- directional antenna, which shapes and focuses each pulse into a stream
- returned pulses that receive antenna picks up and send to a receiver that converts and amplifies them into video signals
- recording device, which stores them digitally for later processing and/or produces a real-time analogue display on a cathode ray tube (CRT) or drives a moving spotlight to record on film.

Pulse frequency is measured in microseconds, about 1,500 pulses/s. Pulse length is the distance travelled during pulse generation. The pulse separator separates the outgoing and returned pulses thus eliminating their natural interference by blocking reception during transmission and vice versa. The antenna on a ground system is usually a parabolic dish. For aircraft this mode of operation is referred to as SLAR (side looking airborne radar). A 'real aperture' system operates with a long (about 5–6 m) antenna, usually shaped as a section of a cylinder wall.

Radar antennas fitted to aircraft are usually mounted on the underside of the platform in order to direct their beam to the side of the aircraft in a direction normal to the flightpath. This type produces a beam of noncoherent pulses, using its length to obtain the required resolution in the flight line (azimuthal) direction. The transmitted beam continuously propagates outwards within a fan-shaped plane perpendicular to the flight line. Figure 5.11 illustrates the Doppler frequency shift due to relative motion of target through radar beam.

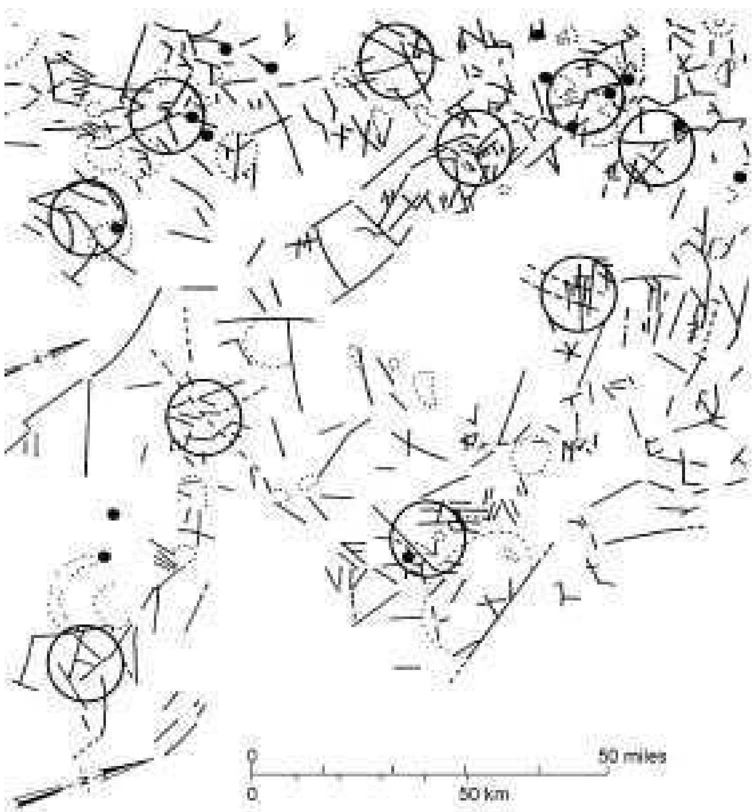
The Landsat multi-spectral scanner system (MSS) is a cross-track scanning system that scans a swath 185 km wide normal to the orbit path. The system



5.11 Apparent motion of target through successive radar beams (after Sabins, 1986).

was first invented in 1972 and used primarily as a vegetation mapping stock. However, while useful for identifying major geological features the technique does not meet the required degree of resolution to provide sufficient detail for topographic cartography. It is thus satisfactory only for coarse spatial and spectral imaging. The resolution cell of MSS measures 79 × 79 m square at 918 km altitude, which ultimately determines its spatial resolution. Reflected sunlight from the terrain is separated by a spectrometer into four wavelengths or spectral bands. The optimum sample interval has been measured at 57 m. Hence each scan line consists of 3,240 pixels each covering an area of 57 × 59 m.

As an example of early Landsat imagery, Fig. 5.12 shows concentrations of fracture intersections and ring structures on a Landsat interpretation map of a gold-mining district in central Colorado. Ten prospective areas, each of 165 km^2 were selected from an initial image covering a 33,500 km^2 area on the basis of the



5.12 Interpretation of Landsat image of central Colorado. Solid lines are distinct lineaments, dashed lines are possible lineaments, dotted lines are curved linear features; large circles are target areas selected for exploration. Solid dots indicate major mining districts (Nicolaise, 1983).

Landsat interpretation. These areas were chosen because of concentrations of fracture intersections or intersections of fractures and circular features. Five of the large circled areas selected for interpretation either coincide with or are adjacent to important mining districts. The other five target areas appear to be prospective for so far undiscovered orebodies.

Thematic mapping

Following the success of the thematic mapper on Satellite 4, the USA offered multi-spectral bands (0.4 μm to 12 μm). These data, in order to be effective, must be displayed so as to provide information in an extractable form and be optimised towards the particular need of the user. The range of applications of thematic mapping includes geological mapping, mineral exploration, geomorphological mapping, and vegetation mapping.

Landsat TM images supply improved spatial and spectral resolution with 30 m-pixel size. They supply seven bands of which bands 5 and 7 provide enhanced geological discrimination, thus making the technique suitable for geological exploration. By recording on both eastbound and westbound sweeps the scan rate is reduced and provides a longer dwell time so improving its radiometric accuracy. East and west sweeps take place seven times per second. TM bands use an array of 16 detectors except band 6, which uses only four. The ground resolution cell of TM is a 30 $\times$ 30 m square, i.e., 900 m^2. An image consists of 5,965 scan lines, each being 185 km long in the scan direction and 30 m wide in the orbit direction. The analogue signal is sampled at 30 m intervals to produce 30 $\times$ 30 m pixels. Each scan line consists of 6,167 pixels and each band consists of 34.9 $\times$ 10^6 pixels (after Sabins, 1986). The seven bands have a total of 244.3 $\times$ 10^6 pixels, which is more than eight times the data contained in a MSS image (Table 5.6).

Band ratios supply a simple but effective means of enhancing spectral reflection. By using colour bands the image is obtained from directly above, no shadows appear and the topography appears flat. The RGB bands for Landsat data are sometimes called false colour images:

- RGB 321 is the closest band combination to natural colour.
- RGB 432 standard false colour image is used for Landsat MSS.
- RGB 741 is the most common composite used for geological interpretation:
  - Red = iron,
  - Green = vegetation,
  - Blue = clay.

The spectral ranges of the six visible and reflected IR TM bands together with spectral reflection curves of vegetation, hydrothermally altered rock and unaltered rock are compared with spectral images of the four MSS bands in Fig. 5.13.

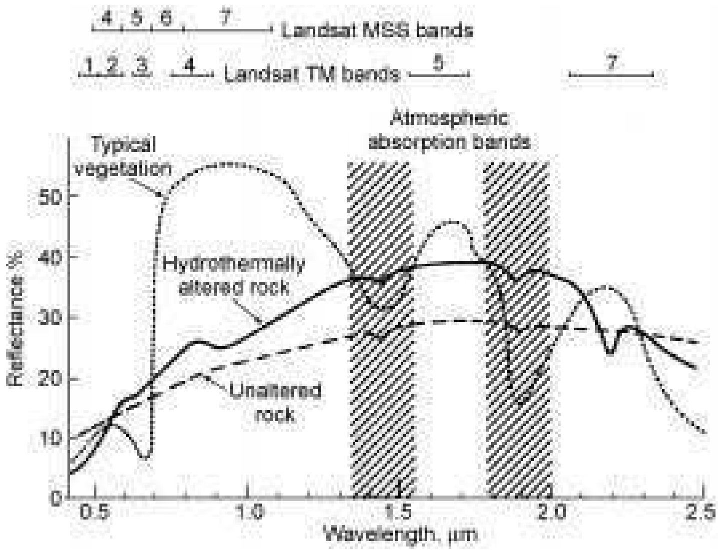
Table 5.6 Thematic-mapper spectral bands (after Sabins, 1986)

Band	Wavelength (μm)	Characteristics
1	Slightly less than 0.52	Blue-green. No MSS equivalent. Maximum penetration of water, which is useful for bathymetric mapping in shallow water. Useful for distinguishing soil from vegetation and deciduous grass coniferous plants
2	0.52–0.60	Green – coincident with MSS band 4. Matches green reflectance peak of vegetation, which is useful for assessing plant vigour
3	0.63–0.69	Red – coincident with MSS band 5. Matches a chlorophyll absorption band that is important for discriminating vegetation types
4	0.76–0.90	Reflected IR – coincident with portions of MSS bands 6 and 7. Useful for determining biomass content and for mapping shorelines
5	1.55–1.75	Reflected IR. Indicates moisture content of soil and vegetation. Penetrates thin clouds. Good contrast between vegetation types
6	10.40–12.50	Thermal IR. Night-time images are useful for thermal mapping and for estimating soil moisture
7	2.08–2.35	Reflected IR. Coincides with an absorptiob band caused by hydroxyl ions in minerals. Ratios of bands 5 and 7 are potentially useful for mapping hydrothermally altered rocks associated with mineral deposits

Synthetic aperture radar (SAR)

Radar is the acronym for radio detection and ranging, the term being used for both the technique and the equipment used. The physical nature of radar was investigated during the Second World War for navigation and target location using a rotating antenna and cathode-ray-tube display. 'Side Looking Airborne Radar' (SLAR) was developed in the 1950s to acquire reconnaissance images without flying over unfriendly territory. The geometry involved in side looking radar governs many of the intrinsic properties of radar images. Stereo radar coverage is obtained by imaging the same area from two different flight altitudes or two opposing antenna flight directions.

Synthetic aperture radar uses an antenna of much smaller dimensions than that used for SLAR. It sends its pulses from different directions as the platform

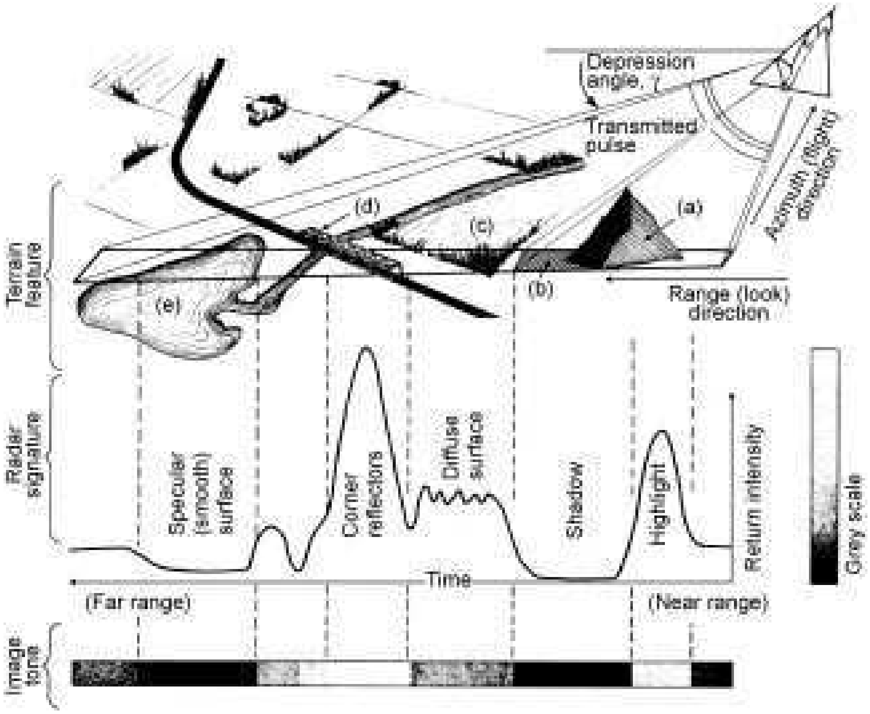


5.13 Spectral bands for TM and MSS systems. Reflectance curves for vegetation, unaltered rocks and hydrothermally altered rocks (after Sabins, 1986).

advances, simulating a real aperture by integrating the pulse echoes into a composite signal. Images of the Earth's surface reflect local differences in the normalised reflectivity of the target. Effective antenna lengths can be simulated up to 100 m and more. The Doppler effect (apparent frequency shift due to the velocity of the target or the radar vehicle) determines azimuth resolution. As coherent pulses transmitted from the velocity source reflect from the ground to the advancing platform (aircraft or spacecraft) the target acts as if it is in apparent (relative) motion. This motion results in changing frequencies, which give rise to variations in phase and amplitude in the returned pulses. The radar records these data for later processing by optical (using coherent laser light) or digital correlation methods. The moderated pulses are analysed and recombined to synthesise signals equivalent to those from a narrow beam, real aperture system.

SAR operates independently of diurnal and weather-influenced solar illumination. Microwave signals in the frequency range 12–10 GHz can penetrate heavy rain, thus having an advantage over optical sensors, which cannot operate in such conditions. Radar measurements taken from high altitudes maintain an almost constant angle of incidence within the swath so as to compare individual features of widely spaced signatures. Typical SAR systems are ERS-1, JERS-1, Seasat and Airsat.

Figure 5.14 describes the nature and beam characteristics of a typical radar system and interpretation of the signal returns as displayed on film or a monitor. The upper part of the figure depicts a strip of land surface being scanned by the

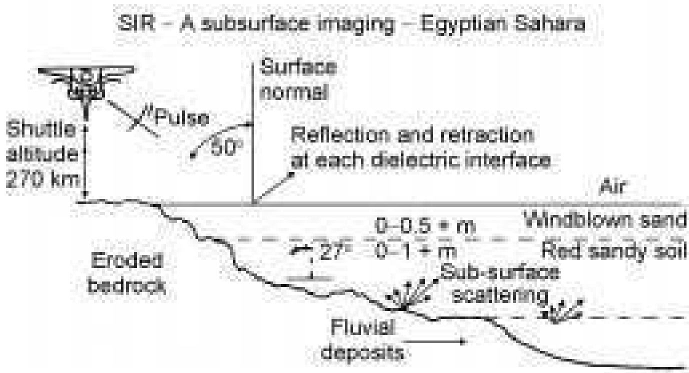


5.14 Nature and interpretation of beam characteristics of a typical radar system as displayed on film monitor, (a) face of high mountain, (b) shadow, (c) vegetation, (d) metal bridge, (e) lake (after Sabins, 1986).

radar beam. The aircraft moves at an altitude above the terrain in an azimuthal direction, while the pulses spread outward in the range direction. The depression (look) angle is the angle between the vertical and that of a ray path. The incidence angle is the angle between an incident radar ray and a line at right-angles to the surface. The direct distance from the antenna to some ground point within the terrain strip defines the 'slant range' to that point. The beam width angle determines how far the illumination of the surface spreads out from near range to far range. Pulse travel times increase outwards between these limits. The duration of a single pulse determines the resolution at a given slant range. The illustration shows that this range resolution, i.e., the minimum distance between two reflecting points along the azmuthal direction, gets poorer outward for specific pulse duration. Thus the resolution improves with increasing depression angles.

Sub-surface mapping relies upon a number of conditions:

- The ground surface must appear smooth to the incident radar beam to avoid backscatter, which would be detected by the radar.
- The sub-surface basement must be rough in order to promote backscatter.



5.15 Diagram showing reflection and refraction of radar waves at air/surface boundary with sub-surface scattering producing an image of the sub-surface (after Blom and Dixon, 1986).

- The layer to be penetrated must not be too thick otherwise the signal will be attenuated; the present maximum thickness is 3 m to 5 m.
- The medium to be penetrated must not have scatterers embedded in it; an ideal medium is aeolean sand with no pebbles or large cobbles.
- The moisture content must not exceed 1.0%; above this level, energy is reflected away from the upper surface/air interface and signal attenuation increases within the penetrated medium by raising the loss tangent of the material; in satellite images, water is typically dark.

Shuttle spacing Radar A images of Egypt and Sudan acquired in November 1981 revealed a previously unsuspected fluvial landscape just below the aeolean cover of the Sahara Desert (after McCauley, 1982). Further investigations of stored data in the Mojave Desert of California revealed other examples of sub-surface imagery. Figure 5.15 (from Blom and Dixon, 1986) describes the SIR-A sub-surface imaging of the Egyptian Sahara to show reflection and refraction of the radar wave at the sub-surface boundary with sub-surface scattering producing an image of the sub-surface.

5.4 Shallow land-based geophysics

Geophysical methods rely upon differences in the physical properties of rocks and measurement of responses to some form of impulse generated artificially in the ground. These measurements can be used in residual and alluvial gold geophysical surveys to:

- map bedrock relief and depth
- investigate the nature of the alluvial materials via their electrical and other physical properties

- detect the presence of false bottoms or continuous clay horizons and/or other relevant features.

Techniques include magnetometry, induced polarisation, radiometry, and resistivity.

The successful use of geophysical methods and equipment depends upon the precise determinations of a theodolite survey and of how well the techniques are selected and applied. The differences may be only subtle and measurements must be taken with care and precision and be presented with an eye to subsequent interpretation. The quality of the data and hence the interpretations deduced from approximate geophysical measurements will remain approximate and be a continuing source of potential error.

5.4.1 Programme strategy

The type and extent of geophysical information required from a residual gold exploration programme will depend upon what answers are needed for some particular geological problems. The questions may relate to the need for more precise mapping of known or suspected horizons, structures or trends. 'Wildcat' investigations may be carried out to identify the possible location of primary gold orebodies or of hidden palaeochannels. The survey might also be intended to predict favourable locations for detailed drilling within practical depth limitations. Survey depths for open-cast or underground methods of mining are generally constrained only by economic considerations. On the other hand, the maximum digging capacity of suitable dredgers and prevailing physical conditions (e.g., depth of the water table, proposed mining rate, etc.) are limiting factors for wet methods of mining. Except where the pond level can be lowered by stripping, geophysical depth profiles will generally be restricted to maximum depths of about 50 m below pond level with a safe bank depth of about 10 m.

Fieldwork usually comprises integrated studies using two or more complementary techniques, the most common of which are combinations of seismic refraction profiling with vertical electric sounding and resistivity profiling. In selecting the most appropriate methods geologists and geophysicists must work together to ensure that the required limits of accuracy will be matched by the precision limits of any proposed instrumentation. Surface features such as swamps and topographic highs and lows must be considered in relation to the spacing of geophones and shot points in seismic refraction surveys. Resistivity surveys are concerned with types and depths of emplacement of electrodes and their configurations and so on. Geophysical techniques developed for offshore tin exploration in S.E. Asian waters are generally suitable for shallow offshore exploration for gold placer in similar type environs elsewhere. Constraints to these techniques are discussed in Section 5.4.2.

Guidelines for interpretation

Interpretation commences with the first inflow of information and is continually updated in line with the geology until the programme has either been satisfactorily concluded or abandoned. A geophysical model provides an indication not a perfect solution and all predictions must be tested rigorously as the work proceeds to allow any possibly anomalous features to be re-examined. Unless errors are picked up quickly and corrected, the model may appear to be believable but the interpretation will be wrong. Only by developing a feel that what is happening geophysically is in fact real will the operator be able to anticipate anomalous features from one line to the next.

Results (depicted as profiles and individual line profiles) should also be charted continuously to ensure that the method is appropriate and the data are good. Even when a project is abandoned the data should be left in a suitable condition for someone else to manipulate and re-interpret at some future time. Since geophysical results are obtained for geological purposes, selection of horizontal and vertical scales for mapping or any exaggeration of the vertical scales should be the same in both disciplines to allow appropriate overlays. Line lengths and spacings are usually large compared with borehole and layering depths and the vertical scale is usually exaggerated for pictorial purposes by a factor of ten. Convenient scales are horizontal 1:1,000 and vertical 1:100.

Method selection

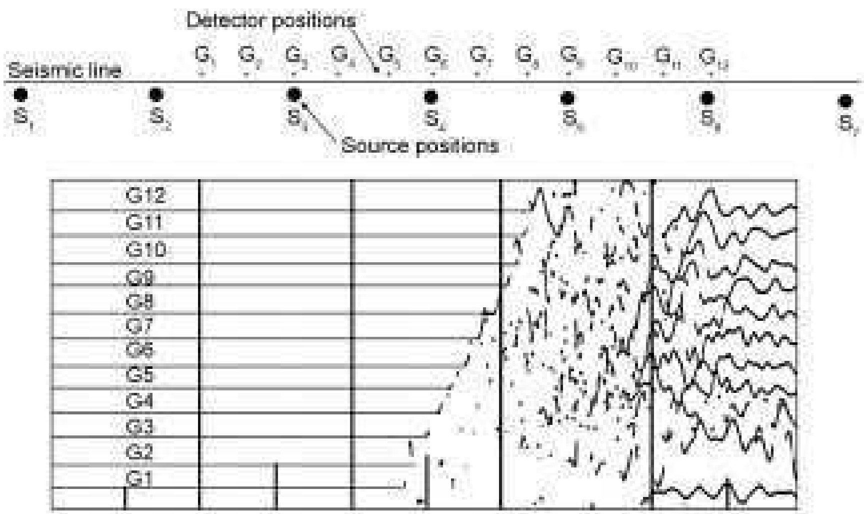
Methods of geophysics may be used to locate specific drainage features and delineate buried channels for which there are no surface expressions. Current methods of land-based geophysics generally offer a choice between seismic refraction and reflection profiling, and earth resistivity. Gravity surveys have some specific applications and ground-probing radar is beginning to be recognised as a useful technique, either alone or in conjunction with shallow seismic refraction. A combination of methods is usually needed to solve a particular problem. Powerful new instruments and digital hardware are available and if it is known what level of response to expect, instrumentation can be selected that is adequate for the task.

5.4.2 Shallow refraction seismics

The method of shallow seismic refraction is a standard procedure for the detection and delineation of buried stream channels. Its purpose is to predict depths to bedrock and other key horizons having rather different responses from those of the bedrock or adjacent strata. The waves are refracted and/or reflected at changes in acoustic impedance. Shock waves generated by an explosion or mechanical impact propagate through the earth at a velocity dependent upon the compressive and shear strength of the soil and rock. Differences in compressional wave velocities between rocks having different densities and elastic constants are used to map adjacent structures and so predict depths to bedrock and other key horizons.

Seismic waves are refracted only from layers having increased velocity signatures with depth. They do not extend to layers in which the velocities are lower than in an overlying bed and may be absorbed in accoustically opaque layers. Such conditions may exist where layers of mud overlying coarser sediments contain bubbles of natural gases, for example, methane produced by vegetable decay. The method can be very selective for unweathered basement rock because of the sharp velocity increase at the sediment/bedrock interface; it does not always succeed where the basement itself is sedimentary.

Important parameters are line lengths, directions and spacings, station spacings and elevations. Lines are marked out by theodolite survey and staked at intervals using wooden pegs (metal stakes are avoided for obvious reasons). The line direction is at right-angles to the inferred palaeochannel course for maximum resolution and must fully cover the width of possible palaeochannels; the station spacing is determined according to the expected complexity of layering. In large reconnaissance surveys the initial grid is usually laid out with lines and stations widely spaced, both for speed and reasons of economy. Reducing the station interval across the relevant sections follows up any suspected palaeochannel development. Figure 5.16 represents the plan view of a typical seismic survey configuration along with the geophone traces.



5.16 Plan of typical seismic refraction survey configuration and geophone traces (from Palmer, 1986).

Wave travel time

Shallow seismic refraction is applied by generating a pulse of energy in the form of an elastic wave at shallow depths. The wave travel time to detectors spaced various distances apart at the surface gives the velocity of propagation of the pulse in the ground. First arrival time intervals are used to calculate the positions of boundaries between regions of different velocity; geophysical boundaries will coincide closely with geological boundaries. Paths for first and later arrivals are illustrated in Fig. 5.17.

The first arrival of the wave front is fundamental to the method. Although ground surface waves are the first to arrive at receiving points in the vicinity of the impact point, the refracted waves travel faster through the denser media and will arrive first at some particular distance from that point. Distances between the impact point and points of first arrival are functions of the strata characteristics and depth. Travel time and distance from the impact point to the sensor determine the wave velocity in a particular layer.

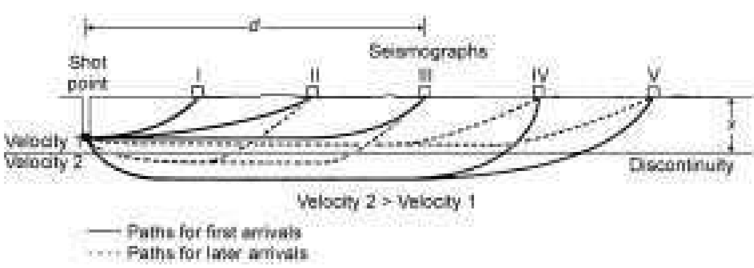
There are many other arrivals but refracted 'P' waves are always the first to be detected on an earthquake record. The 'P' wave velocity is governed by the compressive strength of the material; the 'S' waves depend upon the shear strength and are usually recorded as second events. Reflection waves are incidence waves and are shown along with ground waves and air waves (Fig. 5.18).

Elementary models describing the various phenomena assume complete homogeneity and isotropy in the layers and bedrock. In practice, every response is subject to some interference and distortion due to individual variations in subsurface properties and ground noise. Unambiguous interpretation is either simple, difficult or impossible to obtain depending upon how closely these responses can be identified with the lithology and other factors affecting the propagation of the seismic waves.

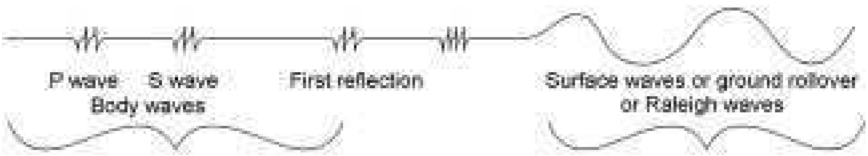
Problems in interpretation

Constraints to the method are its reliance upon assumptions of:

- increasing wave velocity with depth from layer to layer



5.17 Paths for first and later arrivals.



5.18 Wave arrivals – refracted waves are always first.

- constant velocity within each layer, i.e., for the full layer thickness
- layers that are homogeneous and isotropic
- a planar refractor over short distances beneath each geophone
- laterally continuous boundaries off line
- sufficient thickness of each layer to refract energy; the thickness of sedimentary layers above compact bedrock can be determined only for layers that are at least one-tenth of the depth in thickness
- surface detection of the energy refracted from a boundary.

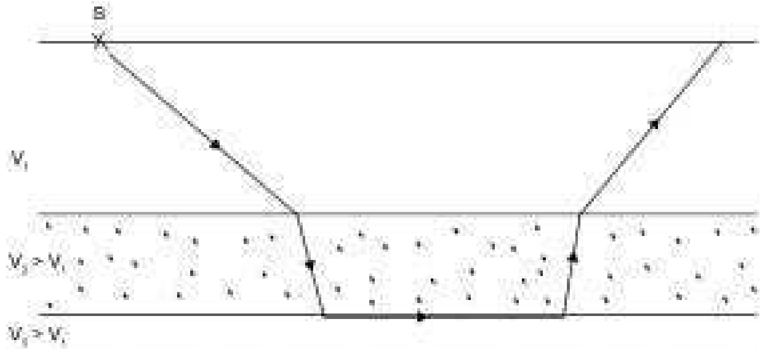
Any departure from these theoretical assumptions limits the use of the method under field conditions. The field data for theoretical models consist of travel time curves with each section representing a different refractor. A depth section is obtained by assigning a refractor to each segment and applying standard formulae. In practice there are likely to be a number of departures from the above assumptions due to one or more of the following.

Insufficient density contrast

The method is unusable because boundaries cannot be detected.

Velocity inversion

Velocity inversion (Fig. 5.19) in the sub-surface is one of the most serious limitations of the seismic refraction method. Inversion can occur whenever a geological layer has a lower velocity than that of the overlying layer. Such a layer cannot be detected by seismic refraction on the basis of first arrivals. Unrecognised inversion layers can create considerable difficulties in depth interpretation. Situations in which the velocity inversions have been reported in shallow seismic refraction studies are quite common and include clay beneath a perched aquifer and peat sequences. Interbedded gravels and fine unconsolidated sediments such as muds and silts represent a very likely environment for the occurrence of velocity inversions. All authors on the subject agree that interpretation of seismic refraction data in conjunction with good geological and drilling control is the only sure way to recognise and solve potential velocity and general hidden layer problems. It may be too late to do anything to adjust to the anomalous conditions if the inversions are recognised only at the stage of interpreting the refraction data.



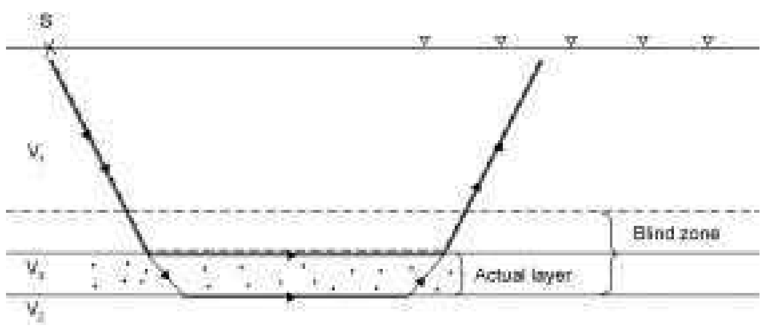
5.19 Velocity inversion. At top of V_2 refraction is towards the normal: interface cannot be seen.

Masked layers

Undetected layers are the most common source of errors in the majority of refraction interpretation methods. Even when all layers are detected, there is still a zone of uncertainty in each layer, the blind zone, in which the velocity is interpreted by extrapolation from the upper part of the layer. Figure 5.20 graphically displays the seismic travel time for an intermediate layer that has become progressively thinner. The loss of the 'straight line segment' due to the central layer can be clearly seen once the layer becomes thin. At this point it is hidden or undetected.

Velocity anisotropy

It is generally necessary for seismic velocities to remain constant for several geophone spacings to ensure the arrivals at each velocity are recognised. Anisotropy is a common feature of the complex layering of flood plain deposits due to lateral variations in the stratigraphic horizons, e.g. from clay to gravel or



5.20 Blind zone. Dashed part takes less time ($V_1 < V_3 < V_2$) then full time path; undetected late arrival (from Palmer, 1986).

fine sand to coarse sand. Significant errors in interpretation can occur if changes are missed.

Gradual velocity increase with depth

This negates the theoretical assumption of constant velocity for the full thickness of the layer. The resulting travel time graph will be curved and its division into straight segments is untenable. The problem can arise from several causes, e.g. gradual decrease of weathering with depth, and gradual increase of compaction with depth.

Shingling

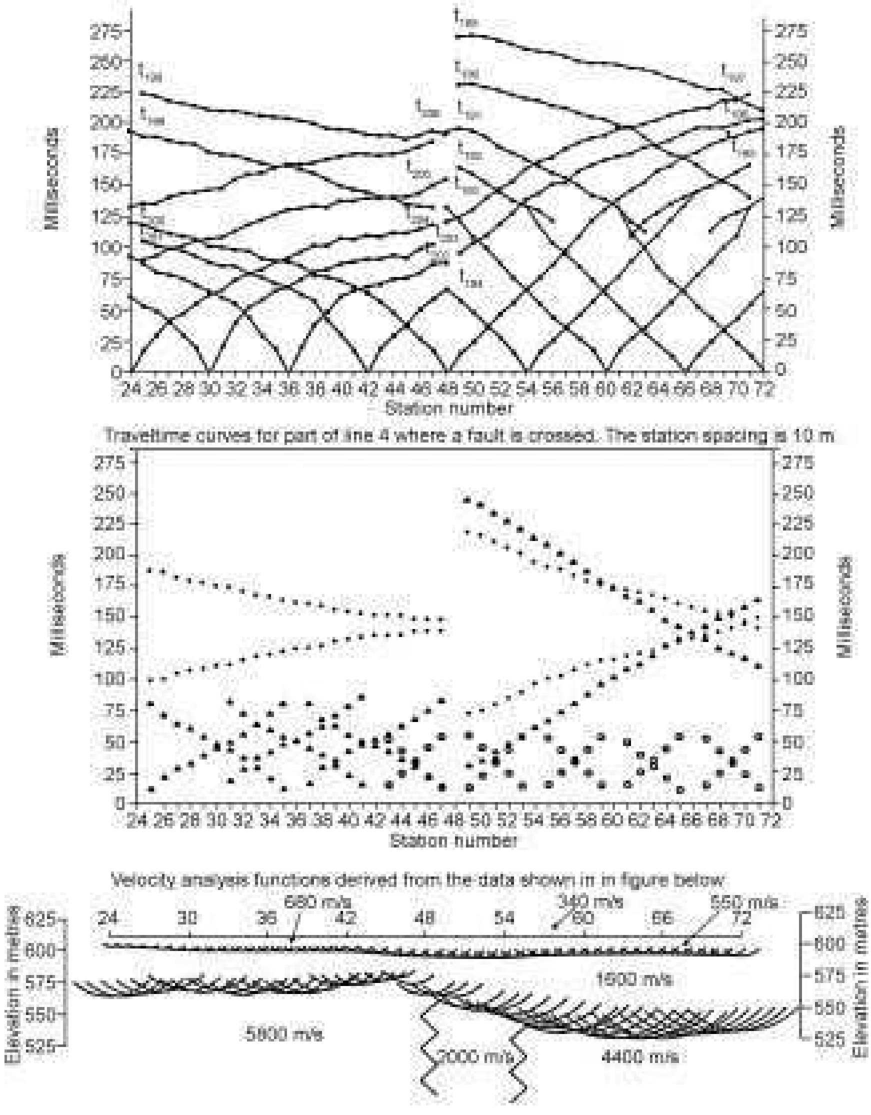
Shingling is a term that refers to discontinuities in the time-distance graphs resulting from phase loss due to attenuation, lateral sub-surface discontinuities, and presence of thin screening layers close to the surface. Generally in any report of refraction surveying, data is presented in the form of:

- time–distance curves or travel–time curves
- velocity analysis curves
- time–depth curves
- interpreted depth sections.

Travel time curves are usually quite complex, however a rigorous analysis can yield a good result, which can closely match the bedrock profile. Depths to the refractor from each geophone station are plotted on an arc because the depth to the refractor from each geophone is the perpendicular distance to the interface. Drafting the depth section in this manner gives a more realistic representation of the true refractor depth profile, which simply becomes the envelope of the arcs. Any sharp variations in the refractor or bedrock topography are smoothed by this method. If rapid changes appear in the seismic refraction interpretation this will indicate serious errors in the data quality or in the processing scheme. Figure 5.21 is a typical field profile of seismic refraction data.

5.4.3 Shallow reflection seismics

The ultimate purpose of this method is similar to that of seismic refraction profiling, i.e. to determine the depth to an interface between two layers. It differs from shallow refraction seismics by relying on travel times of reflected rather than refracted seismic waves. In order to ensure that the responses are due to reflected waves the geophones are spread symmetrically on a straight line on either side of the shot, with maximum detector distance not greater than the shallowest horizon of interest.



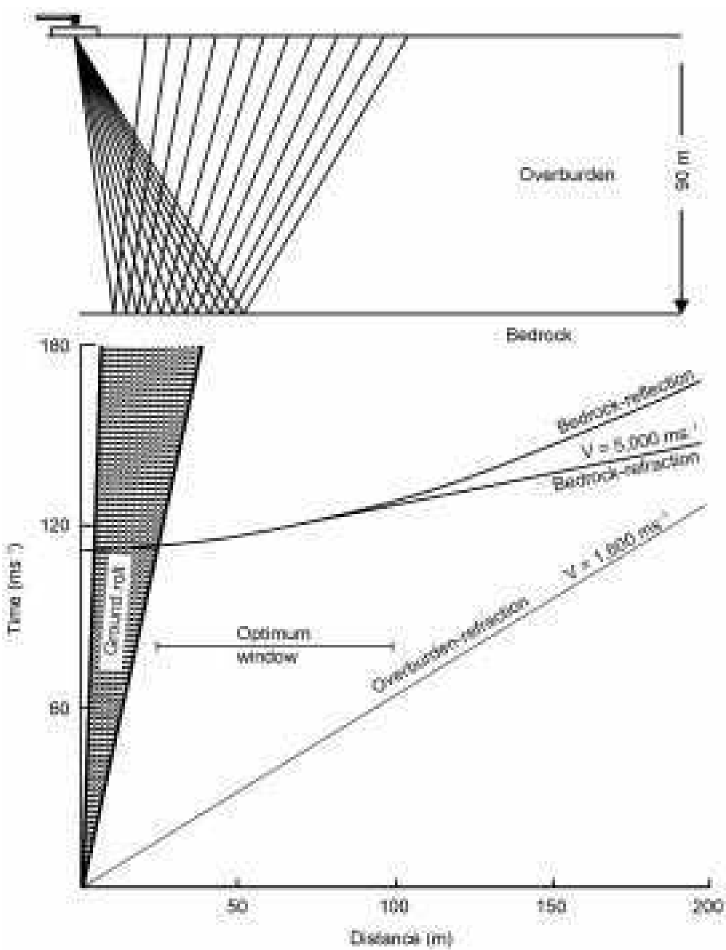
5.21 Field profile of seismic refraction data (from Palmer, 1986).

Instrumentation

The reflection method requires a high-frequency source and the drop-hammer technique is the one most commonly used. This technique involves striking a sledgehammer onto a steel plate firmly in contact with the ground, typically using three or four blows to stack the record with high-energy reflections. Another technique uses a specially constructed gun that fires 12-gauge shotgun cartridges at a depth of about 1 m below the surface. Trigger signals to the seismograph are obtained from a hammer switch attached to the firing rod of the gun.

General requirements for instrumentation are a 12 or 24 channel enhancement seismograph with 100 Hz geophones, a digital tape cassette recorder, and a microcomputer to which the recorder is linked in the field office. The geophones are spaced according to the requirements of depth range and resolution. Computer processing of records involves normal move-out (NMO) corrections, band-pass filtering and static corrections based upon observations of the first arrival refraction.

The seismic reflection model is illustrated conceptually in Fig. 5.22. The model shows the shot-geophone geometry for the optimum window and time



5.22 Seismic reflection model and time distance graph, showing position of the optimum window (after Hunter *et al.*, 1982).

distance graph. Series of expanding spreads are normally shot to view the reflection signature over a wide range of shot-geophone offsets. An 'optimum window' of offsets is chosen for production shooting. This requires the near geophone to be located beyond the zone of 'ground roll' interference. The overburden should also have good transmission characteristics with a sharp discontinuity at the ore–overburden interface.

5.4.4 Earth resistivity

Earth resistivity (ER) techniques measure the potential difference that results from passing a known flow of electricity (DC or low-frequency AC) into the ground. A portable source of EMF is connected to the ground by means of 'current' electrodes and the observed 'potential' electrodes, which measure the potential difference due to the current passed. Measurements are taken at or near the ground surface using a minimum of two 'current' electrodes and two 'potential' electrodes, which may be located in a number of different symmetrical arrays depending upon their purpose.

Resistivity concept

Individual sediment layers and rocks have varying porosities and degrees of saturation, which allow similarly variable amounts of current to pass through them. Water in pore spaces and fissures in the ground account for most of the current flow. Most rocks are good insulators in a dry state and only a few rock varieties and clays containing conductive minerals such as graphite and magnetite are significantly more conductive than other rock types in the dry state. The degree of saturation affects resistivity, which decreases with any increase in the amount of water in the pores and fissures.

Unlike other sedimentary materials, clays allow current to pass by means of weakly bonded surface ions. Porous rocks containing a high proportion of clay minerals behave differently from the same rock types with less clay. Measurements of changing resistivities with depth are used to:

- locate possible false bottom/facies changes
- distinguish between bedrock and overlying sediments
- delineate prospective ore horizons of coarse alluvial gravels.

The latter application is particularly important where seismic refraction fails because of a velocity inversion (hidden layer). That such inversions occur is due to higher velocity gravels overlying poorly consolidated (lower velocity) layers.

Variations in ionic conductivity

The resistivity of naturally occurring water varies from a few tenths of an ohm/m for seawater to more than 100 ohm/m for water in a fresh mountain stream. In the absence of water, the resistivity of dry unconsolidated overburden can reach thousands of ohm/m. Clay beds that retain water are generally more conductive than layers of more granular material near the surface. In practice, the differences are often quite subtle due to the various combinations of lithology, degrees of saturation and water quality.

Prospecting techniques

Shallow alluvial deposits may be investigated using a combination of 'horizontal profiling' and 'vertical electrical sounding':

- Horizontal profiling examines the lateral distribution of electrical resistivity at relatively constant depths and is most effective when the lateral changes in resistivity are large; interpretations are qualitative and instrument precision need not be high unless they are calibrated by additional soundings.
- Electrical sounding allows a systematic examination of sediment resistivities at increasing depths; the technique is used most effectively for relatively flat-lying beds that are laterally homogeneous and extensive; measurements must be taken with great care in order to distinguish between good curves and curves that are distorted by lateral and other effects.

A placer resistivity survey normally commences with one or two lines of horizontal profiling using a dipole/dipole configuration across the channel course as illustrated in Fig. 5.23. The value of 'n' is varied from 1 to 7 depending upon electrode spacing and penetration and is determined in the field.

Electrode arrays

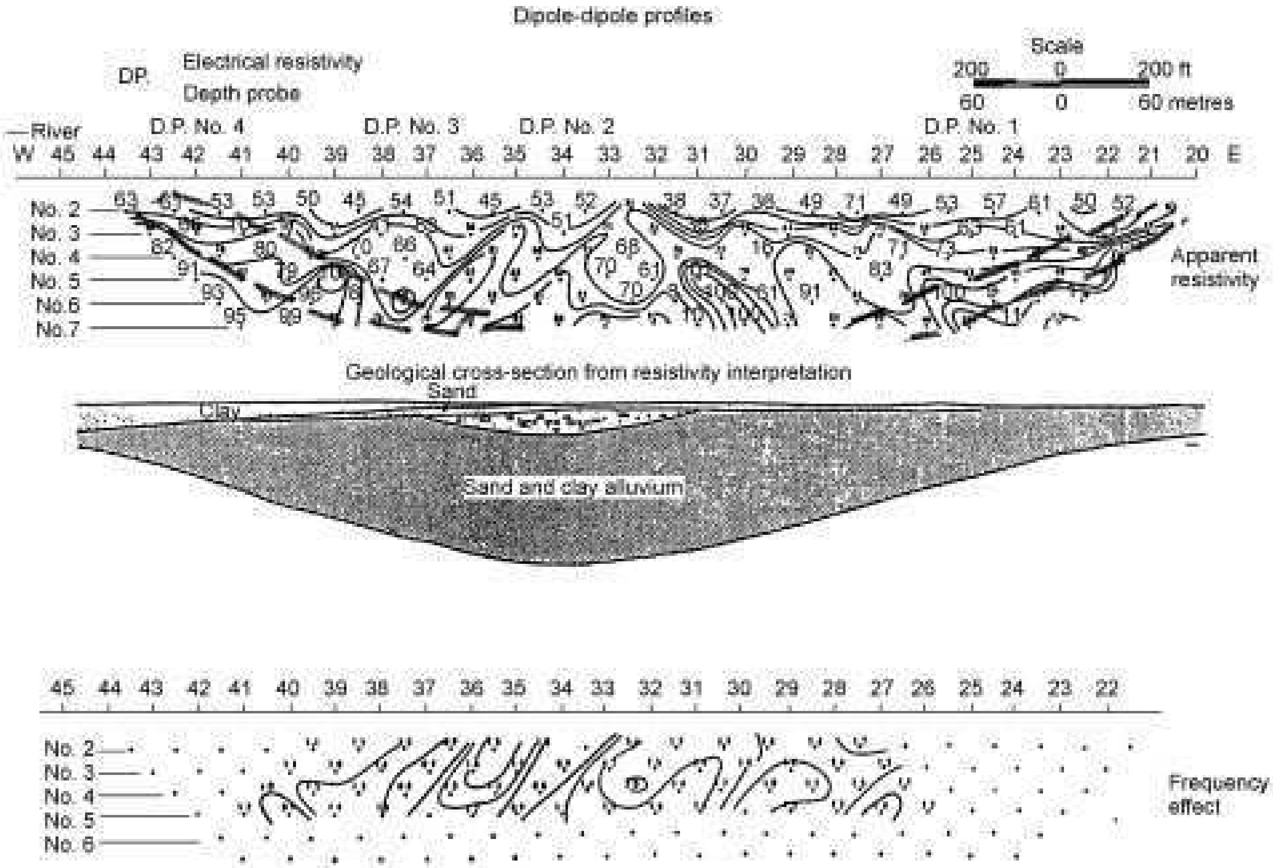
At least four electrodes are needed for resistivity measurements. A known current is passed into the earth through the current electrodes A and B. M and N are the potential electrodes between which a potential difference is measured due to the current passed (Fig. 5.24). The apparent resistivity Pa may then be related to measurable quantities, e.g.:

- current flowing through the ground
- potential difference between M and N
- geometric dimensions of the particular array being used in this figure.

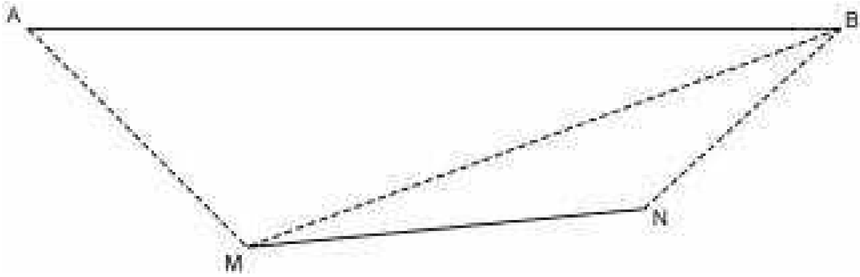
Of the three electrode arrays illustrated in Fig. 5.25, the Schlumberger (a) and Wenner (b) arrays are the best known, various dipole/dipole arrays and the gradient array (c) is used for special purposes such as orientation exercises.

The main constraints to using the earth resistivity method are:

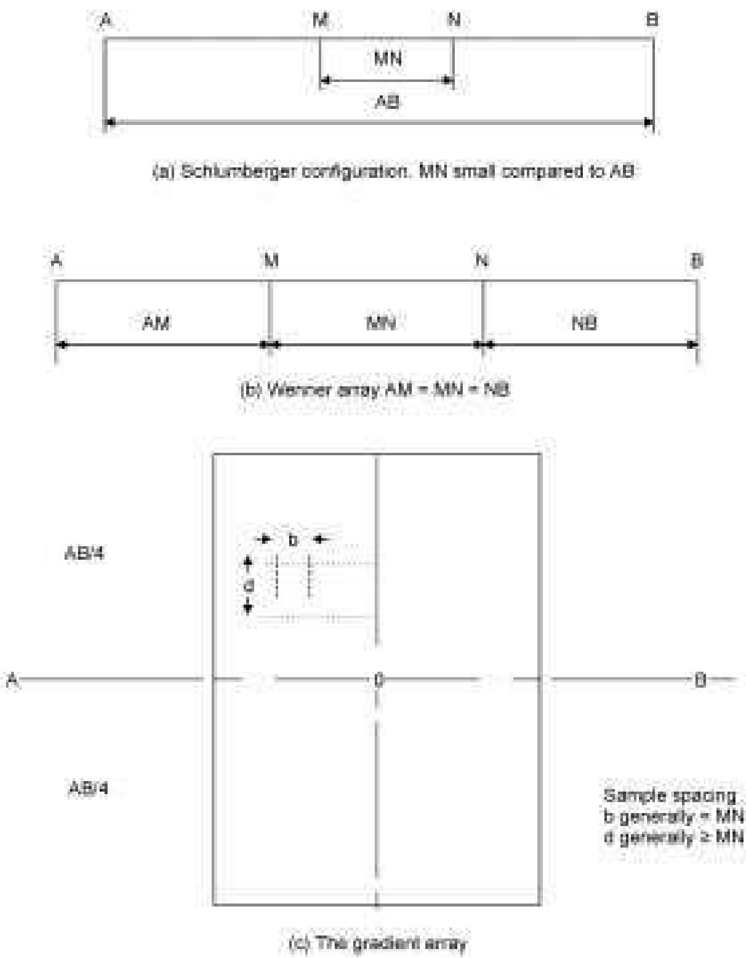
- A basic premise for detailed analysis of the data is the essential smoothing effect of the horizontal cake-layer geology concept; for example, a depth determination for a layer interface at 10 m requires a current electrode



5.23 Dipole-dipole configuration (adapted from Pratt, 1982).



5.24 General resistivity array consisting of two current electrodes A and B and two potential electrodes M and N.



5.25 Principal electrode arrays: (a) Schlumberger configuration; MN small compared with AB. (b) Wenner array; AM = MN = NB. (c) gradient array; sample spacing b generally = MN and d generally $\geq$ MN.

spacing of at least 60 m; any less spacing could result in lateral layer variations over the area of measurement and severely affect the validity of the depth determination.

- Resistances are obtained as average values only for large volumes of a formation; small, localised heterogeneities are detected only if they are close to the measuring array. With increasing depth, the volumes for which the average values can be obtained rise, a clay false bottom having a thickness of less than about one-tenth of its depth below the surface will almost certainly not be detected at 30 m depth; the layer would need to be at least 3 m thick, at 40 m, 4 m thick and so on.
- Dependence upon direction of flow requires derivation of apparent resistivity to account for the paradox of anisotropy; a formation has its lowest resistivity perpendicular to the stratification. In the case of horizontal stratification, resistivity is measured in all directions and from the surface there is no way of knowing if the sub-surface is anisotropic; this leads to non-uniqueness in the interpretation of sounding curves (Van Zijl, 1977).
- Local heterogeneities of rock resistivity commonly result in wide variations in resistivity over short distances, which include variations in the porosity and permeability of sand layers; no precise relationship exists between lithology and resistivity although some generalisations can be made.

5.4.5 Gravity method

Newton's Law of Gravitation states that each body in the universe attracts every other body to a degree that is directly proportional to the distances separating their centres of mass. A body in the vicinity of any other body is in a gravitational field of force and experiences acceleration. The force can be calculated by applying Newton's Law to infinitesimal elements of the body and integrating over the whole volume.

Disturbances to the Earth's gravitational field of sub-surface features that are directly or indirectly related to the direct or indirect influence of mineral deposits may be very small and require both sensitive instruments for detection and careful field procedures. Hence gravimeters are designed to detect field differences rather than absolute values and require very detailed topographic mapping in conjunction with the gravity survey. Because of this, the cost of a gravity survey is very high and because of the relatively small anomalies and pervasive geological noise the results are often ambiguous.

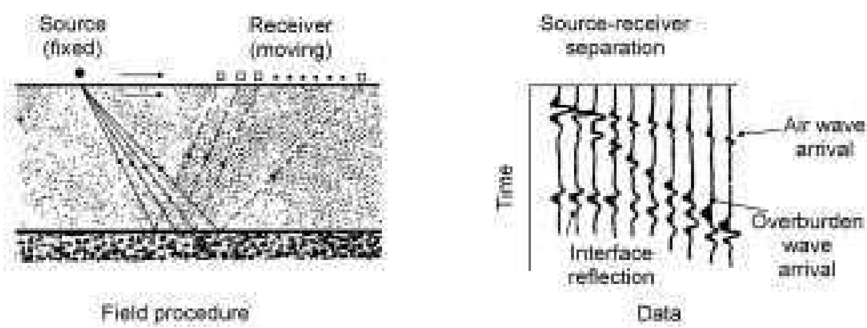
Gravimeters in common use include both stable and unstable types. The 'stable' gravimeter is a highly sensitive balance, which contains a responsive element, e.g. a spring carrying a weight which is displaced from its equilibrium setting when the gravitational force changes. The displacements are magnified optically, mechanically, or electrically so that very small displacements can be measured accurately. The 'unstable' types are designed so that when the displacements occur, the gravity change is measured by the force needed to return the element to its equilibrium setting. Corrections are applied to account for differences in latitude, elevation, topography and instrumentation. Anomalous differences are generally quite small and interpretation is very sensitive to density values. It is very important to obtain reliable densities of the rocks in a prospective area. Density values should be measured on site and not merely assumed.

A gravity survey usually seeks information on targets ranging in depth from 0–3 km below the ground surface, but the technique has been applied to help delineate drainage patterns over wide areas of moderate relief. Using shallow refraction in conjunction with gravity accuracy within plus or minus 10% was claimed by Peterson *et al.* (1968) in determining bedrock configurations between exposed Tertiary gravels in Sierra Nevada.

Daly (1965) found a minimum gravity of 0.6 milligal over a deep cassiterite lead at Ardlethan, NSW Australia. Apparently this was significant enough for accurate delineation of the lead. In this respect, it was in close agreement with seismic findings over the same area. However, in comparing the two studies, Daly found the resolving power of the gravity method to be lower than the seismic method. He concluded that gravity might sometimes be a useful tool for large-scale reconnaissance with seismic refraction follow-up where gravity anomalies suggest the presence of deep leads.

5.4.6 Ground probing radar

Using this method, signals are transmitted back to the surface as both refraction and reflection soundings when a short burst of high-frequency electromagnetic energy is transmitted into the ground. Figure 5.26 is a schematic representation of a radar wide-angle reflection and refraction sounding and collected data. Arrows indicate direct paths of electromagnetic waves through air and



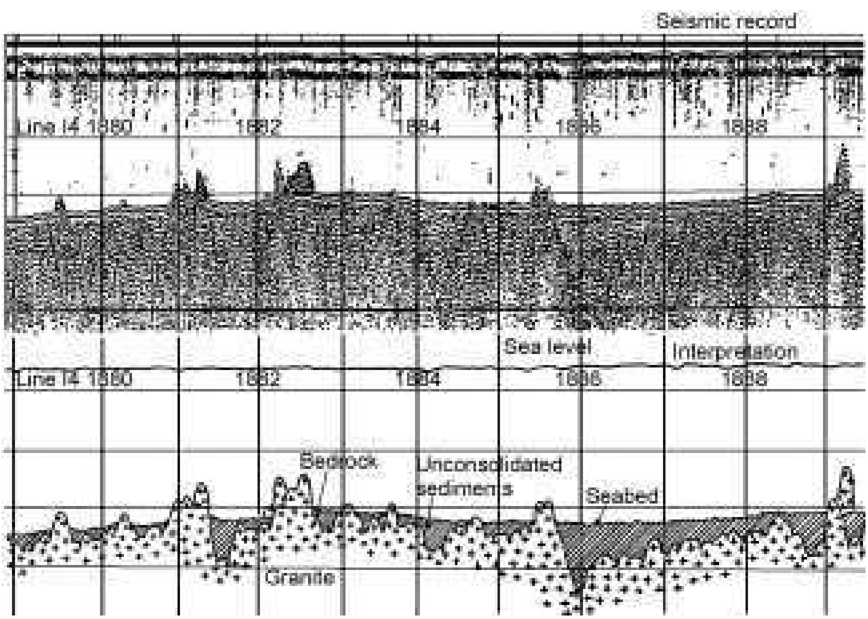
5.26 Schematic representation of a radar wide-angle reflection and refraction sounding and collected data. Arrows indicate direct travel paths of electromagnetic waves through air and overburden and reflected paths from a buried interface. Corresponding events are labelled on the data sketches (from Davis *et al.*, 1987).

overburden and reflected waves from a buried interface. Transmission of the radar signal is governed by the high-frequency electrical properties of the ground, which are primarily controlled by the moisture content of the soil.

The method is still in its infancy and apparently operates best in coarse-grained soils, where the geology is good and the quality of the field data is very good. Penetration depths up to 30 m have been obtained and the above authors claim that stratigraphic horizons have been mapped with spatial resolutions of 0.5 m. Seismic refraction, on the other hand, whilst having lower spatial resolution, will handle a much wider range of ground conditions (e.g. silts and clays) where GPR so far fails.

5.5 Shallow offshore geophysics

Geophysical surveys in shallow offshore areas are based primarily upon seismic reflection profiling with associated magnetics; accoustic mapping and side scan sonar. Variations in measurements of the Earth's magnetic field are mapped as anomalies. Echo-sounding techniques make a continuous recording of depth to seafloor below the survey vessel. Figure 5.27 is a typical seismic profile, which shows features of the granite bedrock assumed to crop out on the seabed of the Andaman Sea, offshore Thailand. A side scan sonar tow-fish is towed behind the survey vessel to identify any changes in the configuration of the seabed that might be attributed to differential weathering of the basement rock. Seabed



5.27 Typical seismic profile in shallow offshore area.

dredgers and sediment samplers – all deployed from a suitable survey vessel, equipped with navigational equipment, towed video cameras, side scan sonar and seismic reflection profiling equipment are used typically for investigating the shallow ocean floor to depths of about 200 m. Geographic information systems rapidly present data from these investigations as maps.

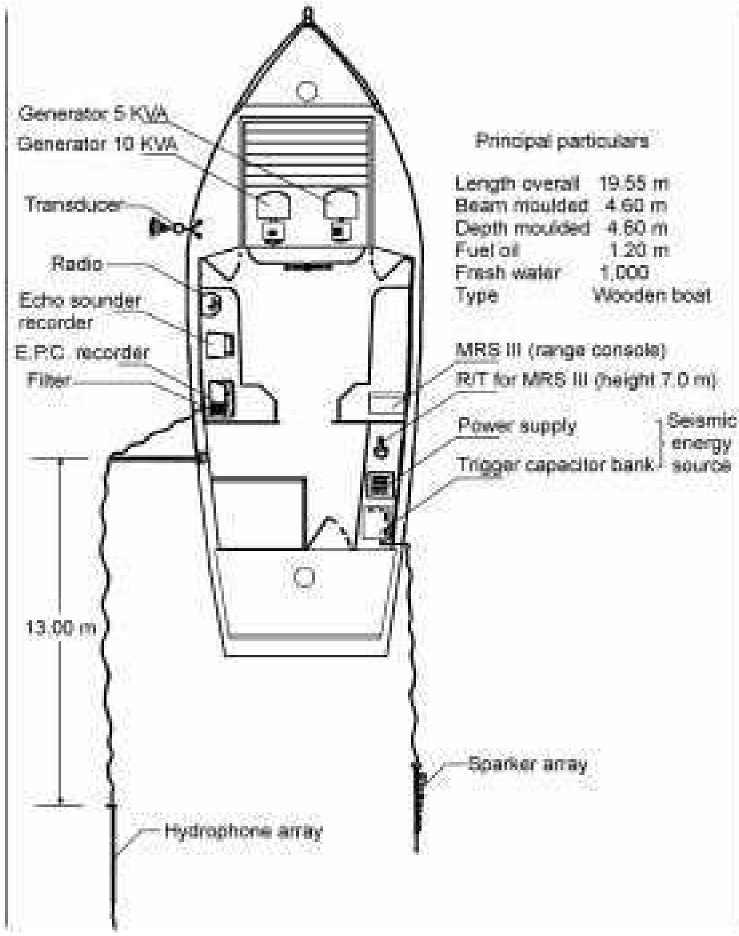
5.5.1 Continuous reflection profiling

The aim of continuous reflection profiling (CPS) is to obtain geophysical information for use in conjunction with geological data to help identify and map the extent of seabed features that might contain concentrations of detrital gold. Information is required of bedrock type and geometry, sediment layering, drowned fluvial systems, channel cut and fill and palaeo-erosion surfaces, drowned sandbars and strandlines. Amongst these features are one or more units of sufficient proportions to constitute suitable targets for follow-up drilling and possibly, additional geophysical surveying. The data relates mainly to former low-level standstill and may also help to elucidate the geomorphic history of the shelf area.

Reflection profiling procedures entail laying out cables on the survey vessel, trials to determine optimum instrument parameters, work scheduling, sound-source hydrophone array separation and tow depth. Consideration must also be given to the influence of the vessel's wake, layout of survey lines in relation to target areas being explored, survey speeds, sea states (waves, tides, currents) and noise levels.

Data are presented to show depth to various horizons and sediment types, contours of depth to bedrock and important sediment layers. Interpretations are made of possible target and/or problem areas and isopach maps and structural interpretation maps are prepared. Seismic patterns are related to the topography at the top of the bedrock and specific bedrock shapes due to weathering effects can be identified and mapped. Rocks such as granite give strong diffractions (rough topography, boulders, etc.); shales have a smooth topography and weak reflection. However, if different 'acoustic bedrocks' can be delineated from the seismic record, their identification can be confirmed only from drill holes Lallier (1985).

Cassiterite exploration in the Andaman Sea geophysical survey was conducted from a survey vessel (Fig. 5.28). The general equipment layout is typical of what is needed for alluvial gold exploration surveys aimed at delineating and mapping extensive palaeochannels on the shallow seafloor. The near-shore Andaman Sea channels were up to 300 m wide, 15 to 20 m deep and extended for several kilometres in length. One zone of inferred coastal dunes was also mapped parallel to the present shoreline. Four groups of break-in-slope occurred in this zone, probably representing drowned shorelines because of their constant depth from profile to profile.



5.28 General equipment layout on geophysical survey vessel.

Energy source

The choice of a good energy source for the geophysical impulses is fundamental and is based on the following premises:

- The depth of penetration of a seismic pulse into the seabed is inversely proportional to the frequency of the pulse, while the seabed resolution is directly proportional to frequency; a single energy source may not be capable of both depth and layer resolution for a particular survey.
- Penetration is reduced or not obtained in seabed soils having high accoustic impedance (e.g. stiff boulder clay in deep water or highly organic soils and peat at any depth); noise from rough seas also blocks out signals.

Systems currently available are sparkers, boomers, pingers and airguns.

Energy systems

Sparkers

Sparkers produce a seismic shock wave by explosive formation of air bubbles resulting from the discharge of stored electrical energy between two or more electrodes immersed in the sea. Multi-electrode sparkers have a comparable depth resolution to light-duty sparkers but with greater penetration. The heavy-duty models are unsuitable for shallow seabed surveys due to long pulse length.

Boomers

Boomers operate by the explosive repulsion of a metal plate, spring loaded against an insulated coil. Passing a current through the coil activates the boomer. They are satisfactory in most applications.

Pingers

Pingers operate with an electric pulse that is generated by the oscillation of piezoelectric or magneto-electric tranducers following the discharge of electrical energy into the transducer. The depth resolution of pingers is good but penetration varies in inverse relation to the grain size of the sediments and may only be 1–2 m in coarse sands and gravels.

Air guns

Air guns produce a shock wave as a result of the explosive release of high-pressure air from a pressure chamber immersed in the sea.

Survey layout

The following range of geophysical and navigational equipment will constitute a typical survey layout:

- navigation system
- seismic reflection profiling system
- marine and recording base station magnetometer
- echo sounding system
- side scan sonar.

Survey procedures

The following procedures are based upon two surveys of platform areas in Thai waters of the Andaman Sea and Gulf of Thailand and one survey in the shallow areas of offshore Bangka and Billiton Islands in Indonesian waters. They are presented here in the form of a checklist with some notes on practical aspects of the fieldwork. The surveys were conducted by the Thai Department of Mineral Resources and the UNDP Project for Regional Offshore Prospecting in East Asia (RAS/771037), (CCOP), and by the Committee for Co-ordination of Joint Prospecting for Mineral Resources in Asian Offshore Areas (CCOPI), Document CCOP XIV/64.

Procedures entail laying out cables on the survey vessel and trials to determine optimum instrument parameters and work scheduling. Other procedures include sound source-hydrophone array separation and determination of survey speeds and an efficient tow depth. Survey lines are planned in relation to target areas being explored taking account of the influence of the vessel's wake, sea states (waves, tides and currents) and noise levels.

Reduction of data

- checking corrections for tide, depth of sound source and hydrophone array
- offsetting of sound source and hydrophone array from navigation, transponder
- checking of intersection of all seismic profiles
- recognition of all important reflection horizons
- correlation with borehole data, or onshore geology near end of lines
- interpretation between intersection points and boreholes Identification of sediment types
- conversion of two-way travel times to depth and thickness
- definition of sub-bedrock structural features.

Data presentation

- profiles showing depth to various horizons and sediment types
- contours of depth to bedrock or various horizons
- isopach maps and structural interpretation.

Interpretation

- results vs. aims of the survey
- recognition of target areas and/or problems
- structural and stratigraphic interpretation – regional correlations.

Identifying bedrock from the seismic records is somewhat hazardous (Lallier, 1985). The differentiation is based upon seismic patterns only, these being related to the topography of the top of the bedrock. Specific bedrock shapes due to weathering effects can be identified and mapped. For example, granitic bedrock gives strong diffractions (rough topography, boulders, etc.), shales have a smooth topography and weak reflection. However, if different 'acoustic bedrocks' can be delineated from seismic records, Lallier stresses that the nature of the bedrock can be identified only from drill holes.

Navigation systems

Accuracy is an essential aspect of positioning and a survey vessel's course must be plotted with a high degree of precision. It may otherwise be impossible to match the borehole locations to the geological/geophysical features they are designed to test. In near-shore locations, Syledis range-range positioning systems are reasonably accurate. The GPS (global positioning satellite) and GLONASS (Russian space forces satellite-based) systems are preferred.

A GPS fix depends largely upon the visibility of enough satellites from which the receiver can obtain survey measurements. With 24 GPS satellites, seven satellites are visible ten degrees or more above the horizon to a vessel in the open sea. The receiver must 'see' a minimum of five satellites to be able to detect an integrity problem. Stand-alone GPS has a demonstrated horizontal accuracy of better than 20 m. This level of accuracy is only available to users through the enabling of 'selective availability' otherwise accuracy degrades to 100 m.

The GLONASS system provides precise navigation and positioning data to users halfway around the globe in a similar manner as for GPS. It has 24 satellites providing two navigation codes but with some important differences including no intentional degradation of accuracy as in the GPS system. A combined GPS+GLONASS receiver, the Ashtec GG24, marketed in Australia by Sagem Australasia Pty Ltd, makes 24 additional satellites available to users. This receiver is stated to enhance the three most important factors for users: availability, integrity and accuracy.

5.5.2 Marine magnetometer survey

Measurements of the Earth's total magnetic field along the survey lines and variations are mapped as anomalies. To be significant, these changes must be associated with inferred bedrock changes. More often, however, they are due to operational factors such as ship turning (i.e. a heading error), diurnal variations and magnetic storms. Trials are carried out to determine optimum field procedures such as distance of sensor behind vessel, heading error check and corrections, recording rate and relation to sampling density and target size, diurnal field monitoring.

Data reduction

- diurnal field corrections
- reduction to datum
- regional/residual separation
- check of line intersections after allowing for offset.

Data presentation

- plotting of values on plan
- contouring profiles and stacked profiles.

Interpretation

- recognition of anomalies
- trends related to geology
- correlation with seismic results.

Ringis (1979) found the method to be helpful in the Andaman Sea survey in only one of three areas surveyed. In this area, significant changes in the magnetic pattern occurred along the regional traverses and appeared to be related to bedrock composition. No sharp changes occurred, associated with inferred bedrock changes in the other two areas of the search.

5.5.3 Echo sounding (bathymetry)

Echo sounding techniques involve continuous recording of depth to the seafloor below the survey vessel. The echo sounder is used at full sweep, usually about 150 m. The main purpose of the work is to obtain information on the morphology of the seafloor. The technique involves the transmission of short sound pulses of pre-selected frequency in rapid succession directed downwards. The reflections of planes of acoustic discontinuity are received, amplified and displayed on continuous recording paper. With the sound source moving, a continuous-profile of the accoustic discontinuities is obtained.

Data reduction

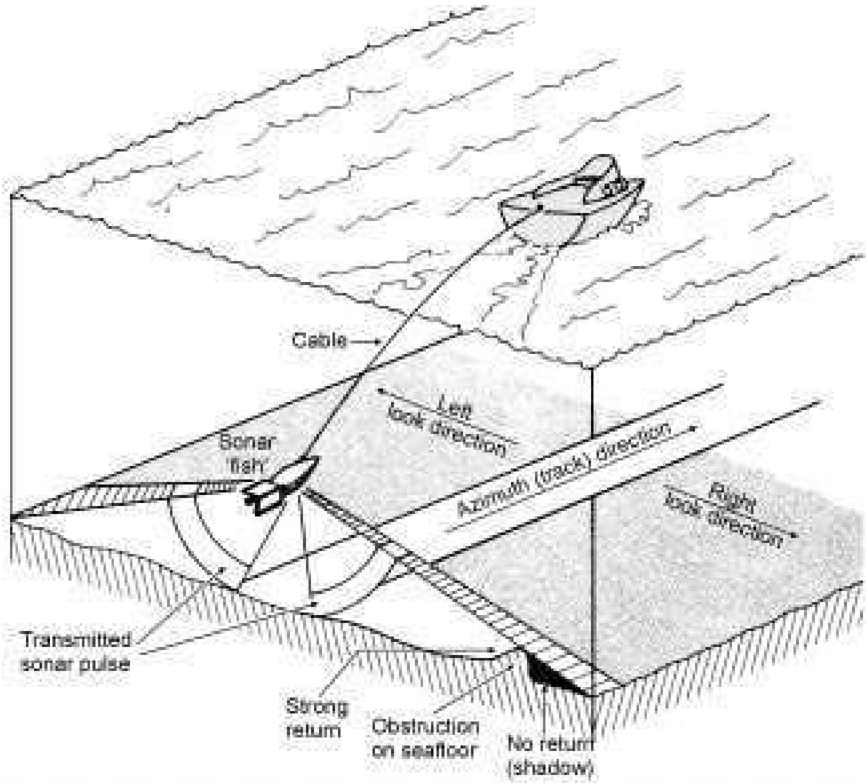
- analogous to topographic maps in onshore studies
- corrections for tides, transducer depth, transducer frequencies.

Data presentation and interpretation

- profiles, stacked profiles and contours
- recognition of break in slope, sand bars, drowned shorelines and other features.

5.5.4 Side scan sonar

The side scan sonar towfish is towed some 60 to 140 m behind the survey vessel to maintain height at about 15 m from the bottom. Holes in the sea floor and small objects projecting upwards by as little as 3 cm can be detected. The seabed is usually covered in sweeps of 30 degrees in width and is described graphically



5.29 Side scanning sonar system (after Sabins, 1986).

in Fig. 5.29. Procedures entail towing the side scan sonar towfish 60–140 m behind vessel to maintain height at about 15 m from the bottom and dual channel recording of pulse length, frequency, range and nature of echoes.

Data presentation and seafloor maps

Interpretation

Types of echoes from different sea bottom features:

- material properties
- slant ranges, distortions and interference
- correlation with seismic profiler and other results.

5.5.5 Synthesis of results

This is a combination of all of the data from the survey including profiles, maps and overlays. Target areas and/or problem areas are identified for more detailed work and consideration is given to the type of follow-up work required.

Case history – regional mapping of Australia's ocean floor

In one major offshore project, regional mapping of Australia's ocean floor is being undertaken jointly by Geoscience Australia, the National Oceans Office and CSIRO Marine Research, which has its headquarters in Hobart. The first section to be mapped is the Southeast Marine Region Mapping Region, which stretches from the sub-Antarctic waters off Macquarie Island, around Tasmania and Victoria up to the waters off New South Wales. The plan seeks to obtain information for understanding and managing Australia's territorial waters, and to assess the geology of the seabed environments as it relates to the known geology of the Australian Continent.

The Cascade Seamount rests on the East Tasman Plateau southeast of Tasmania reaching a height of 1,800 m above the plateau. The great mass of this mountain is believed to have caused the centre of the plateau to buckle, creating a deep depression now filled with hundreds of metres of sediment. The plateau is itself believed to be an ancient fragment of rock from the Australian Continent, possibly related to the much larger feature known as the South Tasman Rise. The Rise is a broad dome of continental rock just to the southwest of Tasmania, a relic of the time when Australia was attached to Antarctica as part of Gondwanaland. Next to the Rise a group of more than 70 seamounts lies on the southern slopes of Tasmania's continental margin. The 450 km long Tasman Fracture Zone along the western flank of the Rise marks the boundary between the deep abyssal floor and the shallower continental rock. Bass Strait, a shallow stretch of water with an average depth of about 60 m, has connected Tasmania with the Australian Continent by a land bridge at various times in the past. The massive Bass Canyon cuts 60 km into the side of the continental margin on the eastern margin of the Strait. The main canyon floor is 4,000 m deep and is connected to the top of the continental shelf by networks of smaller canyons and valleys.

6
Lateritic and alluvial gold sampling

A basic factor in any exploration effort is to increase geological knowledge of the mode of occurrence of gold ore deposits. The second is to examine the surface and near surface geology of any mineralised area and to gain a full set of sample data with measurements that are reproducible within predetermined limits. The principal objectives are to:

- examine selected areas and provide detailed geological data for estimation of resource quantities and grades
- obtain both reliable and representative data for mine planning and design of prototype treatment plant
- identify any morphological or lithological features that might affect proposed methods of mining and treatment
- operate with an eye to possible environmental constraints and produce an evaluation within budget cost estimations that is also environmentally friendly.

Although some deviation from standard methods of testing is normal during evaluation exercises, final studies must provide each constituent part of a deposit an equal chance of representation. Each successive phase of sample evaluation relies upon the results of the previous phase and testing becomes more detailed and systematic as the programme moves towards a final solution. At the outset there can be no certainty as to the eventual outcome and the successful sampling of one potentially viable deposit is inevitably preceded by the rejection of others that fail to meet predetermined standards. This enables work to cease on many unfavourable prospects before significant sums of money have been spent fruitlessly, thus allowing available funds to be allocated more effectively elsewhere. In this respect, the effectiveness of a sampling exercise is properly measured in terms of the time and money expended in arriving at individually correct solutions.

This implies the demonstration of economic as well as geological and engineering integrity at each major successive stage of testing. Safeguards must be built into the system to test the reliability of the data and to guard against any errors of commission or omission that might jeopardise the success of part of or the entire programme. The quality of the data depends upon the suitability of the sampling techniques and equipment and the conditions under which they are used. The adequacy of the data is primarily a function of the number and placement of the samples. Correct conclusions depend upon the methods used for obtaining the samples and their subsequent treatment.

6.1 Sampling criteria

Selected areas of anomalous gold content are prospected in order to determine deposit dimensions in terms of volumes and grades and to identify any morphological or lithological features that might affect proposed methods of mining and processing. Important considerations are:

- delineation of individual zones of high and low gold content
- presence or absence of a water table
- nature of the bedrock
- size range and distribution of the gold
- contamination of particles through textural and other type coatings
- lithology of the surrounding strata.

Exploration geologists are, and should be optimists by nature, but they deal with facts and a favourable potential for economic as well as geological and engineering factors should be demonstrated at each stage of testing. All forms of drilling are costly and field operations must be supervised carefully to minimise both the number of holes drilled and those that must be re-drilled because of faulty techniques or lax supervision. Problems that might arise from adverse ground conditions should be resolved prior to the selection of methods and equipment for evaluation sampling. The churn drill can generally sample with reasonable facility in a wider range of lithologies at a comparatively low first cost than can other drill types but, it must be recognised that other drills may perform more satisfactorily in certain ground conditions.

More exotic geochemical and remote sensing methods might also be needed to reveal the presence of deposits masked by profuse vegetation, or by a cover of later sediments or volcanics. Core drilling techniques are essential for testing residual gold deposits in deeply weathered regoliths where rock type discrimination is a fundamental consideration. The value of fabric as an important means of identifying lithologies of fresh rock, saprock and much of the saprolite is greatly diminished when the weathered rock is pulverised by percussion drilling (Robertson, 1996).

6.1.1 Rules of sampling

Standardisation is essential in all of the sampling techniques used so that probabilities and risks can be evaluated fairly in final economic studies. For example, extraneous materials such as plant fibres and other organic substances, which may be the cause of screening problems (e.g., blinding) in the prototype must be taken into account in any small plant screening processes. Bearing in mind that the eventual outcome of the project depends upon the interpretation of data obtained in the field, very high standards of collection and recording of information must be maintained throughout. The following rules of sampling apply generally to all residual type placers:

- Determine the main deposit characteristics (geographic and geologic) before deciding which sampling method to use.
- Take the largest sample possible commensurate with practical and economic considerations.
- Take precise measurements at all stages of sampling.
- Tie all borehole locations and collar elevations to a common reference point preferably by theodolite survey but never beyond the reconnaissance stage solely by compass and chain.
- Institute checks against intentional and unintentional bias.
- Use corrections sparingly (if at all) and repeat any holes for which the data appear inadequate or badly flawed.
- Standardise all procedures including logging; each drill crew should do and report the same type of things in the same way and each person logging should see and record important features of the samples according to the same set of standards.
- Use sample dressing and analytical procedures that have been proven satisfactory for the particular deposit type.
- Avoid splitting unclassified sample material for quantitative purposes.
- Continue sampling until the additional costs of taking more samples would not be justified by the resulting spread of confidence limits.
- Collate and present all sample data in a form that is suitable for interpretation.

6.1.2 Environmental factors

Climatic, geographic and geological factors are critical to the selection of the most appropriate drilling machines and methods for the particular conditions of the survey. While a selected area might be thought capable of providing adequate quantities of ore at sufficiently attractive grades to justify further testing, the proposed method of exploitation must be environmentally friendly to the particular conditions of the setting. Certain aspects of the project activities can usually be costed out with reasonable assurance, but due allowance must be made for the possibility that presently unforeseen problems associated with complex environmental features may require additional drilling and significantly exceed budget estimates. Possible environmental problems include costs of settling industrial disputes, landowner problems and delays due to adverse weather conditions.

Climatic constraints

Climatic problems that may add significantly to cost and difficulties are:

- episodes of freezing and flooding
- reduced operational efficiency at high altitudes and in sub-zero or intensely high ambient temperatures
- high evaporation rates in desert or windy areas
- cyclonic storms in tropical offshore areas
- possible climatic extremes (e.g., 50 or 100 year events)
- straining financial resources to a dangerous degree when initiating an operation in an unfavourable season because of un-budgeted costs.

The chance that any such environmental problems may arise during the projected life of the mine should be investigated fully at the outset and their possible effects costed out in all ongoing evaluation exercises. Essential information will include historical data relating to the following climatic averages and extremes:

- temperature
  - maximum recorded
  - minimum recorded
- relative humidity
  - range
- precipitation
  - annual average
  - annual maximum recorded
  - monthly maximum recorded
  - daily maximum recorded
  - maximum intensity
- average evaporation
  - summer
  - winter
- winds
  - highest on record
  - normal.

Geographical constraints

Various degrees of difficulty apply to field operations in which features of the surface environment hamper access to field camps or limit mobility around drilling sites. Ideal conditions may be found only where level, well-drained surfaces allow unhindered passage for self-propelled drill rigs and provide firm footings from which to drill. Such conditions are rare and most modern programmes are carried out under such difficult environmental conditions such as those that apply in desert areas, flood plains in humid, tropic regions, frozen wastes of the arctic and sub-arctic and glacial outwash plains at high altitude settings. Logistical efficiency is affected in each case by the harshness of the physical environment and the impaired health of field crews subjected to the hazards of disease, dietary deficiencies and exposure to climatic extremes.

Logistical problems can usually be overcome given ample funds, but only if the prospect is large enough to support the cost. For example, where access entails the construction of roads, airstrips, helipads and barges and other facilities for river transport, only prospects with a sufficiently large economic potential can justify the high cost of ensuring conditions under which men and machines can work safely and well. Significant cost pertains also to the movements of drill rigs in and around the drilling sites where mobility is affected by surface features such as swamps and waterways, and by the low bearing-strength of some soils, particularly when wet. Drilling difficulties and costs increase proportionally with increasing drilling depths. The following case histories illustrate geographical problems involved in drilling in swamp areas and at high altitudes.

Swamp areas

Alluvial gold prospects in low-lying swampy areas are usually difficult to access. Under favourable drainage conditions swamps can be drained and the ground kept reasonably dry during the course of the drilling. In other cases, a suitable means of access must be devised that will allow men and equipment to move freely in swampy areas in and around the selected drill holes. Two of the many examples experienced by the author were at Ampulit, Kalimantan, Indonesia, and in the Mt Kare Alluvial Gold Field, Papua New Guinea.

Ampulit

The prospective area included mangrove swamplands traversed by many small streams. Drilling access was obtained in wide stream sections by sampling from floating platforms and in narrower sections by building causeways and platforms across the low-lying areas and streams using local timbers (Fig. 6.1). In Fig. 6.2 a floating, mechanical Banka drilling platform was used successfully to sample gravel beds to water depths of up to 12 m. The light structures shown in the figure were appropriate for the depth, but would not have been strong enough to support a deeper drilling churn drill, nor would they have been sufficiently rigid for the task of jacking up the longer and heavier casing. A force of as much as 50 to 100 tonnes could be required to lift strings of casing from deeper holes. The

6.1 Local timber access for drilling across swamp; Ampulit, Kalimantan, Indonesia.

6.2 Floating drill platform for Banka drill; Ampulit, Kalimantan, Indonesia.

extractor and its mountings would have had to be able to resist that force, without sinking into the ground. The shallow and narrower sections were drilled from lightly constructed platforms using mobile lightweight Banka drills. Low-cost local labour was available and construction kept pace with drilling at around US$20 to US$30/day/crew.

Mt Kare

The swamp area at Mt Kare is covered by water to depths of 0.5 m and more for protracted periods of time. The ground is too spongy for economic road making and flotation channels were cut across the deposit to float a drilling barge using a platform-mounted backhoe. The underlying wash was sampled using a barge-mounted Banka drill in place of the backhoe.

Drilling at high altitudes

The mechanical efficiency of diesel engines falls away with increasing elevation. Power is lost progressively as the atmosphere thins with height and less oxygen is available for combustion. De-rating at high altitudes is difficult to quantify because of the wide variability of oxygen supply at any one level according to differences in ambient temperature and barometric pressure. De-rating formulae supplied by manufacturers can usually be relied upon only for elevations up to about 2,000 m above sea level based upon standards such as SAE T1349 Standard of 100K Pa (29.62″ Hg) and 25 ºC (77 ºF) and DIN 6270 Standard of 97.8K Pa (28.97″ Hg) and 20 ºC (68 ºF).

Such formulae compare the effects of temperature, pressure and density variations on the performance of engines at sea level and at specific altitudes above sea level according to standard formulae. Weather changes are typically inconsistent and may give variations of plus and minus 20% of average ratings. Land temperature and pressure variations that occur throughout the day and from season to season make it difficult to establish realistic corrections for consistent use. The problem is not one of mathematics (empirical formulae are available for most atmospheric conditions) but is due rather to the lack of comprehensive meteorological data describing the range of atmospheric conditions likely to be experienced in the locations concerned. Violent changes in atmospheric conditions are common at high altitudes and must be allowed for, nevertheless, the general tendency is to underestimate the extent of the required de-rating corrections.

A case in point was the performance of a Schramm drilling unit (refer to Figs 6.17 and 6.18), which operated in a sampling programme at 4,200 m above sea level in the Peruvian Andes. The drill was proven in its performance at low altitudes where it had ample power for most contingencies. At 4,200 m elevation the compressed air output was inadequate for the simultaneous operation of the rotary drilling equipment and casing hammer. The rig carrier also lost much of its climbing power and frequent bulldozer assistance was needed when moving from one drill site to the next. Although the nature of the problems had been recognised in advance, their possible extent had been badly underestimated. Hence a number of costly delays occurred until solutions were effected. The installed air capacity was supplemented using a large auxiliary compressor and a change in transmission gearing allowed the rig carrier to move freely under its own power, albeit more slowly.

Geological constraints

While most onshore geographical problems can be foreseen and compensated for prior to drilling, fresh geological problems, which occur daily, can neither be avoided nor wholly compensated for in advance. No one feature can be dealt with in isolation; the effects of one are closely inter-woven with the effects of others. Techniques and equipment that suit one deposit may fail if applied to a different deposit. The Kinta Valley Syndrome, i.e., employing a particular method, say Banka drilling for no other reason than that it has been used successfully elsewhere, can have disastrous results if it is not equally adapted to the geology of the new deposit. Difficulties are associated with such geological features as:

- alternate sediment layers having widely differing lithologies
- confined aquifers, i.e., water saturated layers of sand and gravel bounded on their upper surfaces by impermeable strata
- auriferous beds in which water logged sands and silts flow freely during any pressure gradient change
- false bottoms, which may be continuous or discontinuous
- barren sediment basements such as hard crystalline bedrock with rock bars and pools that are difficult to identify.

The effectiveness of drilling and sampling in an offshore environment depends additionally upon meteorological and oceanographic factors related to differences in the strength, direction and frequency of windstorms; changing characteristics of waves, tides and currents and variable water depths, distance from shore and seabed conditions. The motion of a drilling ship is profoundly affected by wind-generated waves and it is essential to understand the nature of possible day-to-day variations in the strength and direction of winds as well as the seasonal changes.

Testing sequence

Although there is some overlap, prospecting typically proceeds in stages from early reconnaissance through scout testing to final evaluation. Each successive phase of evaluation will depend upon encouraging results from the previous phase and testing will become more detailed and systematic as the programme moves towards a final solution. At the outset there can be no certainty as to the eventual outcome and the successful sampling of an economically viable deposit is almost inevitably preceded by the rejection of others that have failed to meet predetermined standards. This enables work to cease on many unfavourable prospects before significant sums of money have been spent fruitlessly, thus allowing available funds to be spent more effectively elsewhere.

Practical aspects of evaluation relate to the order of testing and the collection, recording and plotting of data in a suitable form for interpretation (Macdonald, 1983a). The reconnaissance phase of the programme is directed primarily towards testing geological hypotheses formulated from the results of regional exploration and background data related to the provenance of the gold, and to local geology. The ongoing programme of scout testing implies that the results from reconnaissance will have largely confirmed the initial geological predictions and that the proposed scout-testing programme will provide the foundations for obtaining the final proof. If the scout-testing phase of an evaluation programme measures up to the overall project requirements it will be possible to state with reasonable confidence:

- resource quantities of ore in terms of approximate deposit dimensions and grade
- the most suitable pattern of testing for close gridding
- any changes that may be needed for upgrading the methods and equipment used for scout testing
- any special sampling or minerals dressing equipment that may be needed to achieve a high standard of accuracy in samples taken for final evaluation.

The project should of course be reviewed at intervals and terminated abruptly if it becomes obvious at any time that the investment criteria will not be met. Close gridding, as an extension of scout testing basically fills any remaining gaps with additional factual information and a formal valuation commences with the deposit roughly defined in three dimensions. Cost is still an important consideration but it is false economy to take short cuts or to accept grid spacings that are excessively wide to minimise costs expense when basic uncertainties have still to be resolved.

Major undertakings will usually benefit from detailed geophysical test work to help direct the pattern of sampling and so resolve any doubtful issues. Until the geology is known, interpretation of sample results must rely upon geological assumptions that may later be proven wrong. Geological understanding implies the ability to construct a model for predicting within acceptable limits the results that can be expected from further sampling anywhere in the area tested. This position is arrived at only when the results from each line or pattern of holes are consistent with the results from adjacent lines and when individual sample results and measurements are consistent with the results and measurements of their neighbouring samples.

Statistical methods of assessment are available from which to determine the adequacy of sampling within reasonable limits and so avoid taking more samples than are needed. The final document should be an authoritative statement describing all relevant aspects of the geology and geometry of the deposit, the size of the resource and its grade, the distribution of the gold and the processing characteristics of the ore for eventual prototype mining and treatment plant design.

6.1.3 Recording and communication

Evaluation commences with the collection and retrieval of masses of information and measurements, recorded in logical sequence to provide easy access when required. Computerisation improves the handling of exploration data, linking it simultaneously at headquarters with other needs of the organisation. Equipment costs have fallen dramatically and the ready availability of mini and microcomputers and microprocessors of laptop size and smaller, allows explorers to be more self-sufficient in such matters as mapping and contouring and in the modelling of geological, geophysical and economic data. Available programs promote the production of geological and geophysical graphics while retaining the ability of interaction with a centralised system. De Vletter (1983) suggests that the main problem is to find skilled personnel to collect, evaluate, compile and use the information effectively, rather than to find and purchase the equipment.

Design of data forms

The ability of the designer to foresee the essential data requirements and to design the forms accordingly is reflected in their usefulness. The layout of the form should allow results to be noted in logical sequence so that when collated, they can be readily manipulated into a suitable condition for interpretation. Concise and accurate records are kept of the data from drilling and sampling and specific forms are designed for each operation. The standardised recording of information ensures that all samplers record information as far as possible in the same way during the course of an exercise. The data must be sufficiently detailed to achieve the desired level of comprehension for each operational phase. The range and scope of the individual forms should collectively provide an up-to-date picture of the prospect at each stage of its development. Note that while it is advantageous to use the same data sheets throughout a programme without change, small adjustments may be needed. The sampling stage is also a learning stage and some revision may be needed to ensure that the selected methods will work and that they will provide all of the required information. If changes are made, the revised forms will reflect and record those changes and all previous information should be transferred onto them.

Observation and recording

Drilling and sampling log sheets provide most of the required data for mine planning and much of the required information for treatment plant design. In conjunction with the laboratory log sheets they contain a full set of data from which to compute resource quantities and grades. Thus, logging at the drill site and laboratory processing of pit and borehole samples has three main purposes:

- to provide sufficient additional data for the computation of resource quantities of ore
- to provide quantifiable information on the lithology of the strata for geological interpretation and mine planning and
- to establish parameters for the design of prototype treatment plant.

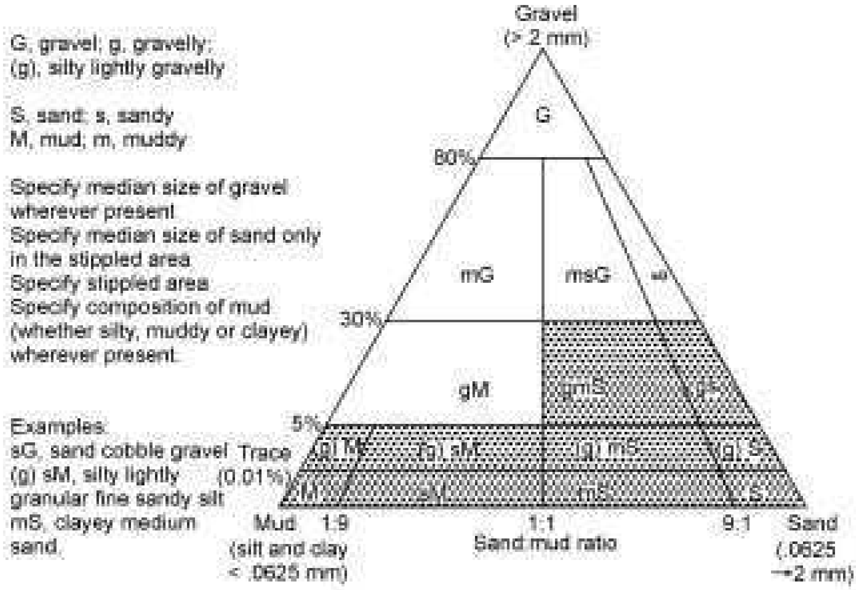
Driller log sheets

These log sheets describe the drilling data for each hole, for example sample date, grid co-ordinates, borehole collar elevation, length and inside diameter of the casing shoe, inside diameter of the piping and map sheet reference number. Critical measurements for individual sample intervals are plug length, core rise, and drive length. Any drilling difficulties encountered are listed in the log together with the action taken to resolve them. Lithologies are delineated spatially and in a consistent manner for geological interpretation and mine planning.

A suitable standard is based upon a ternary system with gravel, sand and mud at the apices. For example, fifteen separate lithologies are described in the Folk diagram (Fig. 6.3) according to their relative percentages of gravel and mud:sand ratios. Each lithology has its own shorthand abbreviation, which can be expanded by the logger in the descriptive log. Thus, a layer of muddy sandy gravel (msg) might be described more specifically as clayey, medium sandy, volcanic gravel for geological interpretation. Facies characteristics favourable for gold deposition are related to clast size and type.

Sample dressing log sheets

Logging in the sample dressing shed should be aimed at providing quantifiable information on the lithology of the strata for mining and process design. Sieving and water displacement techniques are used to size and measure the granular fractions; mud volumes are estimated by difference. Important aspects of the logging relate to the physical nature of possible plant feed materials. Log sheets



6.3 Standard logging alluvials (after Folk, 1980).

will show sample volume recoveries and record the results of sample dressing for each interval tested. Volumes of coarse and fine fractions and numbers of gold particles identified in the pan will be recorded. Estimates of the percentage content and distribution of clay will be required for mine planning and slimes handling.

Laboratory log sheets

Laboratory measurements and investigations should be designed specifically for the design of treatment plant units. Laboratory log sheets record all important measurements such as the precise weights of the recovered gold, the number, grain size distribution and shape characteristics of gold grains recovered from each sample. The gold fineness is usually determined from the analysis of composite and check samples in an independent specialist laboratory. Observations are made of the types and percentage of other heavy minerals in each sample for further reference.

The establishment of precise standards of observation allows all observers to see recorded information in the same way. Objectivity is a basic requirement, and it is essential that what is observed is described clearly and accurately for others to see and understand. Any differences of opinion should be resolved before observations are translated into factual data for recording in maps or tables. Even subtle changes in lithology or in mud colours from one drill interval to another might be of fundamental value to the interpretive process.

6.1.4 Thoughts on interpretation

Thoughts on interpretation should be built into the recording system at the outset and not left until later in the survey when it might not be possible to remedy errors of omission without repeating much of the work. In such cases, the lack of specific data might be more harmful to the interpretive processes than small mistakes in certain of the measurements. A fully integrated day-to-day appreciation of field operations allows any necessary programme changes and other revisions to be made with a minimum of delay. The data are mainly map-orientated and the speed with which the required plans and sections can be prepared is important in determining how soon they are presented for consideration and review. When done manually the process is tedious and time consuming, particularly when revising important geological features or transferring data from map to map. The task is carried out much more expeditiously using interactive computer graphics and computer-stored information to create the required diagrams and other graphic illustrations.

The computerised method has many advantages over hand methods of illustration. The artwork is easier to correct and manipulate and illustrations are free from blemishes such as smudges and erasures. The use of personal computers allows geologists to prepare their own illustrations and develop their own ideas without the delays caused by working through a drafting department. This reduces the number of cases in which expertly prepared hand illustrations are needed. Dramatic improvements have been and are being made in all phases of design including computer memory capacity, access speed, hardware plotting capabilities, software efficiency and sophistication. The major trends are towards producing increasingly powerful and less expensive hardware with more easily used software and more readily generated graphics.

Extremes of interpretation are twofold: intuitive and inferable. Neither approach is entirely satisfactory because there will almost certainly be differences in the operational efficiencies of the field crews and their supervisors. It can be expected that the average skills of workers in an established goldfield will be greater and better applied than those of workers in new locations particularly in remote areas. Management capabilities also vary and different perceptions of the relative merits and demerits of available technologies may profoundly affect the conduct of the sampling process and the quality of the samples.

Intuitive approach

The intuitive approach relies upon the application of correction factors to correct sample data that is inherently inadequate and which may also be carelessly gathered and presented. In a particular placer environment, continued experience by trial and error may enhance the confidence with which a particular correction factor is applied. However, it does not necessarily confer a similar confidence in that factor when applied to sample data from other environments. There may be fundamental differences in geological and geographic features such as the nature and distribution of the gold, sediment lithology, stratigraphy, deposit depth, bedrock geology and the physical environment generally.

Because of the nature of alluvial gold sampling, the application of experience factors tends to increase the uncertainty of any predictions made. There will always be some bias, intentional or otherwise. Even when no dishonesty is intended, problems may arise from data that is not significantly flawed but for which the interpretive processes are flawed. This may occur when correction factors are applied rigidly to data regardless of their quality, simply because someone has used or are stated to have used the factors successfully elsewhere.

Conversely, where dishonesty is intended, factors may be applied to the raw sample data simply to bias the results, regardless of any truths, facts or premises. It is not unusual for data from highly suspect drilling and sampling exercises to be dressed-up using the guise of supposed experience factors, to give an appearance of professional expertise combined with an apparently proper degree of conservatism.

Inferable approach

Implicit in the inferable approach is a perception of the interpretive process as one in which reasonable inferences are drawn only from data and premises that stand up to critical examination. Possibly flawed data is either rejected outright or relevant sections of the work may be repeated and expanded as required to resolve any apparent aberrations or anomalies. It is accepted, however, that no one sample is likely to be representative of the orebody as a whole. By itself, a single sample means very little and only a statistically adequate population of samples provides reliable data for interpretation.

Viewed in association with geological maps and sections, an initial assessment of the mining resource will be made primarily on the basis of:

- the nature of the depositional setting
- apparent trends of paystreaks (high gold values)
- deposit dimensions and location.

The ease and reliability of interpretation depends largely upon the manner in which the above data are organised and presented. Evaluation is an ongoing process and the relevant displays should be successively updated and recorded as a permanent record as further results come to hand. The sampling apparatus may be relatively crude but careful observation of the results will provide all essential measurements of the gravel, sand and clay contents of the ore and the physical characteristics and percentage of gold grains in the heavy mineral concentrates. Columns of figures identifying important facts and features may lack the visual impact of graphic illustrations when plotted on a map and are not necessarily meaningful when viewed in that form alone. Hence useful graphic art forms for data analysis will usually include histograms which present the borehole data. Normally, such figures show the weight of gold recovered from each sample and the conversion of that weight to 'simple' grade (refer to Section 6.5.1) for each interval sampled.

Note that selection of plant and equipment items may be made confidently only when the mineral-processing engineer is able to write down all the quantitative data needed for the design of each component.

6.2 Prospecting methods onshore

Residual and alluvial gold deposits are tested using manual, semi-mechanical or fully mechanical devices, either alone or in combination depending upon practical and economic considerations. The methods include drilling, pitting, trenching and bulk sampling. For small surface deposits the samples may be obtained by hand and processed by panning. For large deposits, no single procedure has the ability to satisfy all of the objectives of the programme and a combination of methods will be used to arrive at a final evaluation. Certain aspects of the methods and techniques for sampling placers, as described by Harrison (1946), Wells (1969) and Macdonald (1983a) are reviewed and enlarged upon in this chapter for the specific problems posed by residual placer gold ores. In all evaluations, the success or otherwise of the investigations will depend initially upon the choice made of sampling methods and equipment.

6.2.1 Pitting and trenching

Sampling from pits and trenches in shallow, dry conditions is cheap and reasonably accurate and the procedures used are also suitable for excavating large bulk samples in dry ground for bench-scale testing. A high order of accuracy is not needed at the reconnaissance stage, but if more representative samples are needed, the selected methods and equipment must be capable of sufficiently accurate and detailed sampling for the required conditions of the survey. The sampling methods include pit sinking with or without the aid of caissons, and trenches excavated at intervals across the deposit by bulldozers or backhoes. Samples may be taken either in bulk or from vertical samples at selected intervals.

Pitting

In good standing ground pit samples closely represent true sections of the material penetrated with the dimensions of either solid cylinders or rectangular prisms with vertical sides. The pits may sometimes be sunk without wall support, but adequate safety precautions are essential if there is any possibility of the sides caving or of rocks being displaced from the sides. Potentially dangerous horizons are timbered in deep pits. Caissons, used for sinking shallow holes, are quite safe in any ground conditions and have the added advantage of providing accurate sample measurement. Flush-jointed steel caissons are generally suitable for sampling river bars to depths of 4 to 4.5 m. Telescopic-type caissons, which allow pitting to be carried to greater depths, reduce in diameter sequentially as sinking proceeds.

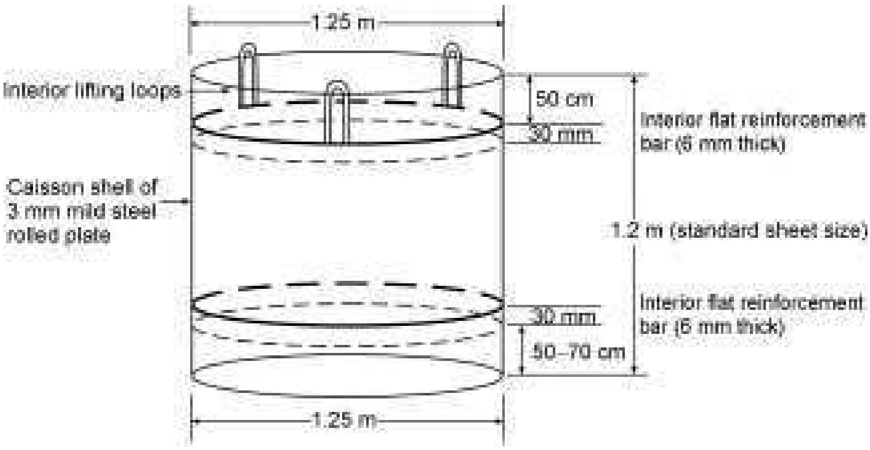
Flush-jointed caissons

Flush-jointed caisson sections are fabricated as complete cylinders in two or more segments with vertical lugs, which allow them to be bolted together. Figure 6.4 shows the driving of a flush-jointed caisson section at Punna Puzza, Kerala State India. Caisson cylinders are usually fabricated in 1.0 m lengths with horizontal lugs to allow them to be connected together. The leading caisson is fitted with a reinforced cutting shoe to protect it when driving.

Telescopic caissons

Telescopic caissons are fabricated from 3 mm thick steel plate rolled and welded into cylinders (Fig. 6.5). The usual height of each cylinder is 1.0 m but may be longer to suit the dimensions of the steel sheets used. The caissons fit (telescope)

6.4 Driving a flush-jointed caisson, Punna Puzza, Kerala, India.



6.5 One section of telescopic caisson shell.

one inside another. An example of how a worker operates inside progressively smaller diameter caissons with depth is demonstrated in Fig. 6.6, taken at Ampulit, Indonesia.

The total depth of a pit is determined by the diameter of the top caisson section. Typically in shallow ground the top caisson has an inside diameter of 1.25 m. Caissons reduce in 50 mm diameter increments for each 1 m sinking

6.6 Reduction in caisson diameter with depth.

depth. Optimum digging conditions are provided when the fit between caisson sections is tight without binding.

Caisson digging procedures

Commencing with the leading, largest caisson the ground is first excavated to a depth of about 200 mm within the caisson, slightly in from the walls, and then enlarged laterally as each caisson is tapped down to a new floor level. Both flush-jointed and telescopic caissons are forced down into the ground by undermining the lower rim and tapping the section down as the sample is removed. Working outwards from the centre helps to limit run-in from outside the caisson walls.

The procedure is repeated until the top of the first caisson section is level with the ground surface. For flush-jointed caissons, the second section is bolted to the first section. For telescopic caissons the next largest section is dropped down inside the previous section. Pitting is continued in both cases until the pit is completed to its full depth. Samples may be recovered at predetermined intervals or at intervals determined by changes in lithology. *In-situ* (bank) volumes are calculated from the pit measurements. An alternative method of sampling involves reducing the size of samples during sinking. An open-ended regular shaped box or cylinder fabricated from 2 mm or 3 mm steel plate is hammered into the ground simultaneously with sample extraction, thus avoiding any contamination. Box dimensions, commonly 300 mm square by 300–400 mm, may be larger if a coarse gravel wash is being dug.

Excavated materials are measured as loose volumes using for convenience an open-ended box placed on a sample mat at the surface. When full and levelled at the top, the box is lifted off the mat leaving the sample behind in a small heap. The process is repeated until the entire sample has been measured. Volume boxes typically measure 500 mm x 500 mm x 500 mm (internal dimensions) so that the number of heaps divided by eight is the volume in loose cubic metres. This volume, compared with the measured pit volume provides a factor for calculating swell. Over the course of a pitting programme, the swell figures provide useful data for mining and treatment plant design. The sample measuring box is shown in Fig. 6.7 at a sample-dressing site at Krung Cuk, Aceh, Indonesia.

Potential sources of error

Contamination from run-in or from rising sand can usually be reduced to acceptable levels by isolating and removing non-sample material separately. The main sources of contamination may be due to:

- dislodgement of wall sediments caused by the action of tapping or hammering caissons down into the ground
- dislodgement of wall sediments when removing large stones that protrude into the pit from outside of the caisson

6.7 Pit sample measurement box at dressing site, Krung Cuk, Aceh, Indonesia.

- surging due to strong water flows in loosely compacted gravel and sand
- rising sand and gravel.

The digging procedures may be modified to suit the particular conditions as follows:

Dislodging loose wall material

Reduce the driving interval to minimise the gap between the floor of the pit and the bottom casing edge. Keep the floor clean around the edges and remove run-in as it occurs.

Projecting stone

Clean the pit bottom in the vicinity of the stone and remove it, first by undermining and then by wedging. Break off that part of the stone that projects into the pit and include it with the sample. Discard any wall material that has run into the pit from the sides.

Rising sands

The effects of rising sand are manifest in high sample recoveries, which reduce the validity of the grade calculations. Contamination by rising sand may occur during periods of lengthy shut down such as overnight or at weekends. Pitting non-stop from where rising occurs until the hole is completed may minimise the problem.

Wet running ground

Dig a pump sump into the pit floor towards one side of the pit. Use a submersible pump, driven from the surface, to keep the water level at or below the pit bottom. Utilise a series of such sumps, each replacing the previous one as sinking proceeds. Note that:

- The pump engine must be kept at a safe distance away from the pit-top to avoid danger from fumes entering into the pit; carbon dioxide is a particularly hazardous gas and being heavier than air will accumulate towards the pit bottom and may cause asphyxiation.
- Any solids entrained in the pump water can be settled out in sumps at the surface; gold recovered by panning this material may be included with the gold recovered from the interval it is derived from.

Trenching

Trenches may be cut by hand but the operation is usually carried out mechanically using back hoe excavators or bulldozers. The method is suitable either for bulk sampling in good-standing, shallow ground or for channel sampling when bulk-sample treatment facilities are not available. Channel sampling involves cutting vertical grooves into the walls at regular intervals along the trench using a template of the correct size. Sectional dimensions range from widths of 100 mm to 300 mm and depths up to 500 mm. The vertical intervals are determined by the thickness of individual layers. The lateral spacing of the channel may be at some predetermined interval such as 5 m or be lithologically orientated. Similar techniques to those used to dress borehole samples are used to dress channel samples. The method is labour intensive but is usually satisfactory for testing shallow, good-standing ground.

6.2.2 Drilling

The most common drill types used for sampling alluvial gold deposits are churn drills (Keystone, Banka, etc.), pit digging drills (bucket, and Bade type) and rotary drills. Ground that cannot be drilled satisfactorily by any known drilling method may be tested using a small-scale mining operation. An operation at this scale would normally be expected to return most of its costs in the sale of gold won. Depending upon the outcome, the mining operation would then be expanded to full size, or abandoned.

Churn drills

Despite serious shortcomings, churn (cable tool) drilling rigs are still the most widely used and accepted device for drilling gold placers and the method has a better record of success in a wider range of ground conditions than any other form of surface drilling. Significantly, most of the economically important alluvial goldfields of the world have been evaluated using churn drills, or their smaller versions, the Banka drill of southeast Asia, and the Ward drill of South America. The size and power of primitive churn drill types used for testing alluvial gold deposits about 75 years ago restricted casing diameter to 100 mm. Keystone, Bucyrus-Erie and similar rigs were later developed for drilling in casing sizes up to 250 mm diameter. The most common size is 150 mm internal diameter casing with a cutting shoe of diameter 180 mm to 190 mm.

Penetration rates for alluvial drilling average about 3 m to 10 m/shift depending upon depth and the type of ground. Drilling depths in wet ground are commonly restricted to around 50 m below the water table, i.e., to the economic depth of bucket line dredging. Drilling depths under dry conditions are constrained only by economic consideration. Indeed, using a large cable tool rig equipped with a 'walking beam' the author sank one exploratory hole for oil in Gippland, Victoria to a depth of 540 m.

Small-scale churn drills are represented by hand (Fig. 6.8) and mechanical Banka type drills by Fig. 6.9. Small-bore hand Banka drilling was found to recover a good cross-section of lateritic material in a semi-dry condition at Royal Hill Suriname to depths of up to six metres. The mechanical Banka is intermediate in size and performance between a hand-operated machine and a full-sized churn drill. The driller uses a manually controlled snubbing winch to provide the percussive chopping and bailing action. The casing is hammered into the ground using the bailer attached to the sinker bar as a guide. After driving for a measured distance the drive head is unclamped and the sample is recovered by bailing. The bailer is fitted with a clack valve inside a removable chopping shoe to retain the sample during the pumping action. Drilling rates to depths of 20 m average 3–8 m/8 h.

An improved mechanical Banka drill developed by Sparkes (1990) is driven through a five-tonne marine-type winch powered by a 15 hp diesel engine. The winch has a heavy-duty clutch capable of handling the required extra drive weight for drilling inside 200 mm casing; it also provides the additional strength required for casing extraction using multiple line blocks. The drill rig is shown in travelling mode in Fig. 6.10.

Operating procedures

Operating procedures for all types of cable tool drills are as follows:

- Site the rig accurately with the mast or tripod centred over the sample point.

6.8 Hand Banka drilling in beach sands, Keimouth, South Africa.

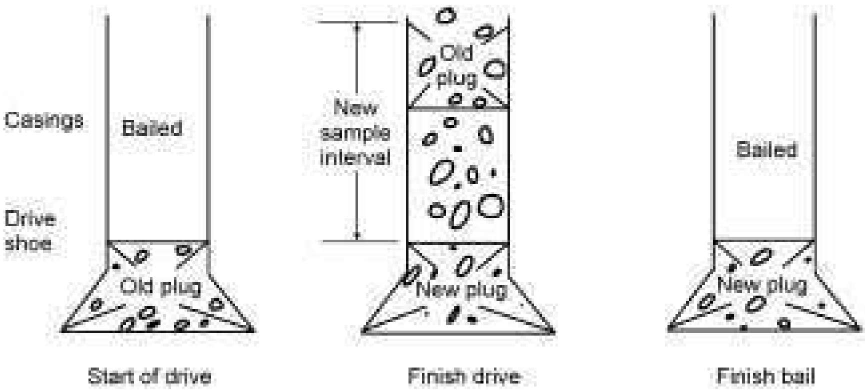
6.9 Mechanical Banka drill.

6.10 Sparkes mechanical Banka drill in travelling mode.

- Dig and sample the first 500 mm by hand and set the first length of casing (fitted with casing shoe) vertically under the drill string.
- Screw the casing head onto the pipe and drive the casing down to ground level; measure and record both the driven depth and the core rise, i.e., the height to which the sample has risen in the casing as a result of the drive.
- Recover individual samples by bailing, maintaining a plug of material in the shoe and lower part of the casing to guard, as far as possible, against contamination from in-flow or from inadvertently recovering core from outside the casing (Fig. 6.11).
- Measure and record the settled volume of the recovered sample.
- Repeat the procedure until basement is reached; pump out and recover the plug; drill far enough into the bedrock to confirm that it is the basement and not a boulder.
- Measure the depth drilled and record.

Figure 6.12 shows how errors may occur because of boulders on the streambed. Borehole A, which just misses the boulder at its upstream may recover only fine sediment largely depleted of gold which has concentrated under and leeward of the boulder. Borehole B, which is located directly upon the boulder will underestimate the depth of wash at this point and fail to recover any of the gold directly beneath it.

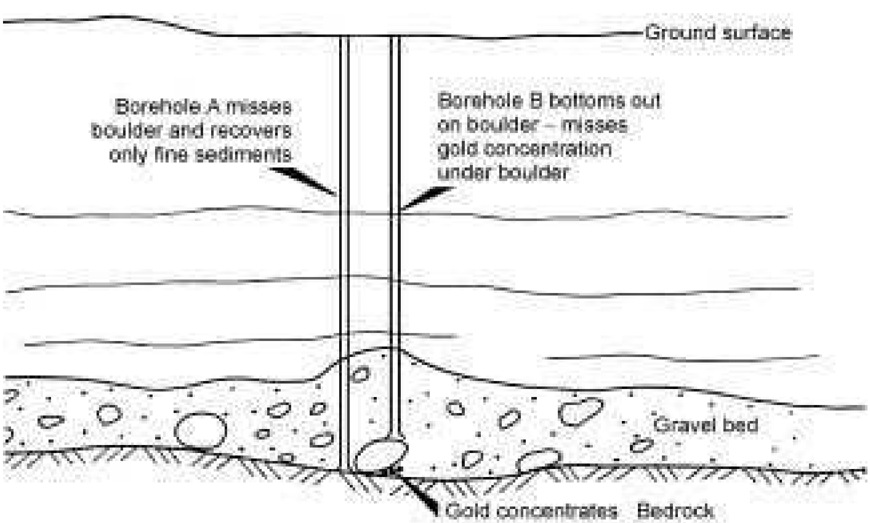
Figure 6.13 demonstrates the effects of different borehole diameters on sample recovery. Constant clogging will occur when some clasts are larger than



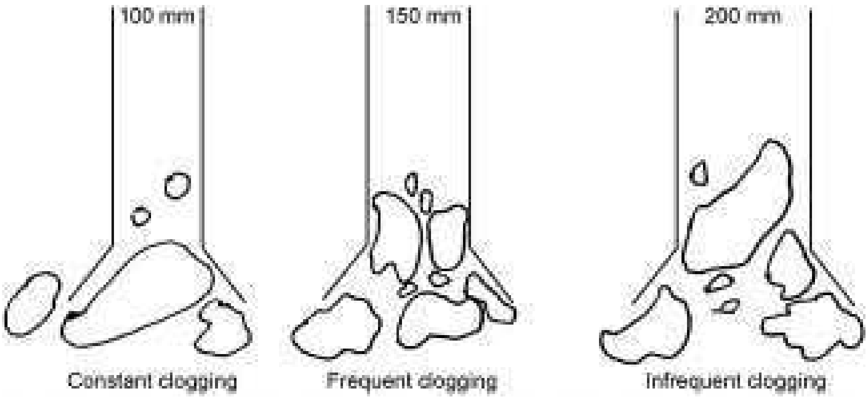
6.11 Plug retention in travelling shoe.

the borehole aperture. Frequent clogging will result from a slightly larger aperture but only when the borehole is significantly larger than the largest clast is drilling virtually unimpeded. The volume recoveries of the samples will be characteristically higher the larger the diameter of the borehole. The average grades of samples taken in all such cases will probably be increasingly understated in the reverse order of borehole diameter. In extreme cases as shown in a section of the Yakatabari Creek at Porgera, Papua New Guinea (Fig. 6.14) no form of drilling would be practicable.

Drive distances are generally at the discretion of the driller who will try to maintain a consistent core rise during drilling. If the core rise is low, say <75% of theoretical he should reduce the drive distance. This problem will normally



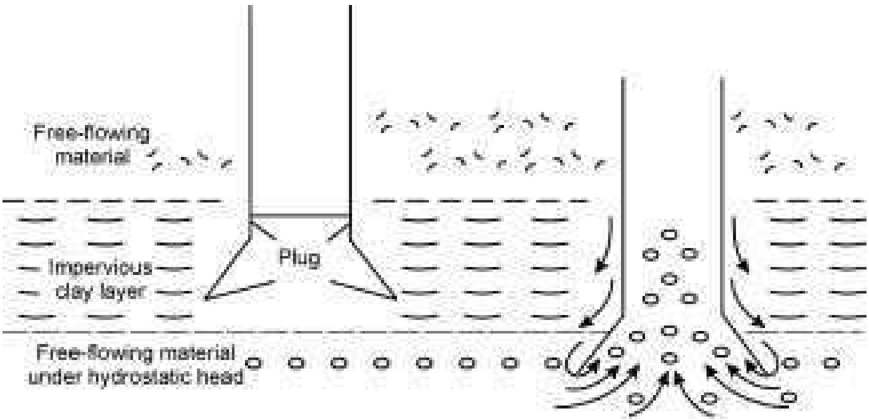
6.12 Uncertainties of drilling in bouldery ground.



6.13 Influence of borehole size on clogging.

result either from the presence of boulders or from a preponderance of cobbles in gravel horizons. If the core rise is >125%, such as when breaking through an impervious layer into a layer of loose unconsolidated sand under a strong hydrostatic pressure head (Fig. 6.15), the ground is most unlikely to contain significant amounts of gold. The drive will usually be continued without recording a sample until a gravel horizon is reached and backpressure in the casing restricts the inflow problem thus stabilising the rise to about theoretical levels.

6.14 Unsampled section of Yakatabari Creek, Porgera Papua New Guinea.



6.15 Gush of material rising into casing when breaking through impervious layer.

In wet ground the casing will be kept full of water to maintain a hydrostatic head against excessive inflow. In itself this may not be enough to hold back rising sands and the core length must be adjusted to suit the conditions. In dry ground, only sufficient water should be needed to slurry the drill cuttings; more than this could provide an excess of head within the casing, which might force some of the sample back into the ground.

Pit digging drills

Large samples recovered by pit digging drills are theoretically more reliable than smaller samples from other drill types, and have the additional advantage of taking individual samples that are large enough for pilot-scale test work. Disadvantages are:

- lack of mobility around the drill sites in difficult terrain
- high purchase costs
- limitations in the type of ground they can handle except at considerable additional cost.

Calweld bucket drill

The Calweld bucket drill is a rotary drill in which the drive is transmitted through a ring gear slightly more than a metre in diameter, to rotate a drive kelly attached to the bucket. The kelly is telescopic and extends to about 20 m before additional stems are required. A hydraulic dumping arm allows the bucket to be discharged into a container or truck on either side of the rig (Fig. 6.16). The combined weight of kelly and bucket provides the necessary downward thrust for digging.

6.16 Calweld bucket drill dumping into sample container.

The buckets are of two types, earth-buckets and rock buckets. Both types have ripper teeth attached to their bases and reaming blades at their sides to give clearance. The base is hinged for rapid dumping. Bucket diameter ranges from 450 mm up to 2,500 mm. Drilling rates for 450 mm to 900 mm buckets average around 30 m/shift although rates have been obtained up to 60 to 65 m/shift in good standing ground (Macdonald, 1983a). In wet ground, necessitating casing, drilling rates reduce to about 15 m/shift. The most suitable conditions for Calweld bucket drilling are provided by:

- stable surface conditions, which allow good access to the drill sites for heavy equipment and trucks
- good standing ground, which can be drilled without caving
- small gravels with a sand/clay matrix that is easily penetrated without significant fall-in; occasional small boulders up to 250 mm diameter can be dug using 450 mm rock buckets but clusters of large cobbles may cause difficulties
- soft or weathered bedrock.

Operating costs are normally less than one half of Keystone churn drilling costs and in suitable ground conditions, the Calweld drill is also much more reliable and representative of the material being sampled.

Rotary air systems

A typical rotary air drilling system comprises three separate units: a self-propelled exploration drill, a trailer mounted air compressor, and a casing driver. The casing driver may be incorporated as an integral part of the drill rig or be mounted separately. The drill unit may be crawler or truck mounted. Crawler mounting is generally preferred because of better traction when hauling the compressor unit.

A Schramm rotary air drill used at San Antonio de Poto (Fig. 6.17) successfully sampled formations of glacial till and outwash material at drilling rates of up to 40 m/shift). Sample material was collected feeding the slurry through a cyclone separator (Fig. 6.18). Important features of the Schramm drill are:

- hydraulically driven rotary drill head with hollow spindle and a speed range from zero to 100 rpm; the drill head is floating in order to prevent galling of drill rod threads when adding or breaking out tools
- a feed carriage with a sufficient length of stroke to handle the required drills; the mast assembly is raised and lowered hydraulically; the operator control panel provides hydraulic control of bit pressure and feed rate

6.17 Schramm drill rig – San Antonio de Poto glacial outwash, Peru.

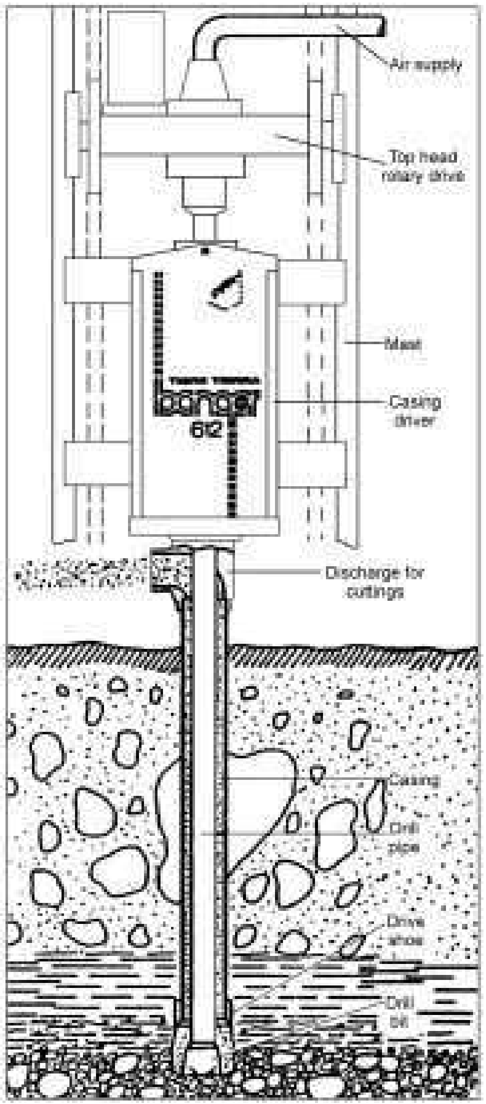
6.18 Sample collecting using cyclone separator – San Antonio de Poto glacial outwash, Peru.

- a heavy-duty crawler assembly with each track independently driven is controlled hydraulically for maximum manoeuvrability
- a suitable power unit drives the displacement pumps, which transmit power to the tracks and various rig functions; engine levelling facilities are available for operating on slopes; an automatic shutdown control operates if the oil pressure drops to an unsafe level; a rear-mounted winch unit, hydraulically operated is an accessory piece of equipment to assist crawlers on steep slopes or through muddy ground.

Casing drivers

The Schramm and similar drill types advance casing in pace with the drilling through the use of a casing driver. The driver is provided with a central vertical passage to accommodate the drill pipe. Each operation, drilling and driving is controlled separately, thus allowing the operator to vary and co-ordinate the drilling and driving functions to suit differences in the various layers penetrated. Formation cuttings are returned through the annulus between the drill pipe and casing. The ascending air velocity is regulated to suit the formations being sampled. The air pressure is also used to tap the casing through boulders or, if required, to exert the full power of the driver when drilling through unconsolidated formations. Either down-the-hole hammer or tricone bits are used to drill the overburden while driving the casing. Casing drivers, when not integral with the drill rig, may be provided with inexpensive mounting devices to suit a particular rig. They are usually designed for all weather conditions and easy field maintenance. The 'Tigre Negro' model 612 casing driver illustrated diagrammatically in Fig. 6.19 can be completely disassembled and reassembled in under two hours.

Drive shoes are made with a reinforced cutting edge and machined from heat-treated, alloy steel. They are designed to withstand high impact loads so



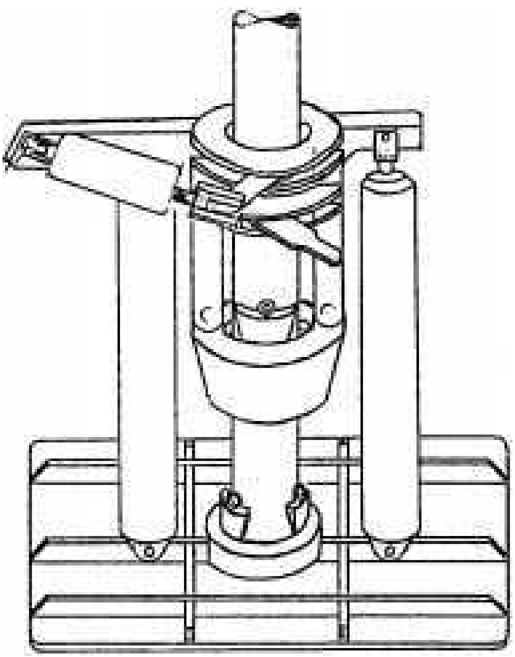
6.19 Tierra Negro casing puller.

that they can be used repeatedly for downward driving in tough ground. For upward driving, the sudden sharp bursts of energy applied to driving demand the use of a heavy-duty, thick-walled casing with strong threads. The disadvantage is that the sharp upward blows damage any threads that have become loose; even with the strongest of pipes, any looseness in the threads can cause them to break easily.

Hydraulic casing pullers

Casing pullers dramatically increase the available drilling time. While casing extraction is in progress on a completed hole, the drill can already be at work drilling the next hole. Using one casing puller in two separate operations (drilling and pulling), it was possible at San Antonio de Poto to average one hole/8hr shift to a depth of 30 to 40 m. In another application in Papua New Guinea a simple casing puller was developed for a cable tool operation as described in Fig. 6.20. The requirement was for a pulling pressure of 44 tonnes and a breaker pressure of five tonnes. The system was in two parts:

- Hydraulic power unit incorporating pump, reservoir, relief valves, controls and filter; the pump could be driven from a tractor power take-off or separate air-cooled engine.



6.20 Avenall casing puller (after Avenall, 1987).

- Breaker puller unit comprising two hydraulic ram mechanisms mounted on the base plate; the spider and breaker are attached to the mainframe, which in turn connects to the jacks; this unit is attached to the power unit via hydraulic hose lines with quick-release couplings.

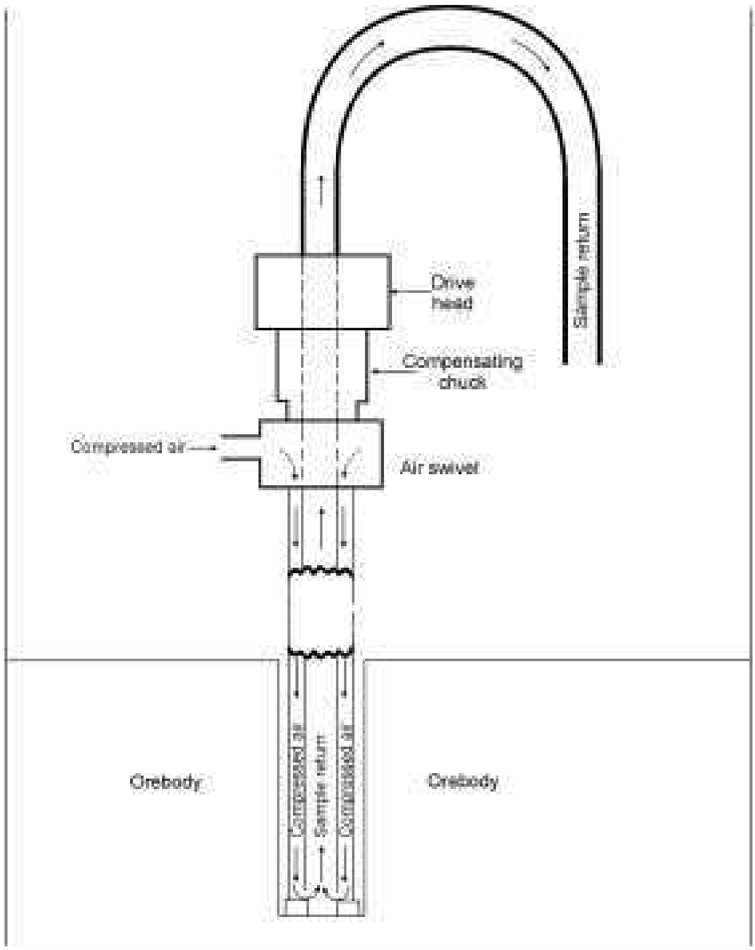
Sample recovery systems

These systems rely upon high-pressure air or water to recover the sample cuttings. Air or water is introduced into the annulus between the two pipes to service a double wall drill string, top drive rotating head and side inlet swivel. If water is the medium, the return flow passes to a settling tank to recover the entrained solids. If compressed air is the medium, the cuttings pass to a cyclone-separating unit for recovery. Water circulation systems are generally less satisfactory than compressed air systems. The Wallis drilling air core system, as illustrated in Fig. 6.21 is designed around the principle of reverse circulation; the sample being returned up a concentric inner tube.

Experience with the Schramm drill at San Antonio de Poto, showed that no undue sampling problems occurred in moderately moist gravel horizons with up to 40% clay content. For an air discharge velocity of 20 m/s a 1.0 m sample interval at 30 m depth could be cleanly blown in from five to eight seconds, the cyclone interior being reasonably clean after recovering the sample. However, pressure hose fittings and in the provision of a latch to hold the respective pieces together while drilling layers containing wet clay and coarse gravels were difficult to sample. Plugging occurred in elbows of the sample discharge line above the drill head exit and in the tricone bit at the bottom of the hole. On encountering such a layer, the abrupt rise in pressure almost invariably resulted in an explosive uplifting and separation of the hammer anvil from the easing adaptor. This allowed the air to burst out violently along with sample material and water. Remedies were eventually found in modifications to the air.

Small-bore reverse-circulation drills

This method, an adaptation of hollow auger drilling may be useful for delineating auriferous horizons in reconnaissance sampling, but the samples are too small to fairly represent gold values in alluvial gravel beds. Reverse circulation drilling is also notoriously erratic in difficult drilling conditions whereas sample recoveries from bucket drills are usually close to 100%. Kitching and Lightweight (1989) claim that five 2.5 inch bore holes spaced over an area of three square metres, should give a broader, and more accurate estimate of grade than a bucket drill of the Calweld type in the same location. This assertion was strongly contested by Gordon (1980) who pointed out that the combined area of five 2.5 inch diameter holes is only about 1/25th of the area of an 800 mm-diameter Calweld bucket.



6.21 Wallis air core drilling system.

Note that hollow augering (Macdonald, 1983a) is more selective than solid augering and cores of the material penetrated can be taken in a relatively undisturbed state. Hollow stems and wireline sampling improve drilling rates, which may be as high as 15–20 m/h in sand and 6–10 m/h in clay. In drilling a sector of lateritic material, it was found possible to sample reliably at 1 m intervals to maximum depths of five metres.

6.3 Prospecting methods offshore

Specific geological predictions of the present location of residual gold deposits on continental shelf areas are strongly dependent upon the results of seismic reflection and magnetometer surveys and correlation with the adjacent onshore geology. As already described in Fig. 6.2, drilling platforms may be constructed as wooden rafts for Banka drilling in calm shallow water, lakes, and sheltered bays. Drilling vessels, which are generally self-propelling in deeper offshore waters, must legally conform to marine standards of safety. The effectiveness of a sampling programme will then depend upon meteorological and oceanographic factors. The most important of these factors relate to differences in the strength, direction and frequency of windstorms; the changing characteristics of waves, tides and ocean currents; variable water and sediment depths; distance from shore and seabed conditions.

Wind-generated waves profoundly affect the motion of a drilling ship or barge and it is essential to have a good understanding of the possible day-to-day variations in wind strengths and directions as well as the seasonal changes. Winds rise quickly during thunderstorms and provision must be made for safe handling of the drilling vessel during stormy weather. Thunderstorms commonly occur in the late afternoon and initially, the direction of the wind is from the sea to the land; but this changes as the storm progresses. Wind forces commonly range to higher than five on the Beaufort scale (see Table 6.1) at such times, and this results in waves of considerable height but short amplitude.

The approximate frequency and timing of periods in which wave heights may reach 1.5 m, 2.5 m, 3.5 m and higher must be obtained in each offshore area selected for sampling. Drilling barges in shallow well-protected marine waters generally have a maximum wave height tolerance of about 1.5 m. For most drilling vessels operating in the open sea, the average practical limit for efficient and effective drilling is not much above a wave height of 1.0 m. High winds cause strong currents to develop and these combined with high waves generated by the winds make drilling impossible and endanger the safety of the ship. The possible number of days per year that can be planned for drilling is normally based upon the statistics for this wave height.

Drilling can be conducted either from the seabed itself or from a floating platform. Factors limiting the choice are the thickness and lithology of sediment layers, sea depth, local oceanographic conditions and local climatic conditions. Drilling from the seafloor itself is carried out using remotely controlled drills such as vibro-corers. The principal drilling types for drilling from platforms are Banka drills (used in shallow calm waters), reverse-circulation drills, and hammer drills. Banka drills operated from simple pontoons are used extensively in the shallow near-shore areas of Indonesian waters. Sample recovery is affected by water jetting in dual tube units. Special bit design has helped to overcome such problems as underflushing and blockages between the inner and outer casings.

Good planning is needed to optimise drilling time by directing drilling operations out into the open sea during the calmer periods and into sheltered areas during more adverse weather conditions. Access for drilling is a day-to-day problem and a hole should not be started without a reasonable certainty of uninterrupted completion during the same work period. Changing weather

Table 6.1 The Beaufort wind scale

Beaufort scale number	Descriptive term	Units (km/h)	Units (knots/h)	Description on land	Description at sea
0	Calm	0	0	Smoke rises vertically	Sea like a mirror
1–3	Light winds	19 or less	10 or less	Wind felt on face; leaves rustle; ordinary vanes moved by wind	Small wavelets; ripples formed but do not break; a glassy appearance maintained
4	Moderate winds	20–29	11–16	Raises dust and loose paper; small branches are moved	Small waves – becoming longer; fairly frequent white horses
5	Fresh winds	30–39	17–21	Small trees in leaf begin to sway; crested wavelets form on inland waters	Moderate waves, taking a more pronounced long form; many white horses are formed – a chance of some spray
6	Strong winds	40–50	22–27	Large branches in motion; whistling heard in telephone wires; umbrellas used with difficulty	Large waves begin to form; the white foam crests are more extensive with probably some spray
7	Near gale	51–62	28–33	Whole trees in motion; inconvenience felt when walking against wind	Sea heaps up and white foam from breaking waves begins to be blown in streaks along direction of wind
8	Gale	63–75	34–40	Twigs break off trees; progress generally impeded	Moderately high waves of greater length; edges of crests begin to break into spindrift; foam is blown in well-marked streaks along the direction of the wind

Table 6.1 Continued

Beaufort scale number	Descriptive term	Units (km/h)	Units (knots/h)	Description on land	Description at sea
9	Strong gale	76–87	41–47	Slight structural damage occurs – roofing dislodged; larger branches break off	High waves; dense streaks of foam; crests of waves begin to topple, tumble and roll over; spray may affect visibility
10	Storm	88–102	48–55	Seldom experienced inland; trees uprooted; considerable structural damage	Very high waves with long overhanging crests; the resulting foam in great patches is blown in dense white streaks; the surface of the sea takes on a white appearance; the tumbling of the sea becomes heavy with visibility affected
11	Violent storm	103–117	56–63	Very rarely experienced – widespread damage	Exceptionally high waves; small and medium-sized ships occasionally lost from view behind waves; the sea is completely covered with long white patches of foam; the edges of wave crests are blown into froth
12+	Hurricane	118 or more	64 or more	Very rarely experienced – widespread damage	The air is filled with foam and spray. Sea completely white with driving spray; visibility very seriously affected

conditions must be monitored closely and all precautions taken to avoid equipment damage or having to set up and re-drill at the same site. Despite the magnitude of the problems, there is a lot of experience to draw upon particularly from drilling in southeast Asian waters. Given suitable equipment, the operator should soon develop the necessary skills and 'feel' for offshore sampling as for land-based operations.

6.3.1 Vibro-coring

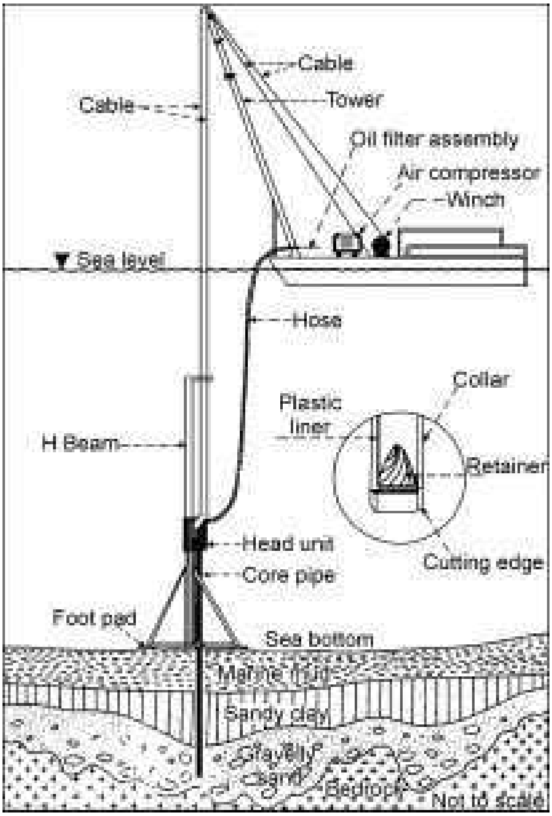
Vibro-coring, is a method of sampling shallow, unconsolidated and reasonably closely sized sediment such as drowned beach sand deposits. The standard rig employs a 4.5 inch O/D core barrel with a rigid plastic core liner 3.125 inch I/D. The barrel is fitted with a cutting shoe and core catcher. Vibro-corers (including recent impact developments) recover relatively undisturbed core samples within their ability to penetrate the seabed. With suitable handling, penetration monitoring and controlled *in-situ* retraction, recoveries should be close to 100% and may therefore be considered as being closely representative of the sequences cored (Pheasant, 1989). The cores may be longitudinally sectioned for calibrating shallow seismic records thereby providing data for understanding upper seafloor lithologies and their Quaternary geology. The drill (Fig. 6.22) is limited in its penetration capability but should be able to produce ~5 m length cores at the rate one core every two to three hours including transit times and anchoring. Properly identified and archived, the cores provide a geological database for future studies.

Vibro-impact

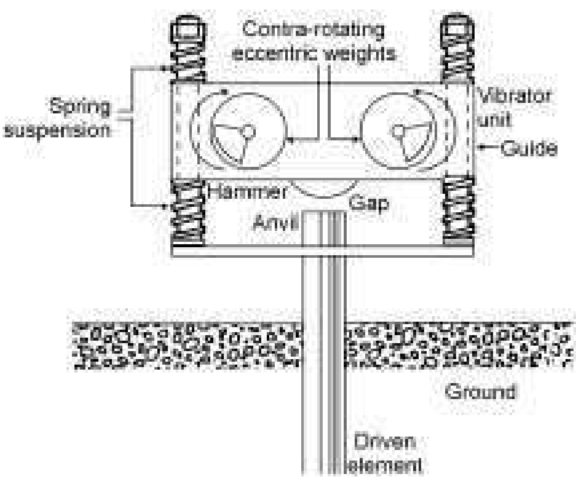
Vibro-impact drilling (VIC) originated in the USSR in 1948 as an improved type of vibrating drill for more difficult drilling conditions. This drill (Fig. 6.23) has since been the subject of a major research programme at Aberdeen University. It uses both periodic vertical vibration and periodic impact to force the casing down into the ground. The incorporation of periodic impact helps to overcome higher resistance soils such as stiff cohesive and dense granular materials that cannot be penetrated by vibro-coring alone. Cores up to 2 m long have been taken from stiff clay of undrained shear strength 150–300 N/m^3 in difficult underwater current conditions. In one investigation the VIC drill achieved a coring rate of 65 m/12 h working day with as many as ten locations being tested in any one day. Drilling from the seabed by vibra-coring is restricted only by the seaworthiness of the drilling vessel in high wave and storm conditions.

Hammer drills

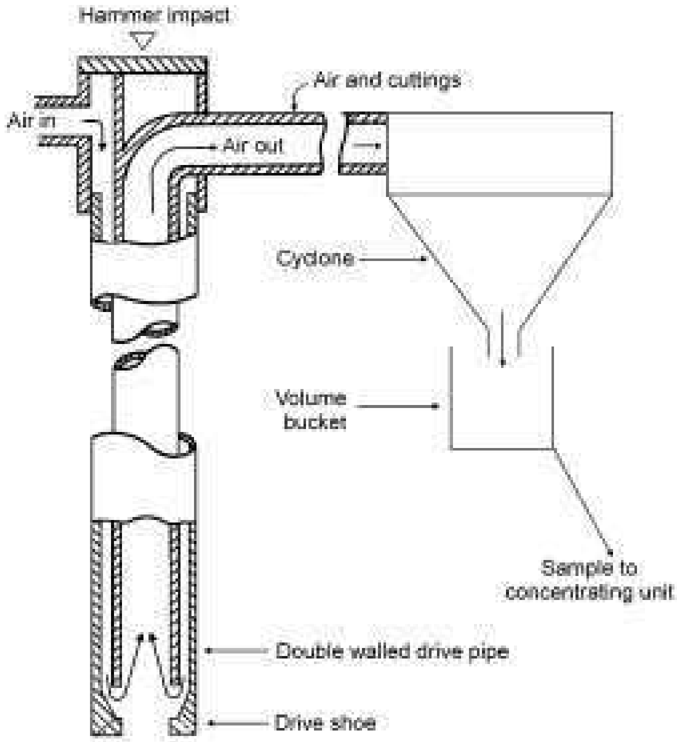
The Becker hammer drill (Fig. 6.24) is the best-known drill of this type. In the Becker system, twin tubes are driven without rotation using a pile-driving



6.22 Vibro-core drilling (after Pheasant, 1989).



6.23 Vibro-impact drilling.



6.24 Becker hammer drill.

hammer operating at 90 to 95 blows/minute. Water or air, under pressure, is forced down the annulus between the two pipes to drive the cuttings back to the surface through the inner tube. The method has generally found more favour for offshore drilling than for onshore testing although Richardson (1988) recommends the Becker drill as the best type for rapid evaluation of a new onshore deposit. Casing diameters of 100 mm to 150 mm are normal; the method, which is three to four times as fast as churn drilling (up to 60 m/day in heavy gravels), can handle much tighter ground. However, both capital and operating costs are high and engineers have mixed opinions on the reliability of the samples. Breeding (1973) doubts the validity of samples taken from coarse placer gravels but refers to favourable reports from a Becker drill used for testing gold-bearing alluvial off the Alaskan coast near Nome, Alaska.

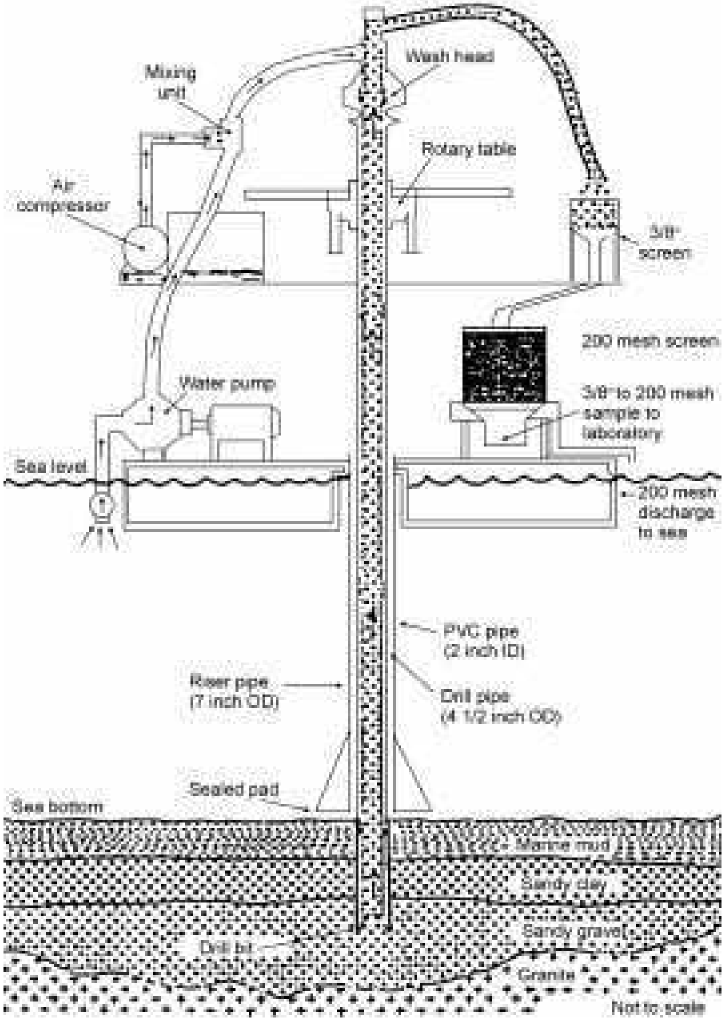
6.3.2 Reverse circulation

Large reverse circulation (RC) drilling systems are finding increasing application in offshore areas. Pheasant (1989) describes techniques that have been developed to flood the drill string prior to cutting off air pressure. This enables rods to be changed under conditions that reduce formation in-rush and blockages that might otherwise occur due to hydrostatic pressure and pore-pressure. With some modifications to their onshore counterparts, offshore RC drilling systems are likely to encounter similar problems related to penetration, sample recovery, etc.

Problems unique to the offshore environment are geographical problems imposed by both meteorological and oceanographic vagaries. Serious constraints are attached to operation from floating platforms. The general tolerance for wave height is a maximum of 1.5 m. Water depth becomes a critical factor at around 20 m below sea level. Increasingly with depth, the drill pipe whips around unless constrained by a riser pipe or in some other way. While not essential, a depth recorder is useful. If properly positioned, the transducer will show a reading of the sea bottom and the casing pad on the chart. This allows the operator to correctly judge the distance from the bottom of the stabilising casing to the sea bottom.

The barge *Supphayakornthoranee* (Fig. 6.25), which has a Conrad-Stork reverse circulating system with counterflush sampling (Fig. 6.26) was provided by The United Nations Development Programme to the 'Offshore Exploration for Tin and Heavy Minerals Programme' in the Andaman Sea, offshore Thailand. The basic system comprised a 178 mm stabilising casing (riser pipe) suspended

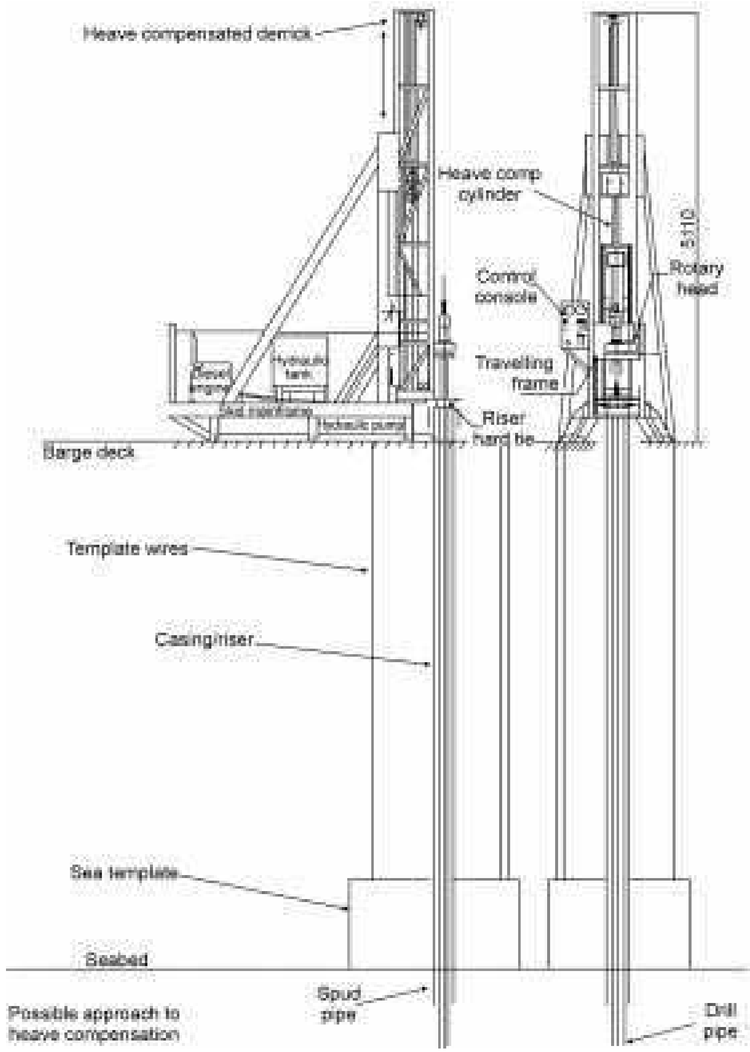
6.25 The drilling vessel *Supphayakornthoranee* – Andaman Sea Offshore Project, Thailand.



6.26 Conrad-Stork reverse circulation system operating with riser pipe, as in the Andaman Sea Offshore Project, Thailand (after Pheasant, 1989).

below the moored drill barge from the lower deck, but held in suspension above the seabed to retain the riser in tension and prevent compressive end loads being induced upon the riser. The riser pipe is mounted on the drilling vessel at its top end and is not allowed to impact on the sea bottom.

The 100 mm steel drill pipe with a 45 mm PVC inner pipe is deployed from the upper deck with a clamped rotary drive head and flexible hoses to the wash head or swivel. Horizontal tangentially mounted chains provide torque reaction to the rotary table, theoretically isolating barge heave from the deployed drill pipe. With the vessel positioned over the drill site the water depth and state of



6.27 Proposal for seabed hard riser (after Pheasant, 1989).

the tide is determined and the riser is lowered over the drill pipe so that distance between the seabed pad and the sea bottom was never less than 2.5 m.

In noting that the 175 mm pipe is strictly not a 'riser', Pheasant (1989), in a review of the project drilling activities, proposes the installation of a seabed hard tie riser (Fig. 6.27) in place of a suspended riser. In this installation process, the hard tie riser comprising a clump weight or seabed reaction template would be lowered to the floor by a twofall wireline. The weight could be fitted with a spud pipe to help avoid sample contamination by superficial sediment while spudding in the reverse circulation or wireline barrel. As the clump weight is lowered the casing or riser would be torqued up to follow the descent. The drill mast would then be hard tied to the riser pipe while a boosted hydraulic system feeds a soft tie hydraulic ram between barge and mast forcing the riser pipe into tension. The seabed hard tie would then provide the driller with similar drilling conditions to those of a land-based rig.

Positioning

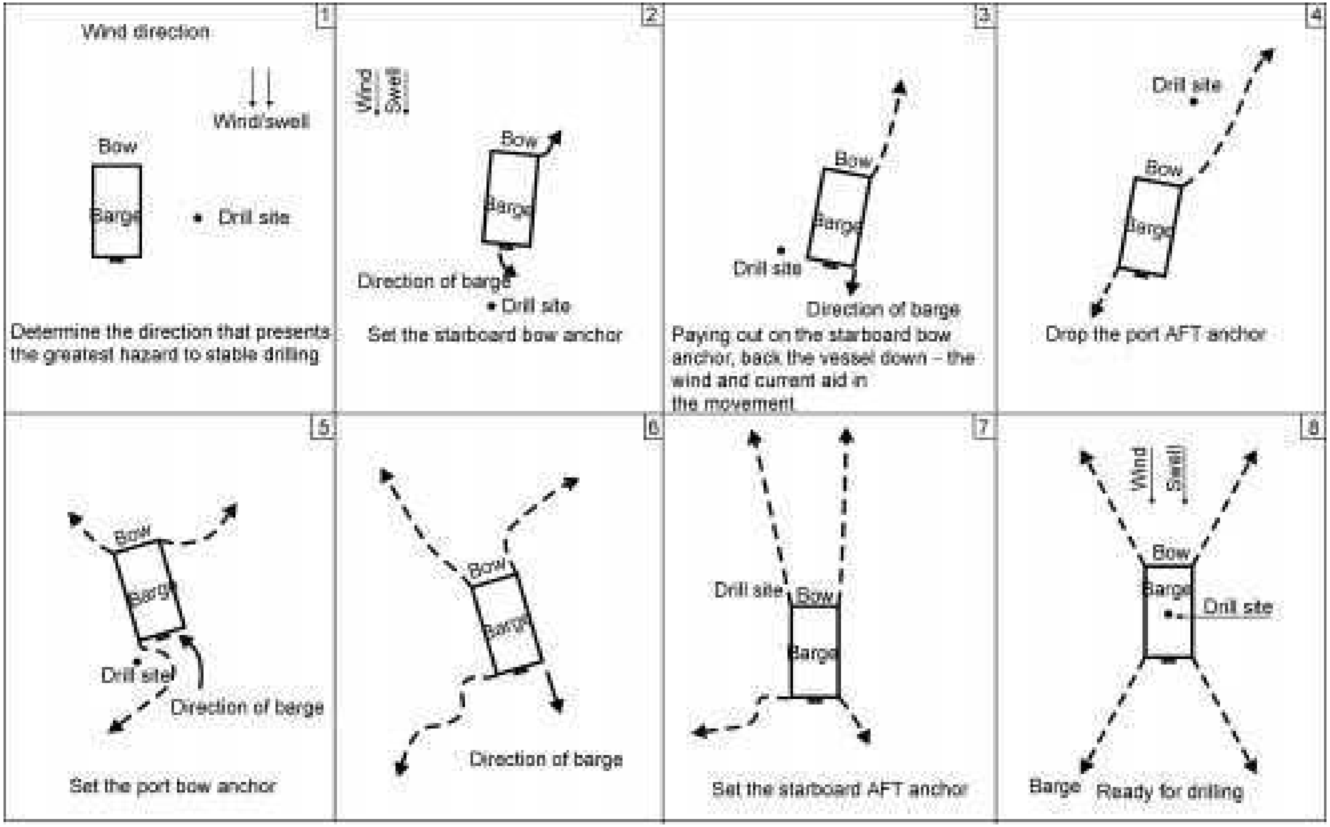
The first US Navy Navigation Satellite System (NNSS) comprised several ground (tracking) stations to continuously monitor the position of the satellites in their orbits. The data obtained were forwarded to a computer centre and combined with data from other sources (e.g., the Earth's gravity field, air resistance, pressure due to solar radiation) from which the various orbit-parameters for the next few hours were computed. An 'injection station' transmitted these orbit-parameters to the satellites about twice a day, where they were stored and re-transmitted at two-minute intervals. This system had an accuracy of about 10 m for repeated measurements for a stationary receiver.

The level of accuracy is improving with modern technological advances. The NAVSTAR satellite Global Positioning System (GPS) consists of a constellation of 18 satellites in 12 orbits (14,650 km altitude) inclined at 60 degrees from the equator. Ware (1987) foreshadowed the present revolutionary ease and accuracy of GPS for positioning measurements (e.g., centimetre-level relative positioning for rapidly moving receivers) and the pocket size of easily affordable positioning equipment that currently attains unprecedented accuracy. Levels of GPS units available to the public include:

- relatively low-cost navigational units that provide an accuracy of a few hundred metres
- hand-held units linked to a fixed broadcast base station; utility companies and some geographical information systems use these units for mapping, they have a positional tolerance of several metres
- real-time kinematic roving, high-precision units are linked by radio to a fixed base station, allowing quick on-site gathering of data; these high-cost units are suitable for topographic mapping with a positional tolerance of centimetres
- geodetic units are used for highly precise measuring of long baselines in difficult terrain such as across rivers and mountains; long observation times and off-site post-processing of data is needed to obtain a sub-centimetre positional tolerance.

Manoeuvring

In order to stabilise the drilling platform in one location a four-point system comprising four positioning winches, each equipped with 700 m of wire rope



6.28 The four-point mooring system.

and a propulsion unit powered by a marine diesel engine, can be used to manoeuvre the platform into position. Once the drill site has been marked (using a float) the direction that presents the greatest hazard to stable drilling is determined and both present conditions and those expected, e.g. tidal changes, local wind pattern, variations, etc. Figure 6.28 describes the sequence of operations:

- The vessel is orientated in the direction of the greatest hazard to stable drilling.
- The vessel moves into position to drop the starboard bow anchor; the distance from the drill site to the anchor set is seven times the water depth.
- The vessel backs down, cable is paid out on the forward anchor cable.
- The vessel is in position to drop the port aft anchor; when this anchor is set the forward anchor is taken in to remove the slack, the aft anchor brake is set to allow even layout, care being taken not to drag it from its position.
- Once the drill site marker is reached, the vessel is moved under its own power into position to drop the port forward anchor.
- The vessel backs down, keeping slack to a minimum throughout the procedure.
- The vessel is in position to drop the starboard aft anchor.
- The vessel is pulled over the drill site and is ready for drilling.

The advantages of this system are as follows:

- Acute angles of the anchor cables are avoided.
- The vessel can maintain a heading close to the current and wind.
- At no point are the cables across the stern.

Care is taken in retrieving the anchors to avoid overriding anchors or tangling the cables. Paying out on the upstream cables retrieves the downstream anchors. The strain on the cables is very great and they should be inspected during retrieval; a flat lay is maintained on the drum during rewinding.

6.4 Sample dressing

It is seldom necessary or even practicable to reproduce the entire range of treatment plant processes on a pilot scale in order to predict the behaviour of residual gold ores under normal plant conditions. Most plant items can be designed with sufficient accuracy from the testing of specific features of the material represented by sample dressing and from known principles on the basis of laboratory data. A field-based laboratory is much more useful than one that is located away from the scene of operations and on-site bench-scale testing of individual borehole samples is usually the most practical way of identifying all significant characteristics of the placer materials likely to influence eventual prototype performance. However, some pilot-scale testwork may be necessary to

6.29 Denver gold sampler, Rio Aurodo gold – platinum placer, Colombia, South America.

resolve some issues involving future prototype operations. Existing commercial operations offer few opportunities for scientific observation and bench-scale experimentation on bulk samples at the drill site is often the best guide to checking the results from borehole sampling on a larger scale. Some manufacturers market mechanical devices for treating bulk samples but although they provide some information when sampling gravelly wash material containing coarse (i.e. >200 micron) gold they lack the refinements necessary for recovering finer and flakier gold particles. Figure 6.29 shows a Denver sampler being used for sampling stream sediment by a team from the United Nations Development Programme at Rio Aurado, Colombia. Figure 6.30 is a larger, mobile sampling plant used in the goldfields of Western Australia for clayey ores.

6.4.1 Bench-scale testing

Under normal plant flow conditions, the sorting characteristics of typically equant heavy minerals (e.g., rutile, zircon and ilmenite) is generally predictable, but the settling rates of detrital gold grains are much more difficult to assess and each gold ore should always be treated as a special case. Although denser than most other placer minerals, gold grains have much wider size, shape, textural and density differences, all of which have different effects on their hydraulic behaviour. The apparatus may be relatively crude, but careful observation of the test results will provide all essential measurements of the gravel, sand and clay

6.30 Mobile sampling plant, Western Australia.

contents of the ore and of the physical characteristics and percentages of gold grains in the heavy mineral concentrates. Important features that can be determined from bench-scale testing are:

- the slurrying properties of the gold-bearing materials and percentage content of slime sized particles
- the proportion and types of other heavy minerals present in the feed
- the size range distribution of both sediments and valuable heavy minerals including gold
- the presence of any organic or other coatings on gold particles that might affect their hygroscopicity or ability to amalgamate with mercury
- approximate estimates of future power and water requirements and residence times for slurrying can be made largely from the ease or difficulty of dispersing the borehole samples.

Important exploration information relates to sample grades, sediment characteristics and other physical parameters. A drilling engineer who receives quantitative data each day from the previous day's operations is better able to deal with any emerging problems than if the results are delayed by weeks or, as can happen, even months for the results of samples that are sent away for processing. Desliming is of paramount importance and the laboratory should be equipped to carry out all essential processes involved with slime settling, and with the identification and physical measurement of fine gold particles. Various simple laboratory techniques include amalgamation with mercury, heavy liquid separation, micropanning, sizing, settling, magnetic separation (hand), microscopy and colour coding as listed in Appendix I. Selection of the plant and equipment items may be made confidently only when the mineral-processing engineer, as a result of the above information, is able to write down all the quantitative data needed for the design of each component.

Dressing shed procedures

Figure 6.31 illustrates the type of long tom device used by the author for dressing pit and churn drill samples. The procedure is generally as follows:

- Allow the drill sample to settle in a calibrated measuring vessel.
- Lower the surface tension of the water through the use of a detergent if any of the gold has a tendency to float.
- Level the top of the settled material and record the depth and volume of the settled material in litres.
- Recover a sample of the slime for examination in the laboratory and decant the remaining water and slime directly to waste.
- Screen the sample at 10 mm and 3 mm sizings in the head box.
- Check the oversize fractions for coarse gold and measure the individual volumes by water displacement.
- Estimate the slime content by difference.

6.31 Long tom sampling arrangement – Ampulit, Kalimantan, Indonesia.

The undersize is washed over the sluice box in the form of thin evenly flowing slurry; rich samples should be passed over the sluice more than once before being discarded. The sluice box is carpeted with coarse jute sacking and expanded metal riffles to catch the gold in concentrates, which are recovered by agitating the sacking vigorously in a dish of water to dislodge all of the solids. Any residues in the sluice box are washed down into the same dish. The water is decanted and the solids are panned down into the form of a rough heavy mineral concentrate (usually about 50% heavies, 50% lights), which is labelled and sent to the laboratory for final processing.

Note that one set of jute sacking is usually sufficient for processing all of the samples from one hole. It is then dried, burned, and panned. Any gold recovered from this process is weighed and the weight is distributed pro-rata between the contributing ore zone samples.

A successful drilling and sampling programme can only be achieved when good communication and compatibility exists with all personnel. Experience, ability and persistence of the drill crew to achieve the desired results are of the utmost importance (Barden, 1990). Because of the scale of possible risks associated with the human equation, the requirements of sample reliability and representivity must take precedence over the total depth that is drilled each shift. The results from correctly designed and conducted sampling operations will always be more accurate than data derived by calculation from general suppositions. Yields and efficiencies are better defined for subsequent financial studies and eventually, there should be fewer start up troubles in the plant.

6.4.2 Reliability

Gold placers are difficult to sample reliably because of the diverse conditions under which they are formed and the heterogeneity of the mixtures of which they are composed. Sample reliability is affected by the geometry of the surfaces over which the original flow took place; differences in the movement and settling rates of particles having different properties of size, shape and density, and both local and regional variations in the original flow rates and stream power. Sub-surface deposits are the most difficult to sample reliably and cheaply because of the lack of surface expression and the diverse nature of the overlying masking material.

Reliability is also a function of the human equation and faulty supervision or inattentiveness at the drill site or laboratory will almost certainly lead to serious error. Ideally, the validity of sampling data is measured by the ability to take and process duplicate samples, closely corresponding in all respects to the original samples. In other words, reliability implies repeatability. In practice, repeatability cannot be obtained for individual holes, regardless of how carefully the check drilling and sampling is done. Because of the sporadic nature of gold deposition, the gold tenor varies from sample to sample regardless of spacing.

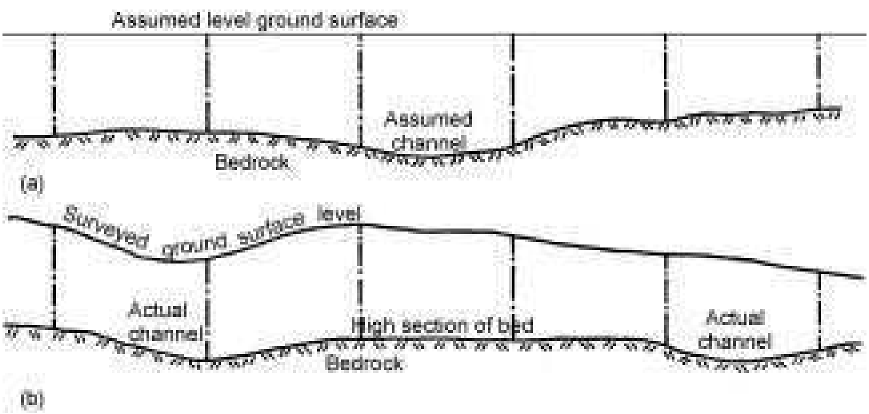
Duplicates, if taken from corresponding depths in adjacent holes, may be closely similar in all physical aspects though the grades are significantly different. Repeatability of tenor, as an average of sample grades, can be expected only from a large number of holes, not from one set of duplicates alone.

Required standards of accuracy

Precise measurement is essential for surveying and subsequent ore reserve estimates. Compass and chain surveys are only suitable for reconnaissance where small evaluation errors are unimportant. Measurements taken for mine planning and final evaluation require that all drill lines, borehole collar elevations and topographic mapping have the precision of a theodolite survey. Figure 6.32 shows how errors in interpretation may result from faulty measurements of borehole collar elevations.

Standards of accuracy at the drill site call for measurements to be within plus or minus a few centimetres for drill string and casing depth. This degree of accuracy allows a measured plug of material of predetermined length to be maintained at the casing mouth thus helping to guard against sample loss or contamination from inflow.

Much higher standards are required for laboratory analytical procedures. Considerable attention has been paid in recent years to recovering alluvial gold in the finer particle sizings and the weight of gold recovered from each size fraction of composite drill sample is normally measured to within plus or minus one-tenth of a milligram. Direct-weighing instruments are available at reasonable cost with detection limits much finer than needed.



6.32 Possible errors in interpretation from faulty measurements or assumptions of bore hole elevations. (a) is plotted for the assumption of a level ground surface. (b) is the true topographical section. The assumed channel shown in (a) is actually located in a high section of the bedrock. The actual channels in (b) are not shown in (a).

A fair estimate of overall property value can be obtained only where any uncertainties, such as anomalous measurements have been resolved by logical explanation or resampling. Instituting a system of random checks and control samples should guard against any possibility of salting, i.e., the addition or removal of gold from a sample or the substitution of false measurements for true ones. The nature of the check sampling and the results obtained should be displayed, and important conclusions reached discussed in the final evaluation document.

Unintentional bias

The risk of unintentional bias is present at all stages of drilling and sampling, either because of the inherent difficulties of sediment sampling or through human error. This author has stressed repeatedly, (Macdonald, 1966, 1983a, 1983b) that human error is best minimised by ensuring that all personnel are kept fully informed on all matters relating to their duties and the duties of those around them. Personal involvement and motivation are essential factors in achieving good sample reliability; operators must understand why something is done in a certain manner as well as knowing how and why it should be done. Disinterest and boredom lead inevitably to slipshod and unreliable sampling.

6.4.3 Representivity

Sample results form a proper basis for evaluation only when, in addition to being reliable, they are closely representative of the body of material sampled. Sample representivity is a statement of the confidence with which the purpose of sampling is satisfied by its samples, within specified limits of allowable error. Ore reserve estimates normally require a confidence level of 90–95% plus or minus 10%. In other words, a 90 to 95% probability that the true value of the material sampled is no more or no less than 10% larger or 10% smaller than the value indicated from sampling. Individual sample representivity is attainable at this level only from samples that are large enough to include a fair proportion of all of the mineral types including gold and numerous enough to fairly represent all sections of the placer. Desliming is of paramount importance and the laboratory should be equipped to carry out all essential processes involved with slime settling and the identification and physical measurement of fine gold particles.

Gy (1956) introduced a formula to calculate the representivity of a sample for geochemical analysis:

$$S_r^2 = f \times g \times l \times m \times d^2 \times M^{-1} \qquad 6.1$$

S_r^2 analysis is the standard deviation S; 95% of analyses should fall in the range $\pm 2S$.

$f =$ shape factor relating to form of mineral grains generally taken as 0.2 for gold.
$g =$ particle size distribution factor: 0.5 for well-sorted material, 0.25 for gold.
$l =$ liberation factor is related to the largest particle size and liberation size and varies from 0 to 1.
$m =$ mineralisation factor related to the density of both the gold and of the host mineral.
$d =$ aperture of screen passing 95% of the sample.
$M =$ sample weight analysed in grams.

This formula can be used to determine:

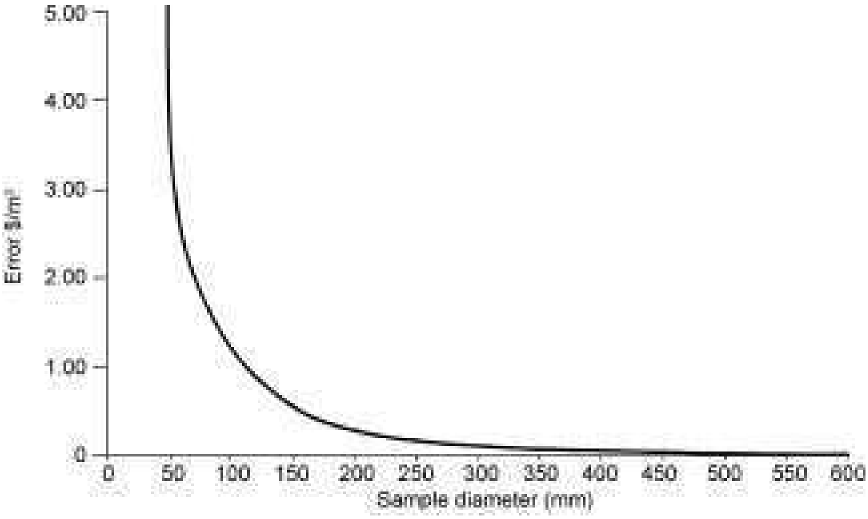
- the sample variability to be expected using a particular sample weight and type of sample
- the weight of sample required for sample representivity
- the necessary physical size for a given sample weight for required sample representivity.

The use of the formula assumes random sampling and implies no bias in the sampling process and takes no account of any analytical errors.

The problem is particularly critical in alluvial gold analyses because of low absolute values and irregular distribution. Gold is very high density by nature and is frequently erratically distributed. Most problems of achieving good sample representivity relate to the small size of individual borehole samples compared with the large volumes of ground that each one is held to influence. Each sample is given enormous authority. For a sample grid of 200 m × 25 m the ratio of sample volume to volume of influence is about 1:280,000 for a 150 mm borehole. Problems of representivity are exacerbated by the high unit value of gold and its low abundance in the ground. One gram of gold in a cubic metre (say 2 tonnes) of ground is present in the ratio of 1:2,000,000. The average tenor of a commercial gold placer may be 160 mg/m and for this grade of material, the gold is present in the sample in the ratio of 1:12,500,000.

Gold grain size

The classical model (Fig. 6.33) demonstrates the effects of the inclusion or exclusion of a particle of gold valued at one cent from one metre length of boreholes ranging in diameter in 1 m^3 from 50 mm to 600 mm. The error is negligible ($0.035/m) for a 600 mm diameter sample, but unacceptably high ($5.09/m for a 50 mm diameter sample. It can probably be tolerated ($0.57/m) for a 150 mm sample provided that the sample is unbiased and is one of a large sample population for which individual errors can be expected to cancel out. For larger particles, the errors would be of significant proportions and could lead to serious over or undervaluation.

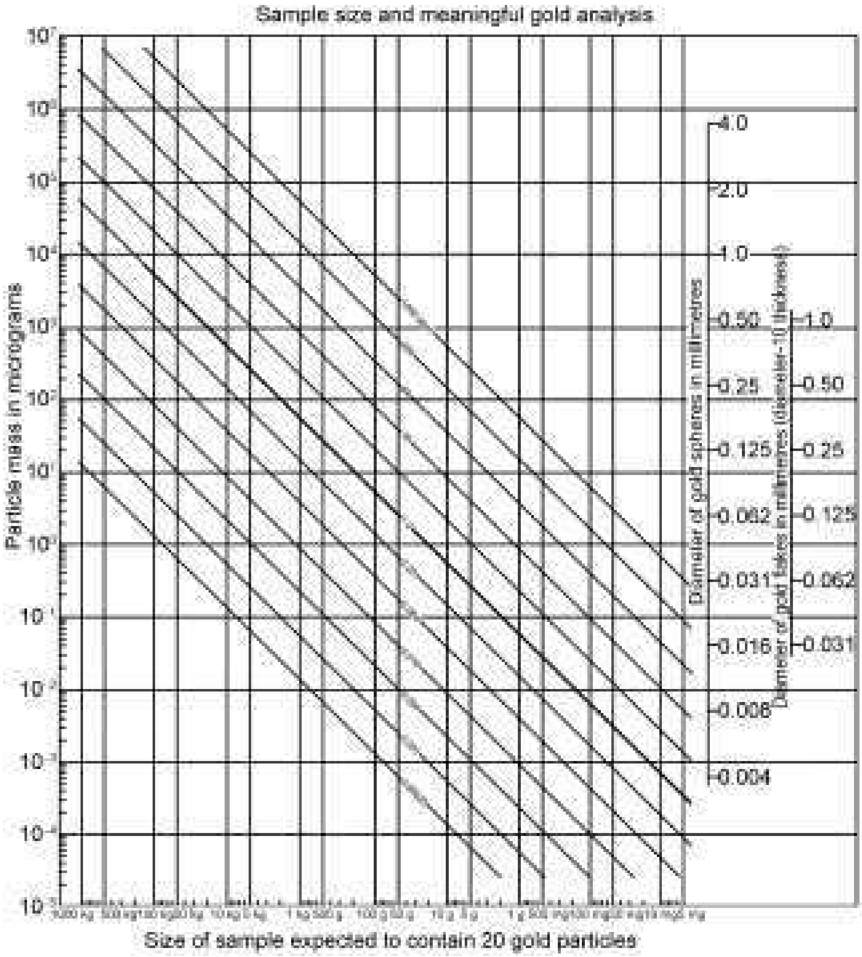


6.33 Errors resulting from displacement of one particle of gold, of one cent value from one metre lengths of boreholes of varying diameter.

Clifton *et al.* (1969) conducted theoretical studies aimed at producing a mathematical solution to the problem of adequate sample size. The precision is determined by the number of gold particles within a sample assuming:

- gold particles are of uniform mass
- gold particles comprise less than 0.1% of all particles
- the sample contains a total of over 1000 particles
- analytical errors are disregarded
- gold particles are randomly distributed within the sample.

The precision obtained from a sample containing 20 particles of gold was found to be adequately representative for most purposes, although a sample with fewer gold particles could provide sufficient information in reconnaissance programmes where less stringent representivity requirements are acceptable. Figure 6.34 presents a graph that relates number of gold particles to precision. The figure applies specifically to gold grain-size relationships in terms of spheres and flakes shown on the right size of the figure. Using this approach a sample weight of 300 g will provide the necessary precision in a sample with gold particles 0.125 mm diameter (20 milligrams weight) and 1 ppm Au. In a sample containing 0.625 mm particles at the same concentration the necessary sample weight would be 2 kg. However, the actual numbers involved may vary within wide limits depending upon the physical characteristics (size, size distribution and shape) of the gold. Precision can therefore be defined as reproducibility and accuracy as a measure of the degree to which data approaches the true value; it is essential that all data relating to the exercise are precise with predetermined limits so that anomalous



6.34 Size of sample required to contain an expected 20 particles of gold as a function of the combination of gold particle size and grade, assuming all gold particles are of uniform size and randomly distributed in the deposit (after Clifton *et al.*, 1969).

areas are reliably identified. The precision also varies with concentration and distribution of gold in the material sampled, deteriorating markedly towards the lower levels of concentration and irregularity of distribution.

Although the ideal conditions assumed for Clifton's study do not exist in nature, the graph usefully demonstrates the increasing difficulties of obtaining good sample representivity with increasing particle size and unit value. As seen by Fricker (1980), 'most theories derive a sample size inordinately large for deposits of low grade, high value minerals'. The problem is particularly critical in alluvial gold analyses because of low absolute values and irregular distribution. Work on the mathematics of sampling gold deposits, both hard rock and alluvial, shows that a sampling formula should contain the smallest number of parameters that need to be determined experimentally and that the determination of these be easy (Royle, 1991). Noting that placer samples are measured by volume rather than by weight, Royle produced formulae for gold spheres and for gold flakes. The sampling volume V for gold spheres is given by:

$$V = 10d^3/A \qquad 6.2$$

and, for gold flakes

$$V = 20\,St/A \qquad 6.3$$

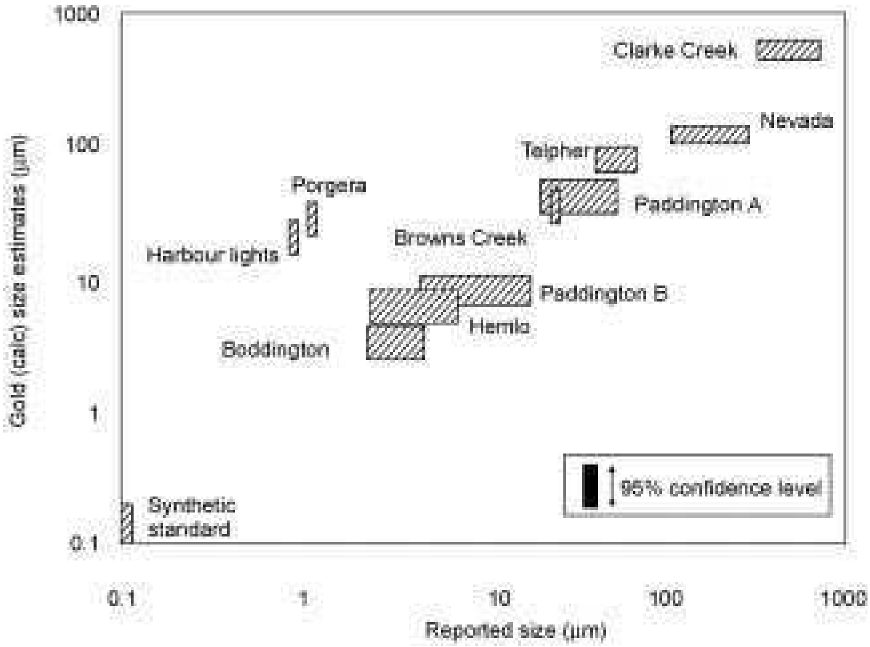
where V is the sampling volume, m^3; d the particle diameter, mm; S is the surface area, mm; and t is the thickness, m. A is the maximum permissible contribution from a single gold flake.

One difficulty with all such formulae is their implicit assumption of uniformity of shape. They may not apply so easily to material containing very irregular grain shape. Royle (1991) notes another difficulty – taking a sample of adequate volume does not by itself assure the avoidance of outlying assays – every stage of sampling and pulp preparation needs to be studied if outliers and skew are to be minimised.

Cohen *et al.* (1996) developed a program 'GOLDCALC' to predict the gold particle sizes, based upon sample replicate gold data. According to Cohen, 'by using a binomial statistical technique on data from replicate analyses of homogenised sub-samples, a direct link is made between analytical variation and the expected number of gold particles present in the sub-sample. From these estimates, particle dimensions are inferred'. The authors' tested this method on a wide range of geological materials and gold morphologies from sub-micron gold in laterites to coarse gold in placers with generally favourable agreement between size estimates obtained by microscopy and other physical sizing techniques (Fig. 6.35). Although the size estimates for 'Harbour Lights' and 'Porgera' sub-samples appeared to be up to two orders of magnitude greater than the reported values, the general result appeared to be valid. The form of the gold did not appear to be a significant factor.

Limitations to the method depended upon the nature of the distribution of the gold particles present and the application. It was considered that incomplete sample homogenisation could lead to overestimation of gold particle size; this could also result from the failure to subtract analytical error from the total error or incorporation of occasional outliers. In situations of high background gold, underestimation could result from chemi-sorbed or very fine gold or marked deviation from a normal distribution. However, no one has yet determined a minimum allowable sample size for a particular sample distribution that can be accepted with complete confidence.

Some investors tend to reject valuations based upon drilling alone and seek to confirm the results by bulk sampling or small-scale mining and where this can



6.35 Comparison between reported and observed gold particle sizes and GOLDCALC size estimates for a variety of styles and sample sizes.

be done it should be done. However, such conditions are limited; factors such as excessive deposit depths, high water tables and loose surface materials preclude the use of bulk sampling techniques in most deeply buried deposits. Bulk samples may be obtained from free-flowing sediment using caissons but depths are usually limited to 5–6 m, rarely to as much as 8 m. Shafts may be sunk to greater depths in dry ground but unit costs are high and may be excessive. Small-scale mining exercises are possible in shallow ground but a full-scale mining operation would have to be mounted in deep wet ground.

Sample splitting

Sample splitting as a method of reducing large alluvial gold samples involves taking aside one shovelful of material from each 'n' numbers of shovelfuls of material from a pit or trench: n may be any number, usually from five to ten. The procedure is repeated until the sample is reduced to the required size. The process is prone to errors of great magnitude, unless carried out with a progressive reduction in particle size. For example, any particulate gold still locked up in the rock could be liberated and thus be unfairly represented as recoverable free gold in a placer gravity circuit. The process has some statistical backing for lateritic and primary gold ores, but large and expensive crushing plant may be needed to effect the reductions and in some cases it would be undesirable because gold may be ‘floured’ or smeared onto other particles in the process and lost.

Note that the process of splitting by coning and quartering involves mixing and shovelling all of the sample material into a conical pile, flattening the top and marking it into four equal segments. The material in either one of the two opposite pairs of segments is discarded; the remainder is again mixed to form a new cone. The process is repeated until the required sample size is reached. The procedure has some statistical backing for small fine-grained samples, e.g. beach sand but is neither practicable nor desirable for auriferous gravel.

6.5 Ore resource estimation

Ore resource estimation is an essential prerequisite to ‘mining reserve’ calculations (Chapter 9) as a basis for mine planning exercises, and is the single most important factor in the gathering of information on which the technical feasibility of the proposed undertaking is established. It will be assumed that in making such estimation:

- all measurements will have been obtained reliably under standard conditions, using proven techniques for taking and processing the samples
- the sampling density will have been such that taking more samples will not have significantly reduced fluctuations in the calculated value of the standard deviation
- given these data, any two valuers will arrive at generally similar conclusions.

Resource quantities for any particular cut-off grade are independent of time and present economics and have no fixed boundaries. Size and reliability of the resource are relatively easy to determine for large homogeneous metal deposits with long lives and relatively long payback periods. Even then, it is unlikely that the resource as a tonnage/grade estimate will be closer than plus or minus 5% in the most favourable conditions. In unfavorable conditions very much closer spacing may be needed to achieve plus or minus 10% accuracy. The degree of difficulty of such estimations for gold ores, in which the valuable metal content is very small and sporadic, is much greater. To obtain gold samples that are both reliable and representative, the drilling-sampling arrangement must be designed specifically for the particular set of conditions.

Each deposit is usually regarded as a series of interconnected blocks for computation purposes (Macdonald, 1983a). Two alternative grid patterns, line grids and rectangular grids are considered on the basis of observed differences in bedrock geology, lithology or geological setting or more simply, according to differences in the drill line spacings. Line gridding follows the course of an alluvial channel and provides cross-sectional views of the channel sediments at selected intervals. Rectangular gridding is generally suited to large deposits having a wide distribution of gold, and to computation methods involving polygons and triangles.

The polygon method of computation involves the construction of polygons around each borehole. The first step in construction is to connect all adjacent drill holes by lines, thus forming a series of triangles. Perpendiculars erected from the midpoints of the lines of each side of a triangle meet at a common point (centroid). By joining the centroids a polygon is constructed around each borehole. With the entire area covered by polygons, each contributes its own area of influence and grade to the deposit as a whole. Polygon areas can be measured by planimeter or computed mathematically by dividing the area into triangles and using trigonometrical formulae for the calculation. The volume of influence is the volume of the prism formed by multiplying the area of the polygon by the depth of its central hole.

The grade of the prism of material is the grade of the central hole. The boundaries of the deposit are defined by the polygons surrounding the outer payable holes. The volume of the deposit is the sum of the prism volumes. The average grade is the cumulative average grade of all of the prisms weighted according to their respective volumes. Where the deposit is of such dimensions as to warrant using the method of polygons or where, for some geographic reason, the holes cannot be spaced evenly, the polygon method of computation is a useful means of computing resource volumes and grades. However, the form of presentation does not lend itself to a pictorial representation of the geology of a placer gold deposit; the method assumes that the sample influence will extend halfway to the next sample and tends to rely more upon statistical relationships for evaluation than upon geological reasoning. Another constraint to this method is the uncertainty of how to close polygons unless it is known where the deposit ends.

Basically, the method of polygons is mostly suited to estimating the resource potential of primary orebodies from randomly spaced borehole intersections. Its successful use in the evaluation of gold placers may depend in any particular case upon how well the data can be adapted for statistical or geostatistical analysis. The method is not suited to the evaluation of stream placers where channels are comparatively narrow.

Methods of computation comprise mainly classical extensions of procedures using weighted volumes and grades. Geometric methods employ sections, polygons and triangles. Classical statistics may be applied to the arithmetic methods to investigate various sources of bias errors in sampling and to throw additional light upon problems of grade estimation including the nugget effect, sample spacing and sample representivity. Geostatistical methods recognise the semivariogram as a measure of sample variance with distance. A process of kriging may derive estimates where semivariograms can be produced for sections of a deposit. All methods make assumptions of a finite relationship of one kind or another between adjacent and neighbouring samples. Most methods rely upon a geometrically designed sample grid to set the pattern for evaluation. None of the methods will produce reasonably accurate estimates if the data from sampling is markedly inaccurate.

6.5.1 Geometric methods of computation

Grade calculations are inherently unreliable because of the difficulties of taking reasonably representative samples. For example, when a mass of sediments is disturbed (e.g. by drilling or pitting) its bulk properties of compaction, moisture, etc., will be changed thereby affecting its volume. Volume recovery measurements tend to be unpredictable and the sample volume measured in the drill pipe often varies significantly from the volume retrieved and measured at the surface. In some cases the pumping action of the bailer pushes some of the sample back into the ground or sucks additional material into the pipe. In others, a high percentage of slimes remain in suspension thus reducing the settled volume of solids. Samples with a high clay content yield lower settled volume recoveries than more granular materials. Although the change in volume is usually a positive value, some loosely compacted sediments tend to swell negatively (i.e. they constrict in volume) when removed from their beds, and thus occupy smaller volumes when disturbed. Conversion of bank volume to loose volume is variable from place to place over a deposit. This may create problems in the conversion of volumes of ore *in situ* to treatment plant measurements (Appendix III). Swell factors, as determined by laboratory methods are of dubious value and although field tests may be more realistic, they are still approximations at best.

Both core rise and volume recovery measurements are affected by the variable and largely indeterminable effect of swell. Normally, if all the *in-situ* column of material is brought into the drill pipe, the core rise is some figure higher than the theoretical rise, depending upon the swell of the material when disturbed. The unreliability of measurement of borehole constants is another constraint.

Borehole constants

Three borehole constants ‘core’ factor, ‘volume’ factor, and ‘drive shoe’ factor are derived from measurements of the drive shoe cutting edge and the internal diameter of the casing. The core factor is the theoretical core rise for a drive of 1 m. The core factor for a drive shoe cutting edge of say 178 mm diameter (area $0.0249\,m^2$) and casing internal diameter 152 mm (area $0.0182\,m^2$) would be:

$$\text{Core factor} = 0.0249/0.0182 = 1.37 \qquad 6.4$$

The volume factor is the theoretical sample volume for a 1 m drive based upon the effective diameter of the cutting shoe. For the above parameters:

$$\text{Volume factor} = 0.0249 \times 1000 = 24.9\,\text{l/m drive} \qquad 6.5$$

The drive shoe factor is the theoretical depth that must be drilled to recover 1 m^3 of sample based upon the effective area of the cutting shoe:

$$\text{Drive shoe factor} = 1/0.0249 = 40.16\,\text{m} \qquad 6.6$$

Based upon 'in-house' experience one major consulting group applies a swell factor of 1.05 to all theoretical values. On this basis in the above example, the factors would be corrected for swell as follows:

- core factor $= 1.37 \times 1.05 = 1.44$ m
- volume factor $= 24.9 \times 1.05 = 26.145$ m
- drive shoe factor $= 40.16 \times 1.05 = 38.248$ m.

By applying the 1.05 experience factor, the assumption is made that all sediments swell to this amount when disturbed and forced into a drill pipe. No doubt in the overall experience of the above group, this factor has been found generally applicable to their calculations. However, it is still an approximation because the amount of swell may vary considerably according to the lithology and compaction of individual layers. Near surface soils may actually be compacted from a loose to a tighter state by drilling. Compaction also influences swell. Samples taken from a layer at 40 m depth will usually bulk higher when disturbed than samples taken from similar but less compacted material at shallow depths.

However, such measurements are often of doubtful validity and engineers differ upon which of the various alternatives might yield the closest estimate of the true value of the ground. Furthermore, most drill samples are much smaller than needed to adequately represent the type of material being sampled. Although an obvious solution is to use larger drilling/sampling equipment, such rigs are often too heavy and lack adequate manoeuvrability for the terrain being explored. They are also too costly for most projects unless modified in some way. One possible approach to keeping weight to a minimum is to modify a multi-purpose set-up for the specific needs of the task in hand. Dividing a large drilling rig into individual segments may sometimes do this. Macdonald (1990) suggested an arrangement comprising:

- a mobile power source for the drill and sampling units
- a track mounted lightweight version of the Bade type of drill stripped down to perform the basic functions of vibrating and hammering casing into the ground and extracting the sample
- a mobile processing plant, which would receive, measure and process each sample as soon as it is taken
- an hydraulic casing puller arrangement, which could move into position alongside the completed borehole and extract the casing while the drill-sampling units move on to the next sample site.

Borehole intersections

The bore of such a drill would need to be large enough say, 1.0 m diameter to extract any occasional small boulders present in the wash. This would provide a reasonably close correlation between sample volume and true volume for subsequent calculation. The basic formula for calculating the value of a borehole intersection is to assume that the amount of gold in the actual sample recovered is the same as in the theoretical sample volume V_t cut by the casing shoe. The theoretical grade G_t is:

$$G_t = \text{Au (Mg)} \times 1000/V_t(L) \text{ mg Au/m}^3 \qquad 6.7$$

In practice, however, the amount of gold recovered is obtained only from the amount of material actually obtained from the sample intersection. This assumption that the grade of the recovered sample is the same as the grade of the cylinder of material cut, is only meaningful if it is also assumed that the material either lost or gained during the sampling operation is of equal tenor to that of the recovered sample.

The 'simple grade' approach assumes that

$$G_s = \text{Au (Mg)} \times 1000/V_a(L) \text{ mg Au/m}^3 \qquad 6.8$$

Wells (1969) notes that most placer engineers correct the gold weight according to either the ratio between theoretical and measured core rise, or the ratio between the theoretical volume and measured volume. The 'modified simple grade' method of calculation takes both core rise and volume recovery into account in calculating the modified simple grade G_{ms}. In each case, the more conservative of the two values is accepted:

$$G_{ms} = (\text{Au (Mg)} \times 1000)/\text{max core or volume recovery } (L) \text{ mg Au/m}^3 \qquad 6.9$$

The weight of gold in milligrams × 1,000 is divided by whichever is the largest of the core rise and recovered volumes in litres, thus adopting the lowest value in each case. This method tends to provide unrealistically low valuations and tends to be favoured by those trying to foster a reputation for conservatism. By taking whichever value is highest in each individual case, the calculated grade is lower than it would be for either core rise volume or volume recovery alone. The method is also disadvantaged by the inability to measure sample volumes closely. Important uncontrolled variables affecting the amount of swell are hole depth, the nature of the sediments, their degree of compaction and moisture content.

Sampling density

Both line and hole spacings are large at first for economic reasons but must be reduced systematically until a sampling grid reaches its final form. Various statistical tests can be applied to the data from time to time during sampling to determine when the optimum spacing is reached. The sample density is normally at a satisfactory level when the depth and grade of any additional holes can be predicted within acceptable limits having regard to the deposit geology in the sectors concerned.

The required sample density varies with the complexity of the deposit geology and may be different in different parts of any one deposit. Possible variations are quite large. In a survey of placer gold practice conducted by Fricker (1980) the range of sample density was 0.4 to 1.6 ha/hole. Lord (1983) suggested a scout drilling density of one hole per 5 to 10 ha or greater for large flat areas of dredgeable ground, reducing to one hole per 2 to 5 ha for close testing. He proposed holes spaced plus 25 m apart on lines of plus 1 km apart reducing to 12.5 m apart on lines placed at closer intervals to define the widths of narrow stream channels. In his opinion, an acceptable spacing for wide channels might be 75 m along lines at intervals of 0.5 km for buried deposits within 30 m of the surface. However, placer formation takes place under variable geographic and geologic conditions and the adoption of a standard sample density for all placer deposits is impracticable.

Examples of drill sample density vs. dredge returns show that some high-density drilling gives poorer correlation than some low-density drilling. For the most part errors can be attributed to the uneven distribution of the gold and the small size of the sample. Examples have been cited of accurate low-density sampling on a dredging property in Idaho where one 44-acre (17.8 Ha) block was prospected by a line of five shafts at each end. The lines were 1,500 feet (457.2 m) apart and the shafts were spaced about 320 feet (97.5 m) apart. The average value of gold for the block was exactly the same for both pit estimates and dredge recoveries, i.e., 9.9 c/yd^3 (7.57 cents/m^3). The ground was dry and the density of sampling was 1 hole/1.78 Ha. But, as in similar examples of recovery estimates in earlier times, there was no accurate measurement of plant losses and the assumed parity between sampling estimates and production was reached only upon a recovery basis. In this case, the pit samples would have undervalued the property by an amount equal to the amount of gold lost from the plant plus that still remaining in any un-mined portions of the deposit. Typically during the period referred to, most of the minus 200 micron gold in the feed was lost. Indeed, judging by the numbers of old workings that have since been re-opened and reworked profitably the overall recoveries were probably quite low.

The assumption that a sampling density that has proven satisfactory for one property will necessarily be equally satisfactory for other properties is similarly impracticable. Each deposit has its own peculiar features and it is important to ensure that a sufficient number of samples is taken from its individual layers in each case to thoroughly investigate the geology of the deposit. To simply grid an area with a predetermined number of evenly spaced holes regardless of its geology is a recipe for failure. Within the range of placer sampling measurements all of the ingredients for undervaluation or overvaluation are present, depending upon how the measurements are taken, interpreted and applied. Individual sample results on their own do not necessarily provide a sound basis for evaluation; only composites of all samples in a suite of samples are meaningful in estimating resource quantities and values.

The diversity of sample recoveries when drilling through gold-bearing gravels has an enormous effect on grade estimates calculated for individual samples. Deposition occurs over time in a wide variety of tectonic and climatic conditions that have important effects on the possible impacts of individual borehole data on total evaluation. The coarsest gold grains occur typically with the coarsest gravel units hence the effects of gold losses in the richest parts of the wash (paystreaks) have more serious effects on the average borehole grade than do losses of finer gold particles from ground deposited under less turbulent conditions. Therefore when coarse gravels are pushed aside as the casing is driven through stony ground, gold is lost from the sample and the ground may be seriously undervalued. However, in less stony ground, gold may be sucked into the hole from outside the casing and the ground may be overvalued, though probably to a lesser degree.

Based upon theoretical values and in the absence of any artificial correction:

- A low core rise tends to ‘undervalue’ due to compacting of loose ground within the drill pipe, exclusion of clasts larger than the drill pipe entrance, clogging of the drill pipe by clasts jammed in the casing shoe, or excessive hydrostatic pressure acting downwards in the pipe.
- A high core rise tends to ‘overvalue’ due to rising sand, a higher than normal hydraulic gradient providing artesian conditions within the pipe.
- A low-volume recovery tends to ‘undervalue’ due to low core rise, loss of sample driven out of the pipe by the bailer, or excessive slimes size particles.
- A high-volume recovery tends to ‘overvalue’ due to excessive core rise, the sucking action of the bailer or excessive swell in the measuring bucket.

Use of experience factors

The best-known experience factor was developed in Malaysia by a noted tin mining engineer from whom the factor, the Radford Factor, was named. It was derived from a comparison between cassiterite sample grades obtained from a 3 ft diameter shaft and those from a centrally placed borehole. The factor may have been reasonably appropriate for relatively equant and finely divided alluvial tin under Kinta Valley conditions. It has had a mixed reception elsewhere; generally accepted by some (e.g. Breeding, 1973), but less so by others (Fricker, 1980, and Macdonald, 1990). There are many other variations and similar illogicalities on the above themes. Some are more optimistic than others and inevitably there are many uncertainties and many failures. Any errors of interpretation, judgement or computation will increase the possible variance.

It must be realised that borehole sampling does not produce an exact assessment or guarantee that the dredging results will not differ in some way from the sampling estimates. The best that can be done to avoid serious error is to generate the data under standard conditions and check and interpret the information carefully in accordance with the deposit geology.

Engineers thus differ upon how to evaluate data from drilling and sampling and so determine the volume and grade of the gold ore *in situ*. Most go through the basic procedures of measuring core rise and volume recovery but there is little agreement thereafter. Large, well-graded deposits are more easily sampled and contain fewer surprises than smaller deposits in which the sediments are predominantly poorly sorted, and the gold is sporadically distributed in a wide variety of sizes and shapes. A common approach to valuation is to apply some form of correction factor to the field data to compensate for sampling errors and expected plant losses (see Section 9.3.1). This procedure, however, is usually based on personal experience or upon the experience of others and frequently leads to estimates of mineable ore reserves, which are quite unrelated to actual plant recoveries and losses. In most cases experience factors are adopted or adapted simply to lend an air of conservatism; many engineers use a factor to downgrade high values but not to upgrade low values. Some, in order to be ultra-conservative, apply further arbitrary corrections to present the worst possible case. One early engineer was most stringent in criticising the motives of operators who reduce their original estimate by 10 to 20% claiming that the practice is done more to allow for defective recovery by a dredge, than through a lack of ability in their own judgement. Cope (1988) noted that the adjusted value given to a placer deposit usually depends upon the engineer's powers of deduction and experienced judgement, rather than on the rigid application of a particular formula or formulae.

However, adequate 'local' experience may allow experienced placer engineers to closely predict, from their own sampling, the total quantities of gold that will be recovered from subsequent production dredging, although individual block estimates may differ appreciably. Gardner (1921) describes a dredging property where the overall recovery efficiency was 93% of the estimate but individual block recoveries ranged from 32 to 149% of the estimate. However, in another example, where dredging on a 195 ha section of the same property, the total recovery was 141% of the estimate but ranged from 104 to 199% in individual blocks. It might be significant that different drill crews were involved in different sections of the deposit even though the same engineer supervised all of the drilling.

Average conditions seldom apply over the whole of a deposit and individual sections should be determined separately and dealt with according to their respective characteristics. The author accepts an occasional high or low value as being natural features of alluvial gold deposition, but tries to more closely define the spatial extent of their influence by drilling closely around all anomalously rich or anomalously poor holes within a possible mining path. Alluvial gold concentrations vary under the changing flow conditions of the depositional environment and considerable doubt applies to the use of any experience factor (see Section 9.3.1).

6.5.2 Statistical analysis

Techniques of statistical analysis can be applied to any quantifiable sample data to help provide a better understanding of orebody characteristics. Available techniques also provide a means of determining how many additional samples may be needed to reach the stage at which the purposes of the particular sampling exercise will have been achieved by the number of samples taken. Useful procedures are provided for examining the reliability of volumes and grade estimations but without always providing unambiguous answers. Shortcomings of the method have been attributed to its faulty assumption of randomness of sample data in space or time. By ignoring deposit geology, statisticians tried to develop a purely mathematical treatment of sample data that would be independent of any bias. They failed for what are now obvious reasons and geostatistics is emerging to take its place.

The first step in analysis is to set up a frequency distribution as a database. All placer sediments have some form of symmetry that allows predictions to be made linking their various characteristics. Hence, if all of the sample grades in a distribution are grouped within suitable class intervals, it is theoretically possible for a sufficiently large number of samples, to predict the frequency with which future samples will fit into each of the classes. Using statistical techniques, a ‘fiducial interval’ can then be determined around the computed deposit grade such that the true grade of the deposit will fall within that interval for a specified degree of confidence.

The size range of a fiducial interval is a function of the number of samples and varies approximately in inverse proportion to that number. It should be noted that the computation of a fiducial interval, and procedures such as Sichel’s ‘T’ estimator assume lognormal distributions whereas most sedimentary distributions are skewed and non-lognormal. For these, the conventional techniques may give more reliable estimates if the individual grades are first weighted (sample length times sample grade weighting) to account for any variations in the length of the sample intervals and then normalised by taking logarithms of the values.

In presentation, the frequency distribution is plotted on a graph. Frequency, as the dependent variable is plotted on the vertical ‘Y’ axis; the grade interval is plotted on the horizontal ‘X’ axis as the independent variable. The normal construction is in the form of a histogram for which the most efficient number of class intervals is between 10 and 25 (Hazen, 1958). The technique is objective in its application and can be used to help solve many practical problems relating to grade estimation. These problems include dealing with extreme assay values and determining optimum sample spacings.

The location of a high point on the histogram is a characteristic that may be measured by a typical value or average value. This is the point of central tendency of the mass of data and may be used as a basis for measuring or evaluating extreme values. Tests include analysis of:

- variance (F distribution)
- chi-square test
- probability level.

Variance

'Variance' is a measure of the scatter of the data about their mean value and is the basis of the variogram. It conveys no information about their spatial variation or their spatial distribution. A short note on variogram structural analysis is appended (see Appendix II). The technique of the analysis of variance requires the comparison of two variances and a test for the significance of the differences between the calculated variances. The larger variance is divided by the smaller variance to give the 'F' ratio.

$$F = {S_1}^2/{S_2}^2 \qquad 6.10$$

Tables are available, which supply required values of the various calculations to facilitate the application of statistical techniques and save the time and money involved in the calculations. One such table is the table of F values. In this table there are fewer than five chances in one hundred that the disparity between the calculated variances at the 5% level is due to chance if the calculated ratio between the two variances (F) exceeds the value for F indicated in the body of the table. If F exceeds that recorded for the 1% level, the probability is less than 1 in 100 that the difference is accidental.

Chi-square test

The chi-square table is used to test for goodness of fit and may be compared with the normal curve distribution to determine if the sample data represent a normal population. The table is also used to test the validity of hypotheses and is based upon the differences between observed frequencies (f_0) and expected (theoretical) frequencies (f_t) as follows:

$$X^2 = (f_0 - f_t)^2/f_t \qquad 6.11$$

Use of the chi-square table indicates the range of probability. A low value indicates a small probability that any differences are accidental or could have evolved through sampling variation. A large probability value indicates that the differences could have arisen due to chance or sampling variation.

The 'standard deviation' (S) is the positive square root of the variance and is the most commonly referred to statistical function in alluvial sampling practice.

$$S = (\Sigma d^2/N)^{0.5} \qquad 6.12$$

Σd^2 denotes the sum of the individual squared deviations from the mean. For a frequency distribution in which the grade intervals are of equal size, the deviation may be taken in terms of grade intervals from a selected mid-point of one of the grade intervals. Each value in the distribution affects the standard deviation, which, by its nature responds to the varied distribution of gold in the placer and to the amount of errors made in obtaining and analysing the samples.

The value S^2 is thus not without bias hence, it is often desirable to use the variance. The standard error reduces progressively with increasing numbers of sample analyses, the amount of reduction becoming smaller with each new set of data until it becomes insignificant. At this point, no additional amount of drilling will reduce the standard deviation for the particular drilling and sampling techniques used, but it will reduce the standard error of the mean:

$$s_x = s/N^2 \qquad 6.13$$

where s_x is the standard error of the mean, s is the standard deviation and N is the total number of analyses in the distribution.

Probability level

The average grade of the deposit is determined for all practical purposes at this point and the only effect of further sampling is to slightly increase the confidence with which the estimates can be accepted. This confidence level relates to the size of the 'fiducial interval' for the conditions of the exercise. The size of the 'fiducial interval' depends upon the number of samples and the standard deviation. It establishes limits above and below the estimated grade and, for any specified confidence level, states that the true grade of the deposit lies within those limits with only x number of chances in 100 of being wrong. For example, at a 95% level of confidence there are ten chances in 100 that the estimate is wrong and so on. The fiducial interval (FI) is given by the formula:

$$FI = Ma + t_{0.05}S_x \qquad 6.14$$

$t_{0.05}$ is the t value of the 95% level of confidence taken from t tables.

Although originally designed for normal distribution Hazen (1958) believes the chi-square test to be a good approximation when used with moderately skewed distributions. Formulae developed by various workers in the 1950s (for example, Sichel, 1951–52; De Wijs, 1951; Krige, 1951) form the basis for most modern statistical ore grade computations.

Geostatistics

The procedure for making a geostatistical ore resource estimation requires first investigating and modelling the physical and statistical structure of the orebody. Concepts of continuity and structure in the deposit are embodied in semi-variograms that are constructed during this first step. The semivariogram is the only simple way of verifying the applicability of geostatistics, trend surface analysis, or even classical statistics to the deposit in question. In short, the construction of an experimental semivariogram should be as automatic a step in ore reserve estimation as the construction of a histogram (Clark, 1979). The second stage of the procedure is the estimation process itself 'kriging', which depends entirely on the semivariograms constructed during the first stage.

Geostatisticians recognised what geologists had known all along, that ore deposits do not occur haphazardly and hence, that mineralisation is spatially related and not distributed randomly. In fact, if the pattern of drilling is such as to describe a deposit adequately for geological interpretation, ore reserves computed by geometric methods may not differ greatly from the geostatistical computations for that deposit.

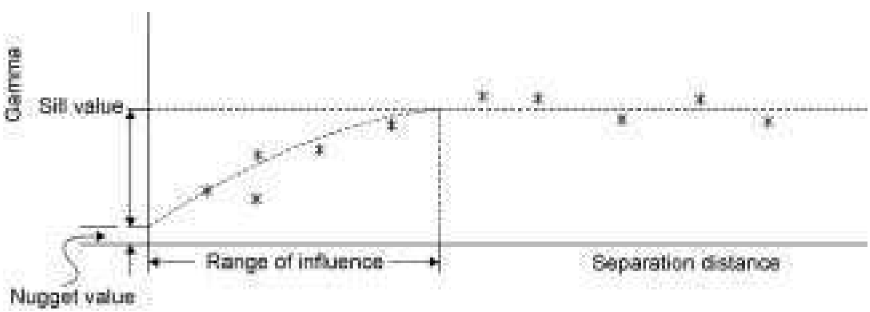
The semivariogram

Geostatistics measures the variance as the sum of the squares of successive differences in spatially related data. The basis of geostatistics is the semi-variogram $Y(h)$, which as a measure of the variance with distance is defined as:

$$Y(h) = [F(x+h) - F(x)]^2 / 2[N(x+h, x)] \qquad 6.15$$

$F(x)$ is the value of position x, $F(x+h)$ is the value of position $x+h$; and $N(x+h, x)$ is the number of pairs where differences have been squared and summed. A common form of variogram (Appendix II) is illustrated in Fig. 6.36.

The regionalised variable x may be any property in the status quo, e.g. grade, grade $\times$ depth, etc., for which a reliable semivariogram can be obtained. Kriged estimates for blocks within the deposit are then obtained by weighting adjacent



6.36 Common form of variogram.

Table 6.2 Comparison of geostatistical and polygonal estimates of section of Taramaku alluvial gold deposit, New Zealand (after Fricker, 1980)

	Volumes ($\times 10^3\ m^3$)	Weight of gold (kg)
Production	8047	774
Estimates		
Polygonal	7471	1174
% production is of estimate	108	65.9
variance	8540	3311
coefficient of correlation	0.51	0.06
Kriged (100 m square panels)	7745	702
% production is of estimate	104	110
variance	8510	231
coefficient of correlation	0.42	0.48
Sichel estimator		
% production is of estimate		59.1
variance		4071
coefficient of correlation		−0.25

blocks using weighting coefficients obtained from the nugget effect and range of the semivariogram and in such a manner as to give minimum estimation variance and to eliminate bias.

Fricker (1980) reviewed data from 370 boreholes covering the whole of the Taramakau alluvial gold deposit in New Zealand. He found that the semivariogram for the whole of the deposit was in a recognisable form but attempts to produce semivariograms for parts of the deposit were unsuccessful. He then applied the semivariogram for the whole of the deposit to derive kriged estimates for contained gold. His summary of the estimates from the geostatistical methods used and estimates calculated by the polygonal method are compared with actual production records in Table 6.2.

The distribution comprised the aggregate of ten successive and adjacent quarterly periods, a total of 69 boreholes. The geostatistical method gave a reasonably accurate account of recovered gold and was of moderate reliability. The polygonal method failed almost completely for sub-sets of this size. The Sichel estimator grossly overestimated the gold content but had greater correlation for the sub-sets of this size than the polygonal method. Fricker's (1980) conclusions are relevant to most geostatistical analyses of gold placer sample data seen by this author:

> Although geostatistics demonstrate in this example that it gives us a better estimate of production and is more reliable than the polygonal method, it hasn't necessarily proved anything. For a start, the variogram was for all of the deposit and not from that section only. We have doubts about the quality

> of our data, how much gold was left on basement and how much was lost by the dredge. The conventional polygonal method to which an experience factor is applied to allow for losses may in fact be a better estimate of what is present in the ground. However, the fact that the kriged estimate is more reliable is significant. The polygonal method takes no cognisance of adjacent values. Intuitively we feel that this is wrong, hence the advantage of the kriged method which does. The triangular method takes cognisance of three adjacent values, and may give more reliable estimates for small sub-sets than the polygonal method. The 90% lower confidence level of the kriged estimate of grade 34 mg/m^3 is too low for making investment decisions. The exercise at least demonstrates the validity of using appropriate statistical methods.

Similar geostatistical investigations by others in New Zealand on a dredged and on an un-dredged prospect conclude that:

- Geostatistical methods tend to underestimate volumes, possibly because of inadvertent reprocessing of a certain amount by the dredge.
- Geostatistical methods give estimates of gold content much closer to the recovered content (as obtained by dredging) than arithmetic methods.
- The borehole spacing used for conventional estimation enables a sufficient level of confidence in global (i.e. whole deposit) estimates so as to make an investment decision but there is nowhere near the same confidence in the conventionally developed dredge path, which is usually selected to dredge richer areas first to help the cash flow.

The confident selection of a dredge path using geostatistical procedures may require a close drilling grid and computations on small blocks (say 50 x 50 m in each case) if the prospect is only marginal. Thus, despite all the seeming advantages of geostatistical procedures, Fricker advocates caution: 'The methods are quite sophisticated and require computers for solution. I am left with the feeling that our data is too crude for such methods and knowledge of our recovery systems inadequate. Nevertheless some improvements on the conventional procedures, particularly the triangular method are necessary. These could include some elementary classical statistics that anyone can use.'

This author has occasionally called upon the services of geostatisticians to help resolve conflicting views on resource estimation. However, the estimators have usually expressed some difficulty in establishing semivariograms that adequately match the theoretical models they are based upon. Fricker's caution is quite understandable because for the most part it seems clear that geological factors eventually hold the key to geostatistical success.

The usual reasons for poor variograms in alluvials are:

- poor data
- failing to appreciate that large high-grade concentrations at bedrock produce a gross nugget effect

- drill holes spaced at distances apart greater than the range of influence
- sample volume being too small to fully represent the nugget component of the variation swamps the spatial component and the variogram looks like that of a pure nugget effect.

It is a matter for human decision, which path to take in arriving at a final assessment of resource and reserve quantities and grades. If geostatistics is the tool used, one set of results may be obtained. If conventional arithmetic/statistical techniques are relied upon, another set of results can be expected that may or may not conform closely to the geostatistical estimates. Clearly, the rights of one or other of the alternatives must be established before proceeding to final evaluation.

7
Mine planning and practice

The ultimate purpose of mine planning is to devise a strategy that will optimise project economics within the physical constraints of the deposit characteristics. Planning commences with the collection and analysis of data from sampling and eventually covers all aspects of mining and mechanical engineering practice pertaining to the design of essential services, infrastructure and environmental protection. The process requires the close co-operation of field personnel, design groups, manufacturers, management and financial agencies in an engineering appraisal of mining alternatives and project economics. In the final operational phase, the successful scheduling of material movement and equipment to meet target requirements will depend upon how soundly the mine plan is constructed, and how well it is translated from the drawing board into the reality of prototype operation.

Methods range from simple hand operations to systems involving large fleets of earth-moving equipment and dredgers capable of digging many thousands of tonnes per hour. The basic systems are similar to those of civil works such as land reclamation, dredging of harbours and waterways, road construction and quarrying. A continuing problem is to achieve a satisfactory balance between the digging rate and the handling capacity of the treatment plant. The digging rate is an average of the rates of ore extraction in both easy and difficult sections of the orebody. The treatment rate varies at any one time according to the nature of material being processed in the feed preparation section. Substantive issues include the co-ordination of mining and stripping operations, minimising downtime, environmental protection and rehabilitation of mined-out areas.

7.1 Planning

A wealth of performance data from non-selective earth-moving operations can be drawn upon when planning a surface mining operation, although such experience must be viewed cautiously when predicting the performance of similar type machines in placer mining activities where selectivity is a fundamental requirement. The first choice is between wet and dry systems of mining.

The principal wet methods of mining are hydraulic sluicing, bucket line dredging and hydraulic dredging (represented by bucket wheel and suction cutter dredgers). Dry mining systems employ almost the entire range of earthmoving equipment used in civil engineering applications. Some operations, particularly small to medium-scale ventures, utilise various combinations of both methods of mining. An overriding operational consideration is to provide a generally compact and closely co-ordinated field administration. Pit-wall stability is vital to operational safety, and any instability due to soil weakness, ground water inflow and variable hydraulic gradient may add significantly to the additional amount of sidewall material required for safe operation and hence to stripping costs. The normal sidewall slope angle for a dry pit is 45 degrees although it may have to be flatter, depending upon the extent of the seepage and composition of the wall rocks. The normal sidewall slope in a dredge pond is variable around 70 degrees from the vertical. Thus, for the same deposit depth, stripping requirements and costs are higher for dry than for wet systems of mining because of the additional amount of material that must be moved to provide safe wall conditions for dry excavations.

Dry mining systems are generally more specific in overburden rejection and clean up than are wet mining systems, and have the advantage of visual control of orebody extraction. Benefits arise from a high degree of selectivity at the mining face, close control of feed to the treatment plant, and the ability to manually clean up and recover pockets of high-grade ore at bedrock. The various methods are positive in their actions, and can usually be relied upon to keep stockpiles at acceptable levels for continuous treatment plant operations regardless of the mining conditions.

Important features of wet mining systems are direct transfer of feed material to the treatment plant and a generally compact and closely co-ordinated field administration. They are less sensitive to ground water movement than are those of dry mining and will usually be more cost effective in terms of material shifted. Constraints to the method are high first cost, lack of visual control at the working face, less efficient cleaning up of gold from bedrock and reduced specificity of feed to the treatment plant. Factors influencing the selection of individual surface mining systems are summarised in Table 7.1. A broad comparison between the two systems is given in Table 7.2.

7.1.1 Data for planning

Raw data for planning are based upon a combination of historical and government records of previous mining activity, investigations of relevant aspects of the geology of the orebody and the geography of its immediate surroundings. Data generated in these fields include sediment characteristics and layering, gold characteristics and distribution, and resource quantities and grade. The inventory of gold-bearing material comprises both resources (not necessarily economic)

Table 7.1 Factors influencing selection of placer gold mining systems

Mining system	Dredging				Hydraulic mining	Dry mining	Hand mining
	Bucket ladder	Bucket wheel	Jet lift	Clamshell			
Minimum volume (M) to justify operations in average values	20,000,000 to 120,000,000	10,000,000	100,000 Sea only	100,000 Land or sea	100,000	1,000,000	Any small quantity
Preferred nature of basement	Soft and even, few hard pinnacles or bars	As for bucket but more tolerant	Less critical than other forms of dredging	Soft and even	Soft preferred but can handle hard	Soft and even capable of supporting heavy traffic	Hard or soft
Nature of mineralised beds	Reasonably free with few large boulders	Unconsolidated gravels and sand	Unconsolidated gravels and sand	Unconsolidated gravels and sand	Can be broken and fluidised using jets	May have small degree of consolidation	Preferably soft but not critical
Preferred nature of overburden	Unconsolidated	Unconsolidated	Unconsolidated	Unconsolidated	Capable of being ripped or broken by jets	Rippable	Preferably soft but not critical
Water requirements	Large	Large	Large	Variable to large	Large	Nil	Variable
Bottom slope	Relatively flat preferably 1:40 for artificial ponds	Relatively flat	Relatively flat	Not critical	Ant degree of slope but preferably around 5º	Not critical	Not critical
Ocean conditions	Maximum wave height 1.25 m	Maximum wave height 1.25 m	Depending on vessel	Not applicable	Not applicable	Not applicable	Not applicable

Table 7.2 Comparisons of dry and wet mining systems (adapted from Macdonald, 1983a)

	Dry mining	Wet mining
Applications	Shallow surface deposits, tightly compacted or indurated sands, irregular geometry, high-level dunes, desert environment	Ample water available for mining and treatment of shallow surface deposits, high-level dunes, marine environment
Equipment system is built around	Bulldozers, articulated front and loaders, draglines, hydraulic excavators, bucket wheel excavators	Pumps and monitors, suction and bucket dredgers, bucket wheel dredgers, clamshell dredgers, jet lift dredgers, hydraulic excavators
Controlling factors for selection	Proposed scale of mining, minerals distribution and value, location and physical chracteristics, slope and texture of mining floor, surface and bedrock geometry, insufficient water for wet mining, position of water table	Proposed scale of mining, deposit size and grade, location and physical characteristics, slope and texture of mining floor. Bedrock geometry, adequate supplies of water for all purposes
Advantages	Ability to handle group of small deposits, constant feed rate under widely different mining conditions, selective mining leads to optimisation of feed grade control, recoveries may approximate 100%	Mining and processing incorporated in one unit. Low unit mining costs, closer supervision and control, only possible method in excess water conditions
Disadvantages	High unit operating costs, inability to handle large volumes of water, requires firm base for vehicle movement, requires large on-site workshop facilities and stock of spare parts	Mining losses sometimes high, less selectivity in mining, high relocation costs, high capital costs, large water requirements, ecological problems may affect large sections of environment

and reserves (presumably economic). An underlying theme is the need for standardisation in all of the techniques used to compile the data for planning so that probabilities and risks can be evaluated fairly in final economic studies. Categorisation depends upon the valuer's opinion of the scope of the sampling data and the degree of confidence given to the expected recovery component. Criteria for testing these estimates include a range of statistical, geostatistical and geometric techniques, which are only as reliable as the data from which they are prepared (Chapter 6).

Perceptions of deposit characteristics and environs and the proposed scale of mining influence the choice of methods and equipment for a particular mining project. There may be several possible choices but generally one particular method is found that suits the conditions and needs of the project better than any other. The test is for both method and equipment to be capable of accurate time scheduling and between them to provide sufficient flexibility for coping with any unexpected problems. Where the choice offers several apparently equal alternatives, their respective strengths and weaknesses should be carefully evaluated before making the final decision. Performance records of mining in similar types of ground may be critical to the decision.

Cost estimates include all capital and operating cost schedules including pre-production development and inventory requirements. Estimates are based primarily upon expected hourly productivity, availability and utilisation for each piece of machinery. The actual selection and sizing of equipment is governed by annual production requirements and selected methods of mining. The mining sequence usually calls for high-grade production in the early years in order to maximise the return on investment (see Chapter 9). Overall, the plan must combine cost effectiveness with optimum productivity while still providing for satisfactory environmental protection and rehabilitation.

A detailed schedule of activities and likely costs of bringing the project through from its resource stage of development to full-scale production is generated in the field. It should be fully representative of the major features and be collected and recorded in a specified form to facilitate interpretation. Data prepared by standard methods are easily tested and frequent checking will usually provide estimates within the limits of normal sampling error. Non-standard data are confusing and tend to promote widely different interpretations.

7.1.2 Mapping

Map types can be categorised into four geoscience-based categories for mine planning:

1. cadastral
2. topographic
3. geological
4. image.

Within each map category a further subdivision relates to scale, i.e. the relationship between ground units and map units. Generally accepted arbitrary scale units are:

- large scale: 1:5,000, 1:10,000, 1:25,000
- medium scale: 1:50,000, 1:100,000, 1:250,000
- small scale: 1:1,000,000, 1:2,500,000.

Cadastral maps

A cadastral map provides the background to a mining tenure application. It has three main functions which are to:

1. illustrate and identify the boundaries of each parcel within a parish or county
2. contain the major drainage pattern of water courses where they form a legal boundary
3. contain a graticule of latitude and longitude so that true north can be identified and will determine the status of the land (private, crown, or reserve) as defined under the Mining Act.

In Australia, Cadastral maps are the graphical representations of the legal cadastre or land tenure framework. They are consulted to determine the nature and classes of land holdings prior to the commencing of prospecting or mining.

Topographic maps

Topography is derived from the Greek 'topos' (place) and 'graphos' (I write). Topographic maps are inventories of the physical features of the Earth's surface and include the names of many features. Conventionally they are printed in colour and colour is used to identify the various features:

- black – cultural features such as railways, fences, buildings, powerlines
- blue – hydrographic features such as rivers, streams
- brown – hypsographic features (relief) shown by contours
- green – land cover such as timber, vegetation
- red – road and track systems.

Like cadastral maps, topographic maps are bounded by meridians of longitude on the eastern and western boundaries and by parallels of latitude on the northern and southern boundaries. Australian maps additionally contain a 1,000 m grid map referenced to the Australian Map Grid (AMG).

Topographic maps are normally available in the following scales/format (longitude by latitude):

- limited coverage at 1:25,000 – format 7.5 minutes by 7.5 minutes
- limited coverage at 1:50,000 – format 15 minutes by 15 minutes
- full coverage at 1:100,000 – format 30 minutes by 30 minutes
- full coverage at 1:250,000 – format 1.5 degrees by 1.0 degrees
- full coverage at 1:1,000,000 – format 6 degrees by 4 degrees.

The date of compilation of the map is important because although most natural features are fixed, some may change due to erosion or excavation, and man-made features such as fences and building are subject to alteration. For example, at Porgera, Papua New Guinea during the 1960s, a section of Yakatabari Creek had shifted about 50 metres since being located by wartime mapping 20 years earlier. Geologists were embarrassed when a horizontal diamond drill hole, sited according to the wartime data, missed the orebody completely and emerged from the hillside after penetrating only about 150 metres of solid but barren ground.

Geological maps

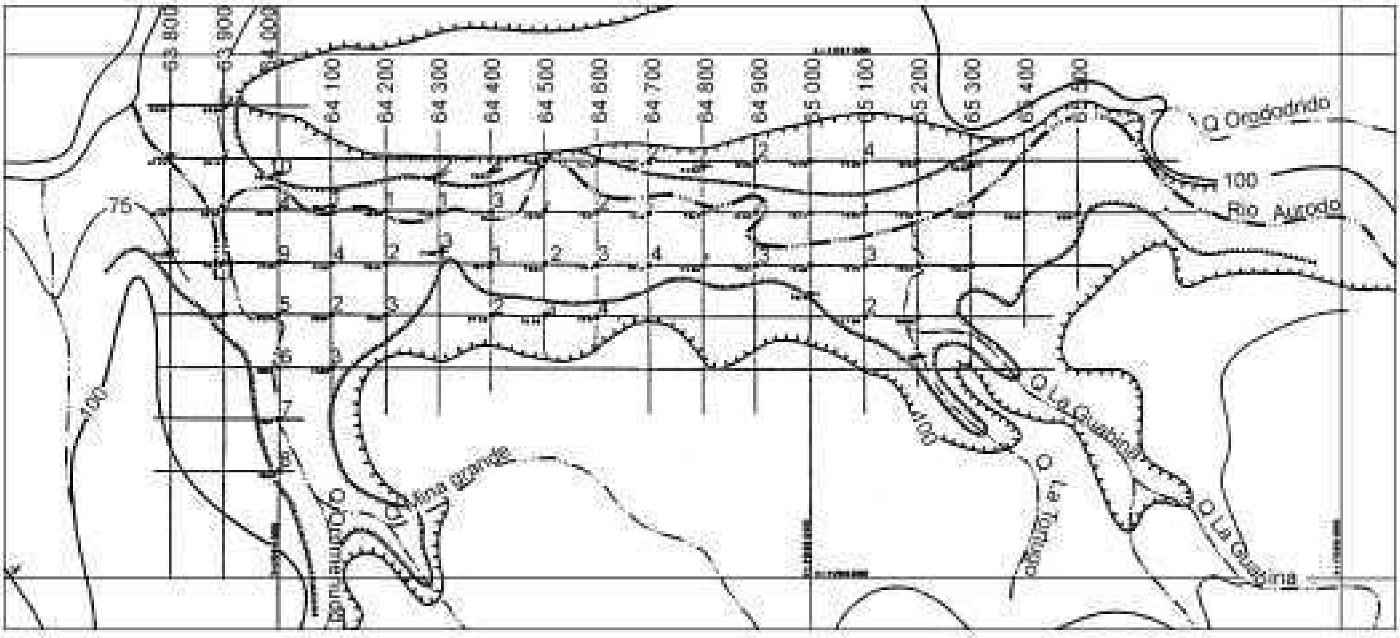
Geological studies investigate the nature of the deposit in terms of ore genesis, mineral association and geomorphic history as a prerequisite to elucidating the local geology upon which the mine plan will be based. The data are displayed on maps and sections accompanied by notes describing such features as the physical nature of the ground, level of the water table, lithology, compaction, swell, and sediment size distribution. Notations refer to bedrock characteristics and the degree and depth of weathering of the various rock types, the occurrence of rock bars and rock pools, slope change, etc. Associated geographical data relate to meteorological records of both short- and long-term records of precipitation, temperature, wind strength and direction, storm cycles, waves, tides, currents, etc. Suitable map scales for mine planning are generally as surface plans and section maps.

Surface plans

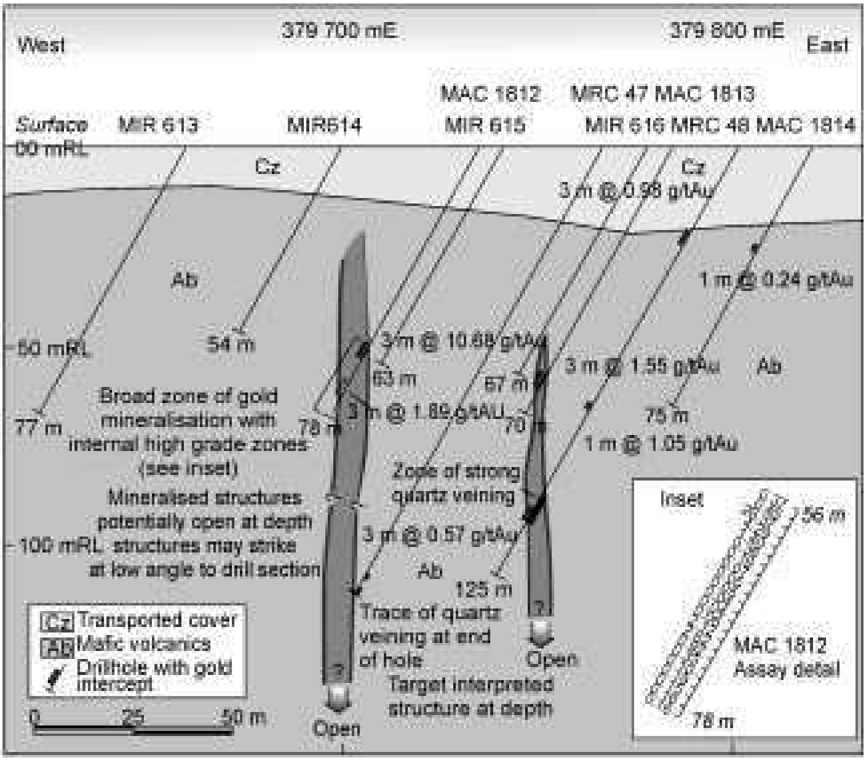
Surface plans (scale 1:1,000–1:2,000) feature the surface contours of orebodies at the main horizons (ground surface, top of ore zone, bedrock surface, etc.). The preliminary ground surface plan of the Rio Aurodo gold placer in Colombia, South America (Fig. 7.1), which locates the drill-lines, sample points and both natural and man-made features such as streams, valley walls and tracks is a typical example of a placer map. Sample data for each borehole in the ore zone are used to compute average grade, ore zone interval grade and depth from surface to the top of the ore zone and to bedrock. 3D-type plans may be developed for all relevant horizons down to bedrock.

Section maps

Borehole line sections are plotted on section maps (scales: horizontal, 1:1,000–1:2,000; vertical, 1:100–1:200) across each deposit. Figure 7.2 represents a line of boreholes across a geological section of Mitchell Basement drill line showing a broad zone of gold mineralisation in mafic volcanics underlying transported cover. Plotted on all of these sections are the subsurface water table, lithology, and borehole sample data and bedrock type. Individual borehole lines can also be plotted longitudinally in straight sections of a placer deposit.



7.1 Preliminary ground surface plan – Rio Aurodo gold-platinum placer, Colombia, South America.



7.2 Geological section of Mitchell Basement drill line, North Prospect, Western Australia (Chalice Gold Mine Limited, 2006).

Image maps

Image maps are derived from aircraft photography coverage and imagery from space vehicles and satellites (refer to Chapter 5).

7.1.3 Environmental protection

The need to protect the landscape from long-term damage and to preserve important species of local flora and fauna is an important factor influencing surface and sub-surface mining. Environmental impact studies examine the effects of any proposals made and point to possible solutions of any problems raised. There is no common standard. Different governments have different views on the required level of protection and some are currently lax in their administration. In practice it is the moral responsibility of the operator to conform to basic requirements and ensure that minimum standards of environmental protection are met with at all times.

Regional considerations

The development of a landscape through time is a natural progression of sculpturing and slope development and is roughly predictable in the short time of a normal environmental cycle. Provided that nothing catastrophic occurs to upset the equilibrium of a particular geomorphic system, change occurs slowly and the total environment will adjust gradually to the change. But if a sudden change occurs the reaction will be rapid and a complete environment may be destroyed. Indeed, any natural phenomenon that causes the base level to change rapidly may induce radical environmental changes that are irreversible. A flood plain may become a lake as a result of damming by a landslide. A mud flow, such as that which followed the Mt St. Helen's volcanic eruption in 1980 may fill valleys with mud, coastal plains may be inundated by the sea; the list is endless.

Natural process is not easily halted and there can be no excuse for actions that invoke rapid and irreversible responses because of carelessness of the fragility of the environment. Provided that the likely impact is known, an engineering solution can usually be found that will safeguard the long-term integrity of an environment. Short-term changes are unavoidable but they should not be such as to lead to the destruction of a landscape or to a worsening of communal life-styles. Responsible mining companies institute restoration processes to help preserve the salient features of an environment, or replace some less useful or unattractive features with more acceptable options. This was shown by Schlemon and Phelps (1971) who described the restoration of dredged areas of the Rio Nechi, Colombia and the provision of elevated tailing areas for the cultivation of plantains and other food crops. The local people (Colomos) have come to rely upon the availability of dredge tailings, piled above normal ground level, to plant crops where previously the soils supported only swamp and jungle plants.

The impact of mining on the marine environment is most importantly associated with the disposal of tailings and slimes. Erosion or accretion of the seabed as a result of mining affects biochemical processes and inhibits marine life in parts of fishing grounds. Navigation hazards may be created by disturbances to the normal pattern of littoral drift. The relocation of large quantities of near shore sediments drastically affects the energy balance offshore resulting in coastal erosion. Other harmful responses include high sediment suspensions, which inhibit light penetration thus reducing photosynthesis and the primary growth of marine life. In all of these matters knowledge of the possible extent of the impact is important to considering how to avoid their worst effects.

Local considerations

The extent to which a particular operation adversely affects an onshore environment is influenced by such factors as its proximity to local communities, waterways and reservoir catchment areas, the possible introduction of toxic substances such as mercury and the need to preserve any unique species of flora and fauna. It is important to minimise noise pollution in settled areas where sound levels for houses should not exceed 30 dB at the outside walls during the daytime. Depending upon how well the house is insulated against noise, this level may have to be reduced further at night when one particular sound is more noticeable. Typical complaints include:

- pollution of streams and other waterways
- unsafe disposal of excess spoil particularly during the opening stages of a mining operation
- problems of water conservation
- inadequate rehabilitation of mined out areas
- cultural shock, i.e., the impact made on the lives of local inhabitants through the incursion of strangers who may not speak their language or may wittingly or unwittingly do things that the indigenous people find objectionable
- health hazards imposed by disease and privation.

Pollution of streams and waterways

Few governments now allow direct dredging in any streams from which the water is used by riverside dwellers for their daily needs. Rules are framed to ban the uncontrolled discharge of dredger tailings and slurries into waterways and catchment areas. If dredging is to take place in such a manner or location as to present a stream pollution hazard, the dredge path and tailings disposal areas must be isolated safely away from the waterways. Dam walls must then be sufficiently robust to prevent destruction by flash flooding.

Land restoration

It is seldom possible and would, in many cases, be undesirable to restore a mined out area to its original state. Instead, consideration should be given to the alternative uses such land could be put to. Alternatives vary from tourist facilities such as parks, gardens and housing developments, to agriculture or afforestation. Cost is seldom very significant if restoration procedures are built into the original mining plan. Restoration can be very expensive if the form it takes is decided upon only after mining has commenced.

Cultural impact

Local tribesmen in remote areas are generally friendly and helpful but most have had sufficient contact with the outside world to be suspicious of strangers. It should always be remembered that these people own the land they exist upon and are entitled to determine who should have access to it. They rightly expect to be consulted and be compensated fairly for anything that is planned and done. Good relations are essential to good productivity at both local and government levels; nothing should be allowed to detract from these relations. Good relations also extend to the responsibility taken by the project team in remote regions to the general health and well-being of the community as a whole and not only of company employees. Diseases like malaria, dysentery and hepatitis are endemic to most of the tropic regions of the world, and sometimes reach epidemic proportions. Diseases such as bilharzia, which spreads by a variety of snail in some limestone environments and sleeping sickness, which is contracted from the tsetse fly, are less general, but equally serious where they exist.

7.1.4 Mine plan checklists

Checklists are prepared to ensure that all aspects of prototype planning operations are of high quality and sufficiently comprehensive for all of the designers' needs. The data date back to the earliest stages of sampling and other exploration activities and no essential information should be dealt with casually or overlooked when proceeding to final design and evaluation. If only for this reason, the planners should be closely involved with all aspects of the project from the start. Apart from helping to organise the work, the planning section can continually monitor its progress so that any deficiencies in collecting quantitative information may be remedied as soon as they are observed. There should not be any need later to replace any steps or conduct further studies requiring further material for critical evaluation.

A comprehensive checklist should apply to each important item. For example, site preparation (Appendix III) for mining involves the following actions relating to the deposit characteristics and environment:

- removal of vegetation and surface or overburden stripping
- setting up a water supply system including slime and tailing dams; dewatering the ground for dry mining or sluicing operations (if applicable)
- stream diversion (if applicable)
- protection from flooding, e.g. drainage around the pit site
- location of infrastructure
- construction of campsite
- construction of roads and communication facilities.

The main elements of an overall mine plan checklist are summarised in Table 7.3.

7.2 Operational concepts and schedules

Channel sinuosity, width, and depth are the main variables of channel geometry affecting the proposed method and scale of mining and the predicted

Table 7.3 Summary of mine plan checklist

Deposit	Water	Overburden
Volume and grade	Depth to water table – mean and seasonal fluctuations	Vegetation – density and type
Size range characteristics of individual layers	Rainfall statistics	Presence of boulders, sunken timbers, etc.
Distribution and volumes – overbrden and ore	Local water sources – plant make-up and potable supplies	
Gold size and size range distribution, distribution of value	* Accessibility – machinery and mining plant	
Depths (average, maximum, minimum) from surface to top of ore zone, and to bedrock, block by block	* Locality factors – People – Culture * Demography – Politics – Environment	
Topography and bedrock features and type		

translation of resource estimates to mining reserves. For each deposit there is a certain minimum width of face that must be removed in one cut. In a dry mining operation the cut must be wide enough to allow trucks to manoeuvre freely for loading and have room to pass one another without hindrance. A sluicing paddock must be large enough for the unimpeded movement of earth-moving equipment for cleaning up and stacking, whilst still allowing for the shifting of water lines and ground race cutting. Bucket ladder dredgers are by nature large and poorly manoeuvrable; turning is difficult, time consuming and costly and considerable space is required for movement except when dredging offshore.

Alluvial deposits are typically sinuous, and depending upon the degree of sinuosity, either of two approaches to mining can be considered. Selective mining of sinuous channels completely within the ore channels reduces dilution but it usually means accepting a lower and less even rate of production. A maximum rate of mining can be achieved by mining in a relatively straight line along the main axis of the orebody, and the more sinuous the orebody the greater will be the dilution and/or the loss of payable material. Small to medium sized channels are usually exploited by hydraulic sluicing or by small-scale dry mining methods. Shallow deposits can be mined selectively, regardless of sinuosity, without significant fall-in from the sides. Fall-in increases with depth and an economic decision may have to be made on the allowable degree of selectivity. Provided that the channels are wide enough to allow for gradual and not abrupt turning, large deposits are usually mined continuously by bucketline dredgers or by scrapers and other large-scale dry mining systems.

Four important factors influence selection of a proposed method of mining – wet or dry:

1. accessibility
2. availability of water
3. stripping and slimes handling
4. dilution.

7.2.1 Accessibility

Alluvial gold depositional systems are made up of various combinations of main trunk channels, divided channels, tributaries, terrace deposits and isolated remnants of earlier channels. As resources, they may all be of potential value at some time in the future. As reserves, they are of immediate value only if accessible within the guidelines of the mining plan. Typical examples are small rich tributaries that are physically inaccessible to a main stream dredger. Such tributaries can be worked economically only by some other method than that selected for the main stream deposits. Apart from being too narrowly confined or too steep and bouldery for large-scale mining operations, small tributaries typically lead to 'dead end' conditions. Dredging through such deposits requires the dredger to re-dredge already worked-out ground in order to re-establish full-scale operations in the main deposit. In the terms set out for the mine plan, such tributary quantities could be classified as resources, but not reserves for the particular venture.

Terrace deposits pose different problems of access. For example, a terrace deposit so located that it can be bulldozed at an affordable cost into the dredge path for treatment with the channel material may be treated as a potential ore reserve. Another deposit of similar size and grade, so located that it would have to be picked up and transported by road for treatment would probably not be considered as a potential ore reserve for this particular mine plan. However, its status could change to that of a potential reserve if it could be exploited under a different mine plan, or become accessible economically following a price rise or some other change in economic circumstances making it profitable to mine.

Residual (lateriticsaprolitic) gold deposits in deeply weathered regoliths comprise a shallow lateritic surface layer 3–4 m deep overlying a barren leached zone that may be up to 60 m in depth before encountering a saprolitic gold-bearing layer at bedrock. A typical mining system will involve mining the lateritic deposits first; stripping and removal of the waste horizons using open-cast extraction methods to gain access for mining the sub-surface, saprolitic deposits. Distribution of the saprolitic gold generally follows the distribution pattern of mineralisation of the primary gold deposit, which it overlies. Though generally of average low tenor, the grades of individual sections of the gold-bearing horizons may vary widely.

Depending upon the geology of the saprolitic deposit at the base of the weathering front, open pit mining may then be continued to greater depths before resorting to underground mining. Wright Engineers of Canada prepared the ten-year mine plan illustrated in Fig. 7.3 for the extraction of the surficial lateritic orebodies at Royal Hill, Suriname.

7.2.2 Water availability

All systems of mining require large volumes of water for processing. The source of water is seepage from external recharge and runoff from natural catchments and ground water. Estimates of supply are based upon the following sources of information:

- maximum daily, monthly and annual rainfalls
- stream fluctuations, run-off depths
- the likely inflow of surface run-off water from basin catchments.

Estimates of maximum flood run-off will determine safe capacities for possible stream diversion and safe heights for protective levees. Probabilities of exceeding average levels from these data are calculated from data obtained at 50 and 100-year flood levels.

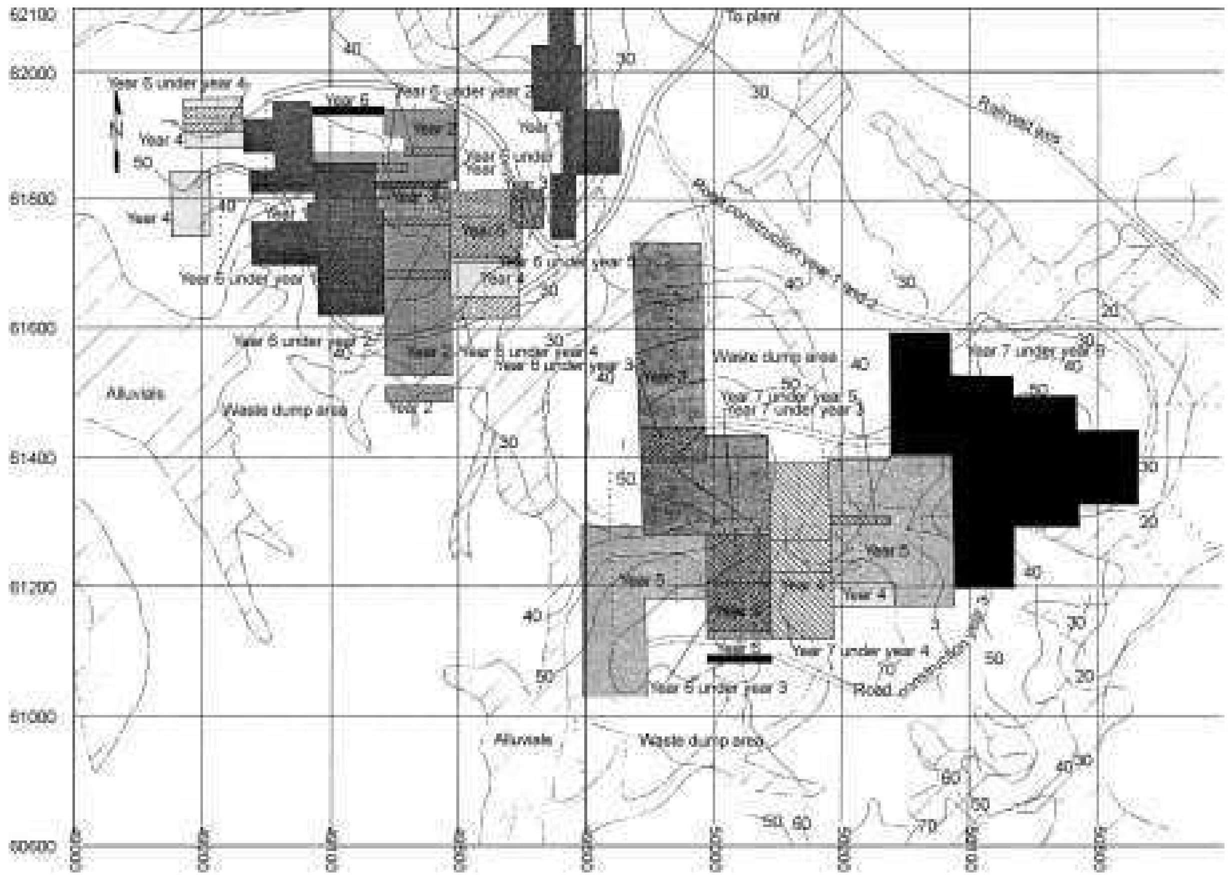
Rainfall

In preparing hydrological estimates, the patterns, amounts and frequency of precipitation as recorded at locations near the project should be similar to those at the mine site. Frequency plots will provide the recorded total annual rainfall and the annual monthly and daily rainfalls for design purposes. Both maximum and minimum daily rainfall figures will be noted, as will records of intense periods of flooding and of drought.

Run-off

The amounts of surface run-off water from catchments in the project area are estimated from hydrological studies, which will also suggest ways of supplementing the supply of process water to the plant. Estimates of flood run-off will determine the required spillway capacities at the tailings and water storage dams. Probabilities of overabundance may be calculated for the following:

- annual maximum daily rainfall
- annual maximum monthly rainfall
- total annual rainfall



7.3 Computerised mine plan for Royal Hill laterite gold deposits, Suriname.

- total annual run-off depths at water and tailings storage areas
- minimum run-offs in one month, two consecutive months and three consecutive months.

Plots of these probabilities are used to estimate the surface run-off volumes available during dry periods having specific return periods. Rainfall intensity-duration-frequency curves can then be derived for estimating flood flows and thus for safe drainage design.

Selection of suitable areas for the catchment and storage of surface run-off water is provisional and subject to some modification when complete topographic data are available. Basically, to avoid excessive costs in providing spillways to handle large flood flows entering into the storage impoundments the catchments should only be as large as needed for a reasonably sized storage facility. Inspection of climatic and hydrological data in temperate to wet tropic conditions will often show that relatively small catchments can supply sufficient run-off water for process water supply.

Seepage and evaporation

The flow of water through sediments of various types is governed by the hydraulic gradient (refer to Chapter 4) and the permeability of the sediments. The hydraulic gradient is a function of the depth below the ground water reservoir and is the height to which water would rise in a vertical tube connected to the exit point. Permeability is a function of the size range and distribution of the particles, their orientation and arrangement. In completely saturated ground the fluid properties affecting flow are viscosity and specific weight.

Due to the heterogeneous nature of sediment and varying degrees of compaction and cementation, permeability and seepage levels may be expected to differ significantly along different planes. Pit design and water controls are both strongly influenced by the number and disposition of aquifers associated with the pit. A high clay content reduces the permeability of strata by reducing the size of openings between the larger grains. Layers of indurated sand, themselves largely impermeable, reduce the movement of water in the vertical plane but may allow movement between layers in the horizontal plane.

Permeability is measured by the quantity of water either pumped from or introduced into a bore to maintain a constant level at selected depths in the casing. With the casing at full depth, the bore is pumped dry and the time is recorded for the water to rise again to those levels. Using perforated casing, seepage rates in the saturated strata are measured in holes bored progressively through to basement.

In laboratory studies, permeability measurements are taken using a constant head permeameter on either undisturbed sample material collected under dry ground conditions, or on disturbed sample material compacted as far as possible back to its original undisturbed state. In the latter case, the degree of success achieved in compacting the sample material back to its original state and ridding it of air bubbles will largely determine the accuracy of the work. The results from such reconstituted samples are usually less reliable than from drill cores because, even if the material is compacted back to its original volume, the orientation and distribution of the mineral grains and hence the permeability will not be exactly as before.

Evaporation and seepage losses may be as high as 20 to 30% of the total usage and depending upon the nature of the ground and the distance water has to be transported in ditches or flumes to the working place. Popov (1971) quotes an average of 0.5 m^3/day/m^2 of wet surface of ditch in sandy ground. In modern undertakings, usage varies widely averaging 20 to 40 m^3 of water per m^3 of ground treated, but ranging from as low as 8 to more than 60 m^3 of water/m^3 of solids for elevating slurries. Possible losses from the tailing pond are calculated in order to assess its role as a source of make-up water. Predictions are made on the general magnitude of seepage losses from the pond and of the range of void ratios to be expected in the tailing deposits. Operational efficiency is strongly dependent upon water reticulation design and water conservation. Prior mechanical stripping and pulverisation of the wash to break down lumps of clay provides a useful means of limiting the amount of water needed for slurrying.

Regardless of the process type, similar amounts of water are required to slurry and process the raw material, plant water losses occur similarly from evaporation and seepage, and similar quantities of water pass out with treatment plant residues. Based upon experience, the overall water requirement will probably not be less than 150 l/m^3 of ground treated and perhaps as much as 1500 l/m^3 for very clayey materials. Where adequate and controllable quantities of water are present the total water usage does not differ greatly between dry and wet mining methods, and the final selection of a mining method will usually be based upon economics.

Water usage also varies with the human equation and the scale of mining. Operators in small-scale ventures are usually less concerned with water usage than larger operators, and seldom do much to improve the efficiency of their methods provided there is sufficient water for treatment and the gold is coarse and easily recovered. Hydraulic elevators are still used in very primitive surroundings despite their very low rate of performance compared with that of gravel pumps. In terms of actual solids lifted, the water usage by hydraulic elevators may be as much as ten times that of a gravel pump used for the same purpose.

7.2.3 Stripping and waste handling

Overburden stripping is usually carried out at a lower unit cost and faster rate than ore extraction, for which the rate of mining is constrained and unit costs are higher. The difficulties and lost time involved in selectively supplying the treatment plant with optimum grade material (see Chapter 8), while cleaning up along the sides and at bedrock are limiting factors. Stripping duty is largely non-selective and machines of the same size and type not similarly encumbered can be worked at maximum economic capacity. Land clearing involves such procedures as bulldozing, tree felling, grubbing, raking and piling. For this service, the variables include the nature of the vegetation (e.g., number, size and types of trees), undergrowth, root systems, etc., bearing capacity of the soil, depth of topsoil, soil type, presence of rocks, water content, topography, rainfall and climate. Table 7.4 is an equipment selection chart for land clearing.

A prime consideration is to return the spoil progressively and permanently to the mined out areas by the shortest practicable route, and to ensure an orderly rehabilitation of the disturbed areas. The responsibility overall is to optimise the value of all overburden disposal operations both in the present and in the future (Macdonald, 1983a). The potential of humus material for vegetal regeneration is too valuable to lose and the first requirement of a stripping programme will be to clear and stack all vegetation and humus-laden soil from the proposed mining area in stockpiles close to the excavations. Temporary safe lodgement must also be provided for other waste material so that it can be returned sequentially to the worked out areas as back filling. The stripped surface soil layer should then be spread across the back-filled material to complete restoration of the mined out ground during the final stages of restoration. All of these operations must be carried out without conflicting with other mine activities (see Chapter 7).

Bucketline stripping

Bucketline dredgers face much greater stripping problems than spud dredgers because of the headline, which holds the dredger against the working face. Headline length is a function of face width. The ratio of headline length to face width is conventionally between 6:1 and 7:1, so that for a dredging width of 300 m the headline will have to be of the order of 1,900 m in length. There is also the problem of headline damper regulations, which will generally not allow any work to be carried out within the sweep of the headline. If two bucketline dredgers are used, one for stripping and one for mining, the stripping dredger must be able to stay clear of the headline at all times. It is common for a headline to fail under stress and the longer the headline, the more easily will it be snapped by stresses imposed by the dredging operation. When this occurs, the broken headline ends flail across the surface of the ground with devastating force and the safest position for the stripping dredger is some position well ahead of the headline anchor point. This is seldom economically practicable because the longer the headline, the larger the ground area to be cleared by the stripping dredger ahead of the mining dredger. Several million cubic metres of material may have to be disposed of and paid for in advance of any income from

Table 7.4 Equipment selection – land clearing

	Uprooting	Cutting at or above ground level	Knocking to the ground	Incorporating into the soil
Light clearing – vegetation up to 5 cm (2″) diameter				
Small areas 4.0 hectares (10 acres)	Bulldozer blade, aces, grub hoes and mattocks	Axes, machetes, brush hooks, grub hoes and mattocks, wheel-mounted circular saws	Bulldozer blade	Mouldboard, ploughs, disc ploughs, disc harrows
Medium areas 40 hectares (100 acres)	Bulldozer blade	Heavy-duty sickle mowers (up to 3.7 cm (1.5″) diameter) tractor-mounted circular saws, suspended rotary mowers	Bulldozer blade, rotary mowers; flail-type rotary cutters; rolling brush cutters	Mouldboard ploughs; disc ploughs, disc harrows
Large areas 400 hectares (1,000 acres)	Bulldozer blade, root rake, grubber, root plough, anchor chain drawn between two crawler tractors; rails		Rolling brush cutter; flail-type cutter; anchor chain drawn between two crawler tractors; rails	Undercutter with disc; mouldboard ploughs; disc ploughs; disc harrows
Intermediate clearing – vegetation 5 to 20 cm (2″ to 8″) diameter				
Small areas 4.0 hectares (10 acres)	Bulldozer blade	Axes, crosscut saws, power chain saws, wheel-mounted circular saws	Bulldozer blade	Heavy-duty disc plough; disc harrow
Medium areas 40 hectares (100 acres)	Bulldozer blade	Power chain saws, tractor-mounted circular saws, single scissor type tree shears	Bulldozer blade, rolling brush cutter (up to 12 cm (5″) diameter), rotary mower (up to 10 cm (4″) diameter)	Heavy-duty disc plough; disc harrow
Large areas 400 hectares (1,000 acres)	Shearing blade, angling (tilted) bulldozer blade, rakes, anchor chain drawn between two crawler tractors, root plough	Shearing blade (angling or V-type)	Bulldozer blade, flail-type rotary-cutter, anchor chain	Bulldozer blade with heavy-duty harrow

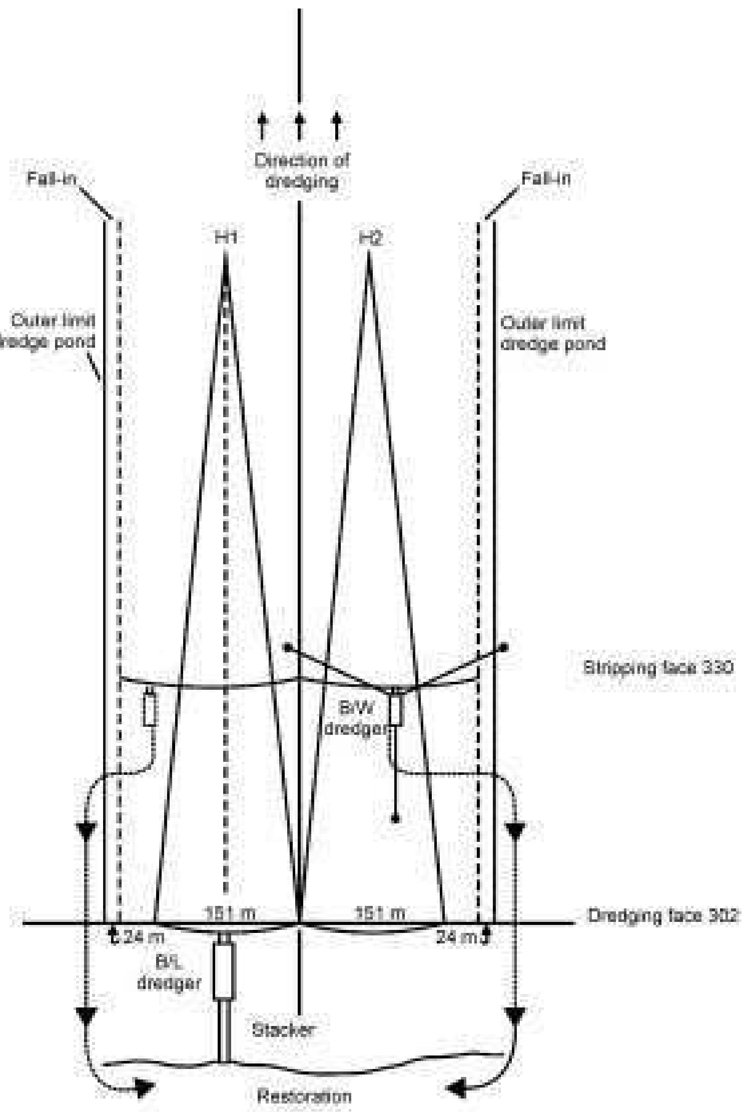
Table 7.4 Equipment selection – land clearing

	Uprooting	Cutting at or above ground level	Knocking to the ground	Incorporating into the soil
Large clearing – vegetation 20 cm (8″) diameter or larger				
Small areas 4.0 hectares (10 acres)	Bulldozer blade	Axes, crosscut saws, power chain saws	Bulldozer blade	–
Medium areas 40 hectares (100 acres)	Shearing blade, angling (tilted), knockdown beam, rakes, tree stumper	Shearing blade (angling or V-type), tree shear (up to 70 cm (26″) softwood; 35 cm (14″) hardwood), shearing blade–power saw combination	Bulldozer blade	–
Large areas 400 hectares (1,000 acres)	Shearing blade, angling (tilted), knockdown beam, rakes, tree stumper, anchor chain with ball drawn between two crawler tractors	Shearing blade (angling or V-type), shearing blade–power saw combination	Anchor chain with ball drawn between two crawler tractors (use dozer blade for trees over 18 cm (7″))	–

Note: The most economical size area for each type of equipment will vary with the relative cost of capital equipment versus labour. It is also affected by whether there are alternative uses for equipment such as using tractors for tillage.

dredging. Very careful planning and execution is needed to co-ordinate the movements of stripping and production dredgers and ensuring that at no time does stripping take place within the sweep of the headline. This practice is inherently risky and can seldom be recommended.

As illustrative of the types of problems that may be encountered, the one headline system described in Fig. 7.4 has the common disadvantage of all such stripping systems, i.e., the very high cost to set up the operation. The ratio of



7.4 Headline dredger stripping.

headline length to total face width, 6.29:1 allows a distance of 175 m to be maintained between the stripping face and the work face. It is a practical minimum distance for moving the stripping dredge and pipeline from side to side when the production dredge moves over to the starboard side. An additional difficulty is that the clearance between the stripping dredger and the headline is small, thus increasing the risk. In both cases the system will function smoothly only if the two dredger operations remain closely synchronised.

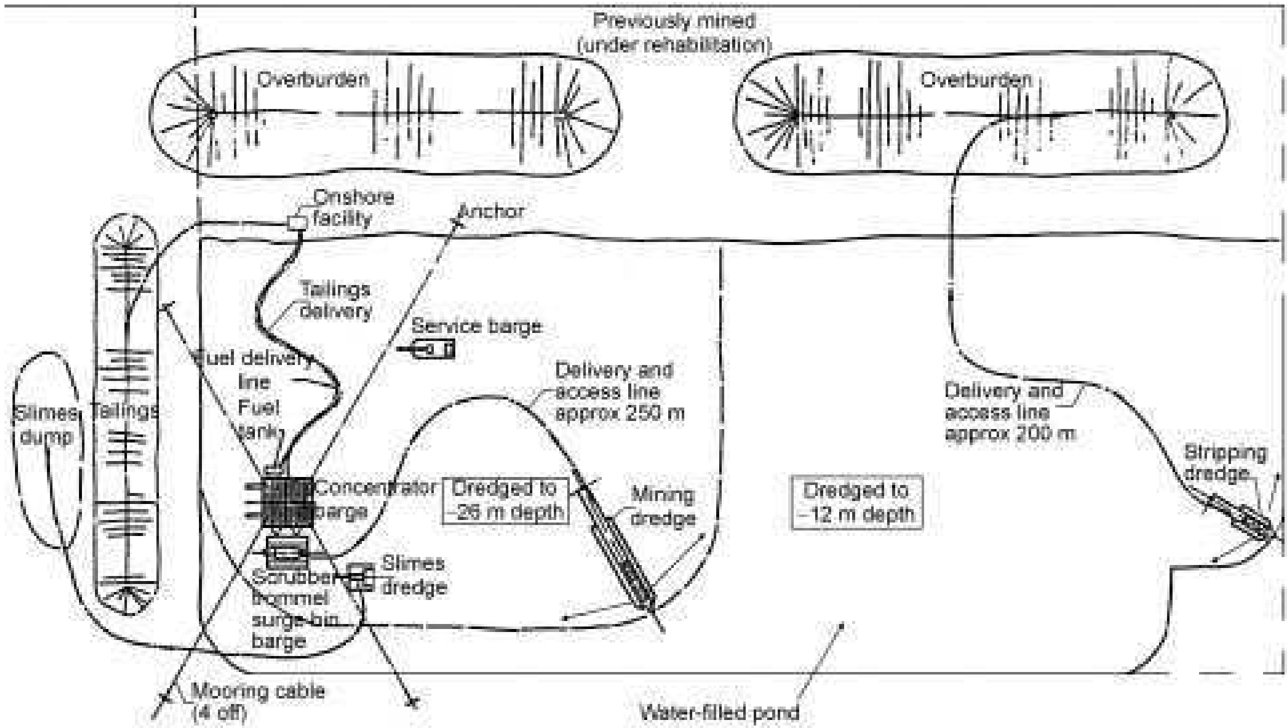
The practice of providing a bucket ladder dredger with an overburden by-pass system and using the dredger for alternate stripping and production is usually a better choice than any combination of separate dredging units. Stripping under these circumstances is usually restricted to clearing and bulldozing the top foot or so of topsoil to disposal sites along the dredge path. This material is returned onto the top of the waste fill in worked out sections of the dredge pond during the final stages of restoration.

Hydraulic stripping

Hydraulic stripping is usually the most cost-effective method of stripping flood plain deposits comprising fine gravels, sands and muds. The method entails the removal of material that can be easily fluidised and pumped through pipelines to the disposal area. Occasional larger gravels and rocks small enough to pass through the pump may be included in the flow but flow rates, power consumption and wear all increase rapidly with increased proportions of coarse sediments.

Although suction cutter dredgers are generally better suited to handling loosely compacted, fine-grained granular materials, bucket-wheel dredgers are usually preferred for digging hard materials, such as compacted clays. Neither method operates successfully in heavy gravels or highly abrasive sediments, for which stripping by earth-moving equipment (back hoes, drag lines, etc.) is usually the preferred method. Figure 7.5 is a schematic representation of a typical stripping/mining operation using two bucket-wheel dredgers, one for stripping the other for mining. The dredgers work independently of one another; the stripping dredger discharges its spoil to stockpiles situated alongside the dredge path so that the overburden can ultimately be returned to the excavation as the top layer. Treatment plant slimes are pumped to a slime disposal area. The mining dredger delivers its slurry to a floating treatment plant, returning the tailing to the bottom of the pond.

Typical problems associated with hydraulic stripping are best explained by actual experience. In the following case history, an example is given of the removal of overburden using a cutterhead suction dredger. All of the stripping conditions for this project (WIDCO Project) were similar to those of typical gold-bearing palaeochannel conditions in flood plain areas.



7.5 Two hydraulic dredgers mining, treating and stripping in the one dredger pond.

Case history

The WIDCO mine location, along the flanks of a drainage system of the Cascade Mountain Range, resembles those of many flood plain placers and like them lies in a swampy setting, covered by scrub and dead trees. In this example, the overburden comprised 4.6 million m^3 of mainly clay, silt and peat with some sand and gravel. The spoil area was an adjacent abandoned pit of 7.05 million m^3 capacity. Six equipment scenarios were considered. The two best options appeared to be a bucket-wheel excavator loading trucks – estimated operating cost $1.41/m^3, and a cutterhead suction dredger – estimated operating cost $1.00/m^3. The cutterhead suction operation was selected on economic grounds.

Site investigations included drilling, which was mainly aimed at locating the ore, and a combined seismic refraction/reflection survey. Two pits were excavated by dragline to provide samples for large-diameter column settling tests and for clay balling tests. In the event, the seismic work could not differentiate between the weathered bedrock and the overlying sediments and was of little value. The drilling results gave a very poor definition of bedrock and an inaccurate quantity estimate. The sediments were identified qualitatively by the pit and drilling samples but not quantitatively in terms of their relative quantities and distribution. The bulking factor was underestimated, as were the obstacles in the path of the dredger.

The dredger used was the spud dredger PARA with 750 mm suction and 700 mm delivery. Power to the cutterhead drive was 600 hp and to the main pumps 2,200 hp. The power was supplied from the mine grid. PARA was apparently well supplied with instruments and was computer controlled and manned by experienced personnel with good technical support. Various consulting engineers and contractors estimated a bulking factor of 1.35. In order to be conservative, the owner adopted a bulking factor of 1.5. However, the actual bulking factor was found to be 1.84, varying in places during dredging between 2.0 and 4.0. This created a problem because the capacity of the disposal site had been designed for a bulking factor of 1.5. Additional water had to be added to the system to make up for the increased volume and raising the rim of the disposal area to increase its capacity resulted in an eight-week delay.

Large obstacles, primarily wood and boulders, posed an additional problem and despite clearing the surface of the ground, tons of cedar (a wood that does not deteriorate with time) were found to be buried in the sediments. Installing a 'knife' in the dredge pump suction to cut the wood into transportable pieces was a first approach to this problem. However, the occasional boulder destroyed the 'knife' and the wood then blocked the pump. These boulders were left behind from man-made fill that had only been partly removed and caused delays of about three weeks.

According to WIDCO Management their exploration programme will be much more comprehensive if they undertake another such exercise. Particular care will then be taken to determine more accurately the horizontal and vertical extent of dredgeable materials and of individual sediment types and distributions. Ground properties will be determined more accurately *in situ* using such procedures as vane, Dutch cone and standard penetration tests. Additional large excavations will be made to identify the presence and location of any likely obstacles to dredging, and a larger number of undisturbed samples will be taken for laboratory testing. Nevertheless, the owners of WIDCO were still pleased with the overall result and the saving, thereby of over US$2 million.

Dry stripping

Dry stripping operations are largely non-selective and machines are usually worked at their fullest and most economic capacities. The choice of machines is usually between wheel scrapers (self- or push-loaded), forward or back acting excavators, drag lines with or without bulldozers for ripping and stockpiling, and trucks of various capacities for loading and transportation. Truck selection is based mainly upon physical parameters such as low rolling resistance, high bearing pressures and good drainage and their effects upon economic and environmental factors. Topsoil is removed and stacked for subsequent replacement and restoration. Roadways are built and storage facilities provided for the solid waste. The overall stripping system should then be co-ordinated with ore production so as to utilise common roadways and avoid bottlenecks.

Medium- to large-scale stripping operations are usually done better by established earth-moving contractors than by mining companies. Contractors are experienced in handling all of the problems of setting up a major operation as well as of operating and maintaining the equipment. They have resources for this work that the mining companies do not have and can offer personal incentives to specific employees for efficiency. Companies cannot offer similar benefits to specialised personnel without providing the same benefits to less-skilled operators in the same undertaking.

It is essential, nevertheless, that pit management is a company responsibility and that any contractual arrangements entered into for mining and haulage be based upon measured volumes of *in situ* material rather than on machine operating time. This means that a company geologist would carry out all measurements related to the depths of stripping and hence be capable of maintaining a close balance between losses of ore and excessive dilution. The contractor's responsibility would be restricted to the physical processes of stripping, haulage, road building and maintenance. Caterpillar and other earth-moving companies progressively update their handbooks with information and tables for determining performance data and operational and ownership costs.

Slimes handling

Slime fractions derived from the weathering of volcanic rocks, especially basaltic and andesitic rock types include a variety of ultra-fine sediment such as clay minerals and silt. The coarse particles settle freely compared with finely divided particles, which settle selectively according to size and, for gold, density. As discussed in Chapters 4 and 8, the settling qualities of individual particles within mixtures of both coarse and finely divided solids are inhibited significantly by the slower settling of the smaller particles. Experimentally, 62.5 μm is the transitional size between Stokesian and Newtonian settling for perfect spheres of quartz ($\rho = 2.65$) settling in still water. Experimentally also, true slime fractions comprise particles smaller than about 38 μm, the size at which the settling of quartz particles is associated with Brownian movement and associated electrical repulsion between colloids.

Such theoretical definitions of settling are clearly too simplistic for gold-dredging operations. In these, the extent to which slime creates problems is determined by the clay content of the material mined, the method of mining and stripping (wet or dry) and the availability of ample supplies of make-up water to replace the water retained in the slimes. Slimes build up rapidly from the action of the digging devices and from onboard treatment facilities, which use hydrocyclones to deslime the primary head feed to concentrators. The slime undersize is either pumped directly to slime dams or discharged back into the pond. The most economic cut-off point for slime separation is a function of the size distribution of economically recoverable gold; predominantly coarse gold ores might be as high as 100 μm or even higher. Slime build-up in a dredge pond is usually minimised by continually pumping away from the bottom of the pond to a settling dam, using a slurry pump located on an independently floating barge. The quantities to be handled may be quite large and the areas selected for disposal must provide adequate space for material that may not settle to more than about 40–50% solids over the life of the mine. Slime disposal areas must also be protected against the effects of 50 and 100-year flooding events as well as flooding from normal run-off.

Estimated space requirements for the rates of disposal of predicted volumes of slime materials are primarily influenced by physical properties such as dilatance and plasticity, which affect the rates of carry over of slime-sized materials. Dilatance relates to wave motions set up during settling. Plasticity affects the rheology of slime and the ease with which the solid/water mixtures deform under stress. The settling characteristics of these materials can be improved by the addition of larger silt and sand-sized sediment; flocculants also enhance settling but nevertheless, losses will occur typically up to 70% of the slime water content. A prime consideration overall is the recovery of surface water as soon as it is clarified sufficiently to meet required effluent water standards. As much as possible of this water is returned to the plant as make up water, any remaining effluent water that may be released, e.g. to streams, must comply with governmental water quality standards. Although the quantities are not large by major civil engineering standards, the percentage recovery of water that can be used as make up water for plant purposes may be of fundamental importance to projects in semi-arid to temperate climatic regions.

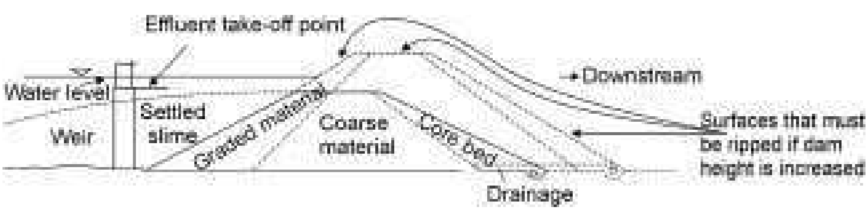
The hydraulic behaviour of slurry containing a variety of ultra-fine sediment such as clay minerals and silt are still being examined. Physical properties such as dilatance and plasticity affect such problems as rates of carry over of finely divided gold particles and the spacial requirements for disposal of predicted volumes of slimes.

Dam wall construction

Figure 7.6 is a sketch showing a method of dam wall construction that has been satisfactory in many existing slime dams. The coarse material core directs the seepage to the bottom of the dam wall, thence into a drainage layer and drainage pipe. If it is found necessary to increase the dam height subsequently, the surfaces marked on the sketch must be carefully ripped in order to avoid slippage between the old and the new layer.

Slurry dam walls should be constructed for controlled seepage, and in such a way as to facilitate any required increase of the wall height if needed. The walls may ultimately undergo considerable hydraulic pressure, and a core of rocks will create the necessary stability using any available boulders and stones for the purpose. The material making up the walls is placed in layers, preferably not greater than 30 cm thick and compacted.

Weir box water levels should be kept at practical minimums for recovering clarified water from the surface of all slime and tailing dams. Should a secondary slime dam be needed to further clarify the discharged water, the wall layer of graded material can be increased in thickness, using the same construction principle. A thin layer of clay could also be advantageous on the inside of the dam



7.6 Construction of slime-slurry dam with downstream raise.

7.2.4 Dilution

The choice between wet and dry mining systems of mining is usually made on the basis of cost and the difficulties of acquiring required volumes of water, and of draining the ground prior to and during the mining operation. Dilution is an important factor influencing working costs. Regardless of applied standards of mining selectivity, auriferous gravels cannot be extracted without including some barren material from the enclosing facies. Inevitably, some of this material passes to the treatment plant thus reducing the feed grade while increasing its volume. Shallow deposits are virtually unaffected by dilution from fall-in and seldom require protective batters along the sides. This changes with depth when fall-in becomes increasingly significant and safety becomes an important and sometimes critical consideration.

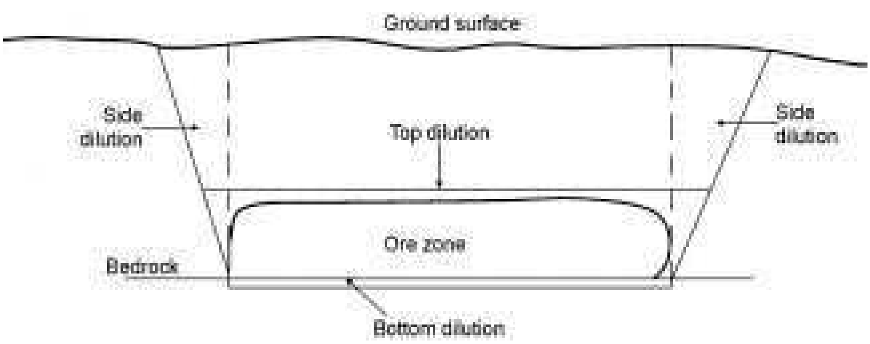
Whilst there is no alternative to the dredging of marine placers and deposits that occur in very wet conditions on land; and no sensible alternative to dry mining in desert areas where water is very scarce, for many other deposits there is a choice. A combination of practical and economic factors helps to resolve any doubtful issues and the selected method will usually be that which offers the most cost-effective method of waste disposal and of slurrying and transporting the mined material to the mill. Occasional exceptions may be made if there is a more ready availability of one type of plant and equipment than another, but only where either will do the job satisfactorily.

Sources of dilution

Dilution is derived from three sources: below the ore zone, above the ore zone, and from both sides of the ore zone (Fig. 7.7).

Dilution from below the ore zone

Cutting into the bedrock for about 300 mm is standard practice except where prevented by physical constraints such as a hard crystalline basement or very



7.7 Sources of dilution.

uneven bedrock. The purpose of undercutting is to recover any gold that may have lodged in cracks or other openings in the rock either during the formation of the placer or during the mining process. Bedrock dilution material is usually brought into mining reserves at nil value because of the difficulty of estimating its grade. Engineers generally regard any additional gold from this source as a bonus, and not as a factor upon which the favourable economics of a project might depend.

Dilution from above the ore zone

It is seldom possible to limit stripping depths to less than about 500 mm without occasionally cutting into the top of the economic gold-bearing horizon. A small quantity of gold may be lost to the waste in this way although it is not usually a significant amount. If there is little or no top overburden the mill feed will comprise all of the material extracted from surface to bedrock including dilution from the bottom and from the sides.

Dilution from the sides

Allowance must be made for material that slumps into the pit from its sides. Safe batter angles for dry and wet mining operations differ markedly. For safe working in dry mining operations the angle of batter may have to be as flat as 45 degrees or even flatter. In a dredge pond a batter of 30 degrees from the vertical is usually taken as the norm for sloughing because of balanced hydrostatic forces. As already noted, sloughing varies directly with the digging depth and its effects are greater for narrow than for wide channels. Regardless of the method of mining the same amount of dilution from the sides is added to the ore in a narrow channel section as in a wider channel section. The resulting differences in volume and grade of the treatment plant feed are thus considerable for narrow orebodies and correspondingly less so for increasing widths.

7.3 Sluicing practice

The sluicing method applies mainly to ground sluicing and hydraulic sluicing small eluvial, colluvial and high-gradient stream placers. Hydraulic sluicing employs high-pressure water jets to break down and treat the wash, either by hand or in association with various combinations of earth-moving equipment such as bulldozers, excavators and traxcavators.

7.3.1 Ground sluicing

Ground sluicing utilises the erosive power of flowing streams of water in open channels to process material broken by hand and is one of the oldest methods of mining. Conventional practice is to construct a dam across the watercourse above the section to be mined and to channel the water along flumes cut into the pay gravels. Material shovelled in from the sides is broken up and slurried manually to release the values. The gold is recovered behind riffles in wooden sluice boxes that are given gradients of 1:12 to 1:10 or steeper. Figures 7.8 and 7.9, respectively, describe typical small-scale ground sluicing operations as carried out by family groups on a point bar in the Lower Waria River, Papua New Guinea and on a hillside in Bolivia.

In larger-scale operations where much fine gold is present, a ground sluice may be sectionalised with the downstream sections acting as scavengers. The slurry flow is stopped and the water is diverted back into the main stream when gold first appears in the final sluice section. The boxes are then cleaned out and the gold is recovered by panning. Periodically, when shovelling distances become excessive, a fresh sluice is dug closer to the foot of the receding bank. The procedures are repeated as necessary until all of the gold-bearing wash has been mined.

Ground textures vary widely and the slopes and dimensions of ditches and other earth channels must be designed accordingly. Channels are usually trapezoidal in section with sides sloping at some angle less than the angle of repose to avoid slumping. This angle may be around 45º for soft ground up to 60º for hard compact ground; wooden flumes may be used when steeper slopes

7.8 Ground sluicing a point bar in Lower Waria River, Papua New Guinea.

7.9 Ground sluicing alluvial deposits in Bolivia.

cannot be avoided. The best hydraulic section has width greater than height; a common W.H. ratio is 2:1. Smirnov (1962) lists critical channel flow velocities for different sized materials in Table 7.5.

A discharge route (tailrace) is common to all sluicing methods. Normally this comprises a channel cut into the soil at a gradient sufficient to carry away all of the waste material. A gradient between 1 and 2 degrees is usually adequate to prevent settling, but it may have to be steeper depending upon the gravel size and the depth of water flowing through the race. If necessary, the race must be cut progressively deeper into the natural ground surface with increasing distance from the face. The

Table 7.5 Critical flow velocities (after Smirnov, 1962)

Average particle diameter (mm)	Velocity (ms^{-1})	Average particle diameter (mm)	Velocity (ms^{-1})
0.10	0.27	15	1.10
0.25	0.31	25	1.20
0.50	0.36	50	1.50
1.00	0.45	75	1.75
2.50	0.65	100	2.00
5.00	0.85	150	2.20
10.00	1.00	200	2.40

slope of the ground is a limiting factor and, at some stage, the spoil may have to be elevated and disposed of by hydraulic elevation or by pumping.

Ground sluicing was practised widely in early Roman times. Army engineers of the day recognised that a natural head of water could be utilised to supply energy at the working face, so streams of water were channelled for great distances in mountain areas to gold mines on which much of the prosperity of Rome depended (see Chapter 1). The method, first described by Pliny the Elder in relation to gold mining in Spain during the first century AD employs a dam which fills slowly and is periodically breached when full. The water is then directed through flumes to the pay gravels (see Chapter 1).

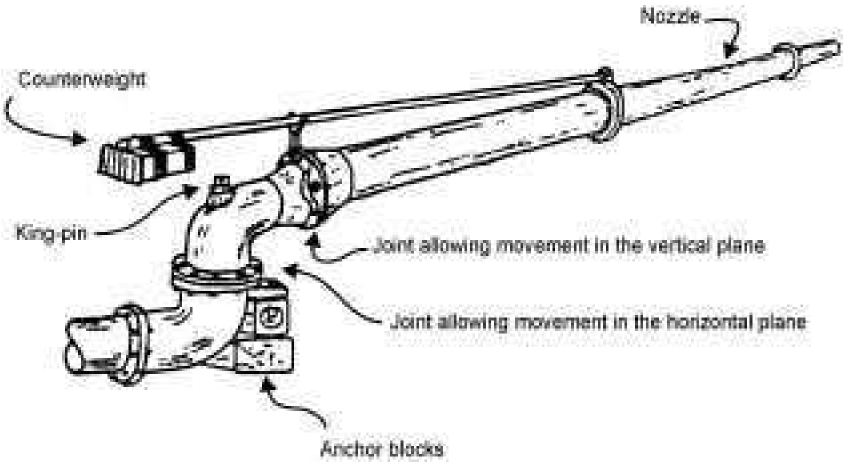
The same method, referred to as 'booming', was used in the early days of some North American goldfields in areas of less intense precipitation, i.e., where run-off and stream flow provides only a small trickle of water. Dams were fitted with lightweight gates (counter-balanced) to which a long lever was attached. A large container was hung from the end of the lever. When the dam filled, water overflowed and filled the container. This activated the lever allowing the water to rush out and scour the channel bottom. In its lowest position the bucket tilted, spilling out the water, thus allowing the gate to reposition itself under its own weight. The gold was trapped behind riffles or stones laid along the floor of the sluice while the light materials were washed away.

Early miners in the Lakekamu Alluvial Gold Field, Papua New Guinea used a different form of ground sluicing to mine surface exposures of gold-bearing fanglomerates. The ground surface in this area was traversed by herringbone patterns of channels radiating out from single channels located in the lowest parts of the terrace. These channels acted as tributaries to collect large volumes of water running off from higher ground during heavy monsoon rain periods. The flow from these channels was directed into a central channel, which cut back into the sluicing face dislodging material for treatment in ground sluices.

7.3.2 Hydraulic sluicing

The first recorded use of pipes to convey high-pressure water to the face was in the USSR in 1830 (Popov, 1971). The method then emerged in the Californian goldfields in 1840 (Wolff, 1976) and soon spread to alluvial goldfields in other parts of the world. Monitors, otherwise called hydraulic giants (Fig. 7.10), were developed to enable high-pressure jets of water to be directed against the face as required. The resulting slurry was washed into a pump sump through races cut into the bedrock. Hydraulic elevators (Fig. 7.11) used to elevate the slurry to a sluice box were very inefficient, and the subsequent introduction of centrifugal gravel pumps extended the availability of gravel pump mining, to any area having an adequate supply of water, regardless of head.

Suitable ground conditions for hydraulic sluicing are provided by small gravelly wash that is easily slurried and soft weathered bedrock in which races



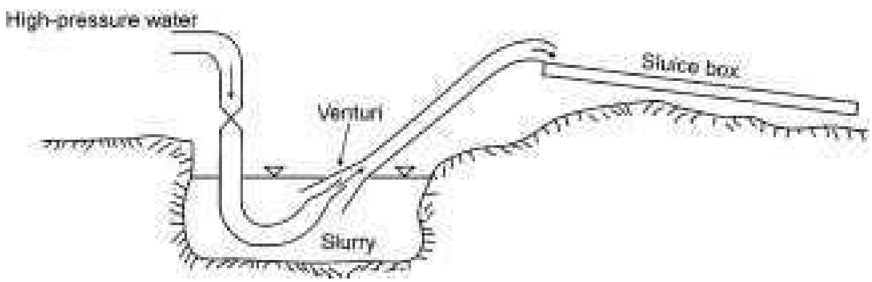
7.10 Sketch of hydraulic giant.

can be cut to direct the slurry from the face to a head feed pump sump. A natural slope of about five degrees from the horizontal is an optimal gradient but slopes may be 30–40% flatter or steeper without seriously affecting the operation. At any such gradients, most of the slurried material gravitates from the face to the sump without excessive surging or settling out of the finer gravels.

Monitors

The monitor unit, or hydraulic giant as it is sometimes called, is a nozzle for directing a stream of high-pressure water against the working face. Some larger units incorporate deflectors to give a better control of jet direction. Various degrees of sophistication are applied to balancing the re-active thrusts developed by the jet, the simplest being counterweights attached to the arm.

Monitors are used to undermine a pit face and so encourage slumping. Material broken by the monitor jet is slurried by the jet and washed down through races (channels) into a gravel pump sump in the pit floor. Riffle boxes



7.11 Hydraulic elevator.

may be placed in the ground races to effect an initial recovery of coarse gold. The larger stones are forked out and stacked along the sides and back of the excavation. A gravel pump elevates the remaining slurry to a gold-saving plant, which may either be a riffled sluice box or a more sophisticated jigging plant. Nozzle diameters range from around 25 mm up to 125 mm and provide jet velocities of the order of 20–50 m/sec. Pressure heads are given by the equation:

$$V = C(2gh)^{0.5} \tag{7.1}$$

In consistent units: V is the velocity at the nozzle outlet, h is the head of water at the nozzle, g is the acceleration due to gravity, and C is the nozzle coefficient. Values for C can be obtained from the supplier; $C = 0.95$ is a general average.

As an example, to find the required head for a jet velocity of 40 m/sec. From eqn 7.1:

$$h = V^2/C^2 \times 2G = 1600/0.95^2 \times 2 \times 9.81 = 90.4\,\mathrm{m}$$

Sufficient additional head is added to compensate for line friction and other hydraulic losses. The total required head might be of the order of 100 m or more, depending upon the length and diameter of the pipe. It is generally wise to add 20% to the calculated value to allow a safe degree of flexibility to deal with puggy clays and partly cemented gravels that might require additional energy for dispersal. The work done by a jet of water varies according to the distance of the nozzle outlet from the point of impact. The jet loses power from the moment it leaves the nozzle. Energy is expended progressively in overcoming air friction and gravity, and the further the jet has to travel the less energy is available to do useful work. Approximate performance figures for jets of water at varying distances from the working face are given in Table 7.6. Distance from the face is a critical factor for operator safety. Because of slumping, the monitor should not be located less than bank height from the face in average ground conditions. This distance may have to be increased if there is any danger of mudflow or of dislodged boulders rolling down into the workings.

An inherent disadvantage of monitoring is the unconfined nature of the slurrying action. The method makes poor use of the available energy because the jet momentum is utilised for only part of the time in breaking down the face. There are practical difficulties in being able to direct the jet continually against

Table 7.6 Monitor performance and water consumption per unit of material washed (after Shevyakov, 1970)

Distance between nozzle and working face (m)	5	10	15	20	25
m^3/hour washed ground	100	93	74	48	18
m^3 water/m^3 ground washed	8	8.6	10.8	16.7	44.5

the unbroken face and excessive amounts of water brought into the pit may have to be elevated out and away from it, thus increasing pumping costs. Large amounts of energy are also wasted in trying to disperse lumps of clayey wash which are moved backwards and forwards by the jet and in having to wash the resulting slurry down to a gravel pump sump for elevation to the plant. The inefficient use of hydraulic power is not critical when an adequate natural head of water is available, however, useful energy usage is often only a fraction of that generated in mechanical operations, some of which face crippling costs for power.

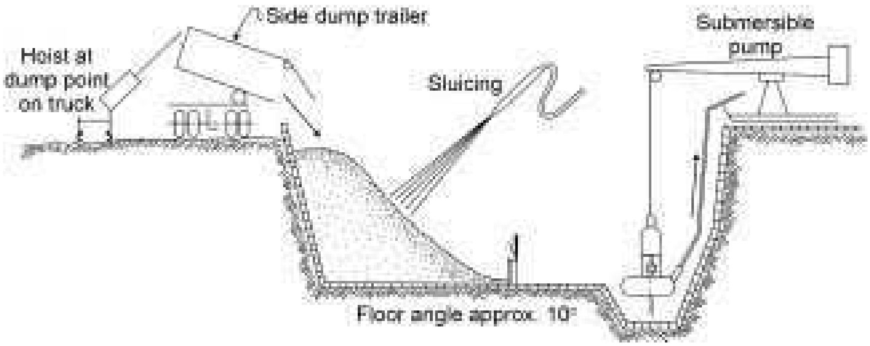
Gravel pumps

Gravel pumps were originally single stage, open impeller, centrifugal types, belt-driven from a diesel engine or slip ring electric motor to give a range of working speed. Pump layouts were cumbersome, difficult to prime and were usually operated close to the limit of their suction lifts. Vertical, submersible types that could be raised or lowered in the sump casing using a simple tripod and pulley arrangement, or block and tackle replaced this pump type. Raising or lowering the pump in the sump regulated flow from the pump to the treatment plant.

Gravel pumps with enhanced priming facilities now operate from rafts floated in the sumps. This arrangement has eliminated most of the pump suction problems attendant upon high suction lifts but new maintenance problems have developed associated with submergence of the electric motor. The main problems are due to electrical breakdowns. Because of the low demand for pumps of this type, there has been little research in trying to develop better insulating qualities for the motors and shutdowns for maintenance add significantly to running costs.

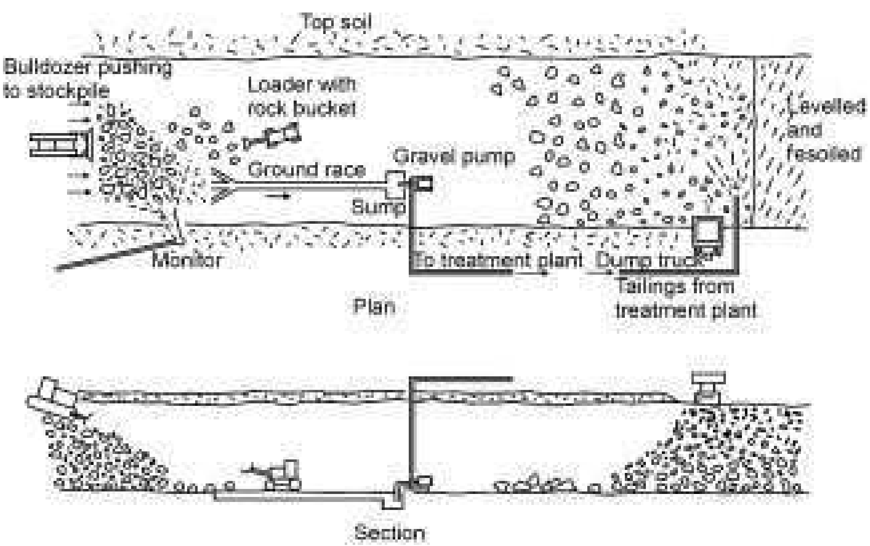
Each sluicing plan is different, but a common denominator is the need to synchronise all of the pit activities. Combinations of wet and dry mining methods of mining often give the best results. Figure 7.12 shows dry feed materials being dumped into a central sluicing paddock for sluicing in the New England District of NSW where a series of small deposits are mined by dry methods over a comparatively wide area. Many difficult materials respond better to jetting if they are stockpiled and fragmented initially by mechanical means (e.g. by bulldozing). Large earth-moving equipment may also be essential within the pit for the systematic exploitation of ground containing numerous large stones and boulders. Monitoring of such activities calls for close co-operation between the head box operator at the treatment plant and the monitor operators in the pit. Since the head box operator alone has a full overview of the working area, there is a clear requirement for him to direct all of the pit activities including the earth-moving functions

A typical sluicing operation (as illustrated in Fig. 7.13 for Yakatabari Creek in Papua New Guinea) commences with the development of a working paddock



7.12 Dry-wet sluicing arrangement New England District, NSW, Australia.

using mechanical earth-moving equipment to move the overburden and open up a face for monitoring. The length of the paddock will probably be about 75 m from face to tailings disposal at the back of the excavation. Slurry monitored from the working face is directed downslope to gravel pump sumps through races cut into the floor of the paddock as shown in the illustration. The width of the cut is held to a practical minimum, according to the variable stability of the sides and face of the channel. In unstable ground there is always the possibility of block flow and monitors will be positioned for sluicing in two or more parallel strips across the full width of the deposit. A bulldozer is used to break down the face of wash ahead of the monitors; large stones and small boulders are stacked along the sides using traxcavators. The ongoing sequence will involve:



7.13 Typical wet sluicing operation as at Yakatabari Creek, Porgera, Papua New Guinea.

- monitoring the broken ground and washing the slurry into the sluices
- bulldozing the washed gravels to the sides of the excavation and stacking the small to medium sized boulders and large stones along the back and sides of the pit using a traxcavator for the purpose
- advancing the face in one strip of the paddock for a distance of 30 to 50 m while slurrying and washing the broken gravels in the adjacent strip into the sluices
- maintaining a sluice along the side of the paddock to channel excess water away
- pumping the sluice box tailings to the top of piled up stone to refill the channel at the back of the excavation
- levelling and resoiling to form a finished surface for replanting.

Increasingly high maintenance and energy costs tend to restrict the gravel pump method to deposits having a natural head of water available at the face. Sluicing is mainly disadvantaged by its very large water requirement, particularly in ground that does not slurry easily. This problem can be alleviated to some degree if the overburden can he removed by stripping. Mechanical stripping of overburden and the handling of heavy stones and boulders in the sluicing paddock can usually be done at less cost than by hydraulic methods. Very tough clays and partly cemented gravels respond better to jetting if broken initially by some mechanical means.

7.4 Bucketline dredging

A bucketline dredger is a complete mining/treatment unit comprising pontoon, digging mechanism, treatment plant and supporting structures. Great strength is needed and bucketline dredgers are massive structures. Weights per $m^3 h^{-1}$ capacity range from about 1.5 tonnes/$m^3 h^{-1}$ for small modern dredgers to about 8 tonnes/$m^3 h^{-1}$ for large deep-digging dredgers. Thus a 130 $m^3 h^{-1}$ dredger will weigh around 200 tonnes whereas the weight of a 1000 $m^3 h^{-1}$ dredger might be as much as 7,800 tonnes. Variations in weight per $m^3 h^{-1}$ capacity are due to differences in the service for which they are designed. Digging capacities are based upon bucket size and speed, availability, and efficiency (Table 7.7).

Dredgers may be powered electrically from an onshore power plant or grid or have its own on-board power plant. Offshore dredgers have their own systems of propulsion and must conform to maritime safety regulations. Onshore dredgers are not self-propelled. Movement is effected through the use of spuds or by manipulating lines anchored to the bank or bottom of the pond. Dredgers are constructed in widely different sizes and capacities according to the nature of the ground to be mined.

Table 7.7 Dredger digging capabilities (after Goh, 1987)

Mode	Stripping	Treating	Treating	Treating	Stripping/Treating		Stripping/Treating		Stripping/Treating	
Technology level	Existing	Existing	Existing	Existing	Existing		Existing		New	
Bucket size FT[3 d]	12	22	24	30	22		30		39	
Bucket size BPM [a]	26	26	26	26	30	26	30	26	30	26
Minutes/hr	60	60	60	60	60	60	60	60	60	60
Hrs operation/month [b]	600	600	600	600	148	452	148	152	148	452
Efficiency [c]	0.85	0.75	0.75	0.75	0.85	0.75	0.85	0.75	0.85	0.75
Capacity YD^3/month	353,600	572,000	624,000	780,000	184,507	430,907	251,600	587,600	327,080	763,830
					615,414		839,200		1,090,960	
Capacity YD^3/year	4.24 m	6.86 m	7.49 m	9.36 m	2.21 m	5.17 m	3.02 m	7.05 m	3.92 m	9.17 m
					7.38		10.07		13.09 m	
Capacity m^3/month	270,336	437,309	477,064	596,330	141,060	329,439	192,355	449,235	250,061	584,006
					470,499		641.590		834,067	
Capacity m^3/year	3.24 m	5.25 m	5.72 m	7.16 m	1.69 m	3.95 m	2.31 m	5.39 m	3.00 m	7.01 m
					5.64 m		7.70 m		10.01 m	

1 m = 3.281 ft; 1 m^3 = 1.308 yd^3

[a] Maximum bucket speeds for tolerable wear conditions are 30 bpm stripping and 26 bpm treating. Wear is actually proportional to line speed, i.e., bpm × bucket pitch. Pitch increase from 24 ft^3 buckets is of the order of 10 to 20%.

[b] Operating time efficiency is taken at 83.3% at 600 hrs/720 hrs month. Stripping/treating dredges have the stripping time:treating time ratio based on stripping to 10.5 m level and treating to 35 m level.

[c] Operating efficiency is taken as 85% for stripping and 75% for treating to determine digging capacity. The higher stripping efficiency is due to undercutting to overfill buckets during stripping and the stripping operation is free of treatment plant problems.

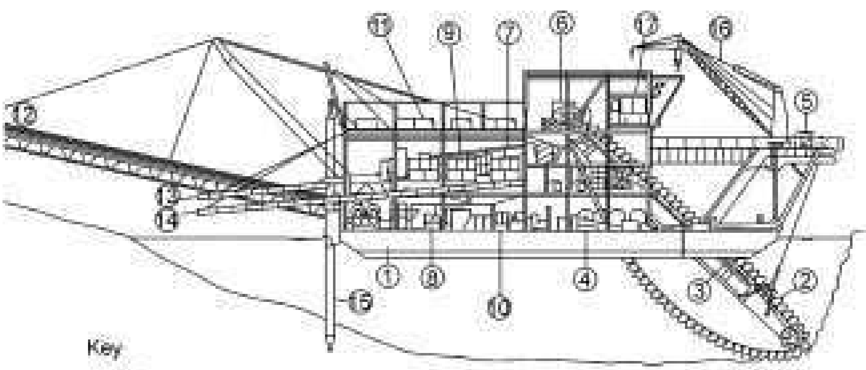
[d] Bucket size capacity is based on volumetric fill at average digging depth (17.5 m for this study). Use of bucket anti-spill flaps can increase volumetric fill by 10–20% especially at low ladder angles during treating.

7.4.1 Design considerations

Factors affecting the design of a bucket dredger are mainly deposit volume, width, depth and the range of depths to be dredged, sediment type and bedrock type. Parameters most affected are hull and ladder dimensions, bucket size and speed and the system of mooring. A typical bucketline dredger is described in Fig. 7.14. Key elements in the dredging system are numbered from 1 to 17.

Hull dimensions

The dredger hull is a rectangular box-like structure, with chamfered sides to facilitate manoeuvring and compartmented for strength and safety. It is slotted centrally for one-third to one-half of its length to accommodate the bucketline and ladder and is equipped with forward and aft gantries and other structures, which support the working units and hold them in place. The main requirements of the pontoon are water-tightness, strength, stability and rigidity. Rigidity is essential because of the wide range and interaction of functions (e.g. digging, screening and pumping, jigging, etc.) involved, all of which impose different types of stress on the hull. The overall structure is designed for the combination of stresses. Hull design is greatly influenced by the digging depth:



Key
1 Hull
2 Bucket system
3 Ladder system
4 Mooring winches
5 Ladder-hoist-winch
6 Main drive
7 Trommel screen system
8 System feed pump
9 Gold recovery system
10 Feed pump
11 Gold recovery system
12 Discharge
13 Course tailings
14 Fine tailings
15 Spud system
16 Main crane
17 Control cabin

7.14 Typical spud bucket dredger.

- In shallow ground, the hull needs only be small to support the weight of digging equipment; the digging ladder must be short to avoid too flat an angle and hence excessive spill; the hull is small and narrow, with tapering bow to dig the corners; capacity is limited because the buckets must be small so as not to exceed the weight limitations (O'Neill, 1976).
- In deep ground, the dredger hull must be large enough to support the combined weight of a longer and more robust ladder and larger bucket band, stacker, drive assembly, etc.; since the cost is also much greater, production rates must be proportionally higher for economic reasons.

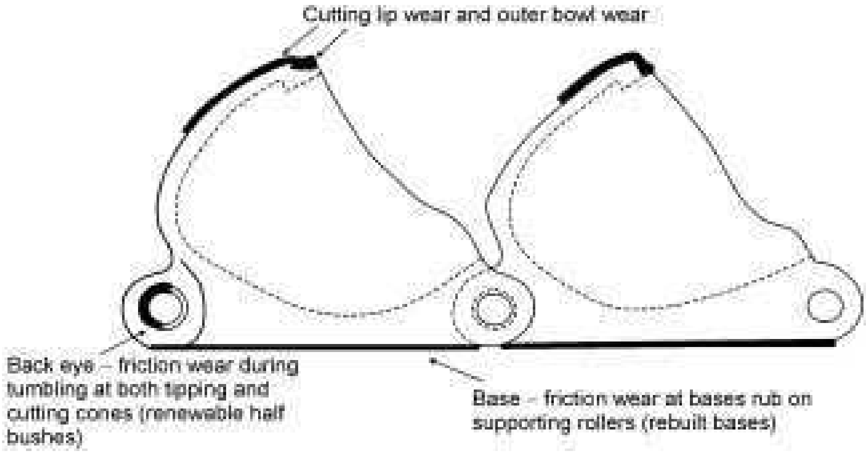
Bucketline

This mechanism comprises an endless chain of steel buckets supported by a box frame of steel girders called a 'ladder'. The ladder is pivoted from a central structure, which also supports the drive. It is provided with evenly spaced rollers on its upper face to support and facilitate the upward movement of loaded buckets to their discharge point. The bucketline hangs free on its return to the face. Buckets are cast from special, high-grade manganese steel. Cutting lips are specially designed to resist both impact and abrasion. Pins, holding the buckets, are machined from high-grade nickel-chrome and other alloy steels and must be very tough and strong. Deep digging bucketlines are long and impose very high stresses due to the catenary pull on the underside of the ladder. The ladder is raised and lowered hydraulically, or by winching using steel cables passing over sheaves on the forward gantry. Caterpillar idlers are installed in most deep-digging dredgers to lend some support to the catenary sag of the buckets and reduce drag. However, Perry idlers, which provide additional support are usually preferred for offshore dredging.

Buckets

Buckets are designed to withstand high impact stresses and wear. The metal thickness ranges generally from 6 mm to 10 mm with as much as 30 mm thickness for the lips, depending upon the size of the bucket. The buckets are either attached to one another to form a continuous chain, or are separated by idler links. The continuous chain type is the more adaptable of the two and usually cuts more effectively into weathered bedrock. The main specifications for a 27 ft^3 (760 litre) bucket, according to Malaysian standards, are as follows:

- material – austenitic manganese steel
- bucket features – lugs for lifting during maintenance
- spill ribs – to direct the discharge of bucket contents
- linkage features – front and back eyes for pin location free of casting defects



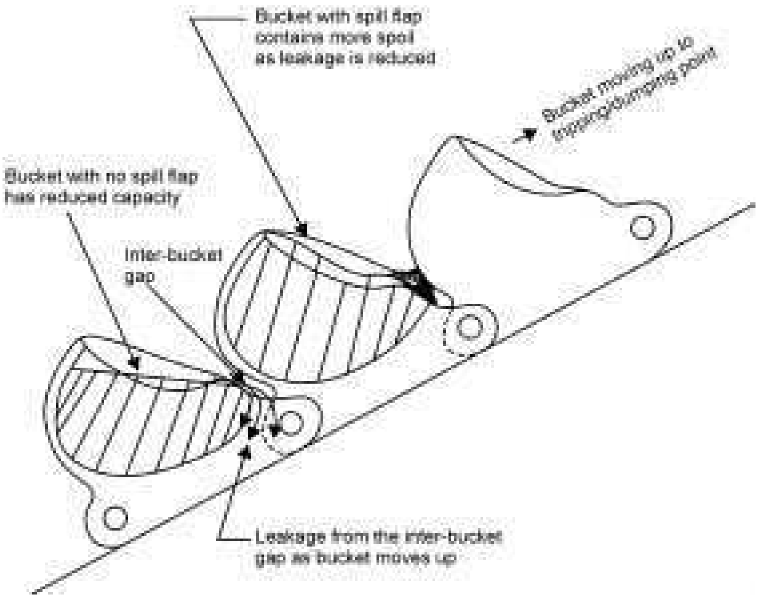
7.15 Bucket wear pattern.

- weight – 2.79 tonnes/bucket
- manufacture – earth bucket a single casting quenched in water at 100 ºC; no part of the bucket should exceed 150 mm in thickness for proper quenching
- service – 24 h continuous at 600 h/month with 120 h/month maintenance downtime.

Buckets with lives of up to ten years in average ground conditions are continually rebuilt with weld to compensate for wear. Current practice favours casting the lip integrally with the hood and base; wear is compensated for by welding inserts into the lip portion. The wear pattern is described diagrammatically in Fig. 7.15.

Bucket size
Buckets are sized in accordance with required dredger output and other practical considerations. In bouldery ground or ground containing cemented wash, rock bars, etc., compensation for high impact stresses is given by using heavy structural reinforcing and oversize drive units or by a reduction in the bucket size for a given size of hull and weight. Because of limitations based upon hull sizes, small buckets are used to dredge shallow deposits. For deep digging dredgers in the USA, the largest buckets used are 510 litre buckets for 40 m depths. In Malaysia, 680 litre buckets are common; 600 litre buckets are used in the USSR for 50 m deep digging at the Urkutsk No. 2 plant and in Colombia the usual bucket size is 400 litres. The Colombia buckets are cast in three sections; base, hood and lip.

The practical upper limit for bucket capacity appears to be around 850 litres. Beyond this size, serious casting problems arise. Weight is one important limiting factor. Bucket sections must be thickened disproportionately to cope



7.16 Use of improved spill flap to reduce spillage and increase capacity (Goh, 1987).

with the very high digging forces involved and the ratio of bucket weight:weight of contents becomes economically less favourable the larger the work load.

Bucket spill flaps

Bucket spill flaps have been introduced to reduce losses from the buckets due to spillage while moving up the ladder from the work face to the tipping point. The first spill flaps were experimental. They were made very simply from rubber tyres and flat rubber sheeting and while they demonstrated the practicability of the method, they tore easily and impeded pouring at the top tumbler. Further studies led to the design of contoured rubber spill flaps that are installed as illustrated in Fig. 7.16. Flaps are attached to the bucket using studs and bolted clamps and have much longer lives than the earlier models. Later improvements were made to the method of attachment, by incorporating a contoured edge in the bucket mouth to seat the flaps.

According to Goh (1987) spill flaps increase the normal bucket capacity by up to 10% depending upon the ladder angle. A 510-litre bucket may be automatically upgraded to around 530–560 litres if fitted with flaps.

Bucket speed

Bucket speed (i.e. the number of buckets/minute) is constrained by ground conditions and in easy digging conditions, bucket speeds may be higher and production greater than in difficult ground without imposing proportionally higher stresses. The modern trend towards achieving increased capacity by increasing bucket speed rather than size is due to the availability of better steels and improved casting techniques. A higher average bucket speed can usually be used in stripping service because of the relative ease of digging overburden. The more difficult digging conditions imposed by mining gravels and cleaning up at bedrock result in slower average digging speeds and fluctuating feed treatment rates.

Bucket pins

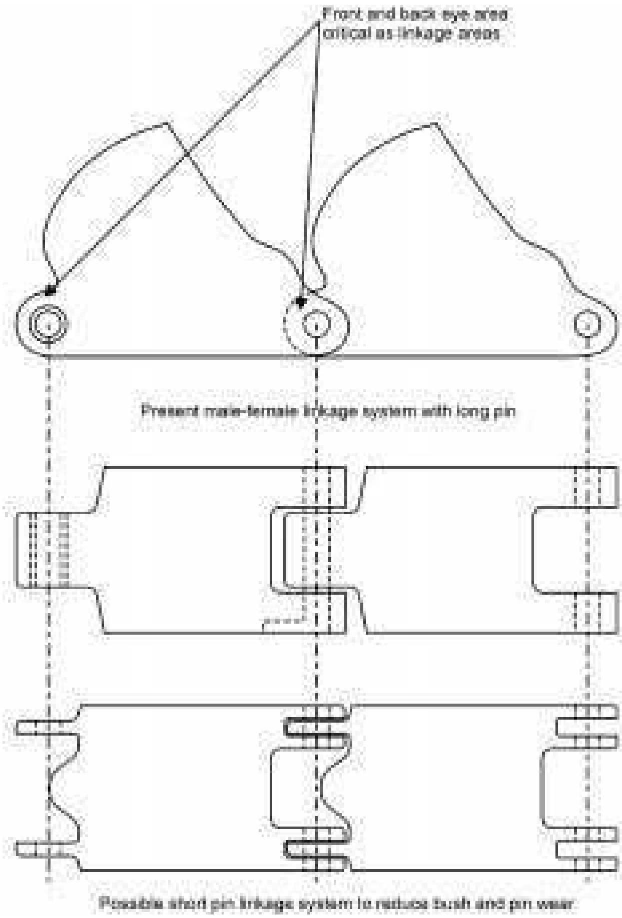
A bucket chain is only as strong as its weakest link and pins holding the links together must be very tough and strong to withstand the great stresses imposed upon them. In order to avoid faulty installation it is crucial to carefully inspect the linkage areas of the buckets during the manufacturing process. Inspection is carried out in the factory using x-rays and ultrasound techniques to test for cracks or flaws in the metal. Bucket pins must also be fitted accurately and be well seated to avoid excessive wear. Pin breakage allows the whole of a bucket line to collapse into the pond thus holding up production until it is recovered. The salvaging process may take several days or even weeks to accomplish.

The usual linkage system employs a male/female joint with a long pin as illustrated in Fig. 7.17. The linkage provides for an inter-bucket gap, which allows the buckets to flex about their pin connections at the top and bottom tumblers. This gap allows spillage to occur between the cutting face and the tipping point at the top tumbler. Spillage usually averages 5–10% at steep digging angles. It increases with lower angles of ladder inclination, as at shallow digging depths, but may be minimised at all angles through the use of spill flaps.

Digging capacity

Dredgers are typically designed for specific sets of conditions and are constrained by economic factors to limit the amount of over-design that would allow increased digging capacity by increasing the size or speed of the buckets. The original design will usually allow sufficient flexibility for the safe use of spill flaps to decrease spillage. Beyond this, there are practical limitations to the amount of upgrading that can be done safely to the digging function. The allowable bucket size is limited by metallurgical conditions, and by the weight of the dredger. The weight and strength of the bucket band limits the allowable increase in the speed of the buckets.

Wear and tear at the digging end is directly proportional to the second power of the bucket speed and the bucket weight (Goh, 1987). Experience suggests that for Malaysian conditions an upper limit of 400–420 ft^3/bpm (11.3–11.9 m^3/bpm) for 15–16 ft^3 (425–453 litre) buckets in average digging conditions, i.e., 26 bpm for 100% bucket fill. Pearse (1985) notes that a typical 300-litre dredger (IHC Holland) has a bucket speed of 30 bpm at an average dredging depth of 11 m



7.17 Linkage system to form chain of buckets.

below pond level. According to Shevyakov (1970), 250-litre buckets are used at the Urkutsk No. 1 plant in the USSR with speeds ranging from 20–34 bpm.

The maximum allowable speed of a particular bucket varies as a function of wear and tear in different ground conditions and the key to achieving maximum output is by controlling the speed. The older gearboxes were generally supplied with one, two or three speeds to select from. The modern trend is for gearboxes that are infinitely variable, throughout a designated speed range. In California, the renovated No. 21 Yuba dredger was refitted with a variable 0–30 bpm bucket speed and 14 ft^3 (396 litre) buckets to achieve greater flexibility and increased throughput. Of the new dredgers, two recent designs are noteworthy: the Grey River dredger, New Zealand and the San Antonio de Poto dredger, Peru. Both dredgers were designed for glacial outwash conditions but very different approaches were taken to the problems involved.

Case histories

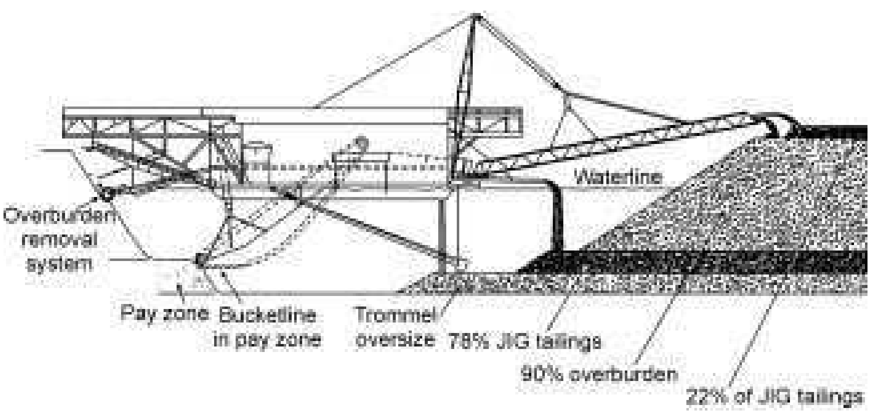
The Grey River dredger (New Zealand)

The world's first dual excavator prototype, the Grey River dredger, New Zealand (Fig. 7.18) incorporated a dual wheel suction cutter head for overburden stripping and a bucket line for ore extraction. The two functions were conducted simultaneously with stripping depths averaging about 16 m above a 10 m thick ore zone. This type of system involved the measurement and control of stresses imposed by two completely different digging systems operating simultaneously from the one hull. The Grey River dredger was a pioneering effort, which put into effect a concept that had previously been considered but thought too difficult to implement. Decommissioning of the dredger in this case left many problems still to be ironed out but some of the lessons learned may be of great future value to the industry.

The main objective of the exercise was to achieve a large throughput without placing too much emphasis on selectivity. The average digging rate was not achieved because some plant units were under-designed as the result of too little experimentation. The dredger availability was badly affected by a poorly designed slurry inlet system that led to frequent breakdowns and blockage. Wear was much higher than expected, particularly in the pumping system. The production record (Table 7.8) highlights some of the deficiencies of the system up until its closure at the end of 1989.

San Antonio de Poto dredger (Peru)

This dredger was still on the design board at the time of writing. Although more conventional in concept than the Grey River dredger it is intended to incorporate 'state of the art' improvements in both dredging and gold recovery technology.



7.18 Grey River dredger (New Zealand) – dual bucketline/bucket wheel facility.

Table 7.8 Grey River dredger Gold Mining Ltd production record

Period to	Op. time O/B hr	Op. time B/L hr	Avail. mech. av. O/B %	Avail. mech. av. B/L %	Avail. o'all av. O/B %	Avail. o'all av. B/L %	Adv. m	Vol. O/B bcm	Vol. B/L bcm	Prod. rate av. O/B bcm/hr	Prod. rate av. B/L bcm/hr	Vol. total bcm	Vol. prog. bcm	Au rec. raw g	Au rec. raw oz.	Au rec. raw prog. kg	Grade rec. mg/m^3	Grade rec. prog. av. mg/m^3	Grade est. dil. mg/m^3	Au est. raw prog. kg	R/E $	R/E prog. $
31.1.89							110	59,000				59,000	59,000	58.2	1.87	0.058	1	1	15	0.885	7	7
2.3.89							66	59,000				59,000	118,000	87.8	2.82	0.146	1	1	15	1.770	10	8
3.4.89							67	83,372	16,898			100,270	218,270	1,686.7	54.23	1.833	17	8	80	9.792	21	19
30.4.89							45	38,726	31,464			70,190	288,460	6,236.4	200.50	8.069	89	28	46	15.828	103	51
2.6.89	271	203			35	26	59	101,840	57,670	376	284	159,519	447,970	11,744.2	377.58	19.813	74	44	83	29.067	89	68
30.6.89	178	221			26	33	48	77,380	60,990	435	276	138,370	586,340	8,784.5	282.43	28.598	63	49	93	41.936	68	68
28.7.89	249	322			38	50	54	84,347	60,277	339	187	144,624	730,964	9,842.0	316.43	38.440	68	53	94	55.530	72	69
1.9.89	314	414	46	59	42	55	92	190,975	115,191	608	278	306,166	1,037,130	18,131.2	582.93	56.571	59	55	79	79.717	75	71
29.9.89	388	452	63	66	54	63	60	137,949	94,560	356	209	232,509	1,269,639	23,165.9	744.80	79.737	100	63	89	100.411	112	79
31.10.89	398	457	77	69	52	60	68	152,984	130,007	384	284	282,991	1,552,630	22,680.4	729.19	102.417	80	66	126	136.068	64	75
30.11.89	404	456	76	70	55	63	63	196,048	146,771	485	322	342,819	1,895,449	26,155.5	840.92	128.573	76	68	113	174.806	68	74

Key to columns
4,5 Avail. mech. = hrs worked/(hrs worked + unscheduled maint.)
6,7 Avail. o'all = hrs worked/total hrs
11,12 Prod. rate av. = vol./op. time (9/2, 10/3)
17 Au rec. raw prog. = sum Au rec. raw (sum 15)
18 Grade rec. = Au rec. raw/vol. total (15/13)
19 Grade rec. prog. av. = Au rec. raw prog./vol. prog. (17/14)
20 Grade est. dil. = borehole block grade estimate, diluted for side batter
21 Au est. raw prog. = sum grade est. dil. $*$ vol. total (sum 20$*$13)
22 R/E = grade rec./grade est. dil. (18/20)
23 R/E prog. = Au rec/ raw prog./Au est. raw prog. (17/21)

The designers have opted for a headline dredger in ground previously considered suitable only for spud dredging. The San Antonio de Poto (Anania) alluvial goldfield, referred to in previous chapters, is a glacial outwash deposit. One section of the field was dredged earlier by the spud dredger 'San Joaquin' purchased second hand from the Californian Goldfields. It operated in San Antonio de Poto for about eight years before sinking and being abandoned. Design specifications proposed for the new dredger are as follows:

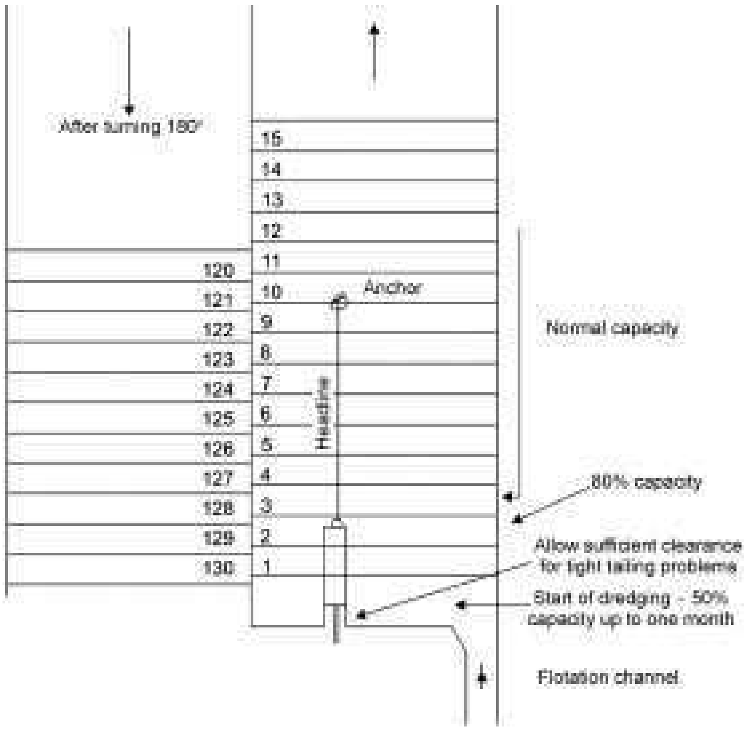
• minimum throughput	3.5 million m/year
• digging depth	20 m + 10 m bank = 30 m total
• hull length	53 m
• hull breadth	23 m
• hull depth	3.25 m
• length overall	15 m
• maintenance power	525 hp
• winches	individual hydraulic units
• bucket size	475 litre
• bucket speed	10–33 bpm (variable); capable of 160% torque at creep speed to facilitate opening and closing of the bucket band
• power consumption	1831 kW (max).

In 1988 in Bolivia, the dredger *Avicaya* suffered serious damage from slumping when the dredging face collapsed onto the ladder breaking it in half. One half of the bucket band was lost, embedded with the lower half of the ladder in the pond bottom. The decking was seriously damaged by the impact; the drive assembly was forced back into the upper deck structure for a distance of almost a metre. The damage was not sufficient to prevent its reconstruction but nevertheless, although consideration was given to purchasing the dredger and upgrading it for this operation, the proposal was rejected, possibly due to doubts of successful reconstruction for heavy-duty operations.

7.4.2 Preparations for dredging

Bucketline dredgers are constructed, or reassembled near to where they will commence dredging. In favourable conditions the dry dock is located immediately adjacent to the deposit so that the dredger may either be launched sideways into a prepared pond or be floated a hundred metres or so to its starting point. In less favourable circumstances, i.e., where there is no suitable ground adjacent to the deposit, the dredger is built at some distance away from its proposed commencement point and must be floated to the deposit along a specially constructed channel.

The outline of a dock for pontoon and superstructure construction prior to completion of a large dredger requires a working space of about 300 m by 300 m.



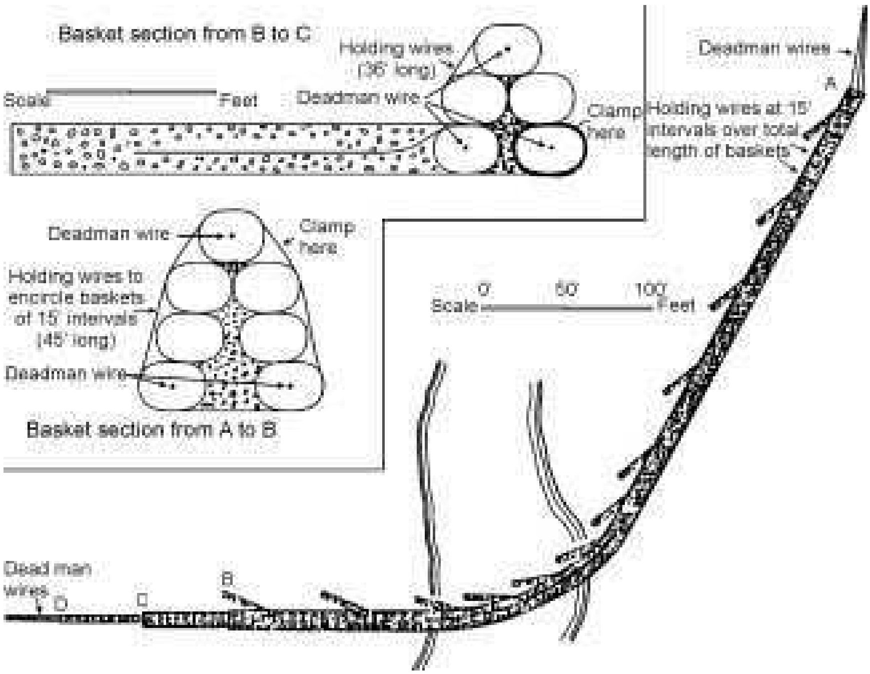
7.19 Open-up path of large bucketline dredger – schematic arrangement.

The dock slopes at 45º (edge piling with poles depending upon ground type). Depth of the dock depends upon ground level and the water level for flotation. Drainage of dock is carried out using drains leading to a de-watering sump. The dredger is towed into the flotation channel prior to installation of its bucketline.

The channel length is determined by the requirement of a stable dry area for dredger construction. The channel can be opened up using an excavator (e.g., back hoe) or small dredger. When floated to the site the dredger opens up a suitably wide strip across the deposit before cutting its way down into the wash and mining according to plan. The opening up path of the dredger is illustrated by the schematic arrangement (Fig. 7.19) at the starting point of production dredging. The assumed capacities and time frames illustrate the general order of parameters for a large-scale dredging operation.

Stream diversion

Conditions suitable for bucketline dredging may require the diversion of a stream traversing the area to be dredged. This operation usually requires filling the original stream channel with spoil after opening up and diverting the flow into the new channel. Where streams are subject to flooding, additional protec-



7.20 Details of rock basket construction and layout for river diversion (Dunkin, 1950).

tion may be given along threatened boundaries using some form of rock basket barriers. The setup illustrated in Fig. 7.20 was used for river control during the dredging of the Bulolo placer gold deposit in Papua New Guinea. 'Deadman' wires, anchored at the ends, pass through the baskets in this setup. Angle iron (350 mm × 10 mm × 10 mm) is clamped to deadman wires at 7.5 m intervals and by rope clamps to a deflector in two places.

7.4.3 Onshore dredging practice

All dredger operations are controlled by the 'Dredge Master' who is effectively the operations manager of a self-contained mining and treatment plant. Conditions in which the dredge pond continues to accumulate slimes due to a shortage of make-up water have an adverse effect on both mining and treatment. The following conditions are required for smooth and economic operation:

- A gently sloping bedrock; although technically feasible to step dredgers up quite steep slopes, such operations are very costly and time consuming; bedrock gradients should not exceed 1:40 for artificial ponds, greater tolerances may be accepted for natural ponds and riverbeds but dredging along flat surfaces is always an advantage.

- Absence of large boulders that might impede dredging; while an occasional boulder may be bypassed or perhaps shattered using explosives, the dredger cannot operate safely or effectively in the presence of clusters of large boulders.
- A bedrock or basement rock that can be cut by the buckets without transmitting undue stress and shock loads onto the digging mechanism; a hard, uneven basement tends to collect gold in the crevasses, potholes and other natural traps; serious losses may occur if the bedrock cannot be scraped down for at least 20 to 30 cm.
- Adequate reserves to justify the large capital expenditure involved; Lord (1983) quotes capacities and costs (manufacture, provide spares, construct on site and commission) for different sized dredgers as follows:

Size	Capacity (m^3/day)	Cost $US (millions)
Medium-large	5,000	20
Large	10,000	25
Two large dredgers	20,000	45

Digging involves slicing from the surface downwards. The dredging level is obtained by lowering the ladder 30 to 50 cm for each slice taken. A conventional digging profile will provide an average face slope of about 45 degrees. Care must be taken to guard against slumping from a steeper face while the ladder is in a deep digging position. This is an ever-present risk; bucketline dredger operators are disadvantaged by not being able to view the digging face and hence by not having full information on the material to be excavated (borehole data only). Job efficiency and bucket fill expectations can be rated only according to standard guidelines, which may be inaccurate for the conditions of the exercise.

Manoeuvring

Dredgers are manoeuvred using winches and landlines with or without spuds. Headlines are now usually preferred to spuds for mooring except in very difficult ground where problems may be experienced in holding the dredger up to the face in digging mode without the solid backing given by a spud.

Headline operation

Headline dredgers are operated through a five-wire mooring system comprising one headline, two forwards and two aft sidelines. In digging mode, the dredger is manoeuvred slowly backwards and forwards in an arc centred on the headline anchor point. Most of the reactive digging stresses are taken up by the headline; sidelines effect transverse movement of the dredger. Advantages claimed for a modern headline mooring systems are given by Anon. (1983) as follows:

- very wide cutting faces in one pass
- ability to move the dredger forward while continuing to dig
- ability to move the dredger continually backward while dredging deeper, thereby creating a stable face slope and facilitating bedrock clean-up; this is especially important when lowering the water table
- working the dredger at an angle to the digging direction to reduce the risk of bucket derailment
- working very narrow channels when moving from one place to another by moving the forward and aft winches in opposite directions
- free disposal of tailings
- gaining an indication of the digging force by the sag of the headline.

Spud mooring system

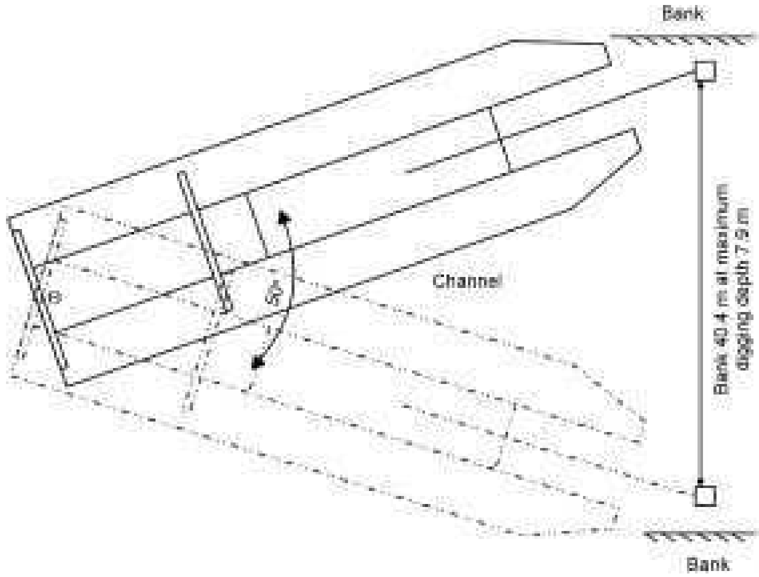
The spud mooring system was developed to correct the tendency of early dredgers to bounce back and forth against the face (yo-yoing) because of headline sag and stretch in tough digging conditions. Combined headline/spud systems are used in some installations to benefit from the best features of each. Spud-operated dredgers are held in position against the dredging face by ‘spuds’ placed at the stern of the dredger. A simple but direct spud arrangement is provided by two box girders, which are raised and lowered vertically from the aft gantry end to hold the dredger in place. Figure 7.21 shows the minimum width of channel that may be cleared by manipulating the sidelines to swing the dredger about the travelling spud. Important design features for spud systems are:

- The spud strength must be sufficient to withstand the reactive digging force of the buckets when they are operating on temporary overload.
- The spud strength must be designed for the ground type and shear strength under all working conditions.
- Spud changeover times should be minimised; this may be done using quick-acting rams or hydraulic winches for raising and lowering.

Headline vs. spuds

The California type spud dredger was adapted from the first New Zealand type headline dredgers for the more difficult conditions of the California goldfields. Spuds appeared to offer better digging capabilities at that time than any upgraded headline system. Each system has its own peculiar advantages and disadvantages but with the more advanced technologies of present times, headline dredging has gained in flexibility over spud dredging. It is also more readily available.

In 1968, Ramanowitz and Cruikshank agreed that ‘while the Malaysian (headline) type dredger was the only possible choice for dredging offshore – this



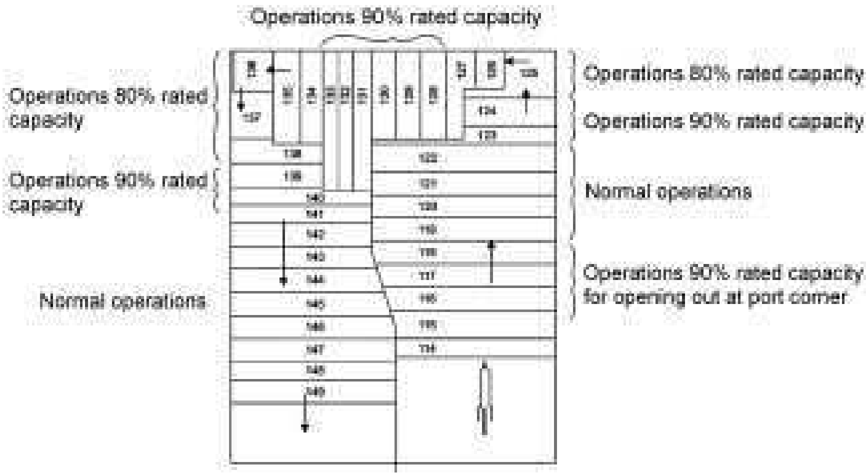
7.21 Minimum width of channel that dredger will clear when digging with travelling spud at extreme aft position.

type of dredger has proven that it cannot produce unit yardages as high as the spud type, but each has its scope of operation'. No statistics were cited for spud and headline dredgers operating in the same ground onshore and perhaps the remarks were relevant only to that time. The modern situation is distinct from in the past. Malaysian headline dredger design has improved greatly, as have the materials of construction.

Turning

Dredgers are not easily manoeuvrable when turning at the end of a run and dredge paths are usually planned to limit the number of turns that must be made. Two main systems are used depending upon whether the deposit is mined in transverse or in longitudinal strips. Transverse dredge paths have advantages for wide placers and for placers in which values extend beyond the expected boundaries. They are disadvantaged by the need to either leave wedges of unworked ground between adjacent cuts or to accept contamination from tailings stacked along those boundaries. This alternative is obviously less attractive for deep, than for shallow deposits because of the increased amount of dilution from fall-in. On the other hand, up to 5% of the total resource may be left in the ground if the wedges remain unmined.

Spud dredgers, mining longitudinally, i.e., along the axis of the deposit, may progress along several adjoining faces over the whole width of the deposit. Each face advances at 10–15 m intervals; transferring from one face to the next takes



7.22 Dredger making a 180º turn.

only 10–15 minutes. The dredge stacker allows tailings to be deposited at a safe distance behind the dredger to allow it to move freely. No intermediate wedges remain and the main difficulty is in working lateral extensions to the deposits outside of the planned boundaries.

Headline dredgers, on the other hand, can mine in wide sweeps across the deposit, the width of each cut depending upon the headline length. In very wide deposits, this length can be about six to seven times the width of the cut to provide optimum digging conditions. For narrower deposits, the headline length is subject to geographical constraints and the ratio will be correspondingly smaller. Figure 7.22 shows the dredger making a 180º turn.

7.4.4 Offshore dredging practice

An offshore bucketline dredging operation faces generally similar constraints as onshore dredgers in regard to dredging depths (e.g. maximum ~ 50 m). Differences include:

- Offshore dredgers are self-propelled.
- Dredging is constrained by the effects of wind, waves and currents; the greater prevalence and impact of atmospheric disturbances; the requirement to conform to maritime standards of safety, particularly when operating within commercial shipping lanes; and isolation from ground maintenance and supply sources.
- Corrosion, due to seawater, is of greater moment offshore; periodic dry-docking for major maintenance is a more difficult operation and may involve longer shutdown periods than on land.

Mode of access to offshore dredgers is by the sea itself. Robust access vessels must conform to maritime standards for the particular areas being mined. The ancillary craft for the more remote offshore waters will not be less than the following:

- tug boats: two (500–700 hp)
- crew vessels: two (300 hp)
- anchor barge: one
- supply vessels: two including one dumb barge for heavy equipment transport.

Note that personal safety is an essential consideration when transferring from the dredger to the crew launch. Swinging from ropes in 'Tarzan' fashion to pass from one vessel to the other (a common practice) is most hazardous in any seas higher than 2.0 to 2.5 m.

Operational considerations

Equipment design should be simple, reliable and effective, and the plant should be easy to maintain. Operational problems are much greater than on land and experience has shown that trying to achieve levels of sophistication beyond the operator's capabilities leads to unnecessary downtime while trying to make some of the features work. Table 7.9 lists design data for four deep-digging bucketline dredgers in Indonesian waters.

The dredger *BIMA* is an example of the additional problems of dredging offshore. Built in 1978 *BIMA* cost US$35 million up to the point of commissioning for a design capacity of 8-million m^3/year. Although considered 'state of the art' for the time, *BIMA* achieved a maximum output of only 7.2 million m^3/year. It was then sold to Inspiration Gold Inc. and towed to Alaskan waters after modification to the treatment plant.

Research for offshore dredging is focused mainly on three areas: (i) design of buffering systems; (ii) ladder and caterpillar design, catenary and digging depth; and (iii) materials technology and engineering. Key environmental factors are wave and wind conditions and currents. In any proposed dredging area the parameters to be measured for both normal and survival conditions are, thus:

- wave: height, frequency, length and distribution
- wind: velocity, frequency and direction
- current: speed, tidal variations, locations
- time: available operating time based upon climatic variations and major repair time allocation.

Specific operating problems are associated with each offshore area based upon the intensity of meteorological and marine conditions for both normal operations and survival conditions. Relevant parameters of wind, wave and current statistics are given in Table 7.10 as recorded for areas around the islands

Table 7.9 Indonesian deep-digging bucket dredgers design data (after Goh, 1987)

Operator	Dredge	Designer	Builder	Remarks
P.T. Riau	*BIMA* 45 m Digging depth 1,00 cu.m/hr 7.2 million cu.m/yr 24 cu.ft buckets 12,000 t weight	M.T.E. hydraulic buffer installation for digging end to enable all-weather operations	Jurong Shipyard, Singapore 1979 completed	Dreding operations ceased in 1985 because of tin quota and inability to dredge continuously throughout the year. Sold to Inspiration Resources Corp. for offshore gold dredging at Nome, Alaska in 1986
P.T.T. Timah	*Bangka II* 46 m Digging depth 675 cu.m/hr treat 794 cu.m/hr strip 5.0 million cu.m/yr 24 cu.ft buckets 12,000 t weight	F.W. Payne no buffer system, fixed ladder	Mitsubishi, Hiroshima, Japan 1978 completed	Operating at Bangka Island in main area during non-monsoon period and escape areas on western end during monsoon period
	Belitung I 50 m Digging depth 675 cu.m/hr treat 794 cu.m/hr strip 5.0 million cu.m/yr 24 cu.ft buckets 12,000 t weight	F.W. Payne no buffer system, fixed ladder	McDermott, Batam Island, Indonesia 1981 completed	Operating at Kundur Island in sheltered waters
	Singkep I 50 m Digging depth 675 cu.m/hr treat 794 cu.m/hr strip 5.0 million cu.m/yr 24 cu.ft buckets 12,000 t weight	F.W. Payne no buffer system, fixed ladder	P.T. Kodja, Indonesia 1983 completed	Operating at Kundur Island in sheltered waters

Table 7.10 Wind, wave and current conditions in operating waters of Indonesia and Thailand

Item	Indonesian waters		Anadaman Sea (Thailand)	
	Operating	Survival	Operating	Survival
Max. tidal range (m)	3.5	–	3.0	–
Max. current (knots)	4.0	8.0	2.0	4.0
Max. wind velocity (knots)	33.0	47.0	40.0	77.0
Max. wave height (m)	1.5	3.0	3.0	9.8

of Banka and Billiton in Indonesian waters, and offshore Thailand in the Andaman Sea. A wind blowing for about ten hours over the surface of the ocean causes the surface water to flow at about 2% of the wind speed. The combination of strong wind and wave conditions makes dredging difficult and dangerous. The Beaufort Scale of wind and sea characteristics (Table 7.11) is accepted globally.

High current forces require more robust mooring winches than are needed for land-based dredgers because of the higher and repetitive stresses involved. The effects are generally slight in open sea conditions, but may pose serious problems in the vicinity of islands particularly between adjoining islands and between islands and the land.

Buffer systems

Methods for reducing the wave effect on the digging operations are described in Table 7.12 for floating breakwater buffer systems and articulated ladder, elongated pontoon and semi-submersible pontoon. The effectiveness of buffer systems for offshore dredging has not yet been proven. The performance of the *BIMA* system was reported to be unfavourable. Before it was switched off problems had arisen in trying to synchronise the damping response to the wave periods. The *BIMA* system also found difficulties in trying to cope with the inherent shock absorbing and bouncing that takes place when trying to recover pockets of high grade ore in the bedrock.

The caterpillar track system is well established onshore but it gives additional maintenance problems offshore because of the lack of space and high rate of wear and tear. Designers have made some improvements by helping to resolve problems of optimum positioning of the caterpillar track to suit the catenary and required digging depth. Perry idlers were used as an alternative to caterpillar idlers in one Indonesian dredger (*Singkep 1*) but appeared to have problems of overheating of the idler bearings and difficulties of access for maintenance.

Table 7.11 Wind and sea characteristics

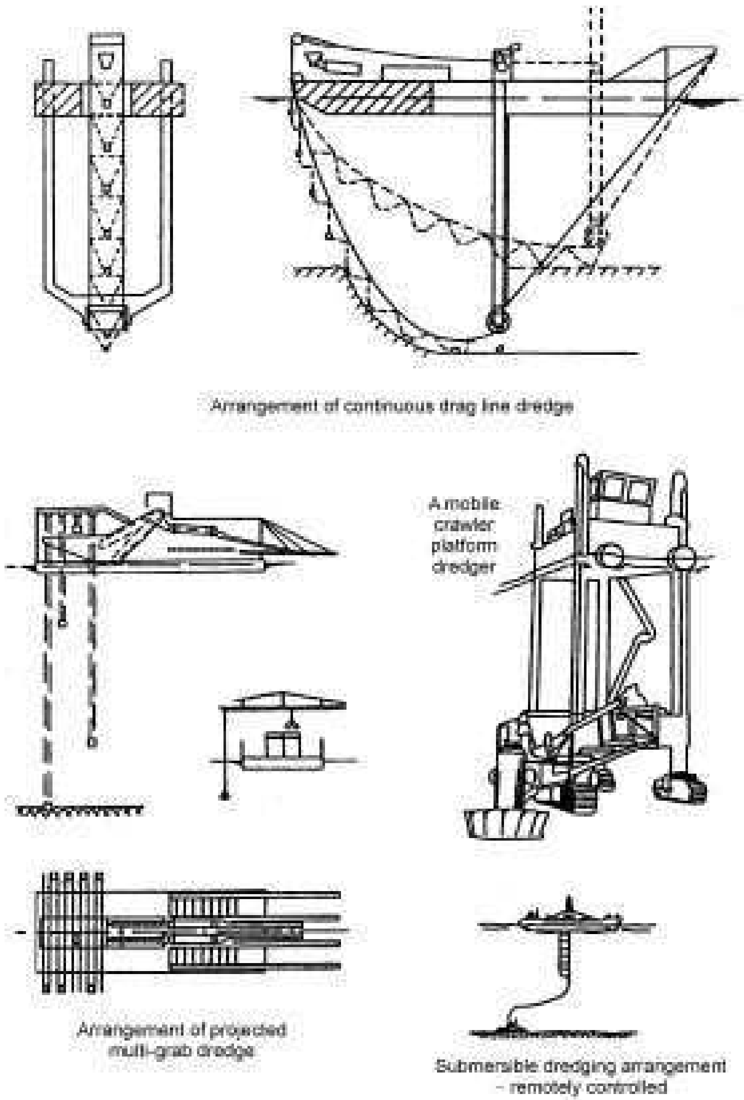
Beaufort Scale	International Code	Wind Type	Wind Velocity	Sea Characteristics	Wave Heights
0	0	Calm	1 knot	Mirror-like	0
1	0	Light air	1–3 knots	Rippled	0
2	1	Light breeze	4–6 knots	Small wavelets	0–1 foot
3	2	Gentle breeze	7–10 knots	Large wavelets, crusts begin to break	1–2 feet
4	3	Moderate breeze	11–16 knots	Small waves, frequent whitecaps	2–4 feet
5	4	Fresh breeze	17–21 knots	Moderate in long form – pronounced whitecaps	4–8 feet
6	5	Strong breeze	22–27 knots	Rough, with large waves, extensive whitecaps some spray	8–13 feet
7	6	Moderate gale	28–33 knots	Sea heaps up with white foam	13–20 feet
8	6	Fresh gale	34–40 knots	Moderate high waves of greater length. Foam in well-marked streaks	13–20 feet
9	6	Strong gale	41–47 knots	Very rough seas with high waves commecing to roll	13–20 feet
10	7	Whole gale	48–55 knots	Very high waves, sea appears white, rolling heavy	20–30 feet
11	8	Storm	56–63 knots	Exceptionally high waves small ships lost to view for long periods	30–45 feet
12	9	Hurricane	64+ knots	Sea completely white with driving spray	45+ feet

Table 7.12 Methods of compensation for waves

Method	Description	Remarks	Cost
1. Floating breakwater	Shields the dredge from the waves. Wave energy is dissipated in the breakwater	Not effective in the open sea; anchoring can be a problem; high mobility of the dredge means breakwater has to be moved frequently	–
2. Buffer system & articulated ladder	Whole digging system including the ladder rests on pneumatic/hydraulic cylinders which absorb the impact of the waves at the digging end	Installed in BIMA and OMO Bodan dredges. Buffers operated under certain conditions of wave height and wave period: High waves with low periods (8 secs) and low waves (=1.2 m) with high periods. Does not cover waves over 1.2 m and period of 8 secs. Operating conditions can be varied to suit Andaman Sea conditions of higher waves and periods. (The articulated L/D idea has yet to be implemented.)	Case 1 A Estimated cost: 1.3 times cost of conventional pontoon dredge
3. Pontoon bow elongated & stabiliser plates underwater	Plates below the pontoon provide a damping effect. Elongation of bow to seal off the bow end of the well provides strength and length to protect the pontoon against long waves	This idea was put forward in an early proposal for an offshore deep-digger bucket dredge in 1972. The project was aborted because of the political climate in SE Asia. Tank testing of models indicated very good damping factors in 2.5 and 3 m simulated waves	Case 1 B Estimated cost: 1.1 times cost of conventional pontoon dredge
4. Semi-submersible pontoon	Reduced water plane area and heavy submerged section reduces the wave effect on the floating structure	Used for oil rigs and big offshore cranes in the North Sea; stable and effective platform. Massive structure and extremely high cost	Case 2 Estimated cost: 1.8 times cost of conventional pontoon dredge

Deep sea dredging

Research has so far failed to produce a commercial model for deeper offshore dredging, although several design possibilities have been investigated (Macdonald, 1987). Interest in the development of deep offshore mining methods was stimulated a few decades ago by the discovery of vast quantities of polymetallic nodules on the deep ocean floor. Three of the concepts tested are described in Fig. 7.23. More recently epithermal-like seafloor hydrothermal gold



7.23 Concepts – deep-sea mining.

ore systems have been discovered in shallow island arc environments of the west and southwest Pacific (see Chapter 2). However no realistic concept for mining such deposits has yet evolved.

7.4.5 Reclaiming a used dredger

A bucketline dredger is usually maintained in good operating condition until the deposit it is working on is exhausted. The residual value then depends upon whether the dredger can be reconditioned and transferred economically to a new location. Both the cost of doing so and the time frame involved must then be able to compete favourably with the cost and time involved with building a new dredger on site. The choice of a new or used dredger follows a period of intense investigation both at the point of purchase and at the proposed new dredging site. A most important factor is the separation between the two points, which may be a few tens of kilometres or many thousands of kilometres, perhaps from one country to another.

Investigation

The history of any dredger selected for upgrading is investigated to ensure its sturdiness and reliability based upon past performance. Expert opinion is sought in order to:

- verify the information of the vendor
- assess the physical status of the equipment involved
- assess the optimum technical and economic performance of the dredger in its present state
- evaluate technically and economically the required modification, reconstruction and or repair requirements and costs
- examine the alternative of a completely new dredger.

Amongst other matters the expert's report will describe the present condition of the main structural members and note what may be safely retained of the superstructure, and what should be replaced. Normally the hull will be replaced in its entirety but some sections may be salvaged if the dredger is not very old. A technical and economic evaluation will cover all aspects of the proposal and the report will advise upon the most appropriate method of upgrading.

It is unlikely, nevertheless, that any used dredger will have all of the required capabilities. The ladder may have to be shortened or lengthened; the production rate may have to be increased; digging conditions may be more or less difficult, requiring different-sized buckets, and so on. However, the matching must be reasonably close. Small changes can usually be accommodated safely with only minor changes to the original design. Any significant differences in operating conditions will require major modifications to the used dredger and almost certainly invalidate the particular choice.

In this respect a ladder will usually accept only a slightly larger bucket band. For example, a 350-litre bucket band would not be replaceable by a 500-litre bucket band. An upgrading of this magnitude would require a completely new ladder along with other accessories such as a new main drive, larger pontoon and larger treatment plant. The end result might be a dredger composed of mainly new parts, but it would still be constructed to a makeshift design, with few of the advantages of a freshly designed unit.

The main drive may be capable of accepting new internals to give a slightly higher speed but any upgrading must be done within the limits of stress safety factors. The same ladder and shell may be retained within safe limits only if the speed change is small. In considering one particular upgrading, the requirement was to increase the existing dredger capacity from 3.6 millions m^3/year to a minimum of 4.0 million m^3/year. The characteristics of the used dredger were as follows:

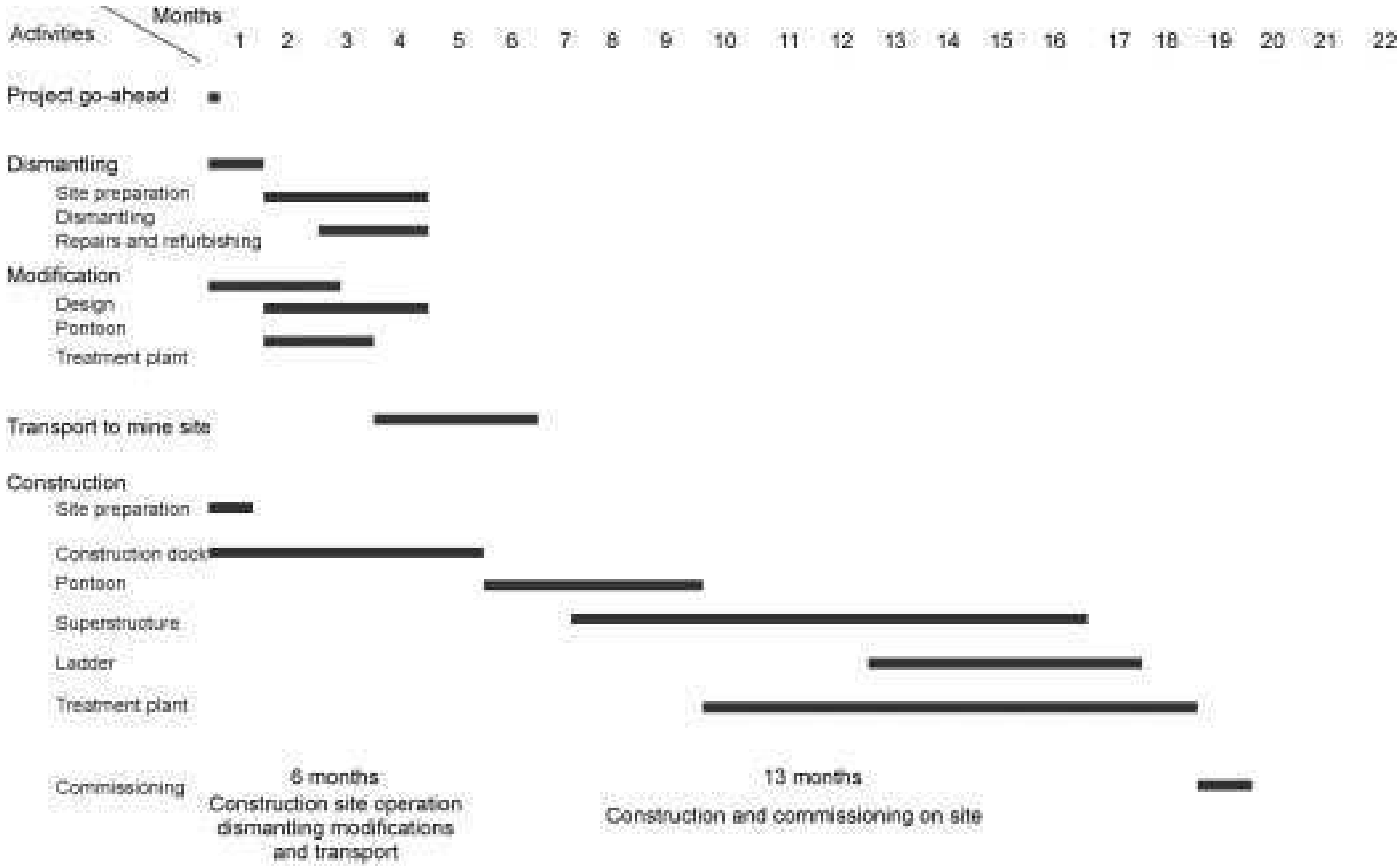
- bucket speed range – 18.9/20.3/22.8 bpm (average 21.9 bpm)
- bucket capacity – 510 litres
- dredger capacity – $0.51 \times 21.9 \times 0.75$ (fill factor) $\times$ 60 (minutes) $\times$ 7,200 (hours) $=$ 3.6 millions m^3/year
- the main drive intervals could be replaced safely to give a bucket speed range of 22.0/23.6/25.5 bpm (average 24.0), giving an upgraded capacity of $0.51 \times 24.0 \times 0.75 \times 60$ (min) $\times$ 7,200 (hrs) $=$ 4.0 million m^3/year.

Advantages of reclaiming

Usually the most attractive feature of any used dredger proposal is its reduced time frame. A refurbished dredger can generally be ready for commissioning within 15 to 17 months from the start of dismantling. The bar chart (Fig. 7.24) shows how good planning can shorten the time frame for project implementation:

- coincident with the dismantling of the used dredger, work commences on pontoon construction and preparations are made for the planned modifications
- site preparation commences at the new dredging location at the same time and continues throughout the shipping period
- the dredger components are shipped first to allow dredger construction to commence and proceed rapidly to completion without bottlenecks.

The larger companies have gained much experience in refurbishing and relocating dredgers and have often found it economically viable to shift dredgers from one property to another, even globally. Companies setting up new operations also look to the advantages of purchasing and upgrading a used dredger rather than purchasing a completely new model. Upgrading offers benefits of lower first cost and reduced project implementation time. A used dredger can usually be purchased for around its scrap value if the owners are



7.24 Hypothetical bar chart for project implementation refurbishing a used dredger.

running out of ground. Alternatively, an agreed value of say $600/tonne could be based upon the weight of the re-usable parts of the dredger.

In general, cost savings may be expected of the order of 15–25% for used vs. new dredgers. Savings of time will generally amount to some 20–40%, depending upon location. In applying various pricing techniques to one particular dredger, separate estimates were reached of US$2.5 million, US$1.68 million and US$0.28 million. Refurbished, upgraded and relocated, this dredger was estimated to have a cost advantage of around US$3.66 million compared with the cost of a new dredger, if purchased for $2.5 million at source.

Dredgers are somewhat akin to aircraft in that they can be kept operating almost indefinitely provided they receive proper maintenance and periodic renewal of worn out parts. Consequently, there is often a choice between either constructing a new dredger or acquiring a used dredger and dismantling and reconstructing it in the required mode. The Yuba Goldfield Company rebuilt and adapted four of its 22 dredgers prior to 1968 for new sets of conditions. Typical of the changes made:

- Yuba No. 17 was converted from a digging depth of 81 feet (24.69 m) to 112 feet (34.14 m).
- Yuba No. 20 was extended to dig to 124 feet (37.8 m)
- Yuba No. 22 was extended to dig to 107 feet (32.6 m).
- Yuba No. 18 425-litre dredger was built in California as a gold dredger in 1925. It operated successfully in California for about 30 years before being modified for tin dredging in Bolivia in 1958. Renamed the *Avicaya*, the 13.75 ft^3 (390-litre) dredger was remodified again in 1966.

Disadvantages of reclaiming

Having regard to the inevitably uncertain condition of a used dredger, a project with a long mine life will usually favour a new dredger. Thus, while a used dredger may be cheaper than a new dredger when re-assembled at the site of the dredging location, long-term reliability is a major consideration and the reclaimed dredger will have the following inherent disadvantages:

- Some of the main components will be in an upgraded and not new condition and will thus be subject to increased maintenance and reduced life.
- The used dredger will be more prone to structural breakdown from metal fatigue.
- All weaknesses may not be detected and corrected during the refurbishing process.
- Only limited modifications may be possible within the constraints of the existing design; a degree of compromise is inevitable and may involve some risk of failure.
- If the used dredger is inherently oversized and overweight for the proposed

service it will have higher unit operating costs and may lack the required degree of manoeuvrability.
- If the used dredger is undersized both mechanical and structural failure may occur, particularly on temporary overload.

7.5 Hydraulic dredgers

Under certain conditions hydraulic dredgers offer a comparatively low capital cost alternative to bucketline dredgers. Hydraulic dredgers are more manoeuvrable, and the smaller dimensions, made possible by having treatment plant separate from the dredger allows operation in smaller channels than bucketline dredgers, which are complete mining treatment units in the one hull. Suitable conditions are provided by:

- sediment that can be easily cut and fed into the dredge pump suction pipe using cutter heads or bucket wheels
- freedom from any buried timber that cannot be broken up by the action of the cutters, or by blades installed at the entrance to the pump itself
- absence of extensive root systems that may make the system unworkable because of frequent pump blockages and high maintenance
- availability of very substantial volumes of make-up water, particularly in clayey ground; the cutting action creates a slime problem, which may be solved only by removing the slime and replacing it with fresh water.

Constraints to the use of hydraulic dredgers include:

- Hydraulic dredgers are power intensive because of the large volumes of water transported with the solids; unit power costs are much higher for hydraulic dredging than for bucketline dredging.
- Pumps and pipelines wear rapidly when the sediments are abrasive; the mining of abrasive materials results in reduced availability and higher maintenance costs.
- Pipeline transport is materially affected by changing conditions of particle size and type.
- Digging conditions change constantly and safe operation requires an ability to adjust flow velocities through a wide range of impeller speeds to prevent the larger solids settling out and blocking pumps and pipelines; the unpredictability of these variables demands a high degree of compromise and the application of generous safety factors in design.

The advantage of hydraulic dredging is that such dredgers can mine small tributaries while the treatment plant remains in the main pond area. In order to mine a similar small tributary by bucket dredging would require excavation over a much greater channel width and would incur an excessive amount of dilution because of its larger proportions.

7.5.1 Suction-cutter dredgers

Suction-cutter dredgers employ rotating cutter heads to break and slurry the face. The cutter head mechanism comprises a cluster of curved, steel blades, drive shaft and drive machinery mounted on a ladder along with the dredge suction pipe. The ladder is pivoted downward at an angle from the pontoon and raised or lowered as required using small hydraulic motors and a gantry pulley system.

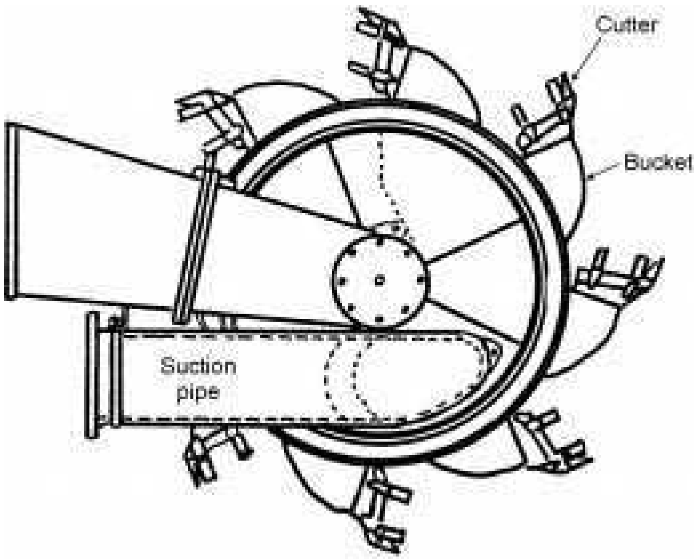
This system of dredging has its main application in the mining of free-flowing sands such as are found in beach sand deposits and drowned sand deposits offshore (Macdonald, 1983a). The cutter head undercuts the mining face and the method then relies upon the sand rilling freely to the suction nozzle. In favourable conditions, suction cutter dredgers mine and transport large quantities of spoil over considerable distances in the one operation. With hull-mounted suction pumps they are limited to a shallow dredging depth of around five metres or so. Deeper dredging is effected using specially designed pumps and drives installed close to the bottom of the ladder. So installed, the suction lift is minimised and the dredging depth is limited only by the weight constraints of the supporting ladder and other design features.

The suction cutter system experiences many problems when used for production purposes in placer gold mining operations:

- Clogging of the cutter blades occurs when trying to dig sticky clays and other cohesive materials.
- Blockages tend to occur from clusters of plant roots and other debris due to poor near-inlet conditions.
- Flow stabilises only at some distance (one or two diameters) inside the nozzle-entrance (Macdonald, 1962, 1966); some solid particles may fall out of suspension and be lost before the flow reaches that point.
- Beach mining experience has shown that not more than 90–95% of heavy minerals (density 3.3 to 4.5) are recovered from the dredge pond. The percentage of gold left behind would probably be much higher.
- The action of the dredger results in low and variable solids/fluid entrainment, less effective cutting in one direction than in the other and a tendency to override more compacted sections of the face.
- The bottom of a suction dredge pond typically becomes pot-holed during dredging thus providing cavities within which the heavy minerals can settle.

7.5.2 Bucket-wheel dredgers

Bucket-wheel dredgers are generally preferred to suction cutter dredgers for production dredging. Bucket wheels are more able to cut harder materials; they clean up more effectively at bedrock and deliver the slurry at a higher pulp density to the treatment plant. Their use is currently limited to a dredging depth

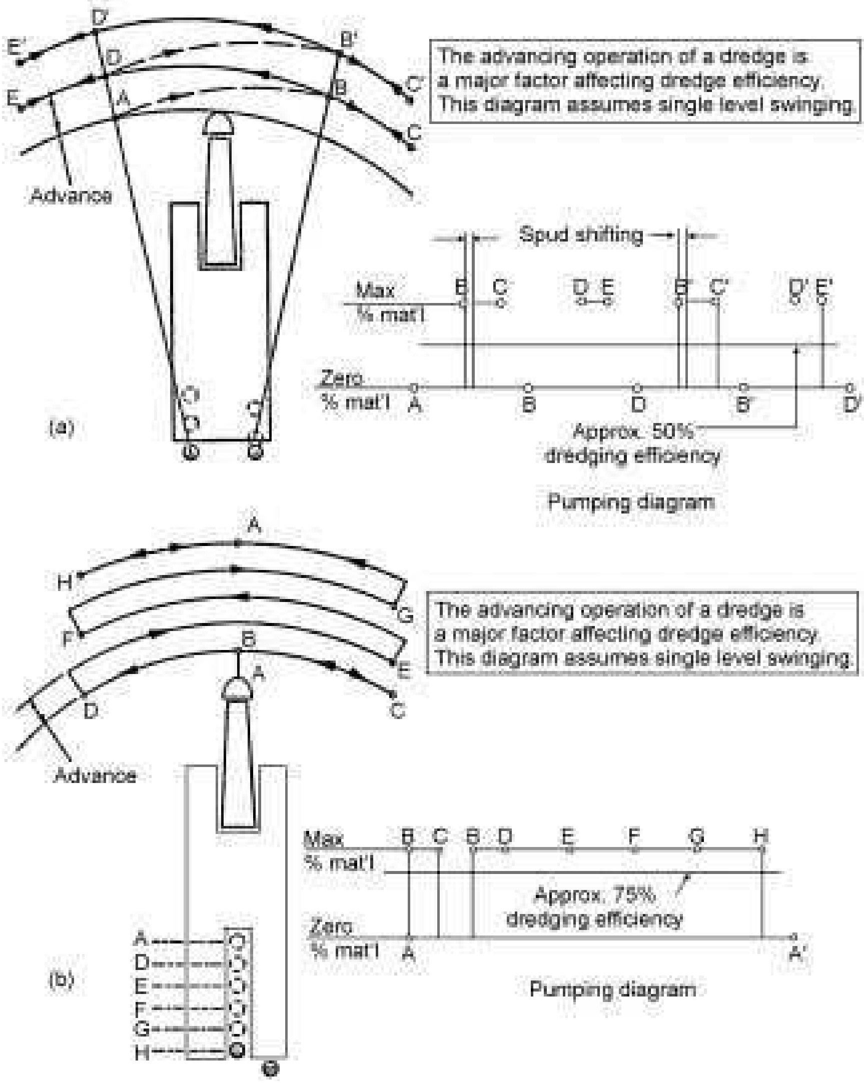


7.25 Typical bucket-wheel configuration.

of about 30 m because of the weight of the wheel and ladder. Although suffering the same clogging and blockage problems as suction cutter dredgers the bucket wheel is more suited to overcoming them. For example, the bucket-wheel configuration shown in Fig. 7.25 may be fitted with clearance fingers to cut through roots and hard clayey fragments.

In the bucket wheel mining operation described diagrammatically in Fig. 7.25, the dredger pumps the spoil through floating slurry pipelines to a gravity treatment plant floating in the same pond. Manoeuvring of both dredger and treatment plant unit is usually effected through a combination of spuds and anchor-lines or by crossed bow, side and stern lines. This particular dredger is manoeuvred using side-slewing winches. The method of advancing an operating dredger is an important factor affecting its efficiency. In Fig. 7.26 the Ellicott Company compares the dredging efficiencies of (a) conventional walking spuds and (b) the Ellicott spud carriage system.

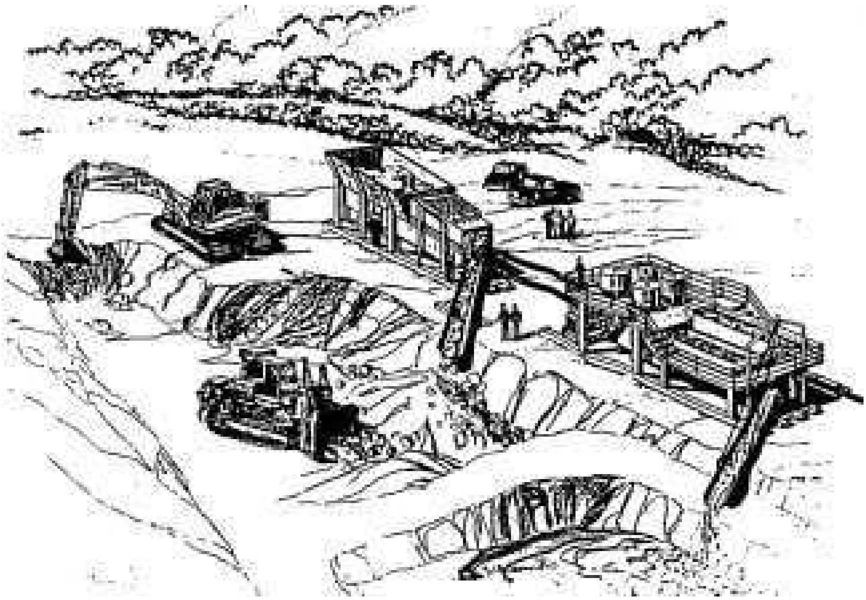
Particle size and frequency are determining factors in pipeline transportation and manufacturers normally supply separate pump performance curves and tables for silts, sands and gravels. These charts offer general solutions for specific physical relationships to assist in preliminary studies. They do not however, offer an unambiguous means of predicting the performance of pumps that are called upon to handle heterogeneous and constantly changing mixtures of sediments from a dredging face. The final pump selection is a more or less safe compromise for the particular set of conditions based upon the results of detailed screen analyses and the manufacturer's experience.



7.26 Comparison of efficiencies of conventional walking spuds (a) and the Ellicott spud carriage system (b).

7.6 Dry mining

The basic systems of dry mining are generally similar to those for civil works such as land reclamation, road construction and quarrying. Only the objectives differ and experience must be tempered with caution when trying to relate performance data from non-selective earth-moving operations to predictions of



7.27 Conceptual arrangement – dry mining operations.

plant performance in which selectivity is a fundamental requirement. While overburden can be removed as in any other mining application, placer gold paystreaks are rarely distributed evenly and the pay dirt must be taken up separately and fed to the treatment plant at a required rate and in a designated form. This calls for greater precision when mining along the boundaries of deposits and in cleaning up at bedrock. A typical dry mining operation calls for the recovery of a rougher concentrate for upgrading in the gold room, and the deposition of tailings in worked-out areas. The arrangement described conceptually in Fig. 7.27 includes a slurry transfer system for greater control and flexibility.

Ground water control is exercised on an *ad-hoc* basis during the cutting of the initial pit until a pump sump can be cut into the bedrock at the lowest point of the pit floor. Sumps are then cut progressively in the direction of mining to keep the pit floor reasonably dry. Pumping requirements are estimated for both the period of pit development and for normal operations. It is assumed that all significant surface run-offs will be diverted away from the pit leaving only seepage water to be dealt with.

7.6.1 Machine selection

Manufacturers of earth-moving equipment must be informed of the particular conditions in which their machines will operate. Usually, it is sufficient to supply general information on the nature of the soils (clayey, sandy, gravelly, cemented, etc.). Machine ratings can then be determined in terms of bucket fill factors, which range from 0.7 to 1.2 according to the texture and compaction of the material; and job efficiency which varies from 0.7 to 0.83 according to likely operating conditions on a scale of poor to good. The variations encountered in any one deposit usually provide an appreciable range of ground types and digging conditions.

The following machines and combinations of machines are in common use, either alone, or in support of some system of wet mining, e.g. sluicing:

- bulldozer/front-end loader/trucks
- bulldozer/wheel tractor scrapers
- back-acting hydraulic excavators/trucks
- back hoe/floating treatment plant (doodle bug).

The proposed scale of mining and cost strongly influence the choice. The type and size of equipment is determined largely by the method of mining and the required production rate. In practical terms this requires machines to be operated at safe maximum capacities without working on overload, except for brief periods. A high level of availability must be backed by good maintenance programmes and the ready availability of spares. Downtime is a major cost factor and high utilisation rates are dependent upon robust equipment having a safe excess capacity over what is required, and a maintenance programme that is designed for the particular needs of the machines.

Important considerations are the location of disposal areas for overburden, plant tailings and slime, the design of haulage roads and transport systems, and the restoration of mined-out areas. The main controlling factors are the physical characteristics of the deposit and its geographic setting, slope and texture of the mining floor, and the volume of water to be handled. Bulldozers, front-end loaders and trucks are obvious selections for small, shallow deposits. Bulldozers are also useful for ripping tightly compacted gravels, indurated cappings, etc., and for restoration. Front-end loaders, useful around stockpiles and in the treatment plant, are general-purpose units in most dry mining operations. For large-scale operations, transport systems may be developed either around wheel tractor scrapers, or back hoe/truck combinations.

Bulldozer/front-end loader/trucks

In this arrangement, a bulldozer is used to break the ground and push the topsoil and overburden to one side. The gold-bearing gravels are piled into heaps for loading into trucks using front-end loaders. Articulated loader types are usually the most suitable types of loader particularly in narrow excavations, which require tight turning circles. In addition to having a high degree of mobility, the articulated types have good digging and transportation capacities in soft ground and gentle slopes and their maintenance costs are relatively low compared with equivalent track type machines. These machines have enhanced traction capabilities and operate better in more adverse ground conditions but are slower in operation and more costly to maintain.

Loading, dumping and manoeuvring an articulated wheel loader takes around 25 seconds in good conditions. Limiting factors are the height of the stockpile face and face compaction, which both affect bucket fill. Bucket fill rises from 0.8 to 0.9 for a low face, to 0.9 to 1.00 for a well-heaped stockpile. Overall cycle times vary with trucking distances to disposal points.

Bulldozer/wheel tractor scrapers

Twin bowl scrapers are the most efficient of the modern units and provide the lowest unit costs. These machines combine the functions of loading and transportation in the one unit and are best suited to large stripping operations on flat level surfaces. Scraper efficiency is dependent upon a rapid turnaround in the pit and loading usually requires some assistance from bulldozers for push loading and/or ripping. Since one pusher unit can usually handle five or more scraper units, the system is best suited to large undertakings. The loading action is essentially non-selective; any attempt at selectivity will almost certainly increase cycle times and unit costs.

A major disadvantage of the wheel scraper operation is the need for very sophisticated garage maintenance to keep the units running to schedule. The system is thus restricted to large operations, which can afford the high costs. Maintenance problems are exacerbated in remote areas where workshop facilities must be extensive, all fast-moving spares must be kept on hand, with highly skilled mechanics retained on site at all times. Because of individual machine requirements, most equipment manufacturers provide special classes to train the mechanics that service these machines. Courses at manufacturers' premises are normally of two weeks duration with re-familiarisation courses at regular intervals thereafter.

Back-acting hydraulic excavators/trucks

A mobile treatment plant offers the best means of minimising haulage distances for ore transport and tailings return. The back-acting hydraulic excavator (back hoe) is the most versatile digging machine for dry alluvial gold operations. It is used in a variety of ways and usually competes favourably against drag lines, forward-acting excavators and other loader types. It mines selectively and can load directly into trucks for haulage to the mill or dumping ground, or into land-based plant hopper. Figure 7.28 shows a land-based back hoe operation at Kim Je, South Korea. The main operating variables of back-acting hydraulic excavators are bucket capacities and types, and digging forces.

7.28 Back hoe operations at Kim Je, South Korea.

Bucket capacity and type

Buckets are rated according to 'struck' and 'heaped' capacities. The struck capacity is the actual volume enclosed inside the outline of the bucket and is independent of any material caught up on the spill plate or in the bucket teeth. The heaped capacity includes, in addition to the struck volume, the amount of material heaped above the strike-off plane (i.e. at an angle of repose of 1:1 in accordance with PGSA Standard No. 3 and SAE Standard 5296). It ignores any material carried by the spill plate or bucket teeth, since these amounts cannot be quantified. Typical bucket payload factors are listed in Table 7.13.

Digging forces

The bucket digging force is a function of the bucket curling force and stick crowd force and a feature of back-acting hoes is their ability to exert high break

Table 7.13 Bucket payload factors

Material	Fill factor (%)
Moist loam or sandy clay	100–110
Sand and gravel	95–100
Hard tough clay	80–90
Rock – well blasted	60–75
Rock – poorly blasted	40–50

out forces at all levels in the excavation. The average bucket payload is determined by the size and shape of the bucket, the nature of the material being dug and the heaped bucket capacity multiplied by a fill factor.

Individual bucket types are available for a variety of soil conditions ranging from easily dug material to compacted gravels, hard clays, and calcrete. Bucket width is a major consideration and generally, the harder the digging the narrower the bucket. Tip radius is important in hard ground. Shorter tip radius buckets are easier to load and provide more total bucket curling break out force (force exerted by the bucket cylinder) than long tip radius buckets. Bulldozers and blasting prepare the ground for loading in very difficult digging conditions.

Digging cycle

The capacity of a back hoe in truck loading service varies according to its cycle time which, in turn, differs with the depth of digging, weight of the material *in situ* (bank), swell factor, type of material and bucket fill. The cycle time for each bucket is made up of excavating time and swing time (loaded), dump time and swing time (empty).

Truck cycle

The truck cycle is the total time taken for the excavator to load the truck, and for the truck to haul the load to the dumping point, dump, return empty and spot, ready for the next load. The spotting time is the time for the truck to be re-positioned for loading and for loading to commence. The main variables of the truck cycle are travel distances, gradients and rolling resistance. The rolling resistance is a function of the state of the road surface and may be as high as 20% for soft, muddy and deeply rutted roads.

Strip mining

This method applies to deposits of shallow (8–10 m) depth with high strip ratios. The area is mined in transverse strips, each about 20 m in width. The elevating scraper removes topsoil, spreading it directly onto levelled backfill. Most of the overburden is dozed into the previously excavated strip. The remaining overburden is cleaned off and dumped on top of the dozed overburden, the wash is then mined using the back hoe and dumped into trucks for haulage to the treatment plant.

Operational problems

The back hoe is a useful production tool in small tributaries or narrow channel sections of large deposits that are being exploited by other means, e.g.

bucketline dredging. Provided that it is operated according to the manufacturer's instructions the main problems are likely to be found in trying to match the intermittent flow of materials fed directly from the back hoe to the treatment plant. For example, if the back hoe has an average digging cycle of, say, 30 seconds, surges of material broken from the face will be dumped into the feed hopper at intervals of about 30 seconds. Each bucket load then has only 30 seconds to be ingested smoothly into the system before the next load arrives. This cannot be guaranteed because the dumped load may contain some surprises in the form of boulders, timber, lumps of clay, fragments of bedrock, etc. The result will be loss of production time and/or loss of gold in unslurried material rejected from the plant.

A second problem is posed by feed rate variations caused by digging at different levels between surface and bedrock. Digging rates in the bottom layers, including cleaning up at bedrock may be less than 50% of the rates achieved in the upper layers. This is significant operationally and the surge capacity must be large enough to iron out any fluctuations in the feed rate that might affect the smooth running of the recovery units. Generous degrees of over-design in the feed preparation section will usually pay handsome dividends in terms of increased gold recovery.

A third problem is limited headroom due to restricted machine dump heights. The usual loading arrangement is not suitable for a feed containing boulders and large trash because of the requirement of an additional screening facility ahead of the trommel. Locating a grizzly screen above the hopper to scavenge out the waste material might even out the flow to the plant but would certainly add to the required headroom. If this were greater than the dump-height of the backhoe, either a two-stage feeding arrangement or a larger back hoe would be needed.

Operational control

In any open pit workings, the individual bench trucking rates vary with depth because of increased ramp haulage distances. Hence, whilst an overall average of back hoe/truck performance may be assumed for a particular exercise, in practice the allocation of machines to benches is a day-to-day operational decision. Transport costs are strongly influenced by haulage distances and by any unnecessary double handling. Transport vehicles travel the same route many times and even small changes in layout can produce significant cost variations. This applies both to opening up operations and with the pit in full operation. Good operational control can be obtained in a number of different ways:

- Greater than 85% availability can be achieved by having an additional stand-by unit on hand at all times.
- Bench widths may be changed to suit the particular circumstances.
- A larger width requires more advance stripping and longer haulage distances but makes more low-strip ore available in case of major equipment breakdown.

- A shorter width requires less advance stripping and shorter haulage distances, however, it is more vulnerable to the temporary loss of some haulage or digging capacity.

Digging is usually extended for a sufficient distance into the bedrock (usually about 30 cm) to recover any gold that has lodged in cracks in the rocks or been carried down during mining operations. Recovery of 'bed-rock' gold, though difficult in some placer settings is an important aspect of the overall operation and is usually done better in dry mining operations than by dredging. Dry methods of mining allow cleanup problems to be assessed visually and attended to using procedures that may then be tailored to the particular situation. In most small-scale operations a considerable amount of gold can be won only by hand scraping in the crevices and hollows of hard, undecomposed bedrock.

Back hoe/floating treatment plant (doodlebug)

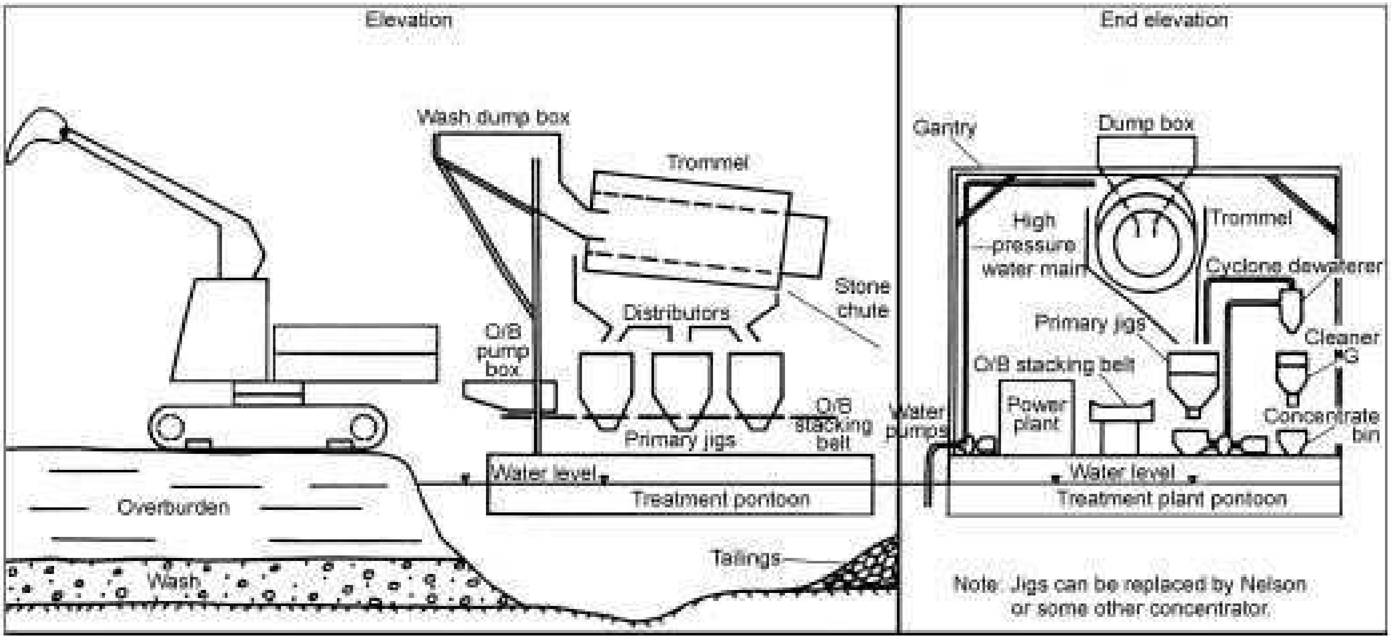
The 'doodlebug' operation (Fig. 7.29) evolved in California during the depression years of the 1930s. Drag lines were used to dig the gravels and load them into the hoppers of small floating wash plants which were moored alongside the banks. Drag lines may still be preferred for mining deeper deposits or deposits with unstable bank conditions where digging machines have to stand well back from the bank for safety reasons. However, back hoe excavators are now used in place of drag lines in most shallow operations because of their increased break out strength and greater accuracy when loading into a hopper.

The back hoe/floating treatment plant method is an extension of the doodlebug type operation. It is applicable mainly to small shallow deposits with maximum digging depths of 8–10 m. The system comprises a back hoe excavator and treatment plant mounted on the same pontoon so that digging, treatment and waste disposal can be carried out with minimum manpower requirements. It is sometimes possible at greater depths to conduct a back hoe/truck stripping operation from the bank and work on two levels with the back hoe/floating treatment plant working within the pond.

7.7 Miscellaneous dredger types

7.7.1 Clamshell dredger

Depths in excess of about 50 m below pond level call for a different type of dredging action from that of conventional bucketline dredgers. The clamshell dredger fits into this category, and some manufacturers offer some types of these units in direct competition with conventional bucketline dredgers. In Table 7.14 IHC Holland makes a cost comparison between their 'grab-miner' and a conventional bucketline dredger in the same service. Based simply upon throughput



7.29 Schematic arrangement hydraulic elevator and floating treatment plant (doodlebug).

Table 7.14 Cost comparison, bucket-ladder and grab-miner dredges – very approximate

	Bucket ladder	Grab miner
Yearly capacity, m^3	4×10^6	4×10^6
Hourly capacity, m^3/hr	700	700
Dredge depth below W.L. in m	35	35
Bucket size in ft^3	20	–
Line speed BPM	27	–
Grab	–	2×8 or 2×10
Investment:		
Dredge only F.O.B.	13–18	7–10
Plan 1×10^6 US$		
Power (kW) max.	1900	1000
ave.	1500	800
Crew:	Depend on PNG condition, say ten per shift	Depend on PNG condition, say four per shift
Maintenance;		
US$/year	1.3–1.8 million	0.7 million

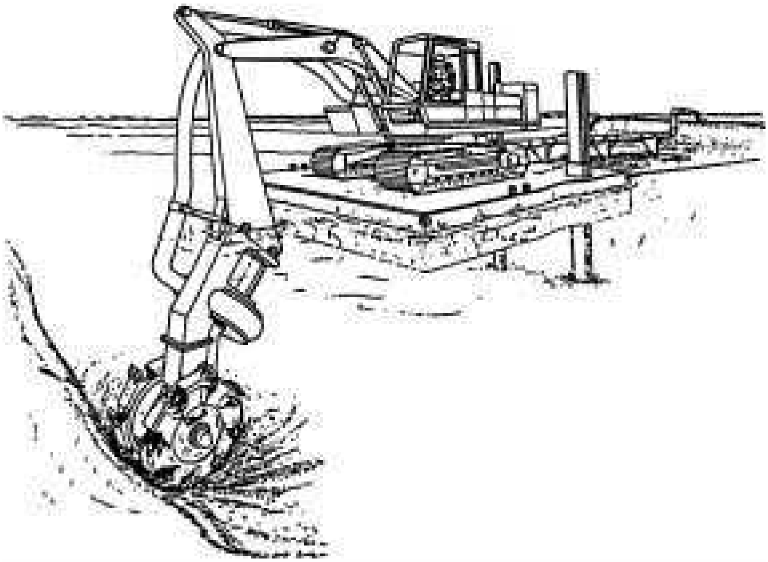
the comparison clearly favours the grab-miner. The number of grabs could be more or less depending upon the production requirements.

The clamshell offers the best present choice in deeper waters, say, 50–100 m despite its lack of specificity. For these and deeper waters, designs now on the drawing board may offer a new concept in which the clamshell is used for stripping in combination with a remote controlled underwater miner. One such device, the C.B.C. scraper, was designed by O and K (Orenstein and Koppel) for test dredging of manganese nodules in the Pacific Ocean in 1977. Another O and K system was developed for mining metalliferous muds in the Red Sea (Pearse, 1985). Some of these units may be brought into commercial use as soon as the existing environmental hurdles (e.g., pollution of marine feeding grounds) have been overcome.

There are, however, certain disadvantages attached to clamshell dredging that may in some cases outweigh its advantages. Important amongst these is an inability to mine closely and make good recoveries along the sea or pond floor. Clamshell dredgers can only compete with other dredger types at shallow (<50 m) depths where specificity is not required and stripped material can be disposed of easily.

7.7.2 Hoe-mounted excavator

A hoe-mounted excavator, as designed by Ellicott to compete with conventional back hoes in some ground conditions, is mounted as an attachment to a standard



7.30 Ellicott hoe mounted on excavator.

track mounted hoe for the continuous excavation and pumping of underwater material (Fig. 7.30). It avoids the cyclicity of conventional back hoe operations resulting from swinging, booming and bucket curling and provides a steady flow of dredged materials that have already been partly slurried. However, it suffers the normal constraints of the bucket wheel in being restricted to mining only that material which can be cut and passed through the pump. It is also power intensive because it elevates several times as much water as solids at a fast speed. A 60 m^3/hr operation is estimated to require 50 hp applied continuously to the bucket wheel to bring the material to the surface.

8

Metallurgical process and design

Fundamentals of the transport mechanism discussed in Chapter 4 relate to natural sedimentary processes involving the development of paystreaks in stream channels. Particles of diverse shape, size and distribution roll or slide along a natural streambed or are transport sorted, arranging themselves stratigraphically with coarse low-density particles further downstream of smaller high-density particles deposited upstream. In this chapter the transport mechanism is enlarged upon and expanded as an introduction to studies of sedimentation under controlled conditions of shear flow. A brief theoretical analysis of the basic physics of gravity concentration processes commences with a statement of formulae associated with flow conditions around a free-settling particle in a quiescent ideal flow. Models based on a diffusion mechanism and also incorporating Bagnold's normal stress are presented to help gain a better understanding of additional problems associated with predicting quantitatively, differences in the hindered settling and concentration of gold particles with a given size and density distribution in a gravity concentration device.

The key to good performance of a lateritic gold treatment plant is the amount of attention given to scrubbing, screening and size classification in the feed preparation section. Design calculations, which call for a hypothetical balance of all materials (solids and water) entering into and leaving the plant, nominate the quantities involved at each stage of treatment. Recovery plant types can then be selected in which the controlling factors balance the probable behaviour of the gold. The best-known separation devices, sluices, jigs, tables, and more recently cones, spirals and centrifugal separators have been developed from research over the past 70 years. Each type of separator has a particular size range of gold particles within which it operates economically but there are many constraints to the recovery of gold smaller than 150–200 microns. Centrifugal separators offer hope for future gravity separation of gold particles down to a few microns in size. Chemical leaching processes, which usually have a gravity component designed to scalp out coarse gold released from crushing and grinding circuits, are finding increased application for the treatment of lateritic/saprolitic ores in which gold occurs both in a primary form and as supergene enrichments.

8.1 Theory of gravity concentration

The settling rates of individual particles in free fall are governed by their size, shape and density but such rates are modified by the presence of other particles and by conditions of hindered settling either naturally or artificially induced.

8.1.1 Free settling of spheres

The theoretical analysis of gravity concentration processes usually begins with an examination of the forces acting upon a single particle settling in ideal conditions, i.e., in an infinite still viscous medium. A simplified equation of motion can then be written from Newton's second law of motion ($F = Ma$):

$$Mdv/dt = (m - m^1)g - R \qquad 8.1$$

where m is the mass, dv/dt the acceleration of the particle, m^1 the mass of the displaced fluid, g the gravitational acceleration and R the drag force for the instantaneous particle velocity v. R is conventionally given by:

$$R = 0.5pv_t^2AC_d \qquad 8.2$$

where A is the projected area of the particle and C_d is a dimensionless drag coefficient.

For a spherical particle with diameter D and density σ settling in water of density p, the mass of the particle and displaced fluid are $\sigma\pi D^3/6$ and $p\pi D^3/6$ respectively, eqn 8.1 can be rewritten:

$$\sigma\pi D^3/6\ dv/dt = \pi D^3/6\ (\sigma - \rho)g - 0.5pv^2D^2/4\ C_d \qquad 8.3$$

The settling particle ultimately reaches a maximum or terminal velocity v_t, at which the acceleration dv/dt is zero, and eqn 8.3 becomes after rearrangement:

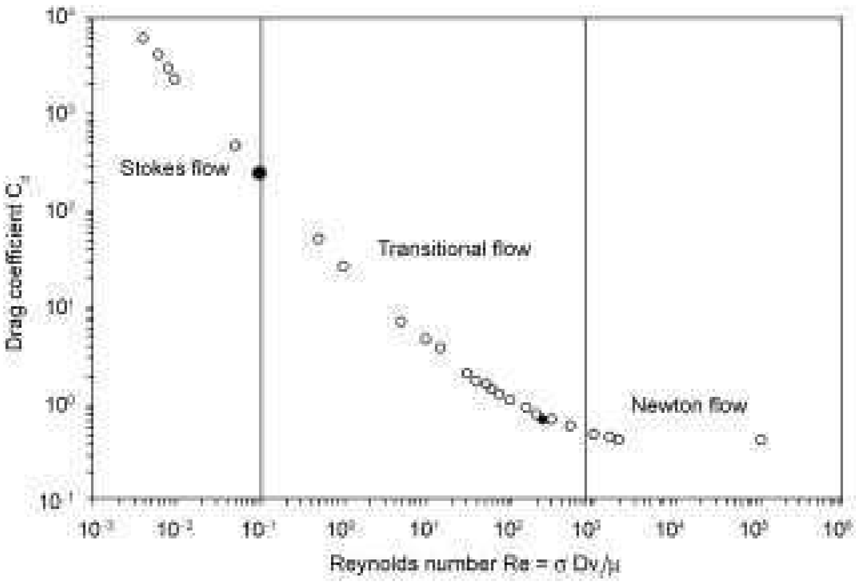
$$V_t - \{4/3(\sigma - p)/pC_dgD\}^{0.5} \qquad 8.4$$

The value of the drag coeficient C_d is dependent on the flow conditions around the particle, which are defined by Reynolds number Re $= \sigma Dv_t/\mu$, where μ is the fluid viscosity. The functional relationship between C_d and Re is shown in Fig. 8.1, the standard drag curve for spherical particles (Clift *et al.*, 1978). This figure shows that the flow conditions around a free settling particle can be conveniently divided into three regimes:

- Stokes flow (fine particle settling)
- transitional flow
- Newton flow (coarse particle settling).

Stokes flow regime

Viscous forces prevail within the fine particle size range, and the resistance to settling is proportional to particle diameter, flow velocity and viscosity. Stokes



8.1 Standard drag curve for spheres.

analytical solution for the magnitude of the drag force R is valid only for very low values of Reynolds number ($<\sim 0.1\,\mu$m) down to the size at which Brownian movement maintains particles in irregular and erratic suspension. This solution gives $R = 3\pi\mu Dv$ and substituting for R in eqn 8.2 and solving for C_d gives $C_d = 24/\text{Re}$ which, when substituted into eqn 8.4, leads to Stokes law of settling:

$$V_t = (\sigma - \rho)gD^2/18\mu \qquad 8.5$$

For quartz density particles, the Reynolds number condition limits D to about $50\,\mu$m whereas if the particles are of gold, the high density is significant and D is reduced to about $20\,\mu$m. Note that Stokes law applies closely to slime settling and to the thickening of fine quartz spheres up to about $50\,\mu$m and, approximately, to most sediments up to about $100\,\mu$m.

Newton flow regime

Viscous forces become negligible for higher values of Re ($\sim 750 < \text{Re} < \sim 3.5 \times 10^5$) and it is found experimentally that C_d is sensibly constant with a value of approximately 0.445 (Clift *et al.*, 1978), and eqn 8.4 can be re-written:

$$V_t = k\{gD(\sigma - \rho)/\rho\}^{0.5} \qquad 8.6$$

where $k = (4/3C_d)^{0.5}$. This equation is often referred to as Newton's law of settling and is applicable for d greater than about 2.5 mm for quartz density particles and 1.1 mm for gold.

Transitional flow regime

Viscous forces vie with inertial forces for dominance in the intermediate (transitional) size range in which spheres settle under conditions in which flow around the particle passes from the laminar to the turbulent state. The standard drag curve for spheres shows that the Stokes and Newton flow regimes are connected without discontinuity by the transitional flow regime. In this regime, the relationship between C_d and Re, although unique, cannot be used to calculate the terminal velocity V_t of a particle of given size D since both size and velocity occur in each of the dimensionless-groups. Holtham (1991) notes that this poses difficulties because the size of gold particles in alluvial gold sediments ensures that significant numbers of them will fall within this regime. A number of methods, however, are available to overcome the difficulty of determining V_t in the transitional flow regime (e.g. Clift *et al.*, 1978; Heywood, 1962; Holland Blatt, 1972).

Particle terminal velocity

Methods used to calculate terminal velocity in the transitional regime can also be generalised to include the Stokes and Newton regimes. Holland-Batt (1972) devised one such method, which may also be easily generalised for accelerations other than those due to gravity. Terminal velocities using the algorithm developed by Holtham (1991) can be calculated for a number of different density particles as shown in Table 8.1.

It is found experimentally that for Re values greater than about 270, the terminal velocity of a free settling sphere can fluctuate from the calculated value as the sphere follows a zigzag or spiral trajectory pattern (Holtham, 1991). Corrections to the value of the drag coefficient are discussed in Clift *et al.* (1978).

8.1.2 Hindered settling of spheres

Consideration of the free settling of individual spherical particles under ideal conditions is unrealistic as a description of practical gravity concentration where

Table 8.1 Calculated terminal velocities of quartz spheres under free and hindered settling conditions (from Holland-Blatt, 1972)

Size (μm)	Terminal velocity (cm s^{-1}) Quartz	Rutile	Cassiterite	Gold
35	0.06	0.11	0.20	0.61
500	7.78	12.28	18.57	38.09
5000	51.53	72.33	98.63	172.98

particle concentrations are normally high. Under such conditions, there is a reduction in terminal settling velocity, the reduction being a function of the volume concentration of the particles. Williams and Amarasinghe (1989) have reviewed some 27 models in the literature describing hindered settling and their conditions of application.

Empirical modifications to Stokes' and Newton's laws

Gaudin (1939) proposed modifying Stokes' law by empirical correction factors in order to predict the terminal velocity of similar sized (fine) particles under hindered settling conditions. The correction factors (which were functions of the volume concentration of solids in suspension) accounted for the reduction in fluid cross-sectional area and the increase in apparent fluid viscosity and density. For hindered settling of large spherical particles in a suspension of very fine particles (which essentially behave as part of the fluid), Gaudin proposed Newton's law (eqn 8.6) with the fluid density ρ replaced by the suspension density ρ_s as a means of estimating V_tC for the larger particles.

The Richardson and Zaki correlation

Richardson and Zaki (1954) found experimentally that the terminal velocity of arrays of uniform spherical particles could be described by:

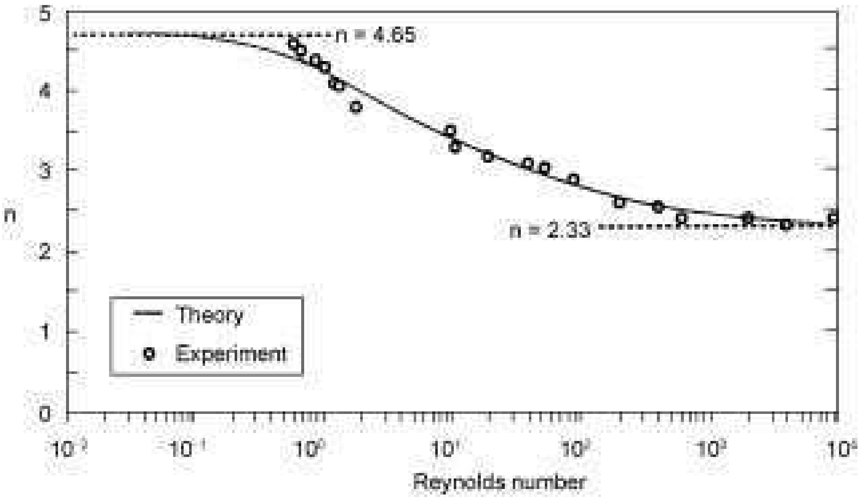
$$V_tC = v_t(I - C)^n \qquad 8.7$$

where V_tC is the terminal velocity of uniform particles under hindered settling conditions, and n is a positive exponent related to the particle Reynolds number.

Maude and Whitmore (1958) showed theoretically that $2.33 < n < 4.65$ and their theoretical curve (Fig. 8.2) agrees quite well with Richardson and Zaki's experimental results. The Richard and Zaki correlation can be readily incorporated into computer programs for calculating terminal velocity as shown in Table 8.1. However, eqn 8.7 processes should be used with caution when used to determine the settling velocity of particles in the non-uniformly sized suspensions that occur in gravity concentration because they were originally derived for uniformly sized suspensions.

8.1.3 Non-spherical particles

The above equations of motion have been developed for spherical particles. In practice, the particles will be irregularly shaped and this will increase the drag and reduce the terminal velocity. The problem of describing and accounting for particle shape is discussed in Clift *et al.* (1978). In the gravity concentration literature, the tabular data of Heywood (1962) is frequently used to make corrections to the drag coefficient for the shapes typical of natural mineral particles.



8.2 The exponential 'n' in Richardson and Zaki's equation for hindered settling (from Maude and Whitmore, 1958).

The Heywood shape factor

Heywood defined a volume shape factor (K).

$$K = \pi D^3/6\ D\rho^3 \tag{8.8}$$

where D is the diameter of a sphere with the same volume as the particle, and $D\rho$ is the diameter of a circle with the same area as the projected profile of the particle lying in its most stable position. The shape factor K has the approximate values shown in Table 8.2. Heywood suggests that although most mineral particles have shape factors in the range 0.20–0.25, gold particles, which tend to flatten into thin flakes, are likely to have lower values.

For the most part, sizing analyses in gravity concentration are carried out by projected area diameter. British Standard BS3406 (Part 4) 1963 suggests multiplying sieve aperture size by a factor of 1.40 to obtain projected area diameter. This factor implies a shape factor of 0.2 and should be applied with caution if the particles are of extreme shapes. Heywood's tabulated data is also readily incorporated into computer programs enabling the terminal velocities of non-spherical particles to be calculated, as shown in Table 8.3.

8.1.4 Particle movement in shear flow

The quiescent fluid condition is inappropriate to gravity concentration devices such as the spiral, cone, sluice, etc., where particle stratification takes place in a shear flow. The assumption that such particles will simply settle to the bottom at

Table 8.2 Heywood's volume shape factor K

	K
Elongate needle-like or very flat	0.1
Tetrahedral shaped	0.2
Sub-angular	0.3
Rounded	0.4
Spherical	$\pi/6 = 0.524$
Cubic	$(\pi/4)^{3/4} = 0.696$

Table 8.3 Terminal velocity of spherical and non-spherical quartz particles (shape factor $K = 0.25$, Heywood's correction for shape applied)

Sieve size (μm)	Projected diameter (μm)	v_t Sphere	v_t Particle
25	34	0.06	0.04
500	689	7.78	5.24
5000	6891	51.53	24.51

After Holtham (1991).

their terminal velocities is incorrect, indeed the finer particles may not settle at all (Holtham, 1991).

Motion of a single particle

Gaudin (1939) analysed the forces acting on a single particle rolling or sliding in a two-dimensional laminar film along a smooth bed. In his analysis the velocity distribution of the fluid film at any depth y can be described by the laminar velocity profile equation:

$$V = \tau h/\mu\{y/h(1 - y/2h)\} \qquad 8.9$$

where τ is the bed-shear stress, ρ the fluid viscosity and h the total depth. Using Stokes' law (eqn 8.5) combined with eqn 8.9, he developed an analytical expression for the downstream distance travelled before a single (free settling) particle settling at its terminal velocity reaches the bed. This demonstrated that smaller low-density particles travel further downstream than larger (but still obeying Stokes' law) high-density particles.

Gaudin also analysed the transport of particles qualitatively at the bottom of the thin laminar flowing film by considering the action of fluid gravitational and frictional forces. He noted that particles rolling or sliding along the bed are transport sorted, arranging themselves with the coarse low density particles furthest downstream, and the fine dense particles left further upstream. The analysis made a number of assumptions that limited the general applicability of the results as a quantitative description of particle movement in shear flows.

Using a multi-exposure photography technique, Francis (1973) followed the movement of a single particle over a fixed rough bed, acknowledging that this approach ignored the effects of particle concentration. He divided the particle motion into three modes:

1. rolling or sliding, in which the particle always maintains contact with the bed
2. saltation, in which the particle made jumps up into the fluid following a ballistic trajectory before once more coming into contact with the bed
3. suspension, in which the particle made longer and higher trajectories that differed from those in saltation; the upper part appeared wavy due to support from turbulent eddies.

With the mode adopted by the particle being established by the transport stage:

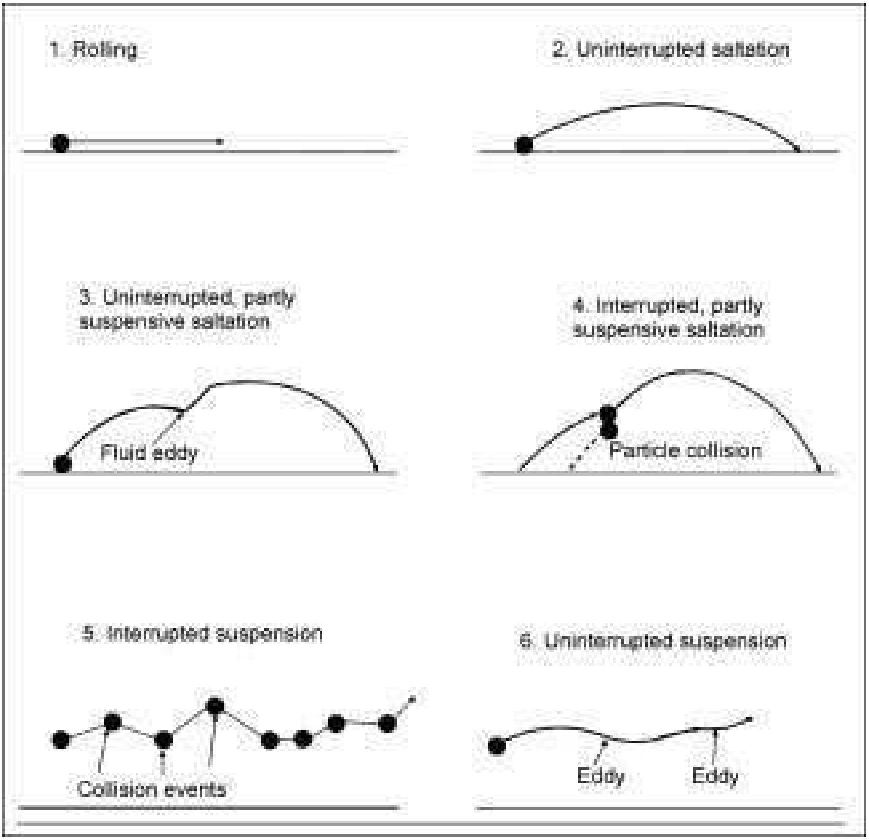
$$\text{Transport stage} = (u^*/u_0^*) \qquad 8.10$$

where u^* is defined as $(\tau/\rho)^{0.5}$ with τ the bed-shear stress and ρ the fluid density is the shear velocity of the observation, and u_0^* is the critical shear velocity for the initiation of motion if the particle forms part of a co-planar fully mobile bed. The value of u_0^* was determined from Shields' criterion (Yalin, 1977). Francis, by the simple expedient of increasing the fluid viscosity, demonstrated that saltation persisted in laminar flow hence this mode of transport cannot be attributed to fluid turbulence.

Multiple particle motion

Leeder (1979) modified and added to Francis' original modes as illustrated in Fig. 8.3:

1. rolling as before
2. uninterrupted saltation (the saltation of Francis)
3. uninterrupted partly suspensive saltation in which the normal ballistic saltation trajectory is modified by the effect of fluid turbulence (the suspension of Francis)
4. interrupted partly suspensive saltation, as mode 3 above, but with the addition of upward acceleration due to inter-particle collision
5. interrupted suspension, in which the particle is maintained in suspension by both fluid turbulence and inter-particle collisions
6. uninterrupted suspension, in which there is true suspension of the particle by fluid turbulence.



8.3 Particle transport modes (after Leeder, 1979).

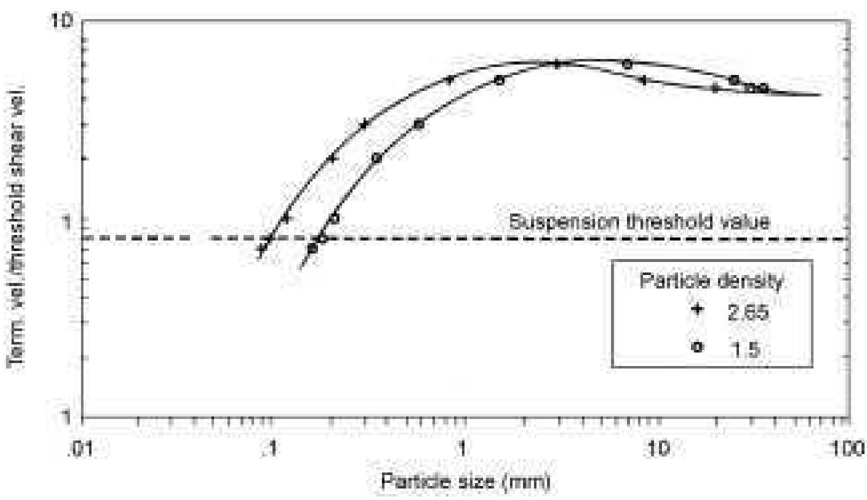
It is important to note that streambed and water surface conditions change constantly; each particle is affected by its neighbours and, because of local velocity changes due to their movement, all six modes can exist simultaneously. The particles will have different values of critical shear velocity u_0^* but the shear velocity u^* at a particular point will be common to all particles, resulting in different transport modes for different particles. Thus large dense particles may be in the rolling or saltation modes, while small dense particles are in the true suspension mode.

Modes 1 to 5 are all modes of bed-load transport. Particles travelling in those modes are operated on by a combination of solid and fluid momentum transfer. Modes 3 to 6 rely upon the effect of fluid turbulence, which is suppressed by high concentrations of particles. Suspended load (mode 6) can be defined as those particles held in true suspension against gravity by fluid momentum transfer alone, i.e., by random eddy currents of turbulence having velocity components normal to the bed greater than the terminal velocity of the particles relative to the surrounding fluid. Theoretically, particles with low particle Reynolds numbers (based upon relative velocity and size) tend to suppress turbulence, whereas the presence of particles with Reynolds numbers greater than about 400 tend to enhance turbulence.

8.1.5 Limiting particle size for unsuspended transport

Bagnold (1966) developed a theoretical criterion for establishing the onset of suspended transport, showing that a particle should become liable to suspension at a transport stage of approximately $0.8\, v_t/u_0^*$. Figure 8.4 with u_0^* calculated from Shield's criterion, shows that for quartz particles greater than about 700 μm diameter, v_t/u_0^* has an approximate constant value of 4.5. Such particles might thus become suspended at a transport stage of about 3.6 (see also Francis, 1983). It can also be seen that the ratio v_t/u_0^* decreases rapidly as the particle size decreases, until for quartz particles finer than 100 μm some suspension should occur at the threshold of bed movement (Holtham, 1991).

Applying Bagnold's suspension criterion to gravity concentration devices suggests that 100 μm diameter quartz particles will be transported entirely in mode 6, turbulent suspension. However, turbulence is considerably dampened by the presence of solids, resulting in delay of the onset of turbulence until a much higher transport stage is reached.



8.4 Suspension threshold value for quartz density particles (Holtham, 1991, redrawn from Bagnold, 1966).

The Bagnold effect

Transport modes 4 and 5 occur with high particle concentrations and the rheology of suspensions of large solid particles in fluids at high shear rates is of interest to many other disciplines, as well as mineral processing. In all cases where particle concentrations are high (e.g., practical gravity concentration devices and the bed-load of streams) there is a significant increase in the shear resistance of the particulate fluid and considerable distortion of the velocity distributions compared with those of the fluid phase alone.

Bagnold (1954) measured the shear and normal stresses developed over a range of shear rates when neutrally buoyant wax spheres suspended in water were sheared in a coaxial-cylinder Couette flow apparatus. Depending upon the value of a dimensionless shear rate group B, the Bagnold number, analogous to Reynolds number, Bagnold defined two limiting flow regimes, macroviscous ($B \leq 40$) and particle inertia, ($B \geq 450$) separated by a transition medium ($40 < B < 450$), where B is given by:

$$B = \sigma\lambda^{0.5}D^2/\mu \; du/dz \qquad 8.11$$

where du/dz is the mean shear rate, σ is the particle density, D is the particle diameter, μ is the fluid viscosity; the linear concentration, λ, is the ratio of the particle diameter to the mean free separation difference between the particles. As a function of the volume concentration C, λ is given by:

$$\lambda = 1/(C^*/C)^{1/3} - 1 \qquad 8.12$$

where C^* is the maximum possible particle concentration. However, Savage and McKeown (1983) have suggested from more recent work that the boundaries of B defining the boundaries of the different regimes may differ from those given by Bagnold.

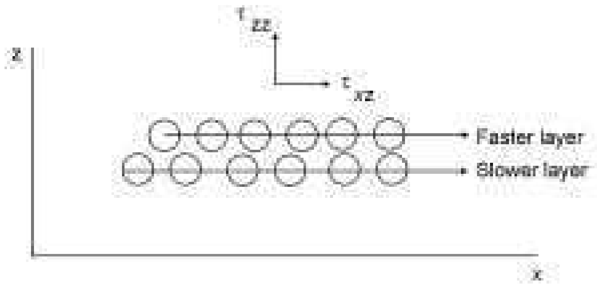
In the macroviscous regime, stresses are transmitted by interstitial fluid friction and are therefore dependent on fluid viscosity, but independent of fluid density. Based on experimental observations, Bagnold proposed the empirical relationships:

$$\text{Shear stress } \tau_{zx} = \lambda^{1.5}\mu \; du/dz \qquad 8.13$$

$$\text{Normal or dispersive stress } \tau_{zz} = 1.3\tau_{zx} \qquad 8.14$$

i.e., the stresses are linearly dependent on shear rate and are independent of particle size and density. Bagnold attributed the normal stress to an anisotropy in the spatial particle distribution.

In the particle-inertia regime, Bagnold (1966) argued that the interstitial fluid plays a minor role and the dominant effects arise from the succession of particle-particle collisions as the particles of one layer overtake those of an adjacent slower layer (Fig. 8.5). By considering the mean paths of particles undergoing rapid shear, Bagnold recognised that both the momentum transfer per collision



8.5 Particle-particle interaction in shear flow (after Bagnold, 1966).

and the frequency of particle collision are proportional to the mean shear rate, resulting in normal and shear stresses quadratic in the mean shear rate.

Experimentally verified, this quadratic stress-dependence as well as the strong dependence of the stresses on the volume concentration of particles for $B \geq 450$ and $1.4 \leq \lambda \leq 14$, Bagnold proposed the following relationships:

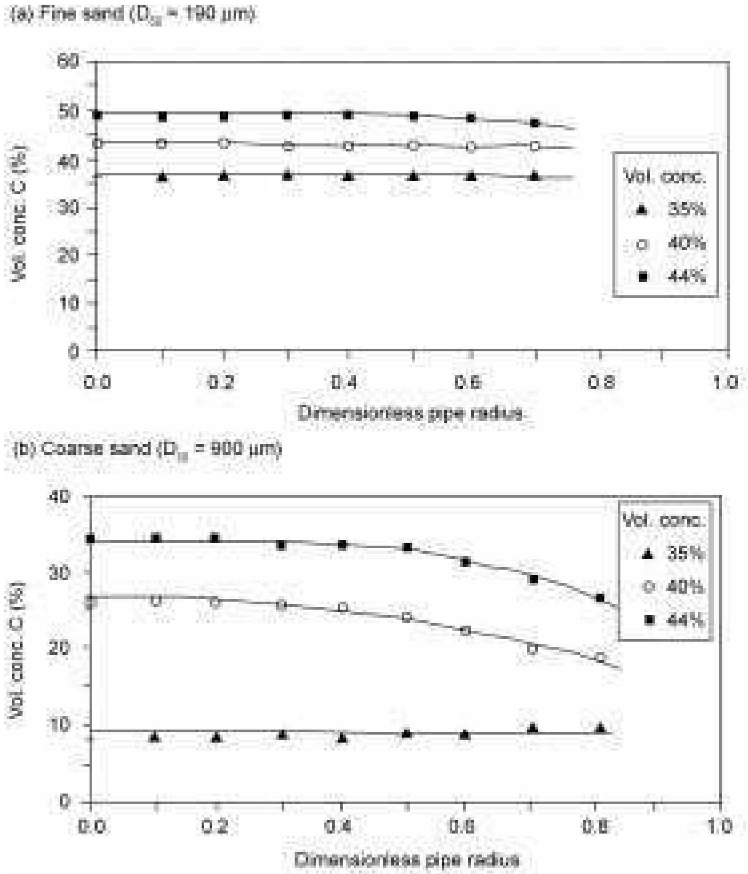
$$\text{Normal stress } \tau_{zz} = \sigma(\lambda D)^2 (du/dz)^2 \quad 8.15$$

$$\text{Shear stress } \tau_{zx} = \sigma(\lambda D)^2 (du/dz)^2 \quad 8.16$$

The stresses are thus independent of fluid viscosity but dependent upon particle size, density and concentration and hence likely to be significant in gravity concentration processes. Bagnold noted that when a range of particle sizes of constant density is sheared together in a gravity flow, the larger particles should tend to move towards the zone of least shear strain (the free surface) and the smaller particles towards the zone of greatest shear strain (the bed).

Although Bagnold did not explicitly consider particles of different density, the likely significance of the Bagnold force for the segregation of particles has been widely noted. In applying Bagnold's results to natural sizing and gravity concentration of quartz and heavy mineral assemblages, Sallenger (1979) found that the data suggested that the Bagnold normal stress could be contributing to the concentration process.

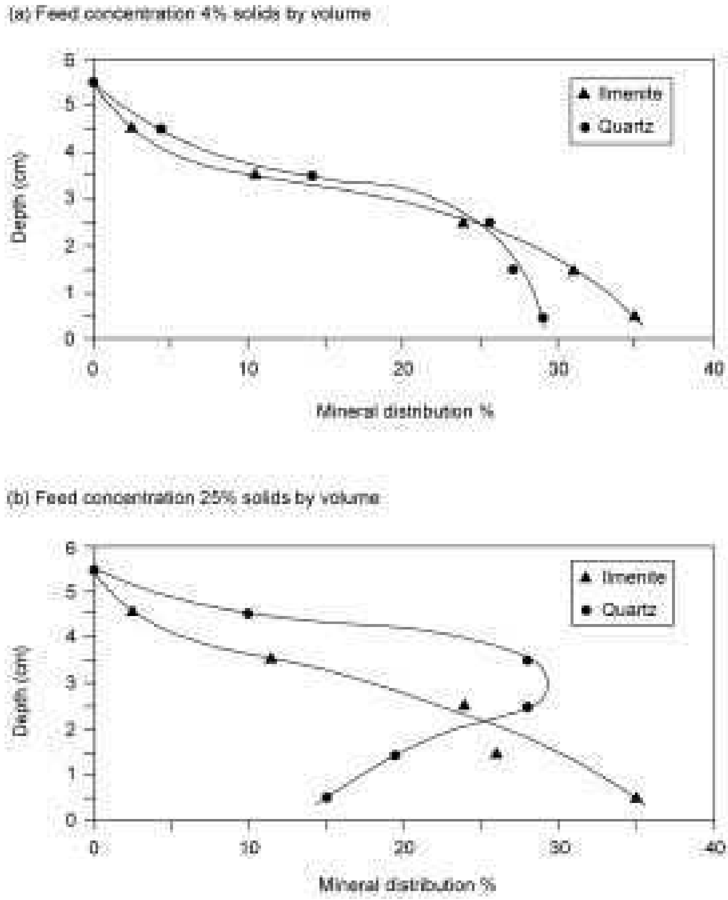
Bagnold's results have stood for some 30 to 40 years under the conditions for which they were formulated: steady, uniform, simple shear flow of neutrally buoyant spherical particles. However, in cases involving more complex flows, application of Bagnold's relationship lead to unrealistic constraints. In particular, in Bagnold's formulation, the stresses disappear for 'vanishing' mean velocity gradient because there is no source for particle velocity variations other than mean shear (Hanes and Inman, 1985). The results of experimental work by Hanes and Inman (1985) and Savage and McKeown (1983) confirm that shear and normal stresses are developed in particulate fluid flow. At sufficiently high shear rates the stresses are quadratically dependent upon the mean shear rate at a given volume concentration, which supports Bagnold's hypothesis that the stresses result from particle collisions (Holtham, 1992).



8.6 Lateral particle distributions (after Leeder, 1979).

Lateral concentration variations in horizontal slurry pipelines, measured by Nasr-el-Din *et al.* (1987), provide evidence of the significance of Bagnold stresses in a realistic engineering application. By thereby eliminating the effect of particle immersed weight; clear evidence was shown of particle migration towards the pipe centreline for distributions of both fine and coarse sand (Fig. 8.6). The authors attributed this to the effect of the normal or dispersive stress component, the effect being more pronounced with the coarser particles as expected from Bagnold's analysis.

Of greater relevance to the significance of the Bagnold normal stress in gravity concentration, pinched sluices and Reichert cones give adequate segregation only at high densities in which a significant normal stress would be developed. The results of a sampling programme on a 'pinched' sluice are summarised in Fig. 8.7. The profiles for ilmenite and quartz suggest the development of normal stress relationships only at high concentration as theoretically required. The feed



8.7 Profiles for ilmenite and quartz in a pinched-sluice sampling programme (Holtham, 1991 from Abdinegro and Partridge, 1979).

particle size was not completely specified but Holtham (1990) suggests that it is possible to estimate the value of the Bagnold number B if a D_{50} of 500 is assumed. For the conditions under which the sluice was operated, the value of B falls within the macro-viscous (low feed concentration) regimes since the mean shear rate was quite low (typically only $30\,s^{-1}$). The value of B would decrease if the feed was in fact finer; Bagnold himself suggests that the dispersive force effect in the particle-inertia regime should disappear when the particle size is less than about 200 μm.

8.1.6 Modelling of sediment transport

Yalin (1977) has reviewed a number of models that relate the bulk flow rate of sediment to the prevailing hydrodynamic conditions for solids, generally assumed to be of quartz density. The requirements are different for gravity concentration modelling. Instead of a description of the bulk flow rate through a device, what is required is a model capable of predicting the particle concentration distributions within the fluid as a function of particle size and density and pulp flow rate and solids concentration.

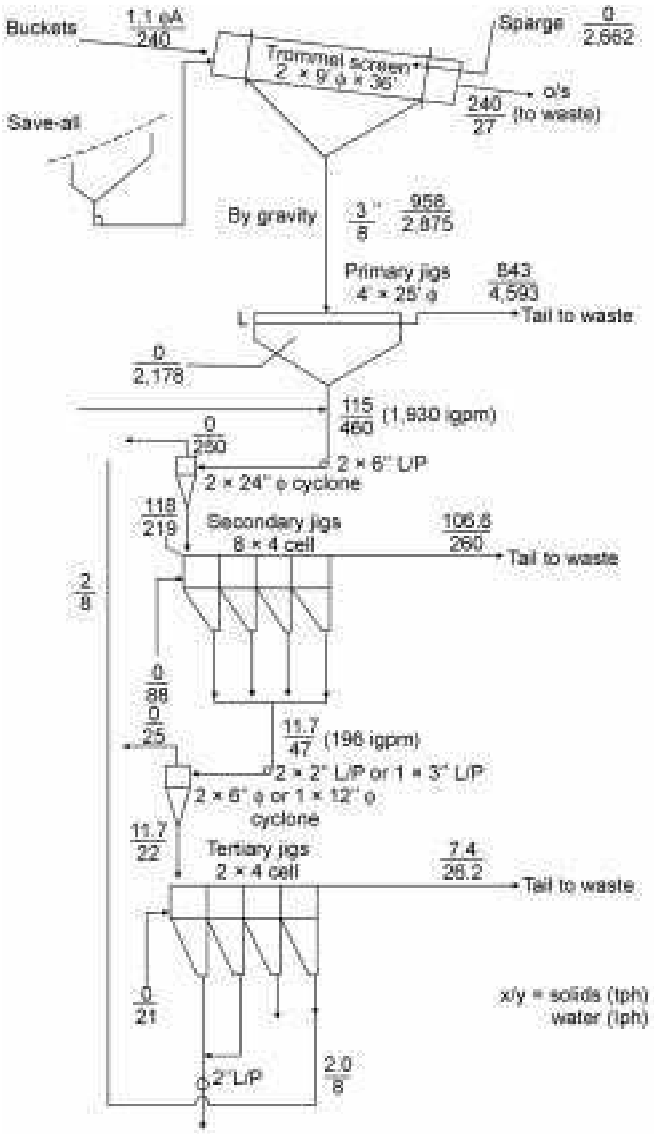
The classical approach to the modelling of particle transport by suspension in turbulent flow (the suspended load) is through a diffusion process (for example Allen, 1983). Wilson and Pugh (1988) combined the diffusion model with Bagnold's normal stress model and generated theoretical concentration distributions with depth for various particle sizes. Good agreement was obtained between predictions from the model and the experimental measurements of Nasr-el-Din. Nevertheless, there is no general theory for hindered settling of a suspension consisting of a range of particle sizes and densities. Also little is known of the important practical case of particles settling in a shear flow. Indeed, the empirical modifications proposed to the terminal velocity equations have very little quantitative value as descriptions of gravity concentration processes.

Bagnold's 1954 paper has been widely cited in the gravity concentration literature as having relevance to the particle stratification problem but the findings do not appear to have been applied other than qualitatively. Only a few authors (e.g. Burch, 1962 and Holtham, 1991) have attempted to evaluate the Bagnold number B for a gravity concentration device and hence determine which flow regime applies to that device. Also lacking are data that describe the extent to which the Bagnold force contributes to the particle sorting process by either size or density.

8.2 Flow sheet and materials balance

The flow sheet component of a materials balance evolves from bench and pilot scale studies of the behaviour of the ore materials during preliminary testing. It nominates the type of plant and equipment and the order in which the various fractions are to be processed, and as shown in Fig. 8.8, it predicts the quantities of solids and water likely to be involved at each stage of the process. In keeping with practical expectations of fluctuating feed rates and variable machine efficiency in the production mode, the basis for design is necessarily a 'weighted' average of the fluctuations but with some degree of flexibility. The effectiveness of the design usually depends upon how quickly the plant adjusts to changes in the quantity and quality of the feed.

The materials balance in the 'production mode' is an average of results from each production period. The flow rates ascribed to materials passing through the plant are predictions based upon interpretations of bench scale test data. The overall quantity of water entering into the plant is obtained from dams set up for the purpose and includes retained water in feed materials and make-up water



8.8 Materials balance for fluvial ore processing showing quantities of solids and ore involved at each stage of treatment.

from slurry settling dams. All other measurements will be approximations based upon the analysis of samples taken at all important points of intake and discharge. The feed rate of material for scrubbing and slurrying is geared to the proposed project requirements. Rates of flow (solids and water) at each significant stage of treatment are constructed from parameters derived from piloted predictions of plant behaviour during the final ‘ore resource’ assessment.

8.2.1 Ore type and location

Two main classes of ore, fluvial and lateritic, are represented by the geology of their surroundings. Fluvial gold ores are formed in networks of stream channels having steep sides and slopes in their upper reaches and lower parts that widen and flatten out with extensive beds of gravel and other sediments. Lateritic gold ores are formed from weathered rock in source areas where the parent materials have undergone mainly chemical and biological change.

Fluvial gold ores

The nature of raw plant feed as derived from fluvial gold concentrations is determined by the type of material dug at any one time and hence is most variable in type, form and gold content. Particle size is reduced progressively with distance of travel and time, paystreaks are common when erosion and transport of gold-bearing sediment occur, but end abruptly with sudden changes in the conditions of flow. A rough distinction can be made between overburden and ore but wet deposits are typically dredged to a predetermined pattern and cannot easily differentiate closely between high and low grade ore horizons in their daily routine. Inevitably, this will include a proportion of finer grained material that moves essentially parallel to it and from sources other than the paystreaks (e.g. bedrock, overland flow, tributaries and caving from the banks).

Palaeoplacers usually provide the most diverse types of mill feed. In addition to normal depositional complexities, the older deposits have been subjected over time to the inter-granular deposition of transported clays, possible dissection, slumping and varying degrees of lithification. Simple diagenetic effects are seen in the adhesion of thin films or coatings of extraneous materials to the surfaces of individual grains. More intensive diagenetic processes such as the lithification of Witwatersrand-type banket deposits have resulted in a cemented and compacted ore that must be crushed and ground to release the gold for recovery by chemical leaching. In between these extremes, varying degrees of alteration require some elements of the raw feed to receive more rigorous pre-treatment than does feed from other sectors.

Optimisation of plant feed depends upon the ability to access ore freely from all sections of the face at all times. For example:

- Refractive materials can be blended with easily processed materials and fed gradually and selectively into the plant in a controlled fashion.
- Very clayey material can be mixed with coarse gravels for better slurrying.
- High-grade ore can be blended with low-grade ore so as not to periodically overload and underfeed the cleaner circuit.
- In extreme cases, very difficult material can be stockpiled and treated separately under specially designed conditions.

Lateritic gold ores

Lateritisation is an associated process in areas of moderate to low relief, particularly in humid tropic regions where oxides of iron and aluminium form as a blanket of indurated material over the weathered rock below (Chapter 3). Variations in lithology, drainage and topographic relief lead to different patterns of alteration and variable free gold content; gold concentration occurs typically in generally flat-lying enrichment zones contiguous with the ferruginous horizon. Constituents of the ore zones are lateritised mixtures of clay, silt and sand with unweathered fragments of rock. The degree of alteration and gold liberation and modification is a function of depth of weathering and time. Gold is concentrated *in situ* by the differential removal of some of the products of weathering. In a shallow, eluvial-colluvial orebody only a part of the gold is likely to be in a free state; some may be locked up in sulphides such as pyrite, arsenopyrite and copper pyrites or be still enclosed in unbroken rock. Under certain conditions (e.g., where descending acid solutions are neutralised or made alkaline) some gold may be remobilised and reprecipitated at the water table. Regoliths in deeply weathered terrains are highly variable in nature and raw feed materials from different parts of a residual orebody vary widely in nature and gold content. The surface rock may be almost completely converted to soil and usually contains the highest values of free gold.

Lateritic materials harden on exposure to the atmosphere, typically forming a tough indurated layer of hard-pan (duricrust) or a mixture of oxides of iron and aluminium in a generally clayey matrix. Differences in distribution of calcrete are attributed to the different conditions of weathering and landscape development (Anand *et al.*, 1989). The gold is commonly fine grained and of high fineness but may also contain coarse-grained primary and secondary gold. Goethite and haematite, usually with quartz and minor kaolinite dominate the mineral assemblage of the ore. Gibbsite is abundant in presently humid areas whereas in arid areas, calcrete materials may be present. Clay minerals are abundant in the underlying mottled zone. In some orebodies the transition from weathered to un-weathered rock is abrupt; in others the textural contrast is gradual.

8.3 Feed preparation

Recovery plant efficiency depends primarily upon the capability of the feed preparation section to produce a balanced flow of size-classified feed materials for treatment. Scrubbing and slurrying processes are designed to meet the needs of the recovery units by sizing, desliming and distribution. Operational control is strongly influenced by the amount of water used at each stage of processing.

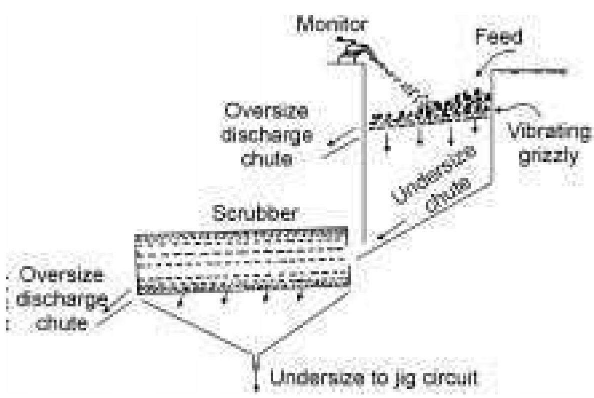
8.3.1 Rejection of trash and boulders

Trash comprises any organic or inorganic material that might interfere with the performance of plant units by causing excessive maintenance, blinding screen apertures and blocking flow lines. Surface trash comprising roots and other vegetable matter is dealt with using either hand or machine methods of clearing. More difficult problems are associated with deep-seated trash for which individual methods of handling are different for different methods of mining. Large bucketline dredgers may push some boulders and buried timber to one side. In other cases dredging may have to be conducted around large obstacles if they cannot be removed physically by drag lines or other such excavators. A prior stage of scalping is usually required to reject large oversize waste material ahead of the scrubbing trommel. The grizzly is placed ahead of the scrubbing trommel to reject smaller waste that might cause blinding of the trommel screens. Fixed (stationary) grizzlies constructed from heavy-duty steel grating with bars made from waste lengths of steel section (e.g., old tramline) are suitable for small operations. Figure 8.9 is a section of a typical scrubbing–sizing arrangement.

Difficult scrubbing applications may require more rigorous treatment than can be given by a stationary grizzly and larger operations require shaking or vibrating grizzly types. Vibrating grizzlies help to avoid blockages and reject more material at reduced aperture, because their shaking movement tends to keep the feed materials in constant motion. Whereas fixed grizzly bar spacings may be 150 mm to 200 mm, vibrating grizzly apertures are usually around 100 mm. Jet water usage may vary from 0.5 to 1.5 m^3/h/tonne of feed, depending largely upon the clay content.

8.3.2 Scrubbing

A belt conveyor or other transfer system (e.g. chute) carries the grizzly undersize to the scrubber. The purpose of the scrubber is to disintegrate clay-bound wash



8.9 Section of typical scrubbing–sizing arrangement.

materials and recover all undersize material while discharging the oversize material to waste in a clean condition. Mechanical puddling devices such as log washers and mechanical puddlers are less favoured in modern plant than in the past because of high maintenance requirements and low mechanical efficiency. For the most part, water jets directed onto the mass of the material as it passes through the scrubber provide much of the required scrubbing energy as well as supplying the medium for fluidisation. High-pressure jets are played over the material being processed and lifters are installed to help agitate the material being processed. Indurated lateritic materials such as hardpan may have to be broken down in a separate crushing-screening section equipped with hammer mills or other impact breaker types. An even solids flow rate is important and a balance must be struck between water pressure and volume to achieve an optimum slurry pulp density.

The scouring action of a jet is a function of its momentum (mass times velocity) and time (duration of the scouring action). Water jets direct the main hydraulic forces onto the upward moving side of the scrubbing cylinder where the unbroken material is thickest and starts to tumble downwards. At this stage, the lumps have no cushion of slimes to protect them against the jet; the tumbling action provides additional energy for scouring and continually exposes fresh surfaces of each unbroken lump to attack. Experience has shown that retaining about 20% of the total load in the form of plus 20 mm rocks helps to break down any hard clays or partly cemented wash. Where such rocks occur naturally in the wash, baffle plates can be adjusted to achieve desired rock-clay proportions. Where the wash is substantially fine grained, rocks may be fed in separately as required.

Some clayey materials have tempering qualities, in that changes in cohesion follow exposure to the atmosphere or soaking in water. Materials that display high degrees of plasticity are typically difficult to slurry when freshly mined. At Moolyella in Western Australia, clayey gravel responds readily to treatment only after a preliminary stage of spreading and sun drying (Macdonald, 1983a). By limiting the residence time of materials in the scrubbing circuit water usage within the plant is also reduced. On the other hand many less clayey materials harden quickly on exposure to the atmosphere and stockpiling adds to the difficulty of slurrying hence the requirement for additional power and water.

Scrubber construction details

A typical rotary scrubber-trommel arrangement comprises a conventional trommel screen to which is attached a non-perforated tubular section at the feed end for scrubbing. The scrubbing compartment is fabricated from heavy wear-resistant steel plate and is fitted with lifter bars, heavy chains (optional) and baffles. Water jets are incorporated in a sparge pipe water system and usually operate at pressures of about 7 atm (100 psi). A retarding ring (baffle) between the scrubbing and screening sections creates an initial semi-autogenous scrubbing section in which the large stones are held back to rumble over and break up the wash. By retarding the flow, the baffle provides a longer retention time for unslurried materials. Varying the height of the ring varies the retention time.

Scrubbing capacity is determined by such features as the length and diameter of the scrubbing section, the internal features (chains, lifters, etc.), jet water supply (volume and pressure), and speed of rotation. Drive mechanisms employ mainly friction rollers, or chain and sprocket arrangements acting through reduction boxes. The optimum speed of rotation is based upon the diameter of the cylinder and is slightly less than the critical speed of rotation for that diameter; its purpose is to present the material most advantageously to the action of hydraulic jets, and to maximise the mechanical rubbing and impact forces.

Secondary scrubbing

Lumps of partly lithified and very clayey materials that are very difficult to disperse effectively in rotary scrubber-trommels may result in the carry over and loss of significant amounts of gold in the screen overflow. The installation of a secondary scrubbing system is sometimes considered as an alternative to increasing the size of the primary scrubbing unit, but such a course of action is seldom successful. Secondary scrubbing systems are both expensive to install and to run and any decision to install a secondary scrubbing system can be justified only by the certainty of thereby recovering sufficient gold to justify the added cost. Although some form of compromise may occasionally be reached, there is rarely any satisfactory resolution to problems that should have been identified and dealt with at the start. If there are any doubts that cannot be addressed at the sample dressing stage, the possible future need for secondary scrubbing can generally be avoided by providing a generous degree of over-design in the primary scrubbing circuit. In dry mining operations where floor space is not a critical issue, the ore may be scrubbed and screened separately, the most difficult materials being given special attention.

8.3.3 Screening

Trommel screens are used universally in primary screening service because no other current screening method provides the required conditions of high capacity, low floor space and high recovery in the treatment of clayey ore materials. Modern bucketline dredger trommels and other trommels of massive construction have walls of heavy wear-resistant steel plate. The plates are punched with round holes that taper outwards to minimise blinding. Aperture dimensions for alluvial gold ores usually approximate 10 mm tapering outwards to 15 or 16 mm. Rubber screen plates held in position by mushroom studs replace the longitudinals in most modern trommel screens, which range in length up to 20 m or more and from 1–3 m diameter. The greater ease of maintenance and non-blinding characteristics of the rubber have encouraged further research and practice has been extended to the rubberisation of the steel shell frame.

The trommel screen rejects a barren oversize fraction of stones and organic trash and recovers an undersize fraction for further processing. Screen apertures are usually sized to reject plus 8–10 mm waste material. The gold particles are mainly smaller than 2–3 mm and occasional nuggets passing out with the oversize can be caught in a nugget trap in the waste chute. The screen sections may be of simple or heavy-duty construction depending upon feed rates and the nature of the feed material. The washed materials, i.e., slurry plus stone plus any surviving lumps of clay flow across the retaining ring into the perforated trommel section. Here the beating action is less violent although some lifters may still be provided and water jets continue to wash the stones and finally clean their surfaces. The screen undersize is directed to the primary gravity concentrating circuit. The oversize drops from the trommel onto a chute or stacking belt for disposal as waste.

Capacity

Features influencing the potential capacity of trommel screens are mainly diameter and length, slope, aperture size and total aperture area. Screen capacity is also influenced by operational factors such as the nature of the feed material, the size range and proportion of feed that is near the critical aperture size, and the presence or absence of plant fibres, peat and other extraneous materials that might cause clogging. Small-scale operations typically utilise light mesh trommels of the type designed to screen surface gravels for minor road making and light construction work. A scrubbing compartment is fabricated and fitted to the feed end and the operators make their own running repairs as required.

Trommel diameter is generally accepted as a determining factor for capacity and length for efficiency of washing. But this is only an approximation and published performance figures such as those given by Fricker (1980) in Table 8.4 can be used only as a very general guide because of the wide range of variables. An added complication is that data supplied to manufacturers are generally too few and unrelated for quantitative analysis although showing an interesting and, perhaps, significant trend in respect of capacity and power. In most cases, for example, the feed materials are described inadequately and too few operating parameters are given. For example, in the treatment of material fed to the Kanieri dredger (New Zealand), screen performance fell away sharply above a certain critical diameter, i.e., somewhere between 2.0 m and 3.0 m for capacity and between 1.5 m and 2.0 m for power. Failure to reach operational requirements is usually due to adopting generalised recommendations from manufacturers without adequate prior testing. Typical problems are:

Table 8.4 Trommel capacities – power requirements (after Hack, from Fricker, 1980)

Diameter (m)	0.91	1.2	1.5	2.1	
Power (kW)	5.60	11.0	19.0	45.0	
Capacity (M^3/h)	30.0	60.0	200.0	400.0	

By way of contrast, the Kanieri (New Zealand) dredger trommel is 3.0 m diameter × 18 m in length, has a 75 kW motor and can handle 550 m^3/hr. Extrapolating from these data:

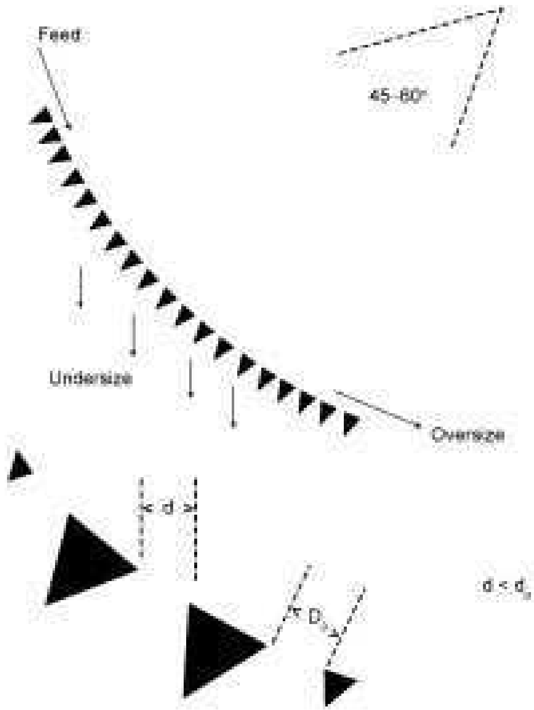
Diameter	0.91	1.2	1.5	2.0	3.0
m^3/hr/m diam	33.0	50.0	133.0	190.0	123.0
m^3/hr/kW	5.36	5.45	10.53	8.89	7.33

- undersized screens
- screen apertures that are either too small or too large for the required service
- failure adequately to slurry clay balls.

The clay ball problem is a feature of screens that are either too short or have apertures so large that insufficient retention time is provided to disperse the wash. Unslurried clay balls rejected from on-going sections of the treatment plant frequently contain entrained particles of gold. Only when the manufacturer is made aware of the true nature of the deposit can design be carried out satisfactorily. In one Indonesian project the wash contained only 15% of gravel larger than 1.5 mm (only occasional pieces larger than 200 mm). Slimes comprised 25% of the feed, the remaining 60% was sized from 1.5 mm down to 52 microns. The screen manufacturer was unaware of the tenacity of the clay and the lack of coarse gravels to act as autogenous grinding material in the scrubber. As a result the screen and hydrosizers were unable to produce a dispersed and well-classified fraction for processing at anywhere near the design rate.

Secondary sizing

The most suitable device for secondary sizing is a sieve bend located between the trommel underflow duct and the head feed pump bin. The sieve bend is a stationary screen type, constructed from profiled wedge bars to form a concave, slotted surface (Fig. 8.10). The profile retards the flow of slurry over the lower part of the screen and creates centrifugal forces that act downwards, thus assisting the screening action where most of the separation takes place. The normal separating range is from 3–4 mm, but may be reduced to 2–3 mm to produce separate feeds in parallel coarse and fine separators. Jigs can usually recover most of the gold reporting with the plus 2 mm fraction. However, significant gold losses in the tailings might occur if through surging, quantities of finely sized particles carry over with the coarser fraction. Because of the



8.10 Sieve-bend dimensions.

screen configuration an aperture made, say 10% larger than a particle that can be passed at low flow rates must be more than 10% larger to pass the same particle at higher feed rates.

8.3.4 Slurrying

The purpose of slurrying is to liberate free gold from the matrix in which it occurs and to produce a water-solids mixture in which all of the particles (including gold) can move freely. Constraints to achieving ideal slurrying conditions are failing to achieve a balance between the amount of water and the type of material being fed to the trommel at any one time. Ideally, the amount of water used should be just sufficient to disperse the clays and provide slurry of the correct pulp density for presentation to the treatment units. Insufficient water will result in inefficient liberation with consequent losses of gold in the trommel oversize. Any carry-over of undispersed feed material due to too much water in the feed will almost invariably be accompanied by losses of entrained gold when the excess water overflows or is purged from the system. Response to such changes is slow without some form of sensing to rapidly trigger control mechanisms to adjust the solids/water flow.

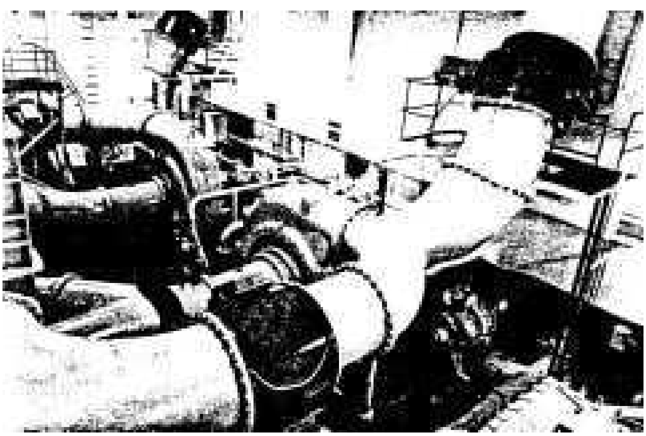
Measurement

Direct estimates of the bulk properties of slurried material are possible in dry mining-trucking operations and in some hydraulic dredging operations where instrument technology has reached acceptable standards for materials in transport. For such materials, a magnetic flow meter measures slurry velocity using a transmitter clamped around the outside of the pipe. An externally mounted nuclear density meter records any change in mass per unit volume of the flowing pulp and the data from both sets of instruments are fed to a single analysing and integrating instrument, which calculates the dry tonnes of solids in the flow. Instrumentation features automatic zero and gain control of the flow signal. Push-button calibration is available through a microcomputer to process mass and flow signals continuously, and provides the relevant information to the operator.

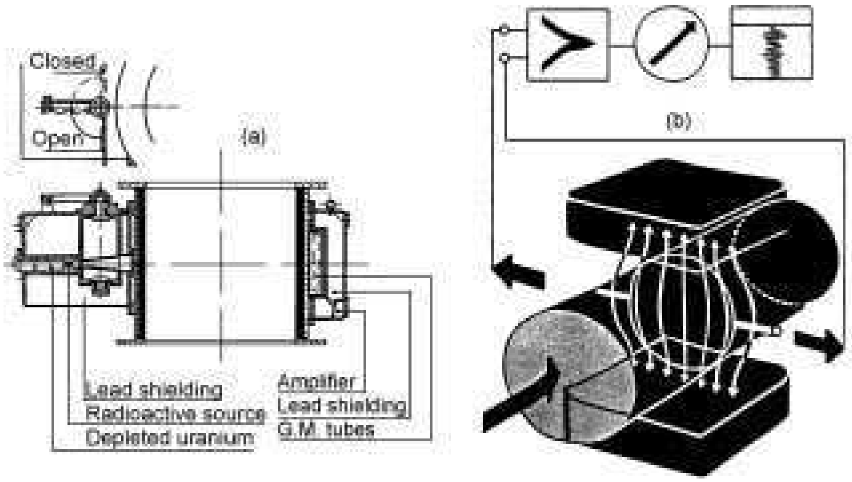
Nuclear density transmitters contain radioactive materials, and all measures must be taken to maintain safe environmental protection. The ‘Class 1 White’ safety grade is obligatory and should be maintained in the worst possible conditions, e.g. open-source shutter and empty pipeline. An example of the type of slurry measurement system used in modern slurry transmission systems is given in Fig. 8.11, which shows the layout of an integrated density transmitter and velocity transmitter system applied to integrated models in dredger pipelines by IHC of Holland. Figure 8.12 illustrates the operational principles of the two systems.

Control

Solids feed rates and pulp density must be closely monitored and controlled for the efficient operation of plant recovery units. In jigging practice for example,



8.11 Density and velocity transmitters – integrated model.



8.12 Principle of (a) density transmission system and (b) velocity transmission system.

the flow of water and solids across a jig is generally optimal at around 25% solids by weight. At reduced pulp densities the flow of top water increases and scouring may occur. At higher pulp densities the flow of top water may be inadequate and increased hutch water will be needed to keep the solids in motion. Losses of gold may occur at both high and low pulp densities that would not have occurred if all of the process parameters had been in equilibrium.

The problem of having too much or too little water in the slurry depends upon how quickly the control mechanisms can be activated in response to fluctuations in the raw plant feed. Clayey material requires more water for slurrying than do gravel and sand and for a rapidly fluctuating feed equally rapid responses are needed to correct the flows of water and/or solids and so maintain design levels of pulp density. A sudden rush of clayey wash should quickly initiate emergency measures to adjust the solids/water flow and re-stabilise the plant operation. Response to such changes is slow without some form of sensing perhaps triggered by a TV monitor or other sensing device. The usual response is an immediate reduction in the raw feed rate or an increase in the jet water pressure to increase the supply of energy per unit mass of feed. Neither measure should be continued for long enough to upset the water-solids relationships in the rest of the plant.

Control is achieved most readily in medium to large dry-mining operations by selective loading of stockpile materials and by the provision of adequate surge capacity to even out the water flow. Hydraulic dredgers are the most affected by excess water. Although manufacturers usually claim an average pulp density of dredged slurry at around 24% solids by weight, this value can be obtained only with the digging mechanism in full production mode. At other times, the solids intake fluctuates widely according to such factors as differences in consolidation of the material being dug, changes in treatment plant requirements and trash handling. Current monitoring technology does not yet have all the answers and the plant operator is usually forced to make arbitrary decisions concerning raw feed rates and water-solids ratios, which may not necessarily be correct. The presence of some excess water in the feed preparation section is generally accepted as a better alternative than the more serious problem of having too little water for adequate slurrying.

Designers of the Grey River Dredger, New Zealand considered adjusting the trommel slope to vary the residence time of the solids and so helping to achieve the optimum regulation of water pressure and volume. An attempt was also made to increase the solids throughput per unit of water by increasing the screen aperture availability. The screens were made of high-density plastic with rectangular holes and the objective was to achieve a total effective screen area of nearly twice that of conventional trommel screens. The results were apparently disappointing.

Distribution

All modes of transport can exist simultaneously in gravity treatment devices when slurry of heterogeneous particles is passing through the unit. As already described the particles will have different values of critical shear velocity, which result in different transport modes for different particles. Higher density solids segregate along the lower parts of pipelines and the coarser gold particles tend to settle out preferentially along the line of flow. Self-driven turbine distributors also suffer from some measure of differential settling. Simple steady heads with an overflow pipe have no means of avoiding segregation and the resulting splits will inevitably vary considerably in both quantities and grades. The problem of distribution is compounded when the total feed from a common source is split between a multiplicity of process units. Although secondary and tertiary plant units of all types deal with much smaller quantities of solids that are both smaller in size and better classified, unresolved problems are still attached to splitting feed evenly through different and multiple feed lines.

The difficulty increases with the number of splits made. Flow to primary jig units are split between a few units only and distribution usually requires only the mechanical division of hydrocyclone underflow in a baffle box. A feed rate of 400 t/h solids distributed between two circular jigs would require only two distribution outlets. Six distributors would be needed for Reichert cones for the same throughout. For spirals, the requirement would be 180 to 200 starts with an equal number of distribution points. The large diameter circular jigs thus have definite advantages over smaller circular and rectangular jig types in respect of simplification of the distribution system. Nevertheless, even in the largest of circular jigs, experience has shown that efficient splitting requires constant attention to flow-velocity and pulp distribution over the total jig area to avoid development of 'dead spots' due to uneven solids distribution. Only rarely do banks of such low-capacity units operate fully loaded without scouring in some compartments and sanding in others.

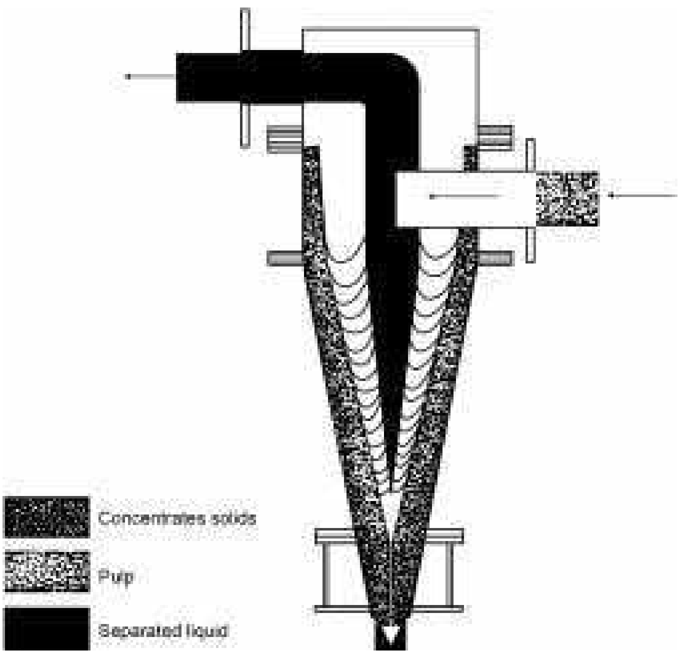
8.3.5 Desliming

Slime to the chemical engineer is any ropy or viscous matter, usually of mud size. To the mineral processing engineer the upper size limit of slime particles is determined by the balance between the D_{50} size of finely divided solids that do not settle easily in still water and the D_{50} size of the economically recoverable gold. The D_{50} parameter in minerals processing is the diameter of a particle that has an equal chance of reporting with either the screen underflow or the overflow fraction of a classifying unit, e.g. hydrocyclone. The level of sophistication attached to the measurement and rejection of slime is often a major key to economic viability in alluvial gold processing plants. Stokes analytical solution for the magnitude of the drag force is valid for largely equant particles with very low values of NR down to the size at which Brownian movement maintains particles in irregular and erratic suspension. For quartz density particles, the Reynolds number condition limits particle diameter D to about 50 μm whereas the high density of gold particles is a significant factor and D is reduced to about 20 μm. But there are other considerations. The Newtonian and Stokes theoretical assumptions are based upon the free settling of spherical particles in a frictionless fluid. However, this is unrealistic as a description of practical gravity concentration where particle concentrations are normally high and slime size particles affect the rheology of slurry through changes in apparent viscosity and density. Increased quantities of suspended solids in the slurry or reduction in water content increase the apparent viscosity of the slurry and make it more difficult for the larger particles to settle and be recovered. Stokes law applies closely to the thickening of fine quartz spheres in desliming applications up to about $D = 70\ \mu$m and approximately to most sediment up to about $D = 100\ \mu$m.

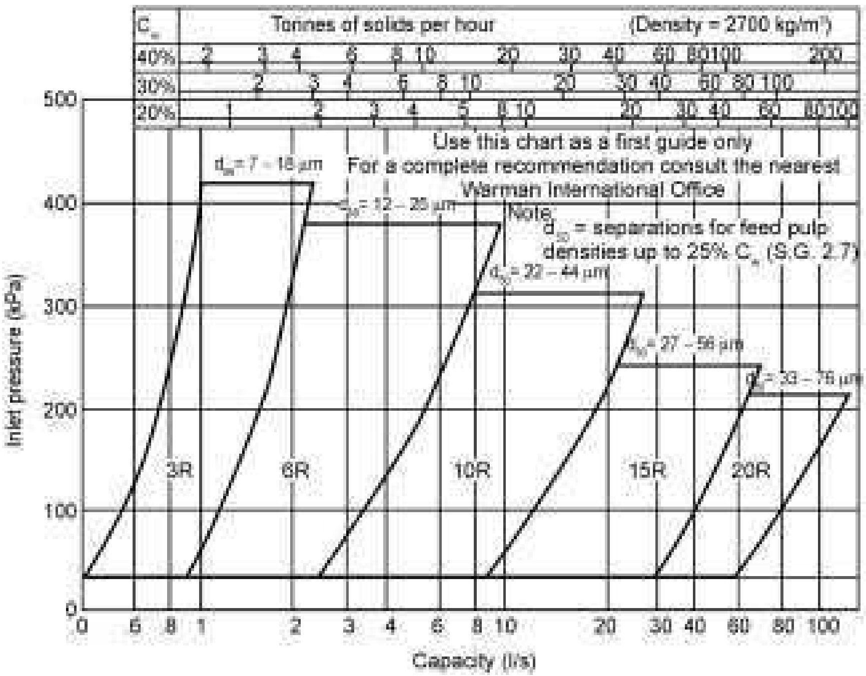
Hydrocyclones

Hydrocyclones are preferred units for sizing or desliming large slurry volumes cheaply and because they occupy very little floor space or headroom. They operate most effectively when fed at an even flow rate and pulp density and are used individually or in clusters to obtain desired total capacities at required splits. Sizing capabilities rely on centrifugal forces generated by high tangential flow velocities through the unit. The primary vortex formed by the incoming slurry acts spirally downwards around the inner cone wall. Solids are flung outwards by centrifugal force so that as the pulp moves downwards its density increases. Vertical components of the velocity act downwards near the cone walls and upwards near the axis. The less dense centrifugally separated slime fraction is forced upwards through the vortex finder to pass out through the opening at the upper end of the cone. An intermediate zone or envelope between the two flows has zero vertical velocity and separates the coarser solids moving downwards from the finer solids moving upwards. The bulk of the flow passes upwards within the smaller inner vortex and higher centrifugal forces throw the larger of the finer particles outward thus providing a more efficient separation in the finer sizings. These particles return to the outer vortex and report once more to the jig feed.

The geometry and operating conditions within the spiral flow pattern of a typical hydrocyclone are described in Fig. 8.13. Operational variables are pulp density, feed flow rate, solids characteristics, feed inlet pressure and pressure drop through the cyclone. Cyclone variables are area of feed inlet, vortex finder diameter and length, and spigot discharge diameter. The value of the drag coefficient is also affected by shape; the more a particle varies from sphericity the smaller is its shape factor and the greater its settling resistance. The critical stress zone may extend to some gold particles as large as 200 mm in size and careful monitoring of the classification process is thus essential to reduce excessive recycling and the resulting build up of slimes. Historically, when little attention was given to the recovery of 150 μm gold grains, carry-over of gold in the slime fractions appears to have been largely responsible for gold losses that were recorded to be as high as 40–60% in many gold placer operations.



8.13 Normal geometry and operating conditions of a hydrocyclone.



8.14 Warman preliminary selection chart.

Figure 8.14 (Warman Selection Chart) is a preliminary selection of cyclones for separating at various D_{50} sizings from 9–18 microns up to 33–76 microns. This chart, as with other such charts of cyclone performance, is based upon a carefully controlled feed of a specific type. It assumes a solids content of 2,700 kg/m^3 in water as a first guide to selection. The larger diameter cyclones are used to produce coarse separations but require high feed volumes for proper function. Fine separations at high feed volumes require clusters of small diameter cyclones operating in parallel. The final design parameters for close sizing must be determined experimentally, and it is important to select a cyclone around the middle of the range so that any minor adjustments that may be required can be made at the start of operations.

The CBC (circulating bed) cyclone is claimed to classify alluvial gold feed materials up to 5 mm diameter and obtain a consistently high jig feed from the underflow. Separation takes place at approximately $D_{50}/150$ microns based upon silica of density 2.65. The CBC cyclone underflow is claimed to be particularly amenable to jig separation because of its relatively smooth size distribution curve and almost complete removal of fine waste particles. However, although this system is claimed to produce a high-grade primary concentrate of equant heavy minerals in one pass from a relatively long size range feed (e.g. mineral sands), no such performance figures are available for alluvial feed material

Table 8.5 Technical data for AKW hydrocyclones

Type (KRS)	Diameter (mm)	Pressure drop	Capacity Slurry (m^3/hr)	Solids (t/h max.)	Cut point (microns)
2118	100	1–2.5	9.27	5	30–50
2515	125	1–2.5	11–30	6	25–45
4118	200	0.7–2.0	18–60	15	40–60
(RWN)6118	300	0.5–1.5	40–140	40	50–100

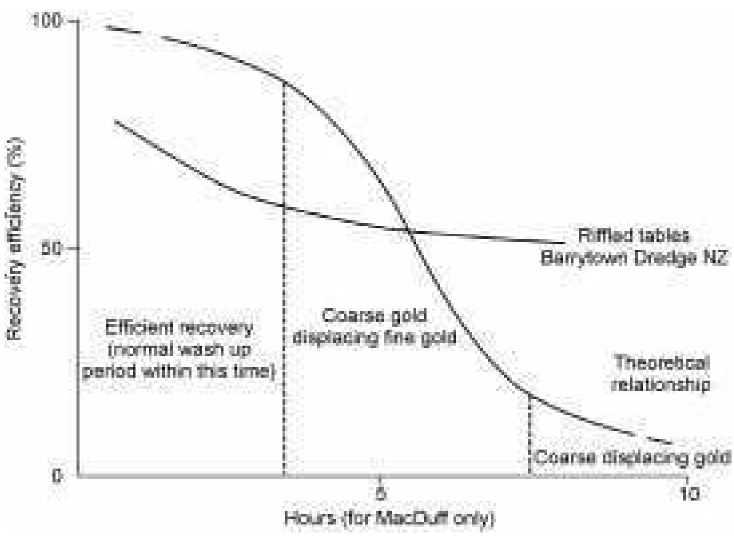
containing fine and flaky gold. Table 8.5 gives the technical data for AKW hydrocyclones for cut-off points between 30 and 100 microns.

8.4 Gravity processing

Gravity processing design calculations are based primarily upon the results of sample dressing at the drill site and laboratory-scale investigations of the sedimentation characteristics of the gold. Significant aspects of these studies include the nature of the feed material and scale-up relationships that incorporate all that has been learned from sampling and small-scale experimental work. Recovery plant types are selected in which the balance of factors matches the expected behaviour of the feed materials at each stage of processing. Typical of these are devices that separate particles of high density from non-valuable particles of lower density in solids/fluid mixtures by exploiting differences in the hindered settling of particles of different size and density. Slime fractions, which usually comprise significant proportions of the total load, account for most losses of detrital gold smaller than about 100 μm. All conventional gravity devices have a minimum size limit for the recovery of fine gold but no device has yet been developed capable of recovering both coarse and very fine gold in the one operation.

Riffled sluices have been used to recover gold since antiquity and are still the preferred units for small-scale mining operations where deposit size and capital cost are limiting factors. Jigs are the best all-round units for coarse gold separation in roughing service but good recovery is limited to gold grains larger than 100–150 μm in well-classified feed. Spirals have replaced jigs in some installations because of their successful use in the beach mining industry but have so far been disappointing in recovering fine and flaky gold. They are also less forgiving than jigs to irregularities in feed rates and other conditions common to alluvial gold operations. Current research in centrifugal jig separation is towards the provision of methods and equipment that might compete favourably with conventional gravity devices in both roughing and cleaning service. Shaking tables, vanners and other thin film separating plant recover finely divided gold under conditions of sub-critical laminar and super-critical laminar regimes of flow, which may occur only where there is a very thin depth of fluid. The main application of metallurgical cyclones is in the desliming circuits of feed preparation systems. More recently, cyclone separators have been developed for use as classifiers within the treatment plant itself. A more extensive use of centrifugal action has helped to resolve many of the difficulties involved with the recovery of very finely divided gold particles. By varying the apparent magnetic field in centrifugal type separators and so enhancing the forces acting on individual particles, operators hope to make future separations down economically to a few microns in size.

Benefication processes have since been extended by investigative research to include compound water cyclones and rotating bowl separators in which centrifugal forces are used to enhance gravity differences, and combined gravity/froth flotation techniques and centrifugal jig separation. An important aim has been to accurately predict the stratification of typically equant heavy mineral particles in shear flow and under the pulsatory conditions of the jig bed. However a common constraint is that all such devices operate by batch processing and hence pass through individual cycles of poor recovery, optimum recovery, and poor recovery before being shut down to recover the concentrate and start the next cycle. McDuff (1945) demonstrates the extent of the problem in Fig. 8.15. His experiments showed that deleterious effects from concentrate build-up occur first with the increasing rejection of fine gold particles as the plant settles down, and finally, with conditions of stagnation in which almost all of the incoming gold passes out with the tails regardless of size. Large quantities of feed materials must often be recycled in order to make high recoveries at required levels of concentration in some units.



8.15 Recovery cycles in batch processing cycle (from McDuff, 1945).

There has also been limited response to the variable effects of differences in size, shape, texture and density on the hydraulic behaviour of finely divided and flaky gold grains. Wang and Poling (1983) suggest that fixed low gold prices, which discouraged early investigations into the recovery of finely divided gold, are largely responsible. Fricker (1984) considers the character of alluvial gold deposits which, 'present severe difficulties and sometimes limitations, on the extent to which metallurgical performance can be measured' to be a contributing factor. A tendency to cling to primitive treatment methods handed down from each generation to the next, virtually unchanged and unchanging may be another reason.

Gravity devices, which separate particles of high density from non-valuable particles of lower density in solids/fluid mixtures, can be broadly divided into three groups:

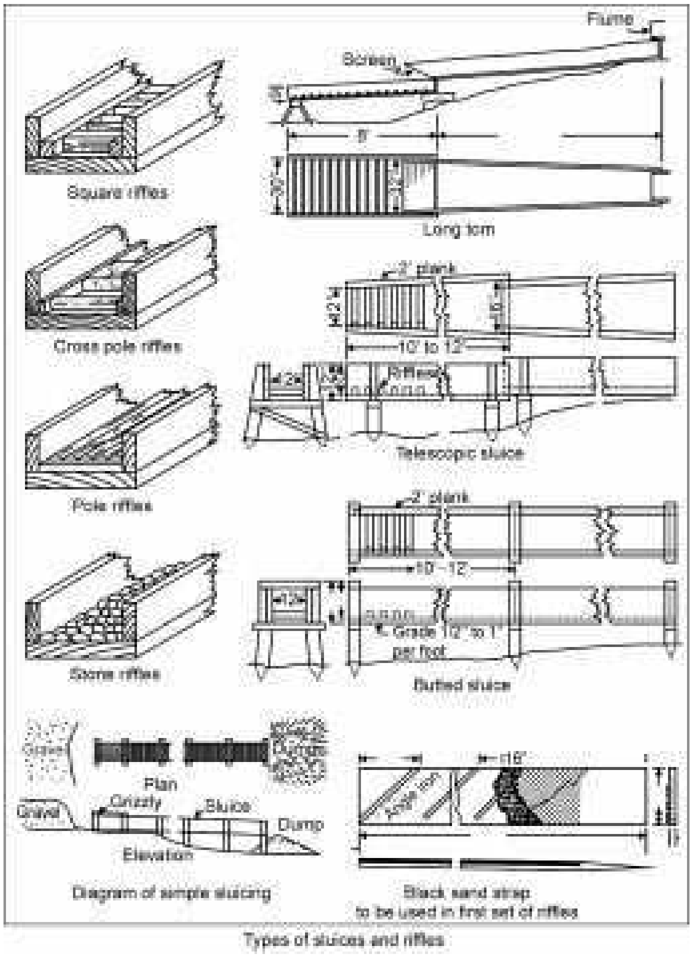
1. devices using shear flow (sluices, cones, tables, spirals)
2. devices using vertical pulsations of water (jigs)
3. devices of both groups that employ centrifugal action to enhance the force acting on mineral particles, particularly in the finer sizings (centrifugal separators, cyclone separators).

8.4.1 Shear flow devices

Sluice boxes

Methods of ground and hydraulic sluicing described in Chapter 7 have typically relied upon sluice boxes for treating the spoil because of their simplicity and low cost. A riffled sluice box is an open launder in which narrow slats of wood or metal are installed normal to the direction of flow. Large sluice boxes are fabricated in sizes up to 2 m wide and 1 m high. The total length is usually 10–30 m but may be more depending upon sediment size, percentage of other heavy minerals and the size of the gold. Small boxes, 200–500 mm in width and up to 250 mm high, are usually constructed in 3–4 m lengths for mobility. The box slope may vary from 1 in 8 to 1 in 20 depending upon feed characteristics, slurry flow rates, riffle type and configuration and the physical properties of the gold.

Boxes may be either tapered or rectangular in longitudinal section. 'Tapered' boxes fit snugly into succeeding boxes, thus allowing a series of box sections to be dismantled and re-assembled rapidly when shifting from one location to the next. Effectively they act as a series of pinched-sluices and it is argued that the sudden deceleration of flow that occurs at the beginning of each succeeding box is beneficial to the deposition of gold. 'Rectangular' boxes, which are joined longitudinally by butt jointing with close fitting cleats around the joints to make them tight, provide an even flow rate down the length of the box. They have the advantage of being easily constructed, but are not as easily dismantled as tapered



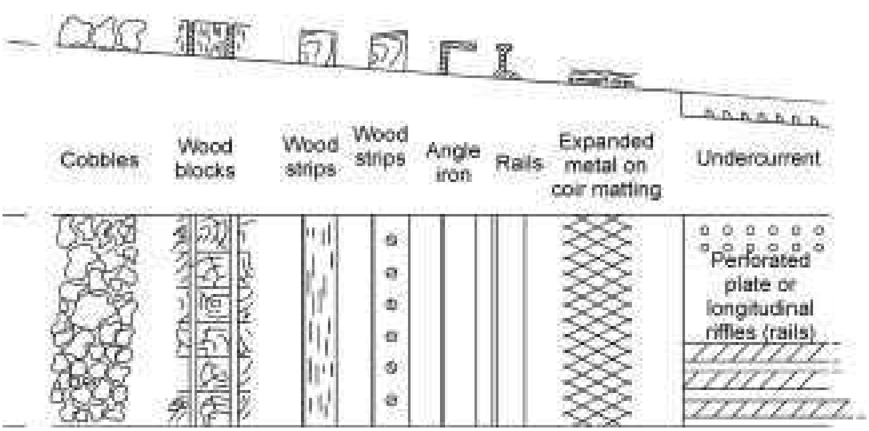
8.16 Sluice box types, rectangular and tapered (after Stout, 1984).

boxes and are thus less mobile. Stout (1984) describes the two box types (rectangular and tapered) in Fig. 8.16.

Gaudin (1939) considered the sluice to be essentially a classifier used as a concentrating device, as the valuable minerals are of one size range (generally the finest) and the waste minerals in another. Large particles (gravel) travel by sliding and rolling over the riffles, with finer particles travelling by saltation. Sand travels by a combination of modes described earlier with some saltation over the riffles. Very fine particles are maintained in suspension by turbulent and inter-particle collision. Riffles function properly only if in the space between them the slurry is sufficiently live (turbulent) to reject the lighter particles, but not so lively that the gold cannot settle. The settling of heavy minerals between the riffles requires frequent stirring to prevent the riffle spaces from blinding. This also disturbs the gold, which then moves progressively down-sluice. Frequent clean-ups are needed to avoid excessive loss. Boxes may be used in parallel to avoid loss of production time. One box is kept in operation while cleaning up in the other.

Riffle type and configuration

Many different types of riffles have been employed, the most common being the block, longitudinal and Hungarian riffle types described by Burt (1984) in Fig. 8.17. The diversity of different riffle configurations is matched only by the variety of conditions in which sluices are used. Peele (1927) describes the dimensions of early typical riffle types in Table 8.6. Cope (1988) describes three types of Hungarian riffles of angle iron, wood blocks and wood with a metal capping in Fig. 8.18. The angle iron riffles lean into the flow at a five degree angle. The wood blocks are tapered similarly and the metal strapped riffles are overhung on the downstream side to give a 'galloping' action to the pulp. This action helps to settle the heavies by causing a slightly lower pressure on the downstream side of each riffle. Stout (1984) describes the fabrication of a Hungarian riffle from 1.5 inch (38.1 mm) by 1.5 × 0.25 inch (38.1 × 6.35 mm) angle iron cut to the width of the sluice box. In line, and one inch (25.4 mm) from the end of each angle, a half inch (12.7 mm) hole is drilled. A long 3/8 inch (9.525 mm) rod is run through each hole with equal length sections of pipe to act as spreaders. This fixes the spacings between the riffles but allows it to be varied as required. Fricker (1984) observes that riffle types dimensions and spacings are matters for personal preference and experience. His review of past and present practice suggested typical dimensions of 50 mm × 50 mm at 100 mm centres for Hungarian riffles, with the top of the riffle at 3–4 degrees to the deck with the tailing edge highest.



8.17 Riffle types (after Burt, 1984).

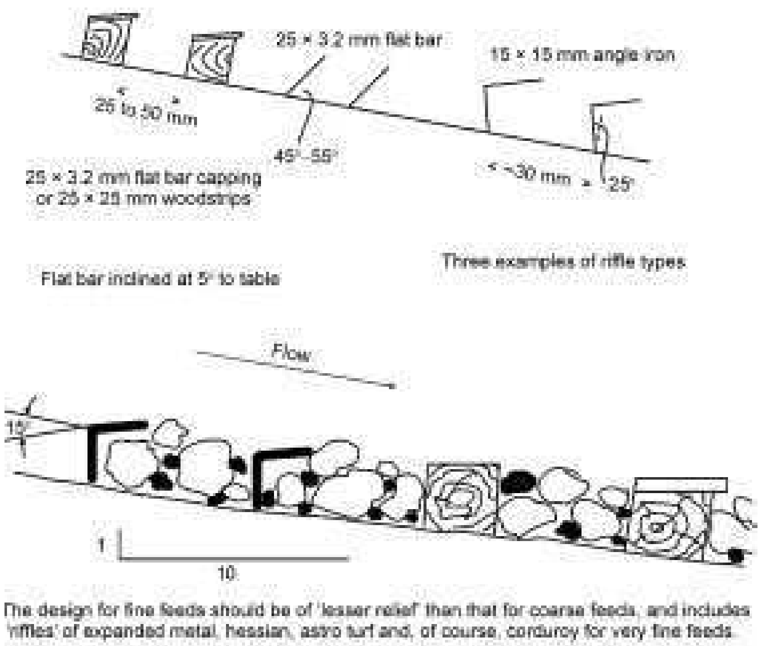
Table 8.6 Typical riffle types and dimensions (derived from Peele, 1927)

Unit	Type	Width (mm)	Height (mm)	Spacing (mm)
1	Wood cross	50.8	152.4	101.6
2	Wood cross	304.8	304.8	304.8
3	Hungarian	50.8	101.6	114.3
4	Angle iron	50.8	50.8	101.6
5	CI bars (1.219 m long)	76.2	31.75	127.0

Capacity

Published literature offers a wide and seemingly inconsistent set of flow rate statistics governing the performance of riffled sluices. This may be due in part to the cyclic nature of sluice box operation, the difficulty of quantifying capacity and efficiency of operations with feed materials that are unique to each operation and the length of time between clean-ups, which may vary considerably from days to weeks.

Feed rates range from 2.5–125 m^3/h/m width, with water/solids ratios from 20–40:1 by volume. Cope (1988) generally looks to a figure of 37 m^3/m of sluice box area/h as a starting point for estimating the size of sluices 'bearing in mind



8.18 Hungarian riffles (after Colp, 1976).

that each deposit is different'. According to Poling and Hamilton (unpublished) sluice boxes typical of the Yukon placer mining industry operate at solids feed rates from 100–700 lb solids/min/ft of sluice width (10–70 m/hr/m width). From the results of pilot scale test work, they recommend using 30–40 m^3/hr/m of width accompanied by approximately 200 US gpm water at a slope of $1\frac{5}{8}$ inches/ft (13.6–15.5 mm/m).

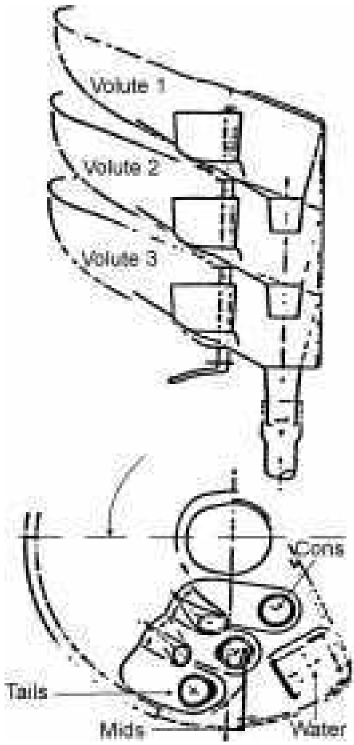
Research has led to many improvements in operations and designs. Poling and Hamilton, on a grant from the Government of Canada and the Yukon Territorial government, studied fine gold recovery of selected sluice box configurations. The major objectives were to determine operating parameters capable of yielding high gold recoveries down to 150 microns in size. Available data suggested that gold recovery in the average sluice box declines significantly for particles smaller than 0.2 mm, and that further processing of tailings may be required in a secondary or undercut sluice.

The authors found that each riffle type has a characteristic scour condition for which gold recovery is optimal. Their experiments covered the use of 1–10 H expanded metal as riffles for fine gold recovery, with conventional angle iron riffles acting only as nugget traps. They also investigated and then recommended interposing short sections of smooth unriffled sluice box bases in their fine gold recovery sections. From their observations this arrangement allows the slurry leaving each riffled section to pre-segregate on the smooth surface; a proportion of the high-density minerals settles out along the base of the flow, before entering into the next riffle section. Instead of employing undercurrent boxes with small riffles at the discharge end to scavenge the tailings of fine gold, they suggested installing expanded metal riffles at the head section of the box to recover the fine gold first. Low water usage was also beneficial to gold recovery; the best recoveries in the pilot scale tests using expanded metal riffles were obtained using water/solids ratio of approximately 4:1 by weight.

However, the sluicing method as a whole is wasteful of both power and water; slurrying is rarely complete and surging is common. Modern practice is to replace sluices by jigs, which separate more efficiently and are less labour intensive. Performance data for the Capital Dredging Company California showed a dramatic rise in recovery efficiency (RE) from 0.76 to 0.99 following the replacement of sluice boxes by jigs.

Spiral separators

Spiral separators were originally developed by Humphreys in 1899 for coal treatment processes in the USA and adapted for separating heavy minerals from beach sands. However, their cast iron construction made them cumbersome and very heavy; the spiral pitch of about 345 mm was not suitable for all classes of feed; the joints between each 120° spiral section were rough and caused turbulence; wear was rapid and wash water distribution was irregular (Macdonald, 1983a).

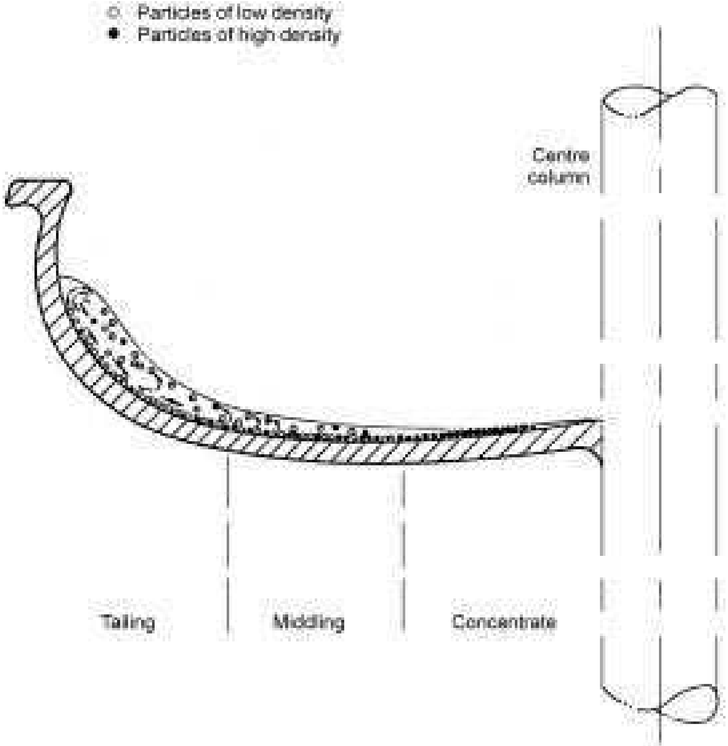


8.19 Schematic view of bottom section of a spiral showing the splitter arrangement.

Modern spirals are divided into two broad types: those with multiple concentrate offtakes using wash water and those using no wash water and splitting the concentrate off only at the spiral discharge end. Multiple offtake spirals generally have ports or splitters every 120º or 180º down the trough, with wash water cleaning the dense minerals by washing overriding less dense minerals back into the main stream. An operational disadvantage is the constant need to adjust a multiplicity of ports or splitters. Figure 8.19 illustrates the splitter arrangement in the bottom section of a spiral. Wash water-less spirals with a single set of splitters producing a concentrate, middling and tailing stream at the discharge end overcome this problem but final selection of any particular spiral type and trough profile is dependent upon the application. A selection of spiral profiles is illustrated diagrammatically in Fig. 8.20.

Spiral flow patterns

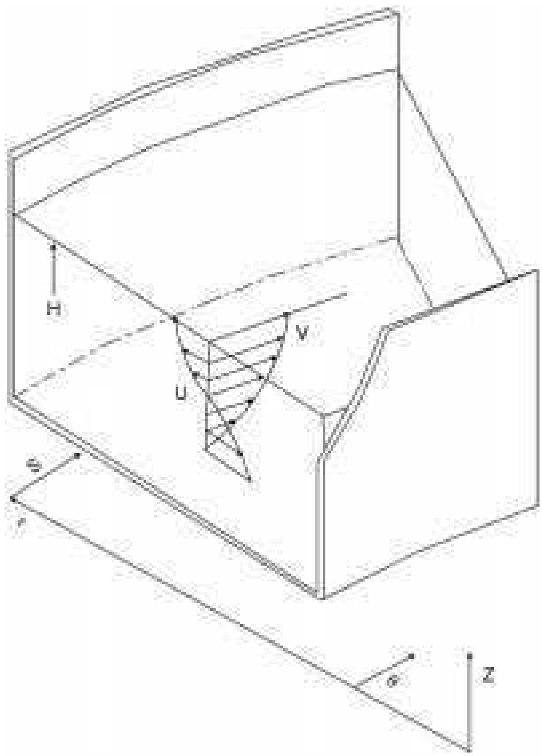
Research by Holland-Blatt (1972) and Holtham (1990, 1991) has gone some way to understanding the complexity of spiral flow patterns and hence the general failure of spirals to live up to expectations in gold processing plant.



8.20 Selection of spiral profiles.

One important observation has been the role played by the secondary current in fluid-particle interactions. Due to the centrifugal acceleration V^2/r as fluid flows around the bend in Fig. 8.21, the secondary flow velocity component U occurs in a plane normal to the primary flow direction. V varies from zero at the bed to a maximum value at or near the free surface where the centrifugal force acting upon a fluid element is also greatest. The tendency is for water to be moved radially outwards and accumulate in the outer portion of the channel, thus giving rise to a second radial force, the radial pressure gradient $\delta\rho/\delta r$. Assuming hydrostatic distribution of the pressure gradient, the radial pressure gradient is proportional to the local radial slope of the water surface $\delta H/\delta r$.

Since the centrifugal force is strongly dependent on the water depth, and the pressure gradient is constant at any given depth, there is a net radial force that changes sign at some depth within the flow. This imbalanced force results in the secondary radial velocity U. However, although this reversal of the secondary component U occurs at some depth within the flow, the direction of the flow at the base of the channel is always towards the inside radius of the bend. Holtham (1990) estimated the magnitude of the secondary velocity component from



8.21 Secondary flow components in spiral channels (from Holtham, 1990).

experimental measurement of the direction of secondary flow and magnitude of the primary velocity. Tables 8.7 and 8.8 give experimental values for primary and secondary velocities on a FGL mineral spiral for a feed rate of $6\,m^3/h$ of clear water.

Table 8.7 Primary flow velocities on the FGL mineral spiral (feed flow rate $6\,m^3\,hr^{-1}$ clear water) (after Nio, 1988)

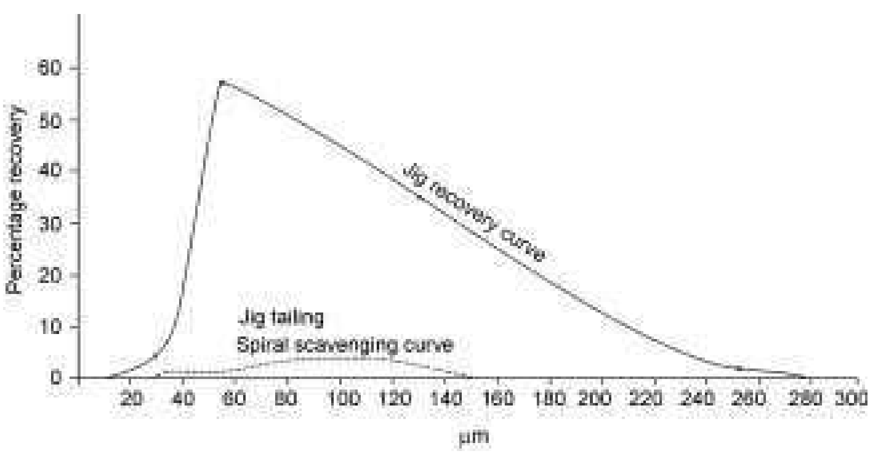
Trough location	Inner	Middle	Outer
Mean velocity (ms^{-1})	<0.1 <0.1	0.1 0.2	0.3 0.9

Table 8.8 Estimated secondary velocities on FGL mineral spiral (feed flow rate $6\,m^3\,hr^{-1}$ clear water) (after Nio, 1988)

Trough location	Inner	Middle	Outer
U trough base (ms^{-1})	0.0001 0.0002	0.001 0.003	0.01 5
U free surface (ms^{-1})	0.01 0.01	0.02 0.03	0.05 0.03

Particle stratification is an unsolved problem because of the added complexity of particle-particle interactions (Bagnold forces). It is known that spiral performance at the finer sizes is poor because such particles are held in suspension by turbulence in the outer trough and swept through to the tailings. Experience has shown also that flat gold particles, even if comparatively coarse, may also be held in suspension and be re-entrained when moving inwards from the outer part of the trough, particularly in fluctuating feed conditions.

The possible role of other heavy minerals in the spiral feed in the entrapment of detrital gold at the base of the trough is another unresolved operational feature. The importance of a heavy mineral fraction in the feed was indicated by the author in Kalimantan, Indonesia, where a bulk sample of gold-bearing stream gravels was passed through a jig circuit to give a concentrate and a tailing fraction. Panning showed the presence of both fine and coarse gold in the jig tailing, which was then processed in a spiral concentrator to recover a spiral gold concentrate. It had been expected that this concentrate would contain any fine gold that had been lost from the jig. This was not so; in fact it contained neither finely divided nor coarse gold. This may have indicated an inability of the particular spiral to concentrate gold under the conditions of the test. More likely however, the failure of the spiral to recover the gold was due to the prior removal of most of the heavy minerals by jigging. With few heavy particles remaining in the jig concentrate, there would have been insufficient heavies in the spiral feed to entrap fine and flaky gold. Figure 8.22 compares the gold recovery by jigging with the recovery of gold from scavenging jig tailing from which the heavy mineral particles had been removed.



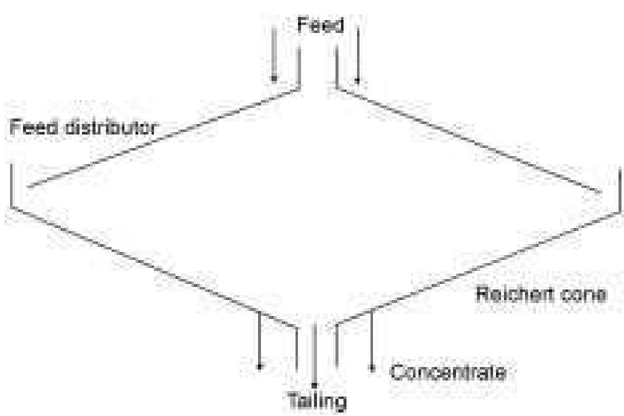
8.22 Particle size distributions for jig recovery followed by spiral processing of jig tailings.

Reichert cone

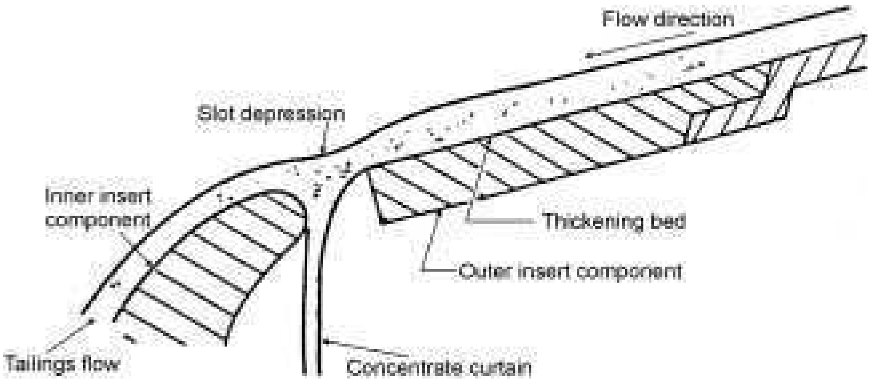
Although used infrequently in gold-recovery systems, the Reichert cone is worthy of consideration in some scavenging applications because of its relatively high capacity per unit of floor space (Macdonald, 1983a). The cone type separator, outlined diagrammatically in Fig. 8.23 was developed by Ernst Reichert in the late 1950s to overcome some of the perceived problems of pinched sluice separators, mainly their poor separating efficiency. This low efficiency was assumed to be due to partial re-mixing of the stratified layers caused by turbulence associated with the effects of sidewall flow.

Cones, like circular jigs, are fed centrally. The concentrate is split off though an annular slot located towards the cone centre at about one quarter of the cone radius. The slot is formed by two annular polyurethane castings, which together form an insert in the cone surface. The inner part of the insert, the concentrate cutter, has a rounded profile and is movable vertically with respect to the outer insert. The manufacturers claim that the curved surface (Fig. 8.24) makes the concentrate split less sensitive to variations in pulp thickness and density, and reduces the tendency to accumulate trash.

Cone units 2.0 m in diameter are suitable for flow rates of 60–90 t solids/h. Larger, 3.5 m cones are claimed to process up to three times the tonnage of the smaller units with similar recovery efficiencies. The feed sizing is preferably between 200 μm and 1 mm for gold although a few larger particles can be tolerated. Like the pinched sluice separators, cones require a feed pulp density of 60–65% of solids by weight. At this pulp density, the capacity of the standard 2 m diameter cone is about 60–90 tonnes of solids/h. Good recoveries of beach sand heavy minerals (e.g., rutile, zircon and ilmenite) are made in the 100–60 μm micron range. This size range does not necessarily apply to placer gold operations because of shape and other factors discussed in earlier chapters.



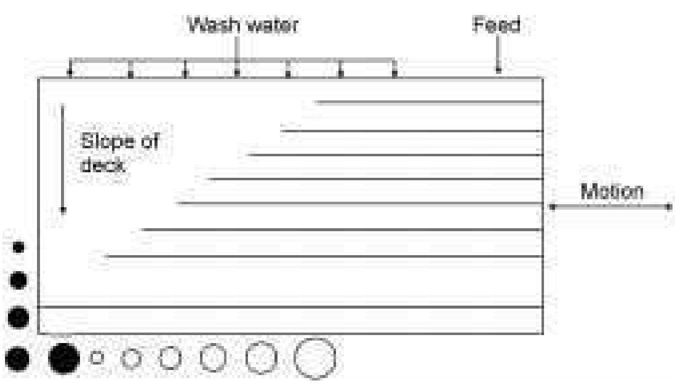
8.23 Diagrammatic outline of flow characteristics over one Reichert cone unit.



8.24 Concentrate split in Reichert cone.

Shaking tables

Shaking tables are essentially low-capacity units for final upgrading. The table is a thin film, shear flow device, which separates granular materials according to differences in density, size and shape (Fig. 8.25). Feed material is mixed with water and fed as slurry averaging about 20–25% of solids by weight onto the deck at its highest point. The tabletop (deck) is given a reciprocating motion along its main axis by means of a vibrator or an eccentric head motion. The deck is fitted with a series of tapered strips (riffles), generally of yellow pine or low-density polythene. Riffles taper in height downwards toward the concentrate discharge end of the table making it increasingly easier for the particles to move transverse to the table axis, thus promoting separation over the full length of the table. Riffle heights are determined according to the duty for which they are required, as are the riffle patterns. The operating variables are solids characteristics, table speed, stroke length, deck inclination, and wash water volume.



8.25 Shaking table – showing segregation pattern of heavy particles.

Preparing several size fractions for tabling is usually achieved in a hydrosizer. If gold is present in both coarse and finely divided sizings at least three, or perhaps four separate size fractions must be treated, each under a different set of operating conditions. Tables operate most efficiently with a closely sized feed. The slurry fans out across a smooth section of the surface until it reaches the riffles. The lighter and very fine particles are washed over the riffles and moved along the riffles by the reciprocating motion imparted to the deck while the heavier particles are held back. The concentrates of heavy mineral and gold are discharged over the end of the deck. Tailings are washed over the lower edge and a middlings fraction is taken off between the lower edge of the concentrate strip and the higher edge of the tailing strip.

Wash water usage is dependent upon the particle diameter and varies from as low as $0.7\,m^3$/t/h of solids for slime decks, up to $5–6\,m^3$/t/h for coarse solids separation. Coarse fractions are usually treated at feed rates of up to 1 t/h using approximately 15 to 20 mm stroke lengths at around 280 rpm (Wilfley table data). The stroke lengths of finer fractions are reduced to 9–15 mm with increased speeds of up to 325 rpm but, because of the corresponding lower film, thickness capacities may fall to around 0.25 t/h. The inclination of the deck is adjusted during operation using a hand-operated tilting device. It is important following each adjustment to allow the table operation to settle down before making a fresh adjustment. The correct inclination is reached when the ribbon of concentrates is clearly defined and remains steady.

The extreme sensitivity of water depths and corresponding current depths to obtain $F = 1$, and the use of stationary tables as primary concentrating units, was probably the main reason for the consistently low (R.E. 60–65%) gold recoveries of early New Zealand dredgers. For such table types, the fluid forces are applied to the streambeds as a whole and ripples form, which keep the sand in orbital motion and provide for the denser particles to sink to the bed. Deposition is most favoured by anti-dune conditions produced by free-surface flow at or near the super critical state. Such bed forms are in phase with the water surface and are produced in the rapid flow conditions of Froude Number $F = 1$. In this state of flow, the bed forms of the upper flow regime are stable. Below $F = 1$ the flow is tranquil and shear forces are reduced. In reviewing recovery distributions of the Arakura and Ngahere dredgers Fricker (1984) noted that some coarse gold reported with the tailing after passing through two stages of tabling and that fine gold did not concentrate noticeably down the line.

8.4.2 Jigs

Jigs are undoubtedly the most important of all types of alluvial gold concentrating device; they are also the most complex and the practice of jigging still tends to be regarded as an art rather than a science. Kelly and Spottiswood (1982) refer to jigging as being

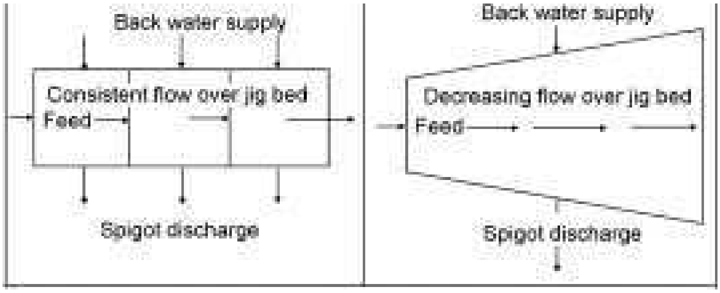
> probably the most complex gravity operation because of its continuously varying hydrodynamics. The mineral bed is repeatedly moved up by the water, expands, and then resettles, the resettlement occurring with the water flowing down at a lower rate (because of the addition of hutch water) than that which occurred on the upstroke. It follows that the waveform itself must be a significant parameter. The manner in which the bed expands is also important because of its marked effect on the particle dynamics. A number of experimental studies are reported in the literature but too often they fail to contribute to the understanding of jigging. This conflicting information appears to arise because many studies were carried out on a narrow set of ideal conditions that resulted in behaviour quite unlike that associated with practical jigs.

Theories of jigging also tend to describe the mechanism of separation in pulsating water in general terms but without any complete understanding of the processes involved. The most favoured explanation is that stratification occurs in the jig bed as a result of a combination of three mechanisms:

1. The bed is lifted *en masse* in the upward flow but differential acceleration occurs at the end of the pulse when the water reverses its flow to a downward direction.
2. The particles are subjected to hindered-settling conditions during most of the downward flow.
3. Interstitial trickling takes place during the final part of the downward flow before the pulse changes direction for the next cycle.

The above mechanisms assume initial upward flow without stratification; the materials being lifted *en masse*. This is probably not so, and since there is no direct experimental confirmation of the actual behaviour of the particles concerned, a complete understanding of the separation process still remains to be achieved. Similarly, while repeated dilation and compaction of the bed takes place as a result of water pulsation, the separation mechanism has not yet been demonstrated theoretically in quantitative terms. Holtham (1991) notes suggestions by workers in coal and other fields that interstitial trickling mechanisms should be looked at more closely. Investigations by researchers outside the field of mineral processing have indicated that the 'interstitial trickling' mechanism is likely to be a strongly geometrically dependent effect – large particles being displaced upwards by small particles when all particles are of the same density (Rosato *et al.*, 1986). It is also likely that the deterministic approach implicit in the existing mechanisms, which effectively consider the motion of a single particle in isolation, will have to be abandoned in favour of a stochastic or probabilistic approach (Vetter *et al.*, 1987).

The practical importance of jigging is that stratification occurs with the denser minerals at the bottom of the bed and the lighter minerals at the top. Two main jig types, rectangular and circular, are linked through a trapezoidal jig form. Figure 8.26 compares the flow pattern over a rectangular jig with the pattern of flow over a trapezoidal jig section of a circular jig.



8.26 Jig flow patterns – rectangular and tapered.

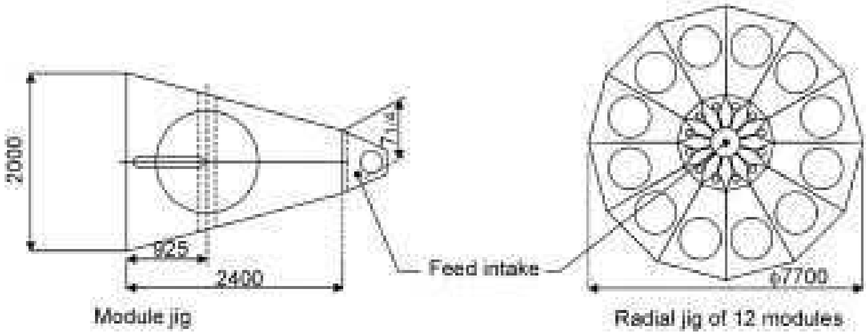
Rectangular

A rectangular jig consists either of one jig-cell acting alone, or a series of cells arranged in modules of two, three or four cells. In a multi-cell arrangement, the tailing from the first cell passes as feed to the second cell; tailing from the second cell is feed to the third cell and so on. Usually, either three- or four-cell modules are used in roughing service. The two-cell arrangement is normally restricted to secondary circuits and cleaning circuits. The designs have not altered a great deal over the years and in modern practice, rectangular jigs are generally restricted to roughing service in installations where low first cost is a major consideration, or where the gold is predominantly sized larger than 200 microns and hence is easily recovered.

Although each rectangular jig type has its supporters, all jigs suffer the common disadvantage of low recovery efficiencies for gold smaller than around 200 microns. This is due to two main factors: increase in cross-flow velocity over the jig bed resulting from the addition of hutch water to each cell in series, and the nature of the jig action. The combination of a continuous supply of hutch water and water displaced by the forward stroke of the diaphragm induces a rapid upward flow of water through the ragging and mineral bed, followed by a slower downward flow on the reverse stroke. The entire jig bed (ragging plus mineral bed) is dilated on the forward stroke and compacts on the reverse stroke. The tendency is for fine and flaky gold to be held in suspension and carried away in the tailing.

Circular

The circular jig arrangement provides the potential for a very large throughput in a single unit, with a single feed point. The result is a feed distribution without splitters and an improved pattern of cross-flow with reduced side effects. The resulting better control of hutch water provides improved recovery of finely divided gold at higher feed rates. The limiting factor is the ability of the jig to recover a maximum amount of recoverable gold in the concentrate, and produce a tailing that can be rejected without the need of further treatment per unit time.



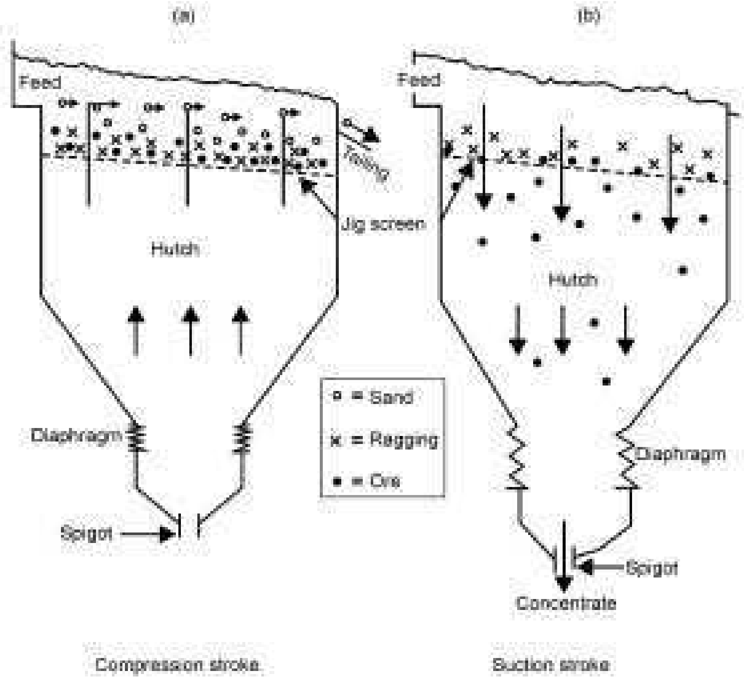
8.27 Construction of radial jig – 12 modules to complete full circuit.

Circular jig types are typically constructed from 12 modules (30º segments of circles) which together complete a full circle (Fig. 8.27). Because flow takes place radially over a steadily increasing jig bed area the flow velocity decreases outwardly thus providing improved conditions for fine particle settling. Mechanical skimmers, which even out the flow over dead spots developed in non-pulsating sectors, attack the problem of dead spots through a combination of improved distribution around the central feed point, closer control of backwater, and the use of non-clogging jig screens. Perforated retarding plates and a longer feed intake length have been found to provide better flow deceleration at the entrance to the jig bed than the early form of distribution through a spigot under pressure.

The modern circular jig is manufactured with diameters up to 7.6 m and capacities up to 250 m/hr. Their main disadvantage is batch-type operation, which lowers their overall efficiency and reduces overall production time.

Jig compartment

Regardless of type, the jig is essentially an open tank filled with water with a slightly inclined screen at the top and provided with a spigot on the underside for drawing off the concentrate. Screen compartments comprise shallow flat trays having perforated bottoms loaded with coarse heavy particles such as haematite pebbles, steel punchings or steel balls to a depth of 25–50 mm. This material, known as 'ragging', is coarser than the screen apertures and with density intermediate between that of the minerals to be separated. Its purpose is to provide a non-cohesive layer of pebbles above which the resulting dilation and compaction of the bed will cause the minerals to stratify with the densest at the bottom of the bed and the least dense at the top of the bed. A hutch, fitted with a concentrate discharge valve or spigot, hutch water inlet valves and a diaphragm of moulded, reinforced rubber is located below the perforated screen plate. The diaphragm is activated mechanically through an external eccentric drive mechanism or through a mechanical/ hydraulic drive (Fig. 8.28)



8.28 Basic features of jig compartment during (a) compression and (b) suction.

Feed

Primary jig feed normally comprises deslimed minus 10 mm screen undersize fraction, although the sizings may be coarser than 25 mm in exceptional circumstances (e.g., if the gold is very nuggetty). Secondary and tertiary jig feeds are classified according to the size of the screen plate apertures. Pulp density may range from about 10–30% solids by weight, but is optimal around 25%. Jigs operate most efficiently when deslimed feed is introduced evenly and at a controlled pulp density across the full width of the live section. The hutch water should not contain more than 1.4% solids. Too little water results in uneven solids distribution and a tendency for the development of 'dead' spots in the jig bed. Slime affects slurry rheology through changes in apparent viscosity and density and is detrimental to the recovery of fines. Allowable slime tolerances are probably much lower than 5%.

Jig bed

The term 'bed' in jigging practice refers to solids on top of the jig screen (Nio, 1988). It is formed during the first stage of separation when the heavies in the jig compartment work their way down into the ragging. The light materials overflow the hutch to be either further processed or discarded. Particles of gold larger than the screen plate apertures accumulate in the ragging and are recovered when the screen basket is cleaned. The smaller, heavy particles are further concentrated as they work their way down through the screen plate and into the hutch. This bed consists of a bottom bed of ragging, which is usually introduced artificially, or in some instances is built up naturally by heavy mineral constituents in the feed.

Ragging variables affecting the separation character of the jig bed are density and size. The ragging should be intermediate in density between the gold and the tailing and sized so as to pulsate and provide open spaces for the gold to move about in. Ragging sizes usually vary from about 5 mm to 20 mm but should not be so small as to cause blinding of screen apertures or too large for its functions to be impaired. As a general rule:

- The thicker the layer of ragging the less concentrate will pass through and gold losses will be higher.
- Very dense ragging reduces the amount of concentrate recovered because excessive velocity of up-flow is needed to dilate the bed; this throws some of the heavies up with the lights making it easier for them to be lost.
- A larger ragging size will increase the amount of concentrate but this will be of lower grade.

A disadvantage of all non-continuous jigging processes is that recovery falls away as the ragging becomes increasingly adulterated by the heavy rock-forming minerals (e.g., pyrite, tourmaline, pyroxene and magnetite). The jig bed becomes a mixture of mainly lighter heavy minerals and heavy minerals (including gold) and ragging. The grade of the gold-bearing concentrate passing through into the hutch rises and eventually reaches a point of virtual equilibrium at which both percentage recovery and concentrate grade are optimal. While this is happening, the lighter heavy minerals continue to accumulate in the ragging, and concentrate production declines as less space remains for movement of the heavies and smaller gold particles. Upward pulsations may be increased to move the heavier jig bed but the operation will ultimately become uneconomic, as more gold is lost to the tailing. At this stage the tray must be replaced with a freshly loaded tray of clean ragging. The contaminated ragging is washed and the gold recovered by panning or amalgamation. In a multi-stage rectangular jig the first compartment will normally require the most frequent cleaning.

Sand bed

The sand bed is a layer of coarse and unclassified sand formed by the feed less the material passing in the ragging and the top flow. Freedom of movement of the mineral particles in this layer is essential for good separation during dilation and fluidisation of the bed. Optimal dilation of the bed is given by a fast, short upward stroke to lift all of the mineral particles and a long, slow return slope allowing them to settle. Constraints to good separation are caused by compaction of the sand bed due to excessive spigot discharge or lowering of the tailing discharge height and loosening of the sand bed as the result of excessive inflow carries with it the risk of lifting fine gold particles into the overflow fraction.

Jig screen

The jig screen is the final point of separation between light and heavy particles smaller than screen aperture size. Aperture size, which typically ranges between 4.5 mm and 6.4 mm, is so arranged as to provide open spaces over about 20% of the total screen area. In trashy flow conditions, blinding causes channeling of the flow and uneven dilation across the bed; losses mount rapidly and production time is lost in frequent shut down periods for changeover and maintenance. To minimise these problems the modern jig screen is fabricated as a rigid punched-hole screen backed by a soft rubber punched-hole screen. More recently, IHC has produced a full rubber screen with a two-layered, hard rubber backing and soft rubber top. The rubber screen has holes smaller than the concentric holes of the stainless steel screens giving each hole a moveable rubber rim. The mineral bed thickness and slope of the cross-flow are regulated by upward and downward adjustments of the tailing weir. A shallow weir increases the top water flow rate, which should generally be less than 0.5 m/s. The average sand flow velocity across the jig is considered to be optimal at 0.076 m/s (Nio, 1988).

Spigot discharge

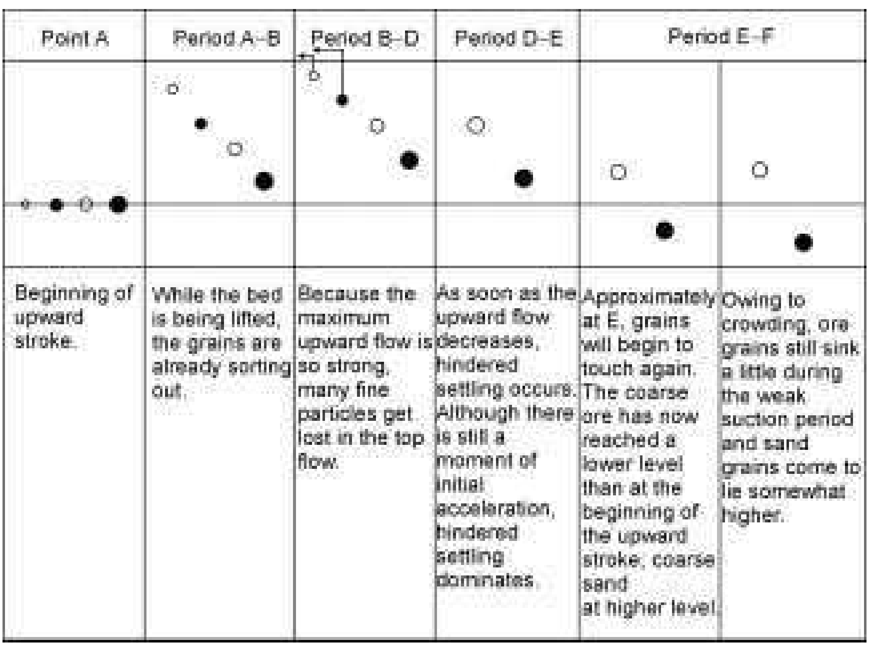
Collection and discharge of concentrates within the jig hutch are accompanied by the inflow of water to compensate for water passing out with the concentrates. Discharge of solids and water from the spigot alters the balance between upflow and downflow. Jig separating characteristics are held steady by increasing the intake of hutch water proportionally to the amount of water discharged. The balance between upflow and downflow is maintained by increasing the intake of hutch water to compensate for the lack of balance caused by removal of solids and water from the spigot. The concentrate discharge can be controlled in a number of different ways. A simple method is the use of a rubber plug with a cone-shaped orifice, generally tapering from around 12 mm to 10 mm at the outlet. Gooseneck arrangements with a simple wooden squeeze clamp are reasonably satisfactory for small operations. ‘Non-chokeable’ spigot cyclones use less water but the flow increases with wear.

Pulsation

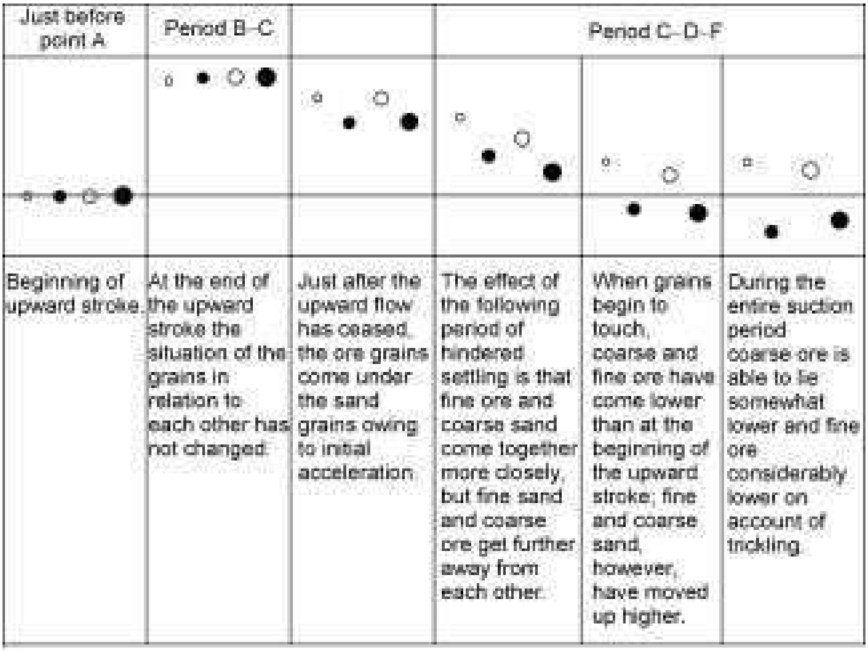
Achieving a reasonable balance between stroke and frequency depends upon the nature of the particles, particularly the size-density relationships. The stroke length is largely determined by the upper size limit of the solids and since this is at its maximum level in the primary jigging circuit, the primary jigs will normally operate with longer strokes than in the secondary and tertiary circuits. Pulse frequency is adjusted inversely with the length of the stroke. Standard stroke lengths vary over the full range of Yuba Jigs from 3.175 mm up to 76.2 mm advancing in increments of 3.175 mm. Frequencies range generally from 350 to 125 strokes/minute.

Drive

The diaphragm is activated by drive systems ranging in type from simple eccentric drives to semi-hydraulic and hydraulic drives. The action of the diaphragm provides a continuous sequence of rapid pulses during which the direction of flow is reversed sharply, several times each second. Modern practice is moving away from the conventional harmonic waveform jigging cycle (Fig. 8.29) to an asymmetrical fast upstroke-slow downstroke pattern. Figure 8.30 illustrates the jigging characteristics of the IHC jig using a special shaped cam drive system to produce a fast upstroke–slow downstroke (saw-tooth) pattern (Nio, 1988). According to Nio, the range of fine gold recovery usually extends from about 60% of 150 μm gold for conventional rectangular jigs to about 85% of 150 μm gold for jigs with mechanical/ hydraulic and fully hydraulic drive



8.29 Conventional harmonic waveform jigging cycles (after Nio, 1988).



8.30 Asymmetrical jigging cycle – fast upstroke-slow downstroke pattern (after Nio, 1988).

systems, which include assymetrical movement of the diaphragm. Current research is aimed at developing methods of monitoring bed dilation to achieve automatic control of the jigging process; this has not yet been achieved.

8.4.3 Centrifugal separators

Metallurgical cyclones in combination with enhanced gravity devices have their greatest application in the circulating load of grinding circuits. Strategic arrangement of these devices ultimately produces a finely divided product for chemical leaching and a coarse gold concentrate for final upgrading and smelting in the gold room.

Metallurgical cyclones

The metallurgical cyclone is the most commonly used classifying device for fine particle sizing and desliming in both alluvial gold processing plant (Section 8.3.4), and for closed circuit grinding in modern chemical leaching plant. Good cyclone separation depends upon control of pressure drop, pulp density and apex size. The 'pressure drop' may vary but should not change rapidly, and is held at safe levels by keeping an adequate sump level. A falling sump level causes cavitation in the pump and reduction in feed rate; pressure drop in the cyclone falls and solids report increasingly to the overflow until the drop approaches zero and the entire slurry stream passes into the underflow. Additionally, while maintaining the required separation parameters, the pressure drop should always be minimised to minimise energy losses, thus reducing pump and cyclone wear.

The maximum 'pulp density' is usually about 50% solids by weight; above that level small fluctuations in density will seriously affect separation. A cone-shaped discharge of 20º–30º reduced angle usually produces optimum conditions for separation. Cyclone control is best obtained by optimising the feed density. With consistent ore types the cyclone feed density is a good indicator of cyclone overflow sizing. A ropy cyclone underflow indicates a very high-density state with a risk of plugging the 'apex'. If control cannot be exercised either a larger apex is needed, or the addition of another cyclone. Striking an economic balance between high D_{50} and low D_{50} separation requires:

- increasing the fine sands content of the overflow and build up of the circulating load until the cyclone feed density increases to the point of coarser separation
- higher hydrocyclone pumping pressures with consequent higher power and maintenance costs; if the mill is unable to grind the ore at the given feed rate the final grind will not be any finer no matter what adjustments are made to the cyclone.

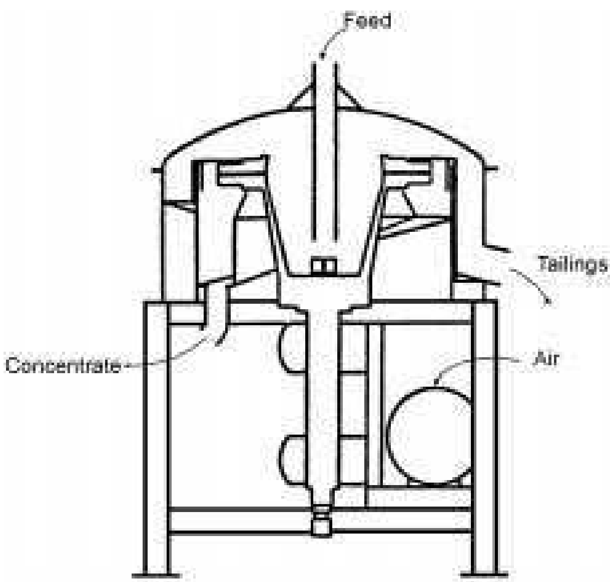
It is important to recover coarse gold as soon as possible to minimise fragmentation and smearing onto other particles during multiple passes through the grinding circuit. Unit capacity cost of an enhanced gravity concentration device is high and in order to minimise the capacity of an installed enhanced gravity concentrating device, conventional practice is to connect such a device to only a fraction of the cyclone underflow. It is considered that, statistically, any free gold escaping the grinding circuit via overflow processes will follow the laws of probability and eventually report to the gravity concentration device after several passes through the grinding mill. McAlister and Sprake (1999) propose that the incoming feed to the cyclone can be scavenged prior to cycloning. The cyclone feed pipeline is so arranged that an outlet under pressure can be installed on the bottom of a line at the end of a straight section that is either horizontal or inclined. Material taken off can be elevated to a sizing screen under its own pressure, the coarse fraction passing directly to gravity separation.

Gravity enhanced centrifugal separators

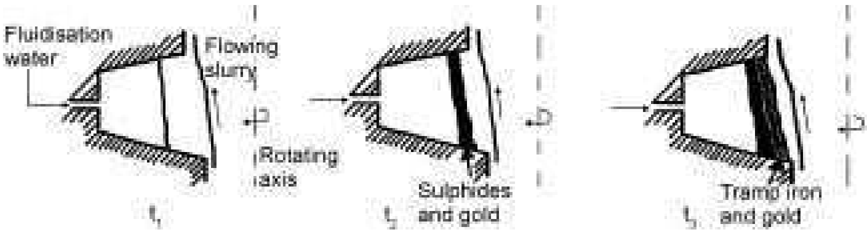
Gravity enhancement is an important design feature of modern grinding/classification circuits. Most gravity treatment devices (e.g., sluices, spirals and cones) operate at a 1 g gravity field, but the method has been extended to recover very finely sized gold particles with the development of devices that use high centrifugal forces at gravity fields up to 300 g. High centrifugal forces increase the terminal velocity of particles, thereby enhancing the gravitational differences between particles of different densities making them easier to separate. Concentrating surface area is the single most important factor determining unit capacity of all centrifugal separators; the larger the concentrating surfaces subjected to high concentrating fields the higher are the unit capacities. The coefficient of friction at the interface between the concentrating surface and the pulp is critical. The lower the coefficient, the lower will be acceleration of the pulp, which will then experience correspondingly lower centrifugal separation forces, thus reducing its capacity compared with that for a unit of similar size with a high coefficient of friction. Nelson, Kelsey and Falcon centrifugal concentrators are the most common types used in gold-recovery systems.

Nelson concentrators

The Nelson batch-type concentrator (Fig. 8.31) consists of a central fluidised cone, which is fed with slurry containing up to 75% solids. The cone rotates at a peripheral speed designed to generate 60 times the force of gravity. Fluidised rings around the cone walls allow the heavier particles to separate and be captured according to their apparent densities. The lighter particles discharge from the top of the cone and pass to the tailing launder. Selective recovery



8.31 Nelson centrifugal concentrator.



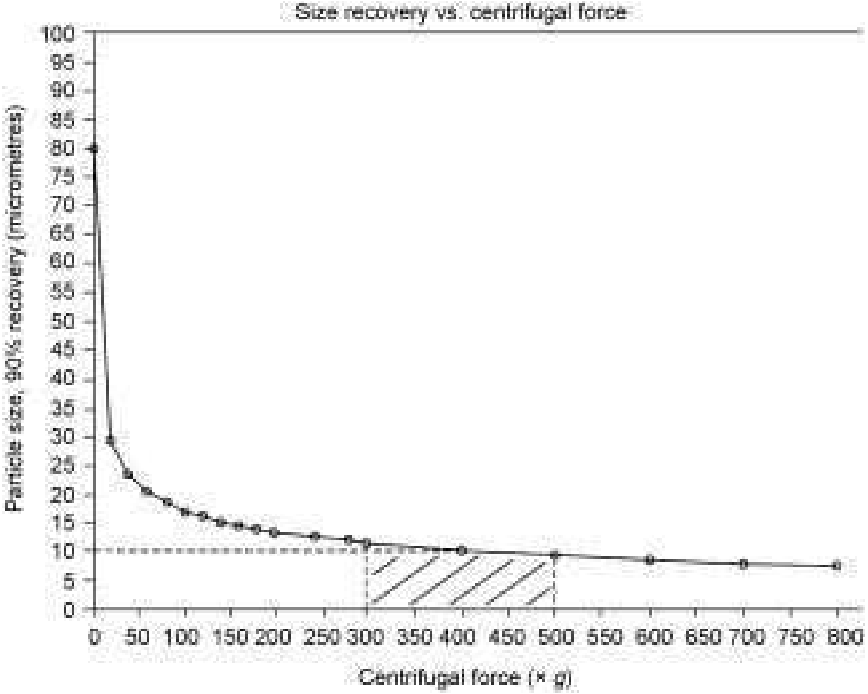
8.32 Schematic representation of the Nelson recovery cycle for a single groove.

commences with the formation of a concentrate bed containing gold and other heavy minerals at the top of the non-fluidised material. After a short time the lighter heavies (e.g. pyrite) begin to experience significant erosion from the top of the concentrate bed, and build up takes place of some high-density minerals and tramp iron. The driving force for separation of metals and mineral diminishes with the increasing grade and density of the fluidised bed and ultimately, as the gold, tramp iron and other heavy minerals build up on the concentrate bed, they themselves experience significant erosion and gold recoveries fall away. A schematic representation of the Nelson recovery cycle for a single groove in the cone is given in Fig. 8.32. The bottom of the groove is initially filled with coarse barren particles with a high settling velocity to form a jig bed. Recovery of this material, which will remain at the bottom of the groove throughout the jig cycle, is due more to particle size than to density. The unit is automatically closed down at the end of the concentrating cycle and the high-grade concentrate is flushed to a separate collection tank. The flushing process should take only a few minutes before the unit comes on stream again.

The Nelson 'Continuous Variable Discharge' (CVD) concentrator has been developed and refined to its present use in largely hard rock gold milling circuits. The difference is in the rotating assembly of the CVD. The most significant differences relate to pinch valve geometry; pinch valves fitted at the base of the fluidised rings are significantly larger than are those of the batch machines. Apart from overcoming some of the disadvantages of batching, the most frequent maintenance requirement is replacing pinch valve assemblies, which can be done easily in an hour or two during stand down periods.

Kelsey centrifugal jig

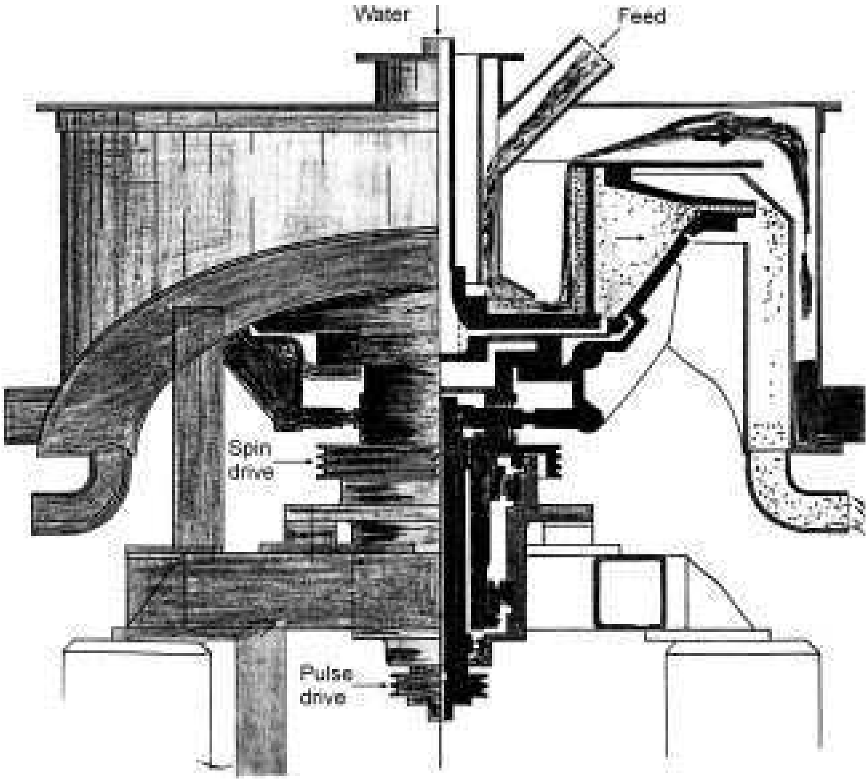
The Kelsey jig has reached an advanced stage of experimentation since the concept was first patented in 1972. Features of the Kelsey jig, considered by the manufacturers to be unique, are flow continuity, feed rate variance adaptability, consistent high recovery efficiency and tolerance to slime. 90% recovery of plus 5-micron size gold is claimed for beach sand ores in a single pass at a throughput of 150 tph. The concept utilises all of the parameters of the conventional jig but changes the apparent gravitational field by applying higher centrifugal forces, up



8.33 Kelsey graph of recovery vs. centrifugal force.

to 350 times gravity, to enhance the separation characteristics. The Kelsey operating curve (size vs. centrifugal force) is shown in Fig. 8.33. The ability to change the gravitational field is achieved by variable control of the spinning rotor. Inside the rotor a screen of parabolic shape is spun co-axially with the rotor. The screen is lined inside with the ragging spread again by the co-axial centrifugal force, evenly across the inside surface of the screen.

The model 'J650' jig (Fig. 8.34) has a nominal capacity of 12 tph solids at 300 to 200 microns. The feed slurry ranges between 15% and 60% solids screened to 1.5 mm, with make up water 120 l/minute depending upon feed composition. It enters centrally into the bowl and is distributed over the ragging. A high frequency pulse applied to the water in the concentrate chambers causes the ragging to dilate and contract at the same frequency. Particles of higher density move through the ragging and retention screen and are discharged through the rotating concentrate spigots. The less dense particles are displaced upwards by the incoming feed and flow over the retention plate into a separate launder. Hutch make-up water, i.e., the additional clean water required for the pulsing flow, is fed through the spinning centre pipe. The largest model 'J2300' Kelsey jig is stated to be capable of processing up to 150 tph of coarse high-grade alluvial ore.

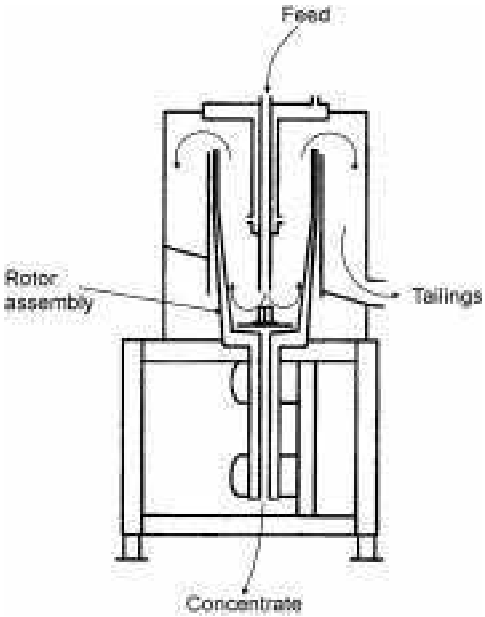


8.34 Schematic diagram of Kelsey Model J650 jig.

Falcon continuous concentrator

The SB model Falcon concentrator (Fig. 8.35) is a fluidised bed spinning bowl gravity concentrator designed for gold recovery from both grinding circuit cyclones and alluvial operations. Accelerating the feed slurries in a rotating cone-shaped bowl induces gravity enhancement. Within limits, the faster the bowl is spun the higher the G forces and hence the greater the tonnage that can be treated by the unit. A sacrificial impeller is provided at the bottom centre of the rotor immediately below the discharge of the feed pipe to ensure that the pulp is given sufficient acceleration. This device, which is relatively inexpensive compared to the cost of other parts of the rotor, is responsible for most of the pulp acceleration and can be changed in less than one man-hour. In addition to the impeller, the wall of the rotor has been lined with a natural gum rubber which, having a soft surface, further accelerates the pulp as well as protecting the rotor from abrasion.

Material fed in slurry form into a high-speed rotor mechanism in the concentrator system migrates up the concentrating zone retaining its desired

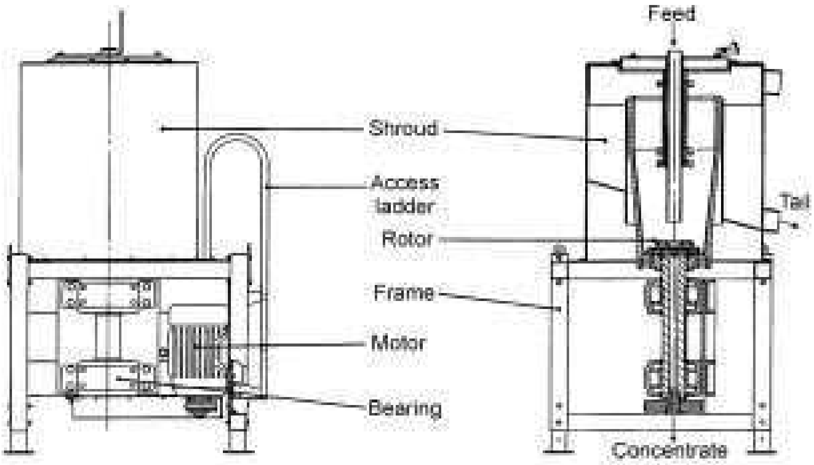


8.35 Falcon flow system.

high-density concentration. A continuous stream of high-density gravity concentrate is bled away from the flow through a bank of hoppers and valves. The bleed-off is electronically adjustable, the valves being operated automatically by compressed air through internal porting. Concentrate is recovered from the rotor automatically or manually at predetermined intervals; reject barren material is conducted out of the rotor reports through a separate discharge port as tailings.

The patented Falcon design (Fig. 8.36) has few moving parts and is easily accessed for preventive maintenance. Process surfaces are rubber lined for maximum wear resistance. Features include up to 300 *g* centrifugal field, no addition of make up water required and minimum operator attention. Centrifugal forces range from 50 *g* to 200 *g* at maximum feed pulp density of 50% solids in water. Solids capacity of 0.25 tph to 55 tph of particles ranging in size from 2 mm down to a few microns. The advantages claimed for the Falcon concentrator are:

- the machine needs never be off-line for rinsing
- high wear resistance of all parts in contact with the flow
- no water added during processing
- minimum operator attention
- few moving parts
- high unit capacity due to high depth to diameter ratio for a given bowl diameter



8.36 Falcon centrifugal separator – nominal specifications.

- availability of a programmable automatic controller to regulate the rotor speed, power draw and ramp up during start-up, and percentage of flow of material reporting to the concentrate
- factory produced metering head assemblies that are available on an exchange basis.

8.5 Lateritic gold metallurgy

All significant characteristics of a lateritic ore likely to influence eventual prototype design must be identified prior to commencing a chemical leaching programme. Preliminary data on how the various materials are likely to respond to conventional methods of slurrying and desliming are provided by pilot-scale testwork conducted in association with drilling and sampling. Laboratory log sheets provide descriptions of the mineralogy and distribution of the other heavy minerals reporting with the coarse gold. Pre-concentration and roasting may be required for some primary ores; coarse (+200 mm) gold liberated by size reduction in the crushing circuit can be recovered by conventional gravity means. Carbon in pulp (CIP) absorption by slow agitation is the most common method used in primary gold leaching circuits, however the following factors usually influence the selection of carbon in leach (CIL) processing methods for the treatment of most lateritic type gold ores:

- Removal of the gold from solution at the time of dissolution minimises its exposure to organic material in the slurry, thus reducing the gold losses associated with those organics; residual gold deposits tend to be contaminated with significant quantities of organic matter and if this material is allowed to reach the leach circuit it will adsorb some of the dissolved gold and tend to block interstage screens; a high frequency screen will reject wood chips or other such debris that might remain in the slurry but regardless of what precautions are taken some contamination of the leach circuit is likely to occur.

- The capital cost of a CIL process in which adsorption of gold occurs in the leach tanks is smaller than that of a CIP process; the CIP process requires an additional smaller series of CIP tanks although CIL operating costs may be slightly higher because of a larger carbon inventory.

8.5.1 Feed preparation

Run-of-mine material is passed over a grizzly screen to remove oversize and trash; lateritic boulders larger than about 150 mm are spalled to a size suitable for grinding. A drum type scrubber is usually preferred for breaking up lumps of clay associated with the lateritic nodules. Any residual organic trash in the product will be removed from the top deck of a double deck screen. Coarse (+2 mm) solids will be discharged from the lower screen and transferred to a coarse ore stockpile as feed material to the grinding circuit. Fine undersize clayey material (−2 mm) scalped from the lower screen will be pumped to a bank of cyclone separators in the grinding circuit; the overflow comprising very fine material will bypass the grinding circuit and proceed directly to the leaching circuit.

Eliminating as much organic trash as possible away from the treatment plant and tailing pond water reclamation areas is a priority because of its deleterious effect on gold recovery. Although large trash is removed during the mining operations and most of the finer trash at the scrubbing-screening stage, the leach plant feed material will inevitably still contain a small amount of dilutionary material including any smaller plant fibres that have not been removed. Possible future problems such as fine screen blinding must be identified so that they can be compensated for in the leaching circuit design.

Sizing

Screen sizing as an adjunct to separation in gold milling circuits is limited in its applications because vibrating screens have increasingly poor performance characteristics at smaller apertures and the flow rates are comparatively large. Mechanically operated screens are restricted to aperture sizes larger than 2 mm and some form of hydraulic classification is preferred for the finer sizings. Vibrating screens are used mainly to remove trash ahead of gravity enhanced centrifugal separators and for the recovery of carbon in the leaching circuit.

Machines of the Dorr-thickener type and spiral classifiers are engineered to provide an effective pool area and overflow velocity for settling in accordance with the particular size separation requirements. However such traditional thickening device types have been replaced by metallurgical cyclones for sizing or desliming large flow volumes of slurry because they are comparatively cheap and occupy little headroom. As integral features of closed circuit grinding, metallurgical cyclones also provide the greatest opportunity for gravity concentration by recovering most of the gravimetrically recoverable gold in the cyclone underflow free of slime. The initial recovery of coarse gold minimises fragmentation and smearing onto other particles during multiple passes through the grinding circuit.

8.5.2 Chemical leaching

Chemical leaching testwork will usually involve continuous pilot plant testing by some selected chemical leaching process (e.g., carbon-in-leach) of batch samples of ore, that are as near as possible representative of future mill feed. Environmental protection measures and hazard costs have militated against the use of cyanides as a gold lixiviant and ammonium thiosulfate is being extensively tested as a substitute to replace cyanide in gold leaching. Bacteria are also used for the purpose but in the following treatment of the subject cyanidation processes have been preferred. Ancillary bond mill grindability testing, thickening and carbon stripping studies will be carried out as ancillary processes. The test work should indicate:

- amount of gold that can be recovered by gravity methods of concentration alone
- size of optimum gold particle liberation
- settling characteristics of the ground ore
- work index based upon bond, ball mill grindability tests
- amount of gold that can be recovered by chemical methods of leaching and carbon-in-pulp processing from the slime and fines particles.

Grinding circuit

SAG mills are particularly suited to the first stage of breaking down residual-type gold ores because they always produce a very fine overflow, which can pass to the leaching circuit as a final product. In addition to the benefits provided by good operational control, SAG mills have the advantage of low capital cost, usually about 25% below that of a conventional crushing/grinding circuit. By eliminating the need for crushing, maintenance costs are also reduced. Steel ball and liner consumption is lower although these savings are somewhat offset by higher power consumption charges per tonne of ore milled.

Conventional ball or rod milling in closed circuit follows autogenous grinding. Fines from the SAG mill and the ball mill are combined with the scalp screen underflow and pumped to cyclone classifiers from which the coarse underflow will pass to jigs in a gravity concentration circuit. The presence of appreciable quantities of coarse gold in the SAG mill cyclone underflow will indicate the need for some form of gravity concentration (e.g. jigs and tables) ahead of the leaching circuit. Coarse gold requires long leach-retention times (up to 24 h) to ensure complete dissolution and removing it will facilitate leach control by reducing fluctuations in the head feed. So called 'rusty' or surface coated gold is highly refactory and may fail to dissolve completely. Metallurgical efficiency need not be high in regard to the recovery of the finer gold particles at this stage because all gravity rejects are returned to the ball mill circuit. Concentrates produced by the jigs are cleaned on a shaking table prior to smelting.

Cyclone overflow passes to the leach circuit surge tank where sodium cyanide solution and slaked lime is added to monitor and control slurry density, solution pH and cyanide concentration. A certain amount of gold is dissolved in the scrubbing and grinding circuit and will be present in the thickened overflow solution. Some recovery of this gold may be made by slurrying the ore using weak cyanide solution reclaimed from the tailing dam. Au stripping is carried out by washing the recovered carbon granules in a solution (cyanide + NaOH) at 135 ºC, (retention time 6–8 h). Carbon reactivating is done in a kiln at 125 ºC to restore the active surfaces.

Calculation of kinetic constants

The kinetic activity of carbon is an important factor in determining the efficiency of carbon adsorption. The use of activated carbon for gold recovery depends initially upon the physical resistance of the carbon granules when submitted to abrasion forces within the pulp and impact forces against agitators, pipes and tank walls. The production of fines results in costly losses of both carbon and of gold carried away by the fines during operation. The adsorption characteristic of activated carbon is a compromise between gold loading capacity and adsorption characteristics on the one hand and hardness on the other. The adsorption of gold from solution onto activated carbon generally obeys the following empirical relationship:

$$[\mathrm{Au}]_c = k \times [\mathrm{Au}]_s \times t^n \; * \qquad 8.17$$

$[\mathrm{Au}]_c$ = increase in gold loading on the carbon (mg/l)
$[\mathrm{Au}]_s$ = gold tenor of solution in equilibrium with the carbon in the contactor (ppm)
k = empirical rate constant dependent upon slurry mixing efficiency and carbon particle size
n = related to gold loading capacity of the carbon
t = elapsed time (hours).

* Although the above model was derived for continuous CIP plants, experience has shown that approximate values for *k* and *n* can be determined by applying regression analysis to the data from the first six hours of a sequential CIL test (Fleming, 1982).

Carbon-in-leach processing

The CIL process enables recovery of gold from slurry by mixing activated carbon granules of a coarse size with the slurry particles. Slurry is pumped from the surge tank to the first of a series of mechanically agitated leach tanks. Laboratory testwork will indicate an optimum leach retention time for the ground material. This retention time is provided by the feed surge tank and by the number of leach tanks in the series. Gold adsorbed from the solution by the carbon will then be recovered from the slurry by screening.

Carbon returning from the stripping circuit is introduced into the final leach tank and is moved through the leach circuit semi-continuously through five of the six leach tanks in a counter-current direction to the direction of the slurry flow. Carbon will advance through each stage to the second leach tank where it will be removed and transferred back to the stripping circuit. The first tank will not contain carbon and will be used only for leaching to ensure that solution gold tenor in the feed to the second leach tank is sufficiently high to enable optimum gold loading on the carbon. This in turn reduces the carbon inventory and the size of the subsequent stripping circuit.

Leached slurry leaving the final leach tank passes to a vibrating emergency screen, which is placed in the circuit to prevent catastrophic loss due to screen failure in the main launders. The aperture of the final vibrating screen will be marginally smaller than that of the launder screens in order to minimise gold losses associated with carbon fines. The carbon fines so retained will be treated separately to recover their gold content. Screened slurry will report to an agitated tailing surge tank with a 30-minute or so residence time, before being transferred to the tailing pond.

Carbon stripping and regeneration

Despite environmental constraints, cyanide is usually chosen for leaching because of its almost universal applicability to all ore types. The use of other leachants (e.g., thio-urea, ammonia and chlorine) could be considered if the testwork does not respond favourably to cyanidation, or if the location is subject to environmental restraints. Gold is stripped from carbon by passing a caustic cyanide solution preheated to 90 ºC through two or more high stripping columns containing loaded carbon on a batch process. The carbon is treated with dilute hydrochloric acid in the working column to remove carbonates that may build up due to the addition of lime to the leach circuit.

Other acid-insoluble fouling agents such as organics may also build up on the carbon and lower its activity; provision is thus made to regenerate the carbon in a small vertical oil-fired furnace at 700 ºC. Regenerated carbon will then be air cooled and screened on a small vibrating screen to remove fine carbon prior to recycling to the leach circuit. The screen is marginally coarser than those in the leach circuit to ensure that carbon particles are effectively retained in each leach tank.

Loaded carbon recovered in the leach circuit is eluted for a pre-determined period in a series of columns, with hot 1% caustic solution containing 0.1% NaCN at atmospheric pressure. Stripped carbon is then washed with dilute hydrochloric acid and regenerated in an indirect oil-fired furnace to remove any impurities such as carbonate or organics that may build up on the carbon during leaching. Regenerated carbon is cooled and screened to remove degraded fine carbon prior to recycling to the last leach tank in the CIL process. Pregnant gold-bearing eluate from the carbon stripping section will be pumped to a pregnant solution storage tank to provide surge capacity ahead of the electrowinning circuit.

8.5.3 The gold room

Coarse gold concentration

Amalgamating barrels, centrifugal separators and wet shaking tables are used to upgrade gravity plant concentrates prior to smelting. Size classification is essential for efficient separation and supplementary equipment will normally include a small vibrating screen and a cyclosizer. Vibrating screens provide more precise size separations in the dry state than any other screen type. Dimensions of width and length, which determine the main separating characteristics of conventional screens, are matched to suit the characteristics of the plant concentrates. Screen capacity is a function of width; efficiency is a function of length.

The associated heavy minerals (e.g., magnetite and ilmenite) are unlikely to cause any screening problems. The main difficulties are found with odd-shaped gold particles, angular fragments of some rock-forming minerals such as tourmaline, and fine rock particles that have held back with the heavies. Such fine particles may give rise to dust problems if dry, and clogging when damp. The Gemini Table is one type of dry shaking table for recovering gold smaller than 1 mm in size. Conventional screening standards usually require surface moisture to be less than 3% by volume.

Slurries of particles larger than 200 microns may be sized on small vibrating screens to provide feed to wet shaking tables at the final stage of coarse gold concentration. Smaller sizings are made using some form of hydraulic sizing. Hydrosizing is a common, if rather inefficient procedure in dressing shed operations where sized fractions are required for feed to such units as shaking tables, belt separators and centrifugal separators. A hydrosizer comprises a series of compartments increasing in size in the direction of flow. The velocity of the surface flow is highest in the first, smallest compartment where only the largest, heaviest particles can settle out. The velocity reduces across each successive compartment thus allowing the solids to be sorted preferentially according to their settling rates in these conditions. Each compartment has a spigot discharge that may be operated normally or mechanically, depending upon the level of sophistication of the unit.

Electrowinning

Gold-bearing eluate is pumped from the carbon strip column to a pregnant solution storage tank. The gold is recovered by passing the pregnant eluate through an electrowinning cell, which houses cathode baskets each measuring, say, 0.9 m × 0.9 m × 1.5 m. Gold will plate out on steel wool contained within each cathode basket up to a maximum loading of about 1300 g Au/kg of steel wool. The maximum flow rate through the cell may be about 15 g/min. Barren solution leaving the electrowinning cell is returned to the barren solution storage tank and recycled to the carbon-strip section.

To ensure maximum recovery of gold, cathodes will be progressively moved from the back to the front end of the electrowinning cell in a counter-flow direction to solution flow. When the steel wool is fully loaded with the gold, it is removed from the cathode baskets and smelted with suitable fluxes in a small reverberatory furnace to produce a final product gold Dore bar.

9
Evaluation, risk and feasibility

Implicit in any framework for economic evaluation is the expectation of eventually recovering the investment plus a rate of return commensurate with the risks involved. Implicit also is the expectation that required finance will be made available when required for each significant phase of the project. All decisions, such as commencing a new operation or introducing a new technology, require an objective analysis of the various options to ensure that all costs and benefits are considered. Distinguishing between financial and technical evaluations is usually the source of most conflict. Inadequate knowledge of the financial evaluation process may result in changes to the technical proposal that will endanger the process or prevent it reaching its planned potential. The overriding and single-most important parameter is determination of the ore resource and its potential for supply.

The process of evaluation thus commences with identification of a geological resource and continues through from exploration, evaluation and development to production of the product for sale. Significant aspects of the undertaking include mine planning and design, minerals processing, engineering design, manpower requirements, capital and operating costs, working and maintenance capital, marketing, environmental management, industrial relations and the social, legal and political environment. Costings become increasingly comprehensible and reliable as ore resources are upgraded to reserve status through consideration of technical and economic data including mining, metallurgy and marketing. Risk reduces progressively as all possible options for the development of the project are explored and all of the external factors influencing methods of revenue flow to the project come under control.

Ultimately, full account is taken of estimates of capital and operating costs and of production as presented in the technical evaluation, and of the estimated revenues as presented in the economic evaluation. Project financing techniques provide a means for sharing risk where conventional techniques are inadequate. The project and details of project activities will be defined so that estimates of all anticipated costs and revenues can be integrated into a financial model of the project. Ideally, the staged progress of the study will have helped identify options, which can be tested, modified or discarded and replaced until the most satisfactory combination of options has been selected.

9.1 Project management

The quality of the technical and management skills at each stage of project development determines the ability of the mining and treatment plant operations to perform up to design expectations. Maximisation of profits is then strongly influenced by grade management, mine life and rate of mining, flexibility and risk. Widely different standards on what constitutes a profit are matters for individual judgement:

- Fundamental Socialist Theory considers a mineral deposit to be payable when its exploitation is warranted from the viewpoint of the benefits it brings to the entire national economy. Conditions of particular import are the quality and quantity of reserves and the geological conditions attending the occurrence and possible importance of its exploitation for the national economy. The importance of any proposed undertaking is the extent to which a state needs a particular valuable mineral from the given deposit.
- Legitimate government objectives, on the other hand, question, intervene and in some cases control the course of mining operations. The joint and respective roles of business, governments and other groups for social responsibilities are fundamental. Exploitation of a deposit must be both economically and environmentally sound; corporations and governments have a common bond to serve the people, a proper balance being desirable between idealism and commercialism. The investor is thus allowed to make a profit while still protecting the rights of society in the context of sustainable development.

9.1.1 Grade management and control

The framework of grade management and control is built into the mine plan, which determines both the starting point and the order of mining operations. The sequence of operations determines the pattern of earnings and has a strong influence over valuation criteria such as present worth, internal rate of return and pay back. The values for all such determinants are greatly enhanced by any action of management that maximises returns during the first few years of the mine life. Concepts of grade management and control fall into two main categories: ‘constant averaging’ and ‘optimisation’.

Constant averaging

The concept of constant averaging, i.e., to maintain as far as practicable a constant grade of ore mined over the life of the mine, does not fit in with present-day techniques of financial analysis although it was once more or less standard procedure. Indeed, when the price of gold was fixed, it was not uncommon for mine managers to hold back some of the gold won during a better than average production period, in order to 'sweeten' the returns from a less favourable production period and provide a constant revenue flow.

Various authors quote references to constant averaging techniques that fix the cut-off grade at a value that maximises the total resource-profit (Ramos, 1977). These include fixing the cut-off grade at a point for which a minimum profit requirement is satisfied (Taylor, 1972), and fixing the cut-off grade for the whole life of the mine under an assumed cost/price relationship, which occurs at a value that maximises the present value at time zero. A major defect of such practices is that they all ignore the time value of money.

Project optimisation

The purpose of project optimisation is to maximise profits through the management of high and low-grade ores. By these means, the most profitable and usually, although not necessarily, the highest grade sectors are mined first and the unmined ore is of decreasing average value until the operation is no longer profitable. By optimising resource quantities the progressive falling-off in value of the material mined is controlled according to a predetermined strategy.

The concept of optimisation of cash flows recognises that the element of time enters into every aspect of the earning power or interest rate on the money involved over the life of the project. Expressed in simple terms, a sum of money earned now has a greater value than at any time in the future. Money in the bank is earning interest; money invested in a project is losing value increasingly with time according to the opportunity cost of investing in that project. The time value of money is a fundamental issue in techniques that rely upon the discounting of estimated lifetime cash flows that represent the costs and revenues of the projects to present value. These cash flows are associated with a specific set of technical options selected to implement one particular version of the resource development.

The robustness of each special version of the project can be tested by the financial modelling by means of sensitivity analysis of the cash flows associated with the major assumptions. There are, however, as many different versions of projects that can be developed in the same resource, as there are significantly different scales of development, mining methods, treatment processes, and marketing niches applicable to that resource. Project optimisation does not depend upon financial modelling alone; technical confirmation that a project version is internally valid is an essential first step in evaluation.

Thus, in putting the optimisation concept into practice, the technical requirements of the optimal project version should consist of the optimal blend of compatible project components, resource definition, mining method, treatment and market niche. This will allow an order of mining to be determined that can achieve the largest margin between the sales revenue per unit of gold produced and the full cost of production per unit of gold produced. The appropriate cost of amortising capital expenditure on the project is included in the full cost of production. Financial models of different projects will react differently to such features; hence care must be taken to ensure that each financial model accurately reflects its specific project. Evaluation of the alternative developments requires a matrix of net present values, internal rates of return, etc., to be calculated for all major parameters.

9.1.2 Mining reserve schedule

Mine life

In general, the shorter the mine life for a given resource the greater will be the economy of scale, the lower will be the fixed charges per unit and the sooner will profit be realised. Additional factors relate to fluctuating gold prices, political uncertainties and sometimes to unrealistic demands for environmental preservation rather than control. In the current financial and political environment, the investor cannot be sure that the conditions under which economic feasibility have been demonstrated will remain unaltered for the life of the mine. This provides an added pressure to shorten the mine life either through increasing the rate of mining or lifting the cut-off grade.

The mining reserves, upon which the mine life depends, are calculated for a particular cut-off grade based upon the projected costs of mining and price of gold. Both parameters are variable and significant changes, which occur from time to time, may require a re-assessment of the cut-off point:

- Lower gold prices or higher production costs call for a higher cut-off grade in order to maintain profit margins; this results in the rejection of some previously payable material, which is no longer of ore grade and the mine life is reduced.
- Higher gold prices and productivity have the opposite effect; cut-off grades are reduced and previously sub-marginal grade ore becomes payable.

Mining rate

The maximum utilisation of plant and labour is obtained on a continuous mining basis, theoretically 8,760 hours/year. Typical dredger time availabilities are illustrated in Table 9.1. However, comparative cost studies, which consider various rates of mining must also assess such possible effects as:

- lost production time due to planned maintenance and plant relocation requirements

Table 9.1 Typical dredger time availabilities (after Goh, 1987)

	Hours/year	Availability
Possible availability	8,760	100
Tin dredgers Malaysia	7,200	82.2
High-altitude dredging:		
Headline dredgers	6,400	73.1
Spud dredgers	5,760	68.8

- lost production time due to unplanned stoppages, breakdown maintenance, etc.
- reduced throughput resulting from adverse ground or climatic conditions, difficult work conditions, etc.

The rate of mining may remain fixed over the life of a small mine but it is important to leave room for flexibility in larger undertakings. The economics of large mines is usually improved by a progressive increase in the mining rate aimed at maintaining or increasing the cash flow. The process may involve ploughing back profits into the enterprise or seeking additional outside funding. The choice may be made in line with company policy or on the basis of comparative studies which show the increase in rate of return for each increment of investment.

The availability for useful work will vary according to the selected method of mining. A typical example in wet mining operations is the choice between 'headline bucket ladder dredgers' and 'spud dredgers'. 'Headline' dredgers normally have an availability of 75–80% although 82.5%, i.e., 7,200 h/yr availability is usually regarded as being reasonably attainable in average ground in the long term. 'Spud' dredgers have lower time availability in average ground conditions due to the inherent inefficiency of spud re-positioning, but may have higher time availabilities where digging is more difficult.

Cash flows

Continued economic justification for ongoing expenditure on the project at each successive stage of its development is essential during the gathering of technical and environmental data, cash flow studies are a means by which this is done. Projections from these studies are based upon estimates of how much the gold will cost to produce and what will be the proceeds from its sale. The studies will consider the effects of various methods of mining and treatment and will test the sensitivities of the projections to possible gold price fluctuations, errors in estimating grades, recoveries and mining rates. Estimates are firmed up continually and costing becomes increasingly reliable and comprehensive as the geological picture unfolds. Quick capital and operating cost estimates and financial analyses for preliminary in-house evaluations can then be used periodically to test the sensitivities of the many variables involved.

Once the relevant data has been gathered and the technical and economic feasibility of the proposal has been established, the findings must be subjected to financial evaluation. This will take account of capital and operating cost estimates and of production as presented in the economic evaluation and will investigate whether or how the project can be financed. While early studies are mostly concerned with striving to maximise profits, the final studies are concentrated on reducing financial risk and tables will be prepared to show the likely profit and loss projects for varying gold prices (currently from US$500–540 per fine oz.). Projects should then be evaluated on the basis of their ability to withstand low gold prices and other adverse possibilities in the business cycle. A detailed environmental impact statement incorporating all the data required by relevant government agencies, local government bodies specific interest groups and individual landowners is also needed before mining can commence.

Environmental impact

Awareness of the importance of safeguarding the environment is only now becoming recognised in more remote areas, and the monitoring of operations still tends to be more lax with distance away from the main population centres. Similarly, restoration standards are higher in some mining districts than in others. For example, in more closely populated areas this may require the surface to be landscaped and sown with quick-growing grasses to provide temporary stability until the slower-growing natural species can develop once more. Unfortunately, however, many operators still reverse the order of re-emplacement of spoil in fertile river flats in some regions by depositing humus-bearing material underneath gravels mined from the wash at bedrock. Similarly, sluicing and other small-scale mining operations in developing countries still involve destroying huge areas of rain forest and stripping the soil from the affected surfaces. Upsetting the balance between transport and erosion in any section of a river or foreshore upsets biological and bio-chemical relationships and destroys marine life and spawning grounds. Fishing industries are destroyed and navigational hazards are created when large quantities of sediments are mobilised and re-deposited in shallow offshore areas. Changes in the energy balance both on land and in the sea lead to conditions that call for a new set of equilibria; even comparatively minor disruptions can create major environmental problems in critically sensitive locations such as are found on hillsides in sparsely vegetated areas.

All such matters should be of as much concern to mine planners as they are to environmentalists, and most environmental impact statements now include detailed studies by specialists concerned with the particular problems likely to be encountered in zones of interest. A typical requirement is for operators to incorporate means of continuously monitoring environmental changes and to simultaneously compare the effects of those changes with conditions in a control area. Particular attention is paid to dredging operations in semi-enclosed embayments, e.g. the Lakekamu Embayment of Papua New Guinea where shallow free flowing stream channels are readily choked with slime and other trash during monsoonal rains. Although some lifestyles may unavoidably be disrupted for a time, proper planning and attention to environmental standards can usually ensure both immediate and long-term benefits that far outweigh any temporary inconveniences. Clearly, if a project cannot support the additional costs of providing an adequate level of restoration to preserve the equilibrium of the landscape against the natural forces that act upon it, it should not go ahead.

9.2 Economic appraisal

Technical evaluation is the basis of the economic appraisal, including the estimation of capital and operating costs, of throughput rates and of the product won. It is a joint effort. All of those involved in the technical evaluation should be skilled in one or other of the disciplines but should also have a good working knowledge of the other disciplines. None is isolated from the other and trade-offs are inevitable between *in-situ* resource and mineable reserve, between mining costs and optimum mill feed-grade, concentrate grade and recovery and so on.

Data for evaluation are derived from components of the mine plan, which considers ways and means of bringing the property into production. Financial projections are based upon proving up a certain minimum resource that may be mined economically in accordance with the requirements of the venture. The most suitable method of mining and treatment is determined for the particular deposit, having regard to its type, size and location. The analysis of investment worth is guided by technical, financial, economic and other studies aimed at optimising the project economics. The time value of money is of prime importance and is the basis for most evaluation techniques. A target is set for revenue expectations based upon current and projected gold prices and production. Analysis is the final step in determining annual cash flows, profitability and the present worth of the project. The discounted cash flow/net present value (DCF/NPV) technique is generally regarded as a suitable yardstick for valuation. Using this technique the proposal must be shown to rank favourably with alternative investment possibilities if it is to go ahead.

Involved with any evaluation of a major project are a review of the original resource data, recalculation of the ore resource estimate and a detailed review of the mining engineering aspects involved in converting the resource into mineable reserves and subsequently into mill feed. The practicalities of the proposed operation are examined and parameters determined that will maximise benefits whilst limiting the exposure to risk. The mining costs should be adequate for the required degree of selectivity and with equipment costs relevant to the nature and the scale of the proposed mining operations. The possible effects of any change in the quantity and distribution of mineable reserves on the mining method or mining costs should be built into the final assessment. Three interrelated disciplines of investigation are involved: ‘technical’, which is very definitely multidisciplinary, ‘economic’ and ‘financial’. These three components of the triumvirate of evaluation are all ultimately expressed in terms of project cash flows and come together in the financial model of the project (Malone, 1992).

9.2.1 Early evaluation techniques

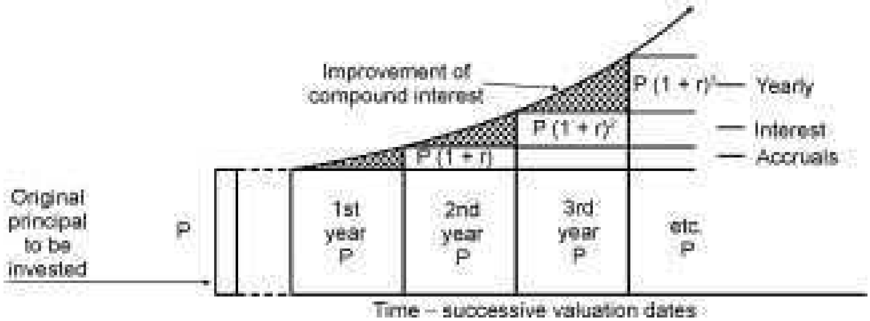
Evaluation techniques have always recognised that regardless of project type, the choice of either proceeding with a project or abandoning it is a business decision. This approach involves an assessment of the outlays and revenues that might result from a particular course of action and measures the benefits of so doing, against the benefits of some other course of action. Up until the 1950s, mining valuations were based upon the principle of ‘capital replacement’.

Capital replacement

The principle of capital replacement was enunciated by Hoskold (1877) in his treatise entitled *The Engineers Valuing Assistant*. The concept applied to any industry subject to the effects of depleting reserves. Its main feature was that it allowed for the replacement of the original capital investment at the expiration of the mine life (or annuity) at either the same or a different rate of interest than the dividend on the capital itself. In 1909 Sperr defined the present worth of an investment under these conditions in a somewhat convoluted fashion. ‘That sum which the exploitation of the mineral will return, together with a fair rate of interest, besides paying operating expenses, taxes, etc., the same fair rate of interest on the required working capital, and redeeming with a fair rate of interest the capital required for equipment and development.’

The Hoskold premise

Hoskold (1877) proposed that the annual income or profit from a mine could be treated as an annuity with a value that could be computed to the present time. His premise presupposed uniform earnings, uniform return on capital and provided for redemption of capital at the expiration of the operating life by annual reinvestment of the balance of the yearly earnings at a safe rate of interest. In other words, it assumed the original investment to be non-recoverable until the end of the life period, at which time it would be returned in full by the sinking fund. The sinking fund was an account to which yearly or



9.1 Increase of principal at compound interest (from Parks, 1957).

periodic payments were made for the purpose of replacing a capital investment. The payments were made from earnings and the account increased at a safe rate of interest so that within the time allowed for redemption, the payments into the fund plus the interest earned would equal the capital invested.

In practice, companies rarely set the monies aside for a sinking fund and either re-invested it in the business or returned it to shareholders as return of capital. This implied that any return of capital to the investor would, as soon as it was received, be reinvested at a safe rate of interest. Figure 9.1 shows the increase of principal at compound interest.

Prior to the First World War the financial environment was relatively stable and the Hoskold premise was considered a valid technique for valuation. It continued to be used between the two world wars, but could not survive in the rapidly changing economic climate. This was because the formulae were entirely dependent upon the accuracy of values estimated for A, r^1, r and n which fluctuated too widely and rapidly. Neither accuracy nor the absolute safety of the principal could be guaranteed, nor could the projected uniformity and stability of returns be relied upon. Hoskold recognised the validity of the time value of money, but his concept was too inflexible for use in unstable money markets.

9.2.2 Modern investment analysis

The modern investment analysis measures the profitability of the project in terms of return on capital, regardless of the source of finance. An all-equity capital is assumed and the potential earning power of the resources committed to the undertaking is analysed. Discounted cash-flow methods are used, thereby reflecting the time value of money. Simple methods of pay back period and simple rate of return are used only for the primary screening of project proposals and are not suitable for detailed analysis.

Investment analysis is thus a multifacted activity of searching for and deciding upon the value of various investment proposals, and includes the making of financing, production, and marketing cost analyses to determine the profit potential of each investment proposal. George Schenck (quoted in Malone (1992)) lists essential considerations at the core of any mineral venture investment analysis:

- **Geological evaluation.** Both mineral reserves and the geologic environment are key considerations in their influence upon costs, length and rate of production.
- **Revenue estimation.** Principal determinants of monetary benefits: production rate, commodity price and plant location.
- **Taxes.** Optimisation of interrelationship between allowable deductions from taxable income (e.g., depreciation, depletion, exploration and interest expense) can significantly increase project cash flow.
- **Cost of capital**. Selection of appropriate after-tax, weighted-average interest (discount) rate of new funds for a project.
- **Timing.** Time necessary to produce revenue from a project. If overoptimistic, the error could result in overvaluation of the project.
- **Discounting.** Cash flows should be discounted at the firm's cost of capital to one common time basis (usually the present) because such cash flows are received periodically and are deferred to occur through time.
- **Inflation.** Explicit provision is essential to provide for the differential effects of inflation on the various revenue and cost categories. Inflation is different from, and reduces the discount rate from, the current (nominal) dollar rate if the analysis is done in constant (real) dollars.
- **Interdependence.** The effects of interdependence amongst revenues, costs, income taxes, financing arrangements, and between the planned project and other activities of the producer should be considered as an integral part of the financial study.
- **Uncertainty.** Specific notice should be taken in a financial analysis that cash flows are predictions based upon current estimates of such matters as future commodity prices and future tax regulations. There is considerable risk that actual results will differ significantly from the forecasts.

Financial assessment

Computerised financial analysis systems provide great flexibility in the creation of cash flows and other economic indicators. Using these systems programmes are written for the three main cost centres: mining, treatment and administration. These cost centres form the basis for financial projections, which can be produced in rapid succession, to compare alternative mining concepts and production rates.

Financial analyses can be structured to show any required type of cash flow projection or financial report and when produced, to test the sensitivity of profit expectations to the effects of variables such as gold price, mining rates grades and percentage gold recovery. The financial model can also be used to quickly analyse any form of financing or taxation system that might apply and be so constructed as to provide a relatively limitless number of options within the scope of the exercise. However, this degree of sophistication extends only to the interpretation of data and to the simplification of conventional methods of accounting. Evaluation processes must still rely upon the integrity of methods of data collection that are much less sophisticated than those used for financial modelling; no amount of sophistication can compensate for data that are inadequate or wrong.

Time value of money

Interest enters into all financial transactions as does the element of time and a sum of money has a greater value now than at any time in the future. Interest is the reward for providing the money or a payment that must be made for its use. Funds invested now are recovered from returns in the future but there is a cost for the use of that money as determined by the rate of compound interest. The longer the period of time the more interest will have to be paid and the greater will be the amount that the money will depreciate in value.

The timing of an investment and its recoupment determines the amount of interest paid and the best way to minimise the amount of interest paid on capital is to repay the total amount in the shortest possible time. The less interest that is paid, the greater will be the available surplus (revenue less costs) for distribution as profit, the higher will be the discounted cash flow value and the greater the present worth.

The number of years selected for discounting varies according to such factors as mine life, investment policy, political and economic stability, and the degree of confidence placed upon forward market projections. The basic data for calculating NPV are contained in the cash-flow statement and the more that is known of the project, the more precise will be the data and the more accurate the NPV.

Note that the payback period is the number of years required for the net cash flows (after tax) from the project to return the original investment. The payback period is one of several complementary evaluation tools. It commences when sales begin and it is achieved when cumulate cash flows become positive. Its main weakness is that it takes no notice of the time value of money. These tests allow estimates of cost and earning power to be compared with alternative investments in less speculative undertakings. Investment worth is appraised in terms of the difference between the money flowing out from the investment and that being returned from earnings.

The DCF rate of return method is an attractive evaluation technique for three main reasons:

1. It offers a measure of expected profitability for comparison with other alternative projects.
2. It provides the real internal rate of return on capital investment.
3. It eliminates the need to determine an acceptable cost of capital.

Instead of using a predetermined interest rate, an interest rate is sought that makes the present value of the aggregated inflows equal to the present value of the combined investment outlays. The method considers the full economic life of the venture and is also flexible enough to discriminate amongst other investments having different cash flow patterns over time. However, its initial assumption that all cash flows can be re-invested at a comparable rate of return is valid only in special circumstances (Rudawski, 1971); in general, such rates are highly variable.

It is also important to realise that DCF estimates of worth are comparative and consider only the economic life of the project and not necessarily the absolute value of the property as a whole. By DCF standards the investment worth of a property having reserves for 30 years is very little more than for a similar property with reserves of only 15 years. Under such conditions the first property is not significantly more valuable to the purchaser than the second property. By discounting the cash flow for the 30 year property to zero at 15 years and paying for it accordingly, the investor then owns a 15 year property for which he has made no investment. He also has an already-operating plant and equipment and infrastructure for which he may subsequently have to pay only nominal costs for refurbishing.

Return on investment (ROI)

Return on investment (ROI) is a simple ratio given by the equation

$$\text{ROI} = \text{Average annual net income/total investment} \times 100\% \qquad 9.1$$

As supplementary measures to assist in corporate decision making, projects with ROI ≥ 1 are acceptable for development. Others are rejected.

Discounted cash flow (DCF)

Using the discounted cash flow approach the project can be subjected to various tests of which the most frequently used are net present value (NPV), internal rate of return (IRR), and present value ratio (PVR). Equivalent annual value (EAV) is effective in comparing operational alternatives. These four discounted measures are related to one another in that a unique stream of cash flows defines each project thus generating NPV, IRR, PVR, and EAV. The IRR is commonly a single result although in some cases multiple internal rates of return are possible. The other three measures are functions of the discount rate chosen as well as the stream of cash flows and hence a continuous array of results is possible (Malone, 1992).

The above evaluation measures are related products of the same cash flow streams but it is possible for them to give conflicting indications in relation to choosing between investment alternatives. The problems, which are usually the result of the selected discount rate, may in many cases be resolved by adopting the appropriate discount rate for choosing between these particular alternatives. The cost of capital may be the appropriate discount rate to evaluate a single investment opportunity but not when choosing between competing, mutually exclusive investment opportunities. The reinvestment rate, the rate at which funds generated can be reinvested, may be significant in choosing the preferred alternative in other cases.

Net present value (NPV)

The NPV of an investment is probably the main evaluation standard in common use. It is calculated by first discounting the expected investment cash outflow and the expected cash inflow, year by year, at a predetermined rate of interest. This rate represents the cost of capital, mine life, growth element and any number of other investment factors. The NPV is then found by subtracting the present value of the outflows from the present value of the inflows, having regard to the discounted salvage value of capital investment items at the end of the project life. A positive NPV indicates a profitable investment; a negative NPV indicates an unprofitable investment.

The purpose of NPV in economic valuation is to enable comparison with alternative investment opportunities. The present value of an investment is a function of the rate at which cash flows are discounted; different discount rates yield different values. Nevertheless, the basis of NPV is meaningful only if it is calculated by bringing all cash flows to a common datum, using an appropriate discount rate. The time datum is commonly the beginning of the first period of life of the project, after which expenditure commences. The solution lies in the choice of a minimum or cut-off rate, which is equal to or above the earning rate (internal rate of return). Thus, for NPV = zero, the internal rate of return equals the cut-off rate chosen for the exercise.

Internal rate of return (IRR)

The internal rate of return of a project is the discount rate that would yield a net present value of zero, i.e., the rate of interest which makes the present value of the estimated cash inflow equal to the present value of the cash outflow required by the investment. Zero NPV means that the cash proceeds of the project are exactly equivalent to the cash proceeds from an alternative investment at the stated rate of interest. The funds, while invested in the project, are earning at that rate of interest, i.e., at the project's internal rate of return.

All things being equal, the higher the IRR, the more attractive the project. In terms of acquisition or future project development, projects generating IRRs greater than a company's target rate of return will be accepted. However, in terms of determining the valuation of a project no IRR can be calculated where all cash flows are positive, as in an operating mine situation. Multiple IRRs can arise where there are significant negative cash flows at other stages of the project as well as at the beginning.

Present value ratio (PVR)

The PVR can be calculated by dividing the NPV of a project by the net present value of the capital expenditure outflows, discounted at the same rate as used for the NPV valuation. In effect it measures the net present value of the project per unit of investment. The alternative method of calculating the PVR is to divide the present value of all cash flows excluding capital expenditures by the present value of all capital expenditures, both streams being discounted at the same rate.

The relationship between PVR (1) and PVR (2) is as follows:

$$\text{PVR (1)} = \text{NPV}/\Sigma \text{ PVs of CEs} \qquad 9.2$$

$$\Sigma \text{ PVs of PCFs} - \Sigma \text{ PVs of CEs}/\Sigma \text{ PVs of CEs} = \text{PVR (2)} - 1 \qquad 9.3$$

PCF is the project cash flow excluding capital expenditure; CE is the capital expenditure. When the discount rate is the company target rate of return, the rule for accepting projects is that PVR (1) should be > 0 and PVR (2) should be > 1. The higher the PVR the better, although it does not help directly in the valuation of projects.

Equivalent average value (EAV)

The equivalent average value of a project can be calculated by multiplying the net present value of the project by the capital recovery factor for the number of years of the life of the project, and for the discount rate used in determining the NPV. The EAV can be used for choosing between different equipment alternatives in which cases the EAVs are equal annual costs and the lowest cost alternative is selected.

Taxation

The influence of taxation is a major factor and its effects on project economics tend to become more onerous with time. Exploration is initially encouraged in developing countries through such means as tax holidays and generous deductions for capital outlays for exploration, plant and equipment, development, infrastructure, etc. Taxation deductions for dividends from certain mining operations may also be allowed to shareholders as a means of encouraging high-risk investment. These benefits tend to be watered down as the mining industry becomes better established and bureaucracies grow. Longer-term risks are then due to government decisions to either raise income tax levels directly or simply 'cream off' additional amounts of company earnings through such devices as resource rent taxes, royalties, excess profits taxes, and the like. Provisions of the various taxation laws are now so complex that predictions of after-tax cash flows require the services of taxation experts.

9.2.3 Project funding

Project funding covers the provision of equity capital by the owners, and borrowings by way of debt financing or similar instrument. The arrangements can vary greatly and may often spell the difference between mediocre returns to shareholders and worthwhile dividends. The funding participants have different motives that are reflected in their perceptions of risk and reward. This must be understood so that a mutually satisfactory basis for participation can be reached.

Equity holders are concerned largely with the maximisation of profit, an early payback period and freedom from debt. Bankers and other term financiers want security first of all. They will look to the borrower's ability to service the loan; to security in the form of a first ranking debenture charge over all of the borrower's assets; or to some form of parental guarantee, negotiable securities or mortgage over certain real property.

The traditional corporate loan structure is based upon a level of interest commensurate with the risk and is motivated by the initial profit on the sale of the plant and longer-term strategies. Equipment suppliers may provide finance in the form of term payments for their products; most modern broking underwriters look closely at share price earnings ratios; a common concern for all participants is the quality of management.

Equity financing

The cost of capital is the weighted-average costs of funds to the enterprise. Except where 100% equity is planned for the cost of capital, the funds comprise generally a mixture of equity and debt. Equity funding applies generally to the early stages of exploration and testing up to the stage at which an apparently viable resource is established. The funds may be sourced from retained earnings, company floats and share placements or from private capital. The work requirements for resource calculation will be land or titles acquisition, geological research and reconnaissance, drilling and bench-scale testing, and development of a preliminary feasibility study. The shareholders of the company provide the equity funding and thus take all of the risk. The return on investment may comprise either or both a capital gain and an income or dividend return. Hence, the investment must offer a sufficiently high rate of return for the risk involved; if the study is unsuccessful the investment will be lost.

Debt financing

With the decision made to raise debt finance, careful consideration of the requirements of the project is needed to obtain the optimum package. Apart from the provision of mining and treatment plant including infrastructure and construction and commissioning costs, there will also be the costs of all additional outlays on current assets affected by the major expenditure in bringing the project into the final stage of production. Included will be working capital and any increase in stocks of raw material and book debts beyond that provided for by equity financing. Sources and types of funding for gold projects at various stages of development from reconnaissance to the final feasibility studies are shown in Table 9.2. Basically, the lenders do not take an equity stake in the company nor do they participate in any profits generated. But they do expect the company to pay regular interest charges, and to discharge its debt at specific times in the future before the equity investors receive the expected returns on their investment.

The appropriate proportion of debt to equity is determined by normal mining industry standards and within the company itself by management's judgement of the capacity of the company to service the debt. The carrying charges (principal repayments and interest) will have to be met by the expected cash flow. Sheppard (1990) places particular stress on the following four matters in considering the sources of finance:

1. ratio of debt to equity, which involves the question of the company's capital structure
2. relative cost of debt and equity
3. mixture of short-term and long-term liabilities in the total debt
4. appropriate level of structuring for balance sheet ratios such as leasing and project funding.

In Australia the most common method of debt financing of gold properties in the late 1980s was 'limited recourse project financing' (Willett, 1988), so called because the period of full recourse to all of the firm's assets is limited. The lender (e.g. the bank) expected that ultimately the cash flow of the project would provide full loan servicing/repayment. Recourse to the borrowing company at the time of greatest risk, i.e. prior to successful commissioning, is via a first ranking debenture charge over all of the company's assets. The bank's security guarantee thereafter reverts to a first ranking mortgage over project tenements and a first ranking charge over other project assets.

Gold loans

To reduce uncertainty arising from gold price volatility a lender commonly expects a borrower to enter into some form of price protection scheme to ensure that the mine would not have to shut down because of a sudden gold price

Table 9.2 Source/type of funds for various stages of gold projects (from Malone, 1992)

Stage of project	Risk level	Type of funding	Source of funding
Exploration Geological research and reconnaissance, identification and acquisition of prospect detailed, geological investigation, drilling calculation of reserves	Very high	Equity	Retained earnings; company float, rights issue, share placement, private seed or venture capital
Evaluation Preliminary economic study, mine plan and metallurgical studies, plant design, costing, final feasibility study	High	Equity	As for exploration stage
Development Mine development, provision of infrastructure, plant construction	Moderate	Limited recourse, project finance, corporate borrowing and/or equity	Bullion house/ trading banks/ merchant banks (gold loans or currency loans) Retained earnings; rights issue, share placement
Mining Mining and ore treatments operations (refinancing or working capital requirements)	Normal	Non-recourse or limited recourse project finance, corporate borrowing and/or equity	Bullion house/ trading banks/ merchant banks (gold loans or currency loans) Retained earnings; rights issue, share placement

collapse. In some cases the price protection scheme takes the form of forward selling such as the purchase of 'put' options. The trend towards debt financing by gold loans gained momentum in Australia with innovative thinking and aggressive marketing by banks, merchant banks and bullion houses. Its great attraction was that it provides a natural hedge against fluctuating gold prices. If the price of gold falls the dollar value of outstanding gold loan repayments and gold loan fees falls proportionally. If the gold price rises, the dollar value of gold loan repayments rises at the same rate as revenue, but the projects operating surplus will rise more quickly than the gold price. This distinguishes the gold loan from the conventional loan for which the dollar value of capital repayments and interest remains unchanged regardless of gold price movements.

Willett (1988) describes the mechanics of a gold loan and supporting letter of credit as follows:

> When a debt financing package involves a gold loan, a bullion house needs to be involved as provider of gold, which is sold to provide currency to enable costs to be paid. The transaction is governed by a gold barter-agreement under which the borrower agrees to return the gold provided and to pay fees, which are also payable in gold. Usually a bullion house that provides a gold loan package will team up with a bank or merchant bank to limit their risk. For a charge, the bank or merchant bank will provide the bullion house with a letter of credit which guarantees payment of currency to the bullion house equal to the latter's exposure in the event that the borrower defaults under its agreement with the bullion house. The letter of credit usually includes an allowance to cover the increased exposure of the bullion bank that would occur if the gold price rose significantly. The bank secures its position by taking a first ranking charge over assets appropriate to the type of debt financing facility. The bank or merchant bank charges the borrower a fee for provision of the letter of credit. The bank also recovers certain costs of putting its facility in place.

9.3 Risk analysis and uncertainty

The Macquarie dictionary defines risk as 'exposure to the chance of injury or loss; a hazard or a dangerous chance'. For the miner the assessment of risk is an integral part of evaluation and its management involves consideration of the scientific evidence from geological and engineering studies and a host of other factors including technical practicability, environmental protection and cost. Governments analyse the possible risks of the proposed operations to the physical and social environment. Financiers are concerned with the ability of the project to repay capital and interest within the scheduled time with the least degree of risk. For management the risk is essentially entrepreneurial and the outcome is either a loss or a gain. Since new companies frequently have to risk all on a single venture there is a tendency to form joint ventures to form larger projects so that one failure will not send the company bankrupt. Two related elements in project risk are the chance of project failure and the size of the potential loss. The greater the size of the potential loss, the smaller is the acceptable chance of failure.

Risk analysis thus deals with areas of gross uncertainty including resource-reserve estimates, market prospects, the world economic environment, the likely commodity price regime, and the local economic environment, particularly in regard to industrial relations and inflation prospects.

The source of the finance is governed in each case by the perceived level of risk. This reduces progressively as the geological picture unfolds and costing becomes increasingly comprehensive and reliable. Thus, while exploration and evaluation are high-risk activities and must rely upon equity capital, funding at the development stage of a project carries fewer risks and may comprise a mixture of equity capital and either conventional corporate borrowing, or some form of limited recourse project (debt) financing (Willett, 1988). At the feasibility stage, both management and lending institutions expect to be provided with proof of the adequacy of the mining reserves, together with the appropriate engineering, economic and financial analyses to demonstrate its economic worth. Risk factors of particular significance to project economics are mining reserve, investment capital and production cost.

9.3.1 Mining reserve risk

Computation of a 'mining reserve' is strongly influenced by deposit geometry, the proposed method and scale of mining and all other such factors that might be involved in the translation of 'resource' estimates to mining 'reserve' quantities. Inventory of gold-bearing material comprises both resources (not necessarily economic) and reserves (presumably economic). Categorisation depends upon the valuer's opinion of the extent and validity of the sampling data and the degree of confidence given to the expected recovery component. Regardless of the sophistication of criteria for testing, the estimates are only as reliable as the data from which the estimates are prepared. Hence, while data, which can be compiled carefully by standard methods, are easily tested and frequent checking will usually provide estimates within the limits of normal sampling error, non-standard data, which tend to promote widely different interpretations, are at best confusing.

Every aspect of sampling involves a degree of risk. The grid pattern may not properly identify all relevant features of the deposit geology; some holes may have been drilled carelessly due to lack of motivation or lax supervision; ground conditions may have been unfavourable for the type of equipment used. Indeed, most deposits offer a wide variety of different materials to be sampled, each presenting its own peculiar problems and exposure to risk. Common errors are to undervalue bouldery ground and to overvalue free-running gravels. Varying gold sizes pose additional problems; coarse gold yields widely different duplicates, finely divided gold is more evenly distributed but is increasingly difficult to recover in the smaller sizings. Palaeoplacer sampling offers a higher exposure to risk than the sampling of younger deposits because of their more complex geology and geography and greater chances of post-depositional alteration.

Ore reserve estimates

Methods of ore reserve estimation comprise mainly classical extensions of mathematical procedures using weighted volumes and grades. Traditional methods employ geometric sections, polygons and triangles. Weighting by triangles is usually more constrictive than weighting by polygons, which usually represent much greater areas. Typically these two procedures are seldom used except for indicating resource potential in large-scale reconnaissance programmes. Drilling on a regular grid is preferential, with close attention being given to the separate evaluation of high-grade and low-grade areas and to the uniformity of the ground in which the deposits occur. In both cases the boreholes should be spaced geometrically so that each hole has a true range of influence equivalent in size to boreholes yielding similar grade material in the surrounding wash. Anomalously high-grade samples must be defined by close drilling to eliminate as far as possible the nugget effect.

Classical statistics may be applied to the arithmetic methods to investigate various sources of bias errors in sampling and to throw additional light upon problems of grade estimation including the nugget effect, sample spacing and sample representivity. Geostatistical methods recognise the semivariogram (Appendix II) as a measure of sample variance with distance. A process of Kriging may derive estimates where semivariograms can be produced for sections of a deposit. All methods make assumptions of a finite relationship of one kind or another between adjacent and neighbouring samples. Most methods rely upon a geometrically designed sample grid to set the pattern for evaluation. None of the methods will produce reasonably accurate estimates if the data from sampling is markedly inaccurate.

The uncertainty of the estimate is a predominating factor; the only certainty is that the estimate will be wrong, the magnitude and sign positive or negative, the variance being the focus of concern. A gold mining project may start with a variance as high as plus or minus 20% or 25%. If the sensitivity analysis shows that the project would be uneconomic at the minus 25% variance level, the project is very risky. The cost of improving the reliability of the particular ore reserve estimate to reduce the variance to a lower more acceptable level by additional drilling could be prohibitive. The appropriate response is probably another drilling programme aimed at expanding the resource. Malone (1992) suggests that in some cases it may be possible to increase the ore resource by about 40% by additional drilling and thus strengthen it to withstand any possible shortfall provided that the project can stand the increased waste/ore ratio caused by shortfalls in the ore resource. Whatever is decided there is no substitute for a healthy margin of error in the ore reserve estimate.

Despite repeated revision of the Australian Code for reporting of identified mineral resources and ore reserves the Code still depends upon subjective judgement. The measured resource and proven reserves categories are dependent upon the judgement of a competent person. The issue is complex and in Australia a Joint Committee of the Australasian Institute of Mining and Metallurgy and Australian Mining Engineering Council issued a *Code of Reporting* in 1988 following several years of deliberations. The Committee considered the relation of mining reserves to mineral resources by first seeking to define a mineral resource and then outlining the process by which it may be upgraded to an ore reserve through consideration of technical and economic data including mining, metallurgy and marketing. The Committee defined the term 'ore reserve' (i.e. mining reserve) as that part of a 'measured' or 'indicated' resource, which could be mined together with dilution and from which valuable or useful minerals could be recovered economically under conditions realistically assumed at the time of reporting.

The Committee pointed out that 'ore reserve' estimates are not precise calculations, being derived from the estimates of 'resources' and modified by economic, mining, metallurgical, marketing, environmental, social and government factors. Ore reserves are quoted under the Code as a single tonnage and grade combination but are actually only proven within limits and these limits will be different for different cases.

The inherent uncertainty of the ore reserve is obvious. But, a frequent observation that the ore reserves are known with certainty only after the orebody has been mined out is only partly correct. The orebody depends upon external variables because it is not fixed in space and time. The best estimate of the *in-situ* resource, which is fixed in space and time, is provided by the grade control (drilling and sampling), which leads to the next critical aspect of any project – the process of converting the *in-situ* resource into mill feed.

The orebody can be defined as the economically mineable component of the resource by application of a cut-off grade, which is a function of changes that occur with operating cost, gold price and economic conditions. This variability of the orebody can result in significant differences from the resource estimate over the life of the project. The 'lowest possible cost' requirement impacts upon the mining method, equipment size, degree of mining selectivity and the style and density of grade control sampling. As the result of these factors the mill feed will comprise a mixture of some proportion of the mineable reserves, including possible dilution, plus a proportion of unanticipated dilution of barren or low grade mineralised material. Further opportunities for error are created in converting overall pit volumes to plant tonnage, estimating plant requirements and costs and in estimating pit volumes for waste disposal. It is intended that the mineable reserves, as calculated from the *in-situ* resource, will match the material that will be mined in practice, so that mill feed will be the same as mineable reserves. This may happen for a time; in the long term there will always be a difference. A periodic reconciliation of ore depleted from the original mineable reserves may record a variance of up to $\pm 10\%$. Any significantly greater variance will throw doubt upon the validity of the original reserve estimate, which in some cases will have to be recalculated to justify future mining.

Considering that mining reserve estimates are made from field studies, it is clear that great care must be taken to reduce the chances of errors in sampling and interpretation. Faulty plant and equipment can be modified or replaced and operating parameters can be revised, but the integrity of the whole evaluation process depends upon the validity of the mineable reserve estimates. Nothing can be done that will rectify the position if the estimates are wrong, and have been seriously overstated or understated.

Criteria for acceptance of ore reserve estimates

In applying the above standards to the assessment of ore reserves, economic studies must provide a satisfactory answer to the following questions (Malone, 1992):

- Will the mining resource be large and rich enough to provide reserve quantities that will support a mining operation at the required level, and yield profits commensurate with the risks involved?
- What is the most appropriate mining method for the size and type of ore reserves, having regard also to essential environmental safeguards?
- What is the likely rate of mining and the assumed mine life for the estimated mining reserve?
- What will be the level of efficiency of available labour and the likely quality of management?
- Where should mining commence and what sequence of mining will optimise project economics?
- What size and type of machines will be required for the various functions of mining, treatment and restoration?
- What infrastructure requirements will meet the needs of the proposed operation?
- What services (water, power, etc.) will be required and what will be their source?
- What additional data are needed to provide realistic costing of the overall project?
- What will be the time frame for project implementation?

These questions all relate to, or are influenced by, the nature of the deposit and the environment within which it exists. They are basic to the computation of mining reserves and the construction of the mine plan. The answers must be both comprehensive and unambiguous so that the plan may be created in the reasonable assurance that it will work within the parameters of the design criteria, and within the guidelines set down for environmental protection.

Data interpretation

Having ensured an acceptable level of sample reliability and representivity the next source of possible risk is data interpretation. As the result of seeking independent audits of a particular mining reserve, financiers may be faced with a variety of different assessments. Various operators approach the factoring of grade estimates differently: some may suggest economic viability, while others cast doubts on the possible success of the project; frequently none of these predictions is made on the basis of logic and common sense. A major difficulty is that acting in all sincerity, many valuers adopt one or other of the many means of factoring borehole data because of reports that a particular relationship has been applied successfully elsewhere.

The R/E factor

The R/E factor is based on a comparison between the actual production of gold over a given period with estimates of feed volumes and grades of material fed into the mill over the same period. The first stage in understanding the differences between expected and actual plant performances is to reflect upon the errors involved in all stages of sampling, measurement and treatment of the particular ore. 'Estimation' errors differ from property to property and within individual properties according to the characteristics of the gold, the method of sampling and interpretation of the results (Chapter 6). 'Selection' errors in planning a dredge course are reduced with increasing sample density. 'Excavation' errors are associated with pit measurements and plant throughput, and are subject to errors due to irregular excavation profiles, regularity of paystreaks in the orebody, quality of supervision, and so on.

Treatment plant problems, such as losses of gold, depend upon the metallurgical performance of the plant units, the physical characteristics of the gold and the human equation. Critical in its effect on the R/E factor is the increasingly finely divided and flakier nature of gold particles in lower stream sections. Additionally the gold may become more refractory and resist amalgamation in the recovery circuit. Records of the Yuba dredger No. 15 shows that over a period of 15 months the dredger achieved a much lower recovery after five months of dredging when the gold was described as rusty or refractory. Probably both factors were involved to some degree. Table 9.3 illustrates the decline in the R/E factor of two dredgers in the USA, largely because of size reduction and increasing flakiness of the gold with distance from source.

Ideally, there will be no errors, and the R/E factor will be unity. Practically, however, the errors may be either positive or negative and if R/E = unity in any particular case it will only be because the differences tend to cancel out over time, not that individual differences do not occur. The estimated grade and gold content of the reserve estimates, which may differ very widely, reflect the ability of each assessor to correctly gauge the size and importance of the individual uncertainties. This was stressed by Garnett (1991) who showed the effect on estimated ore reserve figures of the different approaches taken by three experienced engineers (A, B and C) to defining the reserve definition for a

Table 9.3 Effect on R/E factor of occurrence of refractory gold which inhibits the recovery process, as illustrated by records of No. 15 Yuba gold mining company (from Garnett, 1991)

Years	Dredging period (months)	'Refactory' gold	R/E factor achieved
1937	3	Absent	1.49
1937	2	Absent	1.47
1937	3	Present	0.87
1937–38	3	Present	0.56
1938	2	Present	0.60

particular unworked resource in Sierra County, New Mexico, USA (Table 9.4). Quoting from Breeding (1973),

> Experienced placer engineers process drill logs in different ways. The process used by conservative California operators was to correct logs negatively to reflect excessive cores, but not positively to reflect deficient cores. Some used 88 feet of drive per cubic yard (the theoretical factor) though others used the Radford Factor or 100 feet per cubic yard. When positive corrections are not used, it was common for unit values (valuable mineral/cubic yard) recovered by mining to exceed prospecting results. I [sic] have seen and used positive corrections as high as 240 percent (save in the initial drive); however, I have generally limited positive corrections to 100 percent and where much coarse gravel is present to 50 percent and 25 percent.

In referring to such 'rule of thumb' and 'hunch' type techniques, it has been remarked 'They have no scientific rationale, no legitimate parentage'.

An inherent constraint is to use the R/E factor as an exploration tool to measure the performance of a mining operation at the feasibility or pre-feasibility stage. If used in this application, the expected recovery will be estimated from an assessment of volumes and grade at some arbitrary level of confidence based either upon the prior experience of the assessor himself or that

Table 9.4 Effect on estimated reserve figures for a placer gold deposit of employing different reserve definitions on a resource, as illustrated by data for unworked property in Sierra County, New Mexico (from Garnett, 1991)

Estimator identity	Estimated volume of reserves, 10^6 m^3	Estimated grade of reserves, mg/m^3	Estimated gold content of reserves, 10^3 oz.
A	13.71	1261	564
B	23.51	1424	1064
C	14.76	407	192

Table 9.5 Average long-term R/E factors achieved by gold dredgers in the USA between 1900 and 1990, as identified from limited sources available (from Garnett, 1991)

Locality and time period	Arithmetic average R/E factor achieved
All states of USA before the Second World War	0.71
California before 1920	1.01
Other states of USA before 1920	1.34
All states during 1920s and 1930s	0.77
All states during 1930s and 1940s	0.96
All states of USA since the Second World War	1.15
Total arithmetic average	1.00

of some other assessor. Its possible accuracy depends upon the relevance of that experience to the particular task in hand. Large negative variances in the resource estimate of mineable reserves and mill feed, which are related but not identical problem areas, are the greatest threats to project viability. Estimates of these parameters must be checked and rechecked to ensure their reliability. Most aspects of mining projects are interrelated and both upstream and downstream consequences of every change or possible change to every assumption must be investigated.

As part of a detailed review of global gold and tin dredging operations, Garnett (1991) showed that the average R/E factors achieved by a variety of dredgers in the USA between 1900 and 1990 could be combined to achieve an overall average value of unity (Table 9.5). However, performance figures achieved by some gold dredgers in the USA (Table 9.6) demonstrated that important divergences occur for individual dredgers; variations of as much as plus or minus 50% were common during a single year. Hosts of possible contributing factors affect the relationship between plant feed and gold recovery and in predicting an R/E factor as part of a feasibility study the individual problems must be considered stage by stage. The long-distance transfer of established procedures from one locality to another is unlikely to be feasible and fresh standards must be applied to each environment.

In the author's view, the economic viability of a proposed mining operation should not rely upon the application or non-application of a correction factor. If that or any other such arbitrary method of correction appears to be a controlling factor, the data should be re-examined and the grid closed in until the matter is resolved. Presently used borehole correction factors add greatly to interpretation risks, and the best means of avoiding serious error is to generate the data under standard conditions and check and interpret the information carefully in accordance with the deposit geology. It is usually advantageous, nevertheless, to have a safe reserve tail beyond the debt repayment period in case the project life is threatened by a shortfall in mineable reserves.

Table 9.6 Historical R/E factors achieved by some gold dredgers in the USA since the mid-1930s (modified from Garnett, 1991)

Location	Dredging company	Dates of dredging activity	Number of dredging years or seasons	Number of dredges	Average dredging depth, m	Long-term average R/E factor achieved	Notes of range of annual R/E
California	Yuba Gold Dredging	1937–1958	22	No. 17		1.21	Range: 0.87–2.04
California	Yuba Gold Dredging	1959–1967	9	No. 17		1.32	Range: 0.92–1.75
California	Yuba Gold Dredging	1954–1987	17	No. 21		1.33	Range: 0.74–2.49
California	Natomas	1945–1960	15	5		1.04	
Alaska	Alaska Gold Company	1952–1960	9	No. 1	6.85	1.13	Range: 0.84–1.40
Alaska	Alaska Gold Company	1950–1954	5	No. 2	3.08	0.83	Range: 0.61–1.39
Alaska	Alaska Gold Company	1949–1952	4	No. 3	2.98	1.27	Range: 1.13–1.47
Alaska	Alaska Gold Company	1949–1990	25	No. 5	21.04	1.05	Range: 0.49–2.38
Alaska	Alaska Gold Company	1957–1990	18	No. 6	10.52	0.81	Range: 0.51–1.24
Alaska	Nyac Dredging Company	1945–1946	2	No. 3		1.83	
Alaska	Nyac Dredging Company	1959	1	No. 4		1.22	
Alaska	Nyac Dredging Company	1982	1	No. 3		0.72	

Addressing mining reserve risk

The first task is to standardise the techniques and equipment used for sampling. Skilled personnel strictly supervised should carry out all procedures so that the raw data can be used for grade assessments rather than data that have been tampered with. Where drilling conditions are difficult because of the nature of the gravels, individual boreholes should be redrilled from time to time using larger diameter boreholes in order to test the validity of the sampling method used. It is essential that sample data is sufficiently comprehensive to adequately describe its geology, and any apparent discrepancies should be resolved by re-sampling as soon as the problems are noted. In a completed exercise, the overall risk will probably lie within reasonable bounds if the synthesis of results is reproducible, even though individual duplicates vary widely. The temptation to continue the exercise only long enough to identify a trend and to use that trend to develop a correction factor for the data as a whole is extremely hazardous and should be resisted regardless of cost. The acceptance of any such trend from insufficient data will only add to or reduce the risks of accepting or rejecting the initial sampling results. Only for a statistically large number of check samples, can any pronounced bias be capable of scientific explanation and so be accepted as a valid basis for interpretation. No precautionary measure entirely eliminates error but the range of measures, if carefully applied, should provide generally reproducible sets of data for evaluation through some form of sensitivity analysis.

Note that test pitting instead of drilling is sometimes used to check borehole results in shallow ground; even in deeper ground an occasional shaft may be dug to examine some unexplained feature that cannot be explained by drilling. In some cases, a series of trial mining exercises may be conducted as a final check in marginal ground.

9.3.2 Investment capital risk

Investment capital risk is determined by the accuracy of capital cost estimates and the measures taken to avoid cost over-run. Capital intensive projects are the most vulnerable. Regardless of size or other consideration, capital-intensive projects have less margin for error, and are more prone to failure than projects that may require an initial high capitalisation but which, relative to the projected mine life and profitability, are not capital-intensive

Revenue predictions could once be based with reasonable assurance upon forward projections, which related spot price levels to past price trends thereby assuming a continuance of present and past conditions. The uncertainty of predicting gold price futures in the present world market poses much greater risk through price fluctuations that occur relatively unexpectedly, often quite drastically. In order to reduce marketing risks, most companies now determine the income for any given production rate in the short term by forward and spot selling at prices that are governed by market conditions at the time of negotiation. A good blend of forward to spot selling is dependent on the state of the market.

Recent history has shown that projects are particularly sensitive to risks associated with political unrest. No new undertaking can be entered into in developing countries with full assurance that certain premises upon which the project parameters depend will not be suddenly voided. Typical problems are related to land ownership, new and more onerous environmental rules applying social and increased taxes and royalties. Natural difficulties may result from the lack of historical environmental data. Unexpected setbacks such as flash flooding or prolonged drought conditions may seriously affect building schedules.

Within the developed countries, a tendency to underestimate capital requirements for new ventures makes them vulnerable to sudden wage and material cost blow outs and industrial unrest. A study of engineering construction performance by the Maddock Committee (Anon., 1989) found that, for 21 major projects (>$20 million) completed during 1982–1988 in Australia, 60% were completed within budget estimates, 80% were completed within the upper accuracy range of these budgets and 20% over-ran their budget plus stated accuracy limits.

Cost centres

Capital expenditure decisions are based upon the findings and recommendations of the project feasibility study in relation to:

- mining plant and ancillary equipment
- treatment plant and buildings
- services (e.g., workshops, power, water reticulation, communications)
- site preparation including roads of access and handling facilities
- infrastructure including housing, recreation, office buildings and stores
- inventories
- planning, design, overheads, supervision and commissioning
- freight and installation
- environmental requirements
- contingency
- working capital.

Engineering studies examine all aspects of the above items and estimates are made of costs for mine development, construction and commissioning. Each component of the exercise is considered both for itself and its relationship with other components in arriving at the estimated capital requirement and time frame for the project. The principal uncertainties are associated with the structuring of capital cost and the attention paid to possible causes of cost overrun.

Structuring capital cost

The preparation of a construction schedule requires extensive experience of construction techniques and site management. It is based upon many considerations relating to the environment in which the construction is to take place, the size and type of the plant, and the required time frame for the exercise.

The need for extreme accuracy is minimal in the early stages when design data are few and preliminary estimates of project costs are obtained mainly from information extrapolated from known projects of similar dimensions. Initial estimates are seldom more accurate than plus or minus 30%. But, as the project advances through to the pilot plant stage and the relevant systems are developed in more detail, additional field data are available from which to upgrade process flow sheets. Plant and equipment sizes may then be estimated in accordance with a materials balance, which shows the expected flow conditions through each unit of plant. Reasonably detailed bills of materials and labour can be taken out from design details and specifications prepared for tendering purposes. Cost estimates based upon preliminary equipment specifications and conceptual arrangement drawings may then have improved to plus or minus 15–25%.

When technical and economic feasibility has been determined, the infrastructure requirements will conform to basic objectives and to the broad plans formulated for the project. Installation costs can be prepared from current estimates of materials and labour. Engineering design and supervisory costs are estimated by rule of thumb as a percentage of the overall estimated cost. Freight and insurance costs are based upon the items and costs listed. At this stage, the order of accuracy for the cost of upgrading an existing facility should be around plus or minus 10–15%.

The construction phase begins with the final acceptance of successful tenders. Most or all of the uncertainties should have been resolved and costs are prepared for process equipment, construction materials and infrastructure based upon firm quotations including freight and possible government charges. Engineering design should be close to completion, and construction estimates based upon the skilled and unskilled labour available to the site, should have an accuracy of 5–10%. An allowance for working capital is estimated from detailed analysis of required inventories and cash extended credit.

Working capital

In order to establish a mine as a going concern there must be in addition to the cost of all buildings, plant and equipment, services, etc., an additional allowance for the cost of all consumable materials and labour during start up. This allowance (working capital) is designed to pay for all project outlays in the period leading up to full-scale production and steady income from the sale of the gold. It may be funded and accounted for separately.

Working capital is intimately tied to production and is subject to the risk that production targets will not be reached within the allotted time frame. Reserves of cash are required to pay for wages, raw materials, utilities, operating supplies and miscellaneous expenses. Extended credit must be available to be drawn upon as needed and within reason for as long as needed. Few projects proceed exactly as planned and sufficient capital must be available as cash and credit to establish the operation as a going concern.

Cost over-run

Capital cost estimates rely upon the completion of each phase of the programme according to specification and within the time frame allotted to the project. The duration and man-hour requirements for the various functions involved are graphed in a time frame. Contingency allowances are provided to take care of possible changes to the general social and economic environment and in accordance with the perceived level of expertise of the construction team. A higher contingency allowance will be applied to projects with extreme environmental problems such as operating at high altitudes or in wetlands (see Chapter 5).

Cost over-run is usually due to increased expenditure for items that are outside the estimator's control and not covered by contingency allowance. Typical items include design alterations, schedule changes due to transport delays, strikes and other disruptions, all of which cause construction delays and lead to system bottlenecks. Excessive overtime payment, double handling, inefficient use of available machines, penalty rates and increased interest are only a few of the resulting problems.

Investment in shoestring projects is a gamble, and not a quantifiable risk. 'Shoestring financing' is financing of a project according to bare essentials without making adequate provision for cost over-run. Such projects are necessarily brought into production prematurely because of financial pressure, which forces the acceptance of the cheapest instead of the best alternatives. Most fail because they lack the cash reserves or borrowing power to deal with unexpected problems. Decisions are either 'hunch' decisions, or are based upon crude, or otherwise unsatisfactory analysis. Both are equally hazardous and may be due to a lack of any better method of valuation (Middleton, 1965).

Addressing investment capital risk

Investment capital risk is linked with all other project risks and hence can be minimised initially by improving the quality of all the data at their source. The accuracy of scheduling is a function of the skill and experience of the project engineers and of the approach taken to risk evaluation. Typical economies for large projects can be made as follows:

- Reduce construction time to a practical minimum; this will minimise the costs of interest on capital prior to start-up.
- Avoid unnecessary overtime; overtime rates commonly cost up to 2 to 2.5 times the ordinary rates.
- Maintain an adequate but not an excessive workforce; productivity actually falls as a result of over-staffing.
- Utilise heavy machinery to its fullest capacity, particularly if hired; get it off the site and off the costing when its task is completed.
- Work to a systematic job sequence that will minimise double handling and wasted effort.

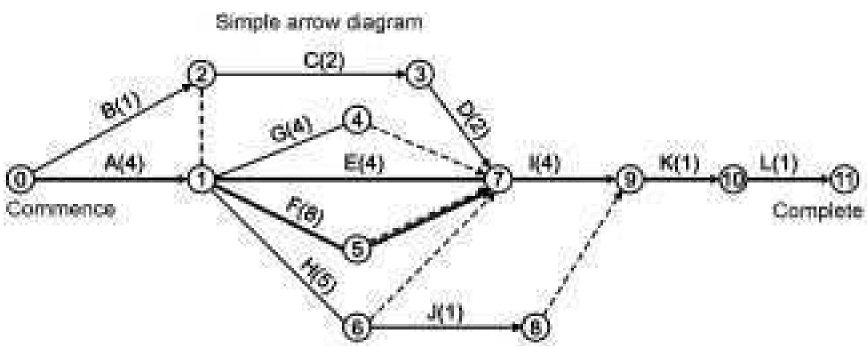
For small projects additional economies might include:

- semi-permanent instead of permanent structures
- plant design orientated to initial low cost–high upkeep instead of initial high cost–low upkeep
- contract mining rather than company ownership operations.

Critical-path technique

The critical-path technique provides the necessary background data for the bar chart. As its name suggests, this technique is a critical factor in making time estimates that are realistic for each of the various jobs. The role of the various functions is examined from the decision to proceed through to final commissioning. There is no substitute for skilled and experienced estimating staff.

The critical path shows the starting time for each job and indicates the consequences of any delays, thus drawing attention to those jobs that threaten completion. A simple example is illustrated in Fig. 9.2 for a project involving design, procurement and installation. The jobs are listed in order (Table 9.7) to prepare the arrow design. An arrow in this figure represents each job. Arrows start and finish at events and work cannot begin on any job until all jobs leading



9.2 Critical-path diagram.

Table 9.7 Example of critical-path scheduling

Project – to design, procure and install fresh water header tank and pump. Jobs listed in order to prepare arrow diagram.

Description	Identification code	Estimated normal duration (weeks)
Design system and select equipment	A	4
Clear and prepare site	B	1
Design concrete foundations	C	2
Construct concrete foundations	D	2
Fabricate tank, structures, etc.	E	4
Delivery time, pump and motor	F	8
Delivery time, piping and valves	G	4
Delivery time, electrics	H	5
Erect pump, motor, tank and piping	I	4
Install electrics and wiring	J	1
Painting	K	1
Testing	L	1

Data from arrow diagram compiled for feeding to computer:

Code	Event start	Nos. finish	Duration weeks	Description
A	0	1	4	Design system and select equipment
B	0	2	1	Clear and prepare site
G	1	4	4	Delivery time, piping and valves
E	1	7	4	Fabricate tank, structures, etc.
F	1	5	8	Delivery time, pump and motor
H	1	6	5	Delivery time, electrics
C	2	3	2	Design concrete foundations
D	3	7	2	Construct concrete foundations
J	6	8	1	Install electrics and wiring
I	7	9	4	Erect pump, motor, tank and piping
K	9	10	1	Painting
L	10	11	1	Testing

to its start event have been completed. Jobs on the arrowed path are given an identification code A to L. Dashed lines link events that are interdependent but normally have no time value.

The critical path in Fig. 9.2 can be determined by inspection and is shown as a heavy line. Events (start and finish times of jobs) are numbered accordingly. If, for example job F, which is critical, could be reduced from eight to four weeks it would cease to be critical and the new critical path would pass via job H or (0), (1), (6), (7), (9), (10), (11). By the same token, no extension to the time of any of the jobs on the critical path can be made up without incurring cost in attempting to reduce the time for others of the jobs to be completed. Essentially in this regard, the critical-path technique addresses the risk of failure to complete a project on time by organising the work so that individual responsibilities can be assigned to each job and no job that might delay the project is overlooked, however small.

Contingency allowances

It is common practice to deal with investment risk by applying rule of thumb methods (contingency allowances) to compensate for possible cost over-runs due to probable errors in purchasing, scheduling and implementation. These allowances are applied both as a percentage of fixed capital or as a percentage of combined annual operating cost estimates, and the highest figure is generally adopted. However, while this approach to cost estimation was once considered to be reasonably satisfactory, it is now known to be too simplistic and one or other of the techniques known as 'absolute values interpretation' and the 'probability review' (Slattery, 1990) now takes its place. Of these two techniques, the probability review is usually the most recommended.

The absolute values interpretation considers that the percentage accuracy assumed for each stage costing represents the extent of the risk of possible over-run. Thus, for an estimated cost of $1 million and a stated accuracy of 10%, the actual cost would lie between $0.9 and 1.1 million and finance would be arranged to $1.1 million. This apparently conservative approach becomes more and more risky with the increasing complexity and uncertainty of international relations and global economics.

The probabilistic approach recognises that the estimated project cost is only one of a whole range of potential outcomes. Slattery assigned potential ranges to ten significant variables or work items in the above example and then simulated the possible outcomes of 1,000 repetitions. The results can be plotted as a histogram to show the number of times various project costs occurred. Once in the thousand samples the cost was as low as $0.725 million and once it was as high as $1.19 million. Analysis of the results showed that for an actual mean estimate of $947,120 the standard deviation was $71,140. There was about a 68% chance that the actual cost of the project would be within the range of one standard deviation from the mean, i.e., within the range $875,980 and $1,018,260. There would be a 95% chance that the actual cost would be between two standard deviations from the mean, i.e., $804,840 and $1,089,400 and a 99.7% chance of the actual cost being within three standard deviations from the mean, i.e., $733,700 and $1,160,540.

Sheppard (1990) suggests that over and above these considerations, an additional reserve coverage ratio will ameliorate the risk of error by ensuring sufficient reserves to compensate for other adverse developments. This ratio, the reserve life ratio, is defined as the ratio of the 'total saleable reserve' to the 'loan life saleable reserve'. A ratio of the order of 2:1 is usually considered satisfactory.

9.3.3 Production cost risk

Production costs comprise project operating costs (the costs incurred in day-to-day mining operations), and investment costs (the day-to-day costs of servicing both equity and debt capital). Costs are both fixed and variable. Fixed costs such as building, depreciation, interest and property charges are independent of production and continue to accrue regardless of whether the mine operates or not. Variable costs depend upon the quantity and value of the gold recovered and the volume of material mined.

Cost centres

Variable costs may be grouped under a number of headings:

- supplies; raw materials, reagents, fuels and other consumable materials
- services; electric power and water
- labour and administration
- repairs and maintenance
- company overheads including marketing, royalties, insurance, communication, and taxation
- freight and shipping.

Structuring operating cost

Table 9.8 presents cost data from a variety of sources as being generally indicative of relative costs at that time. The data apply to commercial-scale operations covering a variety of detrital minerals including gold. Capital and operating costs and water consumption figures are recast in this table as percentages. The standard for comparison, bucketline dredging, is taken as 100%.

The high end of the range of capital costs compiled by Fricker is fairly indicative of 1989 costs based upon recent quotations. The operating costs/m have risen steeply and, might be more than double the $US 0.50/m average for a single bucketline operation up to 1980. No data were given for bucket wheel dredgers. The figures quoted in the above table are from that author's experience up until 1988. Furthermore, they relate only to the cost of dredging not, as for the bucketline dredge, for a combined dredge/treatment operation. Data for suction cutter dredges have been omitted from the table as being generally inapplicable to placer gold mining. Where ground conditions are suitable for some form of hydraulic dredging, bucket wheels would almost certainly be preferred for their better digging characteristics, particularly in cleaning up at bedrock.

The percentage comparisons are generally applicable today but like the capital and operating cost comparison must be viewed in proper perspective. Unit costs for labour and diesel power (major cost centres) vary by up to hundreds of percent

Table 9.8 Commercial scale mining cost comparison (adapted from Fricker, 1980)

Method	Capital costs $/m³/year	Capital costs %	Total costs $/m³	Total costs %	Water consumption m³/m³	Water consumption solids %[a]
Dry mining						
F.E. loader	0.60	14	0.80	160	3–15	100
Scraper/dozer	1.40	33	1.20	240	3–15	100
Bucket wheel/conveyor	1.60	38	0.80	160	3–15	100
Walking drag line	2.00	47	0.90	180	3–15	100
Wet mining						
Gravel pump			1.50	300	10–15	139
Drag line dredge			0.90	180	3–12	83
Bucketline dredge	2.0–6.5	100[b]	0.50	100	6–12	100
Bucket wheel dredge[c]	2.0–5.0	78	0.65	130	10–12	122

[a] Average 9.0 m^3 water/m^3 solids – 100% standard.
[b] Average 4.5 $/$m^3$/year – 100% standard.
[c] From the author's experience (dredge costs only).

from country to country. In the USA, Richardson (1988) suggests US$1.50/yd (US$1.951m) as the bottom line for dredging plus recovery, considering such factors as environmental constraints affecting material handling and reclaiming the area after dredging, stripping or clay problems that might affect the production and recovery rates and whether the dredge selected might clean the bedrock without compounding the problems mentioned above.

It should be noted that Richardson's bottom line cost of US$1.95/m applies only to USA conditions. It contrasts unfavourably with 1988 production data from Malaysia where the cost of grid power averaged as low as US$0.07/kWh/day, US$0.03/kWh at night and labour US$6.04/day. The 1988 operating cost of a single dredge operation in Malaysia is US$0.53/$m^3$ (mining and treatment) plus overheads of US$0.12/$m^3$. Pre-strip dry mining costs at that time averaged US$0.38/$m^3$.

Addressing operating cost risk

Operating costs are influenced by changes in rates of production and gold recovery but not in every case, in direct proportion to the change. Economies of scale relate to costs for labour and administration and the production rate can usually be increased substantially for a very small increase in workforce numbers and costs. On the other hand, service charges for the purchase of electric power and water, and freight charges, which are normally paid for on a weight/distance basis will more closely vary in direct proportion to changes in the quantities mined. Another constraint is that operating costs are also greatly affected by the quality of labour and management and by the avoidance of process bottlenecks and unplanned maintenance.

The level of operating costs is strongly affected by the method of mining. Operating costs for bucketline dredging are generally lower than for other mining methods, provided that the deposits are large enough to justify the large capital cost. In some circumstances a bucket wheel dredger/floating treatment plant combination may be strongly competitive at a lower capital cost. However, this particular method is restricted in its scope, and is a generally higher risk operation than bucket-ladder dredging.

9.3.4 Financial analysis

Financial analysis considers the capital structure and financing aspects of the project. In simple terms, investment analysis is a study of the use of money and measurement of the returns; financial analysis looks into the ways of raising the money. Different financing alternatives are analysed to provide the optimal structure for the different periods of the project life.

Uncertainty analysis

Investment and financial analysis determinations are necessarily based upon assumptions of such parameters as production rates, capital investment, operating costs, revenues, discount rates, etc. The uncertainty analysis identifies the key variables and analyses them for uncertainty in terms of probability, sensitivity and break-even values. Uncertainty refers to how wide is the range of potential outcomes of a project. But while the various risk elements are identified, the uncertainties cannot easily be quantified.

Various methods of dealing with them include raising the selected discount rate or application of experience factors, using a sensitivity analysis, and assigning a probability function. Raising the discount rate is intuitive and the expected profitability and uncertainty are essentially independent indicators of the worth of an investment alternative. They should be separately measured. To combine them in the evaluation is to obscure the true basis for the mine development decisions.

Responses to uncertainty

All projects are afflicted by uncertainty and assumptions used in constructing the financial model must be realistic. The base case assumptions are only estimates and will prove to be wrong in some respects. Different techniques are used to allow for uncertainty. One of these is to raise the discount rate used in the discounted cash flow evaluations of potentially risky projects. Another response is a shorter payback period to justify investment in the project. This has the overall effect of raising the required return but there is no specific evaluation of the components of risk. A shorter payback period is also a comforting response to achieving a quick return of the investment in developing countries such as Papua New Guinea. High grading is a logical and justifiable technical means of obtaining a solution, if less than optimal to the problem. The responsibility of the country concerned to remove the country risk in order to prevent wasteful exploitation of its resource base is paramount.

Probability distribution

The probability distribution (i.e. the Monte Carlo simulation technique) is an assigned value given to the variables in a given investment opportunity. Using this distribution, the probability distribution of the profitability measure can be determined. Uncertainty can then be expressed in terms of numerical probabilities, i.e. 1.0 for an event that is certain to occur; 0.0 for one that cannot occur. Betting on coins, the probability of a head or a tail facing upwards is 0.5 in each case. For the possibility of more than one outcome, the combined probability must add up to 1.0.

The main difficulty of the Monte Carlo method is that it simulates random variations by using random numbers involving human judgement. Estimates are subjective and are made by analysts who may assign different probabilities to the occurrence of the same event. As a result, each may predict a different measurement of risk. This is effectively another form of experience factor based on past association with more or less similar projects. No method is foolproof and while evaluation techniques using cash flows and allowances for time value of money are superior to the earlier paybacks and methods, they are still not perfect (Rudawski, 1971).

Sensitivity analysis

The sensitivity analysis aims to identify such critical variables that if changed, can considerably affect the profitability measure. Individual variables are then changed, usually conservatively, and the effects on the expected rate of return (ROR) are computed. Profitability is most influenced by changes in ore grade, product prices and throughput. All investment decisions are based upon data that are subject to error to a greater or lesser degree. Analysing the effects of uncertainty and incorporating them into the investment evaluation is the hallmark of an adequate risk analysis technique. In addition to errors in borehole values, uncertainties are also associated with such items as capital cost over-runs, variable gold prices, variations from design digging rates and productivity. It must therefore be accepted that what is hoped will happen, may not happen exactly as planned.

Sensitivity analysis is carried out at two different levels. One is to consider the individual effects on the project of a specific fluctuation of each parameter from the base case assumption; although it cannot measure its uncertainty, i.e., the probability that the uncertainty will occur, the analysis will determine the effects on DCF of small changes in one variable whilst holding the other variables constant. The other is to consider the extent of change of all of the parameters of the base case in a best case and a worst case situation; since many of the parameters are interrelated, the arrays of each of the best and worst case parameters must be internally consistent; three values will be generated representing the most likely or base case; the best and the worst cases representing the extreme limits of the actual project.

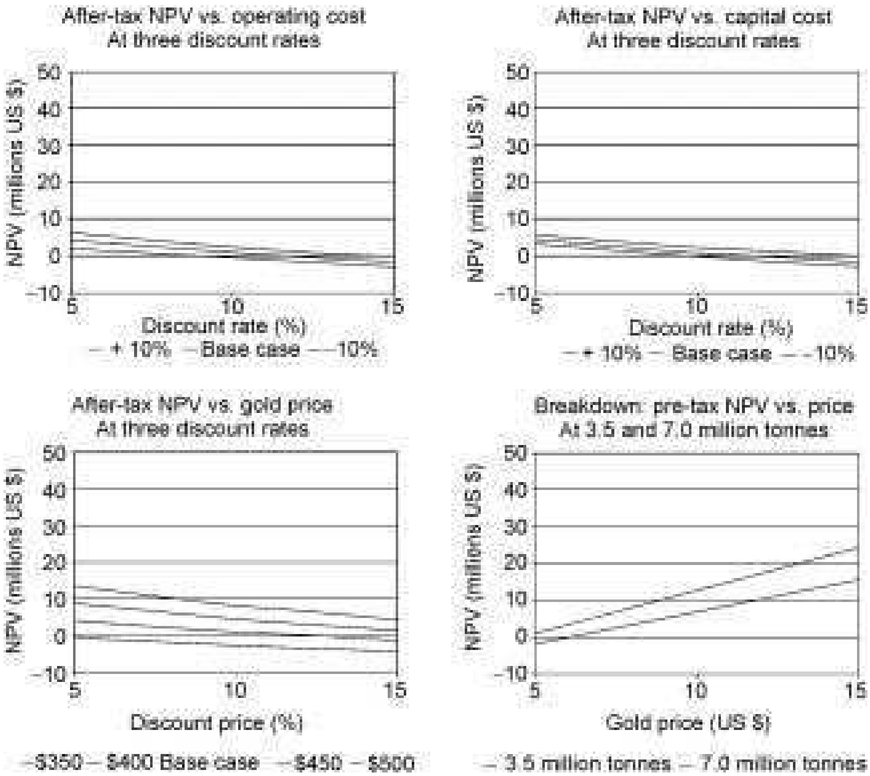
The advantages of sensitivity over the risk adjusted discount rate and shorter payback period is that individual areas of uncertainty are identified, thus allowing special attention to be given to their control (Malone, 1992). In the first format, which identifies the critical variables, the impacts of specific percentage changes can lead to improvements in the reliability of those variables. In the second format, the likely maximum variation from the base case assumption for each parameter is individually considered and integrated into holistic best and worst case scenarios. The analysis is a vital factor for decision making in marginal situations. It is also useful for the technical and financial control of bonanza-type deposits for which there might otherwise be a complacent acceptance of some areas of risk. If a possible change in a parameter would render the project uneconomic, that parameter should obviously be redefined before deciding to proceed.

Case history – calculating IRR for the all equity investment

The financial analysis chosen for this discussion required the estimation of year-by-year cash flows to the end of the projected mine life of a lateritic gold deposit in Suriname, South America. Individual cash flows were discounted to 31 December 1984 dollars to yield a net present value (NPV). Four of the detailed flow outputs are graphed in Fig. 9.3. A base case and 12 sensitivities were undertaken for both pre-tax and after-tax results. The following base case assumptions were made:

- reserves of 3.5 and 7 million tonnes
- mining rate 1,000 tonnes per day
- gold recovery of 95%
- gold payment factor of 97%
- gold grades variable – average 2.4 g/tonne for reserves of 3.5 million tonnes and 2.28 g/tonne average for reserves of 7 million tonnes.
- taxes include a 1% net smelter return and 49% of profits after recovery of capital
- constant dollar (no inflation in prices and costs) revenue risk.

The analysis established a base case, after-tax net present value of $0.8 million and a pre-tax net present value of $4.0 million dollars. These values showed extreme sensitivity to the gold price and to discount rates as shown in Table 9.9.



9.3 Flow output types graphed for calculation of IRR for all-equity investment.

The degree of sensitivity of the project to the price of gold can be highlighted by noting that the value (using a low 10% discount rate) of the project on an after-tax basis, rises approximately $1 million dollars for every 14% rise in the price of gold. Similarly, on a pre-tax basis, the value increases by $1 million for every 9% rise in the price of gold. A discussion on gold price fluctuations is included in

Table 9.9 Before and after taxation effects on discount rates

	Before Tax NPV (discount rate)			After tax NPV (discount rate)		
	5%	10%	15%	5%	10%	15%
$350 Gold	1.2	−1.8	−3.8	−0.6	−3.0	−4.6
$400 Gold	8.9	4.0	0.8	4.2	0.8	−1.6
$450 Gold	16.5	9.8	5.2	8.8	4.3	1.3
$500 Gold	24.2	15.6	9.7	13.4	7.9	4.1

Appendix IV. The analysis also shows that the value is sensitive to reserves. Doubling the reserves to 7 million tonnes (and assuming the same production rate) while assigning a lower grade to the additional reserves quadruples the after-tax present value. It should be noted that if the expected reserves were 7 million tonnes, the production rate should also be increased significantly and the net present value would increase by a greater amount than that indicated above.

Using a 10% discount rate and the base case assumptions, the break-even graphs show a pre-tax breakdown gold price of about \$360/oz. and an after-tax break-even price of about \$390/oz. As one would expect, the break-even prices of gold decline as additional reserves are added. With the assumption of a 7% inflation rate in prices and costs (and 100% equity funding), the net present value at a 15% discount rate is \$1.5 million. With the introduction of debt financing (100% debt at a 12% interest rate), the after-tax net present value increases from \$1.5 million to \$3.7 million at a 15% discount rate. It should be noted that the 12% interest rate referred to is a combination of 7% inflation and a 5% real interest rate. Inherent in the results shown in Tables 9.10 and 9.11 is the

Table 9.10 NPV after tax

(a) Base case reserves (3.5 million tonnes)				
			(Discount rate)	
Constant dollars (100% equity)		5%	10%	15%
\$350		−0.6	−3.0	−4.6
\$400		4.2	0.8	−1.6
\$450		8.8	4.3	1.3
\$500		13.4	7.9	4.1
Operating costs	+10%	2.1	−0.8	−2.8
	−10%	6.2	2.3	−0.4
Capital costs	+10%	2.8	0.6	−2.9
	−10%	5.5	−2.0	−0.3
			(Discount rate)	
Inflated dollars		10%	15%	20%
\$400 gold (100% equity)		5.0	1.5	1.0
\$400 gold (100% equity)		5.6	3.7	2.5
(b) Double the reserves (7.0 million tonnes)				
			(Discount rate)	
Constant dollars (100% equity)		5%	10%	15%
\$350 gold		2.9	−1.4	3.8
\$400 gold		10.1	3.6	−0.2
\$450 gold		17.1	8.3	3.2
\$500 gold		24.1	12.9	6.6

Table 9.11 NPV (before tax and royalties)

(a) Base case reserves				
			(Discount rate)	
Constant dollars (100% equity)		5%	10%	15%
$350 gold		1.2	−1.8	−3.8
$400 gold		8.9	4.0	0.7
$450 gold		16.5	9.8	5.2
$500 gold		24.2	15.6	9.7
Operating costs	+10%	5.6	1.5	−1.2
	−10%	12.2	6.4	2.6
Capital costs	+10%	6.9	2.2	−1.0
	−10%	10.8	5.8	4.6
Inflated dollars		10%	15%	20%
$400 gold (100% equity)		11.2	5.9	2.3
$400 gold (100% debt)		10.1	6.7	4.6
(b) Double the reserves				
Constant dollars (100% equity)		5%	10%	15%
$350 gold		7.3	1.1	−2.3
$400 gold		19.0	8.8	3.1
$450 gold		30.8	16.5	8.6
$500 gold		42.5	24.2	14.0

concept that the risk associated with the probability of a deviation from the base case greater than that tested for is small. This concept allows the risk and sensitivity analyses to be combined.

9.4 The feasibility study concept

The feasibility study is a process for gathering information and using that information to estimate and forecast the likely results of proceeding with the development proposal. Budget projections are based upon proving up the minimum size deposit believed to be mineable in accordance with the requirements of the venture and are necessarily made without having the full advantages of local experiences yet to be gained. Instead, assumptions are made on the possible geometry of the deposit and its lithology in order to assess the costs of exploration from grassroots to final evaluation. Since all predictions are tentative as much geology as mathematics is used in the early stages.

Decision points are reached from time to time during the evaluation process and the initial estimates will establish a broad time frame for the future investigations and expenditure on capital items; operational costs will be budgeted for accordingly. Capital items will comprise all material and equipment for setting up a main camp and out-camps, transport, roads, drill rigs, casing, tools, sampling equipment, laboratory, workshop and stores. Operating costs are those incurred for labour, technicians, and technical support, field supplies (consumables, spares, etc.) fuel, food, communications, transport in and out of the project area, medical supplies and so on. Preliminary cash flow projections will be updated at appropriate times and critical decisions upon whether or not to proceed may be made a number of times prior to obtaining sufficient data for a full feasibility study. Such decisions will be influenced by the gradually unfolding geological picture and by costings that become increasingly comprehensive and reliable with time and with the acquisition of more complete data. Each successive interim study will update what has already been done and will incorporate fresh data and consider fresh possibilities.

Budget control is essential at all stages of the exercise and no operation can function effectively without up-to-date financial statements. This entails the formulation of financial procedures relevant to the particular project and the design of financial data sheets that fit the requirements of both field and company in-house accounting. Treatment and marketing are the most suitable of possible alternatives. All of the costs associated with the project are estimated to determine the cost of production and hence the price that will provide the required return on investment. Most importantly, the author of the proposal must display his knowledge of the resource, and provide evidence that the proposed methods of mining are the correct ones in order to demonstrate the technical feasibility and economical viability of the project.

In practice, feasibility means different things to different people, and those who provide the finance determine feasibility in different ways. 'Blue Sky' projects, which are inadequately prepared or that fail to address the significant issues, rarely get through the financing stage without some degree of study. Normally, the lender considers the ability of the project to repay principal and interest within the scheduled time with the least degree of risk, and under a possibly wide range of economic, political and technical conditions. Banks that are active with mining companies typically have their own specialist groups consisting of mining, economic and banking consultants who will analyse, measure and evaluate the project risk according to their own levels of knowledge and experience. Basically, however, they still protect themselves by forcing sponsors to provide completion undertakings and or sufficient equity to overcome any problems that may develop.

9.4.1 Stages of feasibility

The feasibility study is a detailed technical plan and financial budget for the construction and operation of the project. It is carried out in a number of stages, which reveal the sources of the finally selected options and how many alternative options were considered. The first stage commences as soon as sufficient resources are identified to justify a large-scale testing programme. A disciplined analysis is undertaken to ensure that all of the various issues that contribute to feasibility are properly addressed in everything that is done from that point onwards. Sampling procedures incorporate physical and metallurgical studies of all materials that may eventually be fed into the treatment plant. These studies include slime settling, ground water investigations, sediment particle size distribution, gold size relationships and fineness; in fact, all factors needed to establish technical feasibility before proceeding to the second stage of mine planning investigations. Economic studies will be developed concurrently with the technical studies in order to demonstrate both technical and economic feasibility before advancing to the next stage and ultimately to completion. An environmental impact statement will state clearly the costs of what must be done to protect the environment from any possible damage that might be caused by the proposed operation.

Physical performance and financial performance vs. budget are measured against the feasibility predictions throughout and beyond the construction period for a few years. More detailed budgets will be prepared annually from the commencement of operations. The cost of a feasibility study, if it is comprehensive, authoritative, reliable and accurate, is high, and may represent a significant proportion of the total engineering cost of the project. Quantitative data are expected to be accurate within a minimum of 15% where much of the engineering work has already been done.

The developing situation will by this time have provided the basis for an investment decision. The project and details of project activities will be defined so that estimates of all anticipated costs and revenues can be integrated into a financial model of the project. Ideally the staged progress of the project will have helped to identify options, which can be tested, modified or discarded and replaced until the most satisfactory combination of options has been selected. In practice, many projects fail to meet feasibility study prediction for a number of reasons:

- Various pressures may have resulted in a less than viable selection of options; cases are referred to by Malone (1992) in which major gold mining companies, with great experience, have made major write-offs of capital as a result of feasibility predictions that were very wrong.
- In some notable instances variability against prediction ranged from as much as +20% to −100%; e.g. The Big Bell open pit mine in Western Australia, for which there was already voluminous information from previous mining.
- Some projects come on stream with mill throughput significantly above design without any additional capital expenditure.

For those who prepare feasibility studies the usual explanation of such variability is the inherent difficulty of predicting outcomes from an array of unrelated parameters, all of which are subject to a degree of uncertainty, the compounding effect being a wide range of variability. A cynical view is that the rare event of the actual result matching the feasibility study predictions is probably due to compensating errors rather than to correct predictions. It is safer to assume that feasibility studies are relatively inaccurate and that a conservative approach is essential when making predictions based on other than factual data. The difficulty is not to be so conservative that projects worthy of development are unnecessarily rejected. In fact, by going to extreme lengths to protect companies from making unprofitable investments, the companies will also miss out on many profitable investments and could soon disappear from the mining industry.

The answer may be to review the feasibility more carefully, consider the potential for compounding the uncertainties and determine if the project has or has not a sufficient margin for error to survive that range of uncertainty. By these means the assessor will be attempting to establish a range of possibly viable outcomes rather than trying to predict an absolute value. Management, in its attempt to establish a value for the project, must then decide upon the minimum outcome that will safely yield an acceptable rate of return. This may mean that the margin for error in the assessment process will be improved by establishing a lower purchase price for the project. At the other end of the spectrum, it might also mean that some projects could come on stream with mill throughput significantly above design without any additional capital expenditure.

9.4.2 Final feasibility

The final stage of a feasibility study is reached when all of the pre-mining information on the deposit and its development appear to demonstrate technical and economic feasibility:

- As much as 50% of the engineering design work will have been completed to form the basis for construction cost studies.
- Mine planning and scheduling of feed materials to the mill will be sufficiently advanced that it can be used confidently as input to the feasibility study.
- Marketing studies will include the basis under which the gold can be sold.
- The environmental impact statement will have been prepared and where needed modified as the result of feedback from government bodies and other interested parties.

All aspects of importance are covered including those dealing with the ore resource, mineable reserves, mine plan, metallurgy, capital and operating costs, working capital, maintenance, marketing, infrastructure, environmental management, industrial relations and the social, legal and political environment. A proper assessment of the study will scrutinise all aspects of the inputs and their interrelationships, as well as the background and philosophy of the team carrying out the study before deciding upon the reliability of its projections as the basis for a financial model. The complete study must cover all aspects of importance to the project and, as already discussed in this chapter, deal additionally with projected capital and operating costs, working capital, maintenance capital, marketing, infrastructure, environmental management, industrial relations and the social, legal and political environment. The feasibility study is an expert report and is intended for an expert audience. Hence, the importance of the study is in what is written; irrelevant descriptive material is a waste of time and money and if present, is often intended to distract attention away from the actual facts.

Technical aspects of the study

The contents of the section dealing with deposit geology should be described in sufficient detail to allow an experienced assessor to form a considered judgement on matters relating to the resource estimation, the manner of testing, the proposed mine plan and metallurgical treatment. Details of each resource estimation should be given in terms that can be understood by experts who understand the calculations used, and what should have been done to arrive at the conclusions made. Bench and pilot-scale test work on ore samples should be described, with particular care given to demonstrating the reliability and representivity of the samples tested to the range of qualities expected of the orebody. It should be stated if this representivity applies to the orebody as a whole or to only one part of the orebody. Any differences in ore zoning, whether in the nature of likely mining or metallurgical performance, must be described in relation to what will be done to achieve the predicted production. The possible impacts on mining or metallurgy of any required changes must be individually identified and the means of handling it addressed. Measures taken to achieve representivity in terms of predicted metallurgical performance should be described in detail; the results of bench and pilot-scale test work should be correlated with the nature of the ore samples to confirm the relationship.

The nature of the samples submitted for test work and their level of representation of the orebody in part or as a whole must be stated. If the samples relate only to particular zones of the orebody having distinctly different characteristics and/or metallurgical responses, then the study must show how the zoning will be handled to achieve the required production. Any feature of the orebody such as a lateritic capping that will require selective mining and individual treatment must also be addressed, as should be descriptions of the proposed mining and metallurgical methods that will be applied to the specific material.

Financial aspects of the study

The above discussion has concentrated on technical aspects of the feasibility study. To the financier, the feasibility study is the basis for financial evaluation and obtaining finance for the project and is commonly referred to as a 'bankable' feasibility study. Financiers expect that the study will present a critical appraisal of accuracy and reliability of the input data and interpretations and of the associated technical, managerial and financial risks to be assumed by the company. However, they will rely upon the results of a particular feasibility study only to the extent of the technical and management skills that can be applied to the development of the project.

Although financiers tend to concentrate on the capital and operating cost estimates they will want to be assured of the ability of the cost structure to withstand the lows of business cycles rather than simply the expectation of future profits. Amongst the uncertainties that affect predictions made in feasibility studies are mine design faults such as open-cast cave-ins that should have been anticipated and accounted for in the mine design. Lower than forecast gold prices can have similarly catastrophic effects unless hedged against by forward selling contracts. The economic outlook at the time of carrying out the study can influence the reliability of the assumptions made. This was a major problem in the bull-market of the 1980s when gold prices rose to unrealistically high levels for a brief period, and then fell dramatically. In addition, management constraints, such as in relation to the size of the project, may not properly consider the true dimensions of the project. Many instances are known of projects that failed because of operations that were commenced prematurely in high-grade material, simply in the hope that these values would continue at economically sustainable levels.

Definitions of terminology and the basis and methodology for establishing the estimates include the following for capital costs:

- date of estimation
- grouping of costs consistent with technical description
- nature and number of quotations for various items
- sources of major plant and equipment items
- allocation of freight, sales tax, landing costs and duties
- construction labour productivity, working schedules, and assumptions on rates, benefits and on costs
- timing and distribution of the expenditure.

At any given date the base estimate is increased by allowances for contingency and escalation. The feasibility study should detail how they are calculated although when estimated it may not be clear where they should be charged. Escalation should be charged from the date of estimation to the expected date of commencement of construction and thereafter to the remaining expenditure to the end of the construction period. Escalation applies to the contingency allowance also.

Correct estimating of production costs is generally of greater importance to the project than accurate estimation of capital costs. Operating costs are estimated in relation to logical cost centres representing all phases of the operation. Descriptions of estimates should include:

- methodology of cost estimation
- fixed and variable components
- manpower requirements
- wages, benefits, on costs by jobs classification
- allowances for training, absenteeism, turnover
- equipment running cost estimates
- materials, power, fuel and consumables usage factors and unit costs
- freight, royalty and selling expenses
- overhead cost estimates
- indicated inflation trends for various components
- any anticipated changes in costs futures.

In the final feasibility study the accuracy of the estimates should be known, the only uncertainty being rates of change in the future.

Appendix I

Field laboratories and techniques

Basic equipment

Items of equipment for routine studies are relatively inexpensive and easy to use. The possible benefits of some of the more exotic procedures must be weighed against cost and the difficulties involved with their use. The following items are suitable for medium-to large-sized programmes:

- micro-panner
- fume cupboard
- binocular microscope
- best of screens and shaker
- direct-reading assay balance
- simple bulk weighing balance
- hot plates
- small amalgamating barrel.

Useful laboratory items include:

- standard glass slides
- long thin bar magnet
- small plastic vials with lids
- self-adhesive labels plus waterproof felt-tipped pens
- tubes of strong adhesive glue
- latex glue or rubber cement
- small stainless steel anvil and tack hammer
- heavy liquids (see heavy liquid separation)
- separating funnels, beakers and miscellaneous glassware
- assorted evaporating dishes
- graduated slide or graticule for measuring particle size.

Techniques

The following methods may be employed in the study of gold placer concentrates where panning is still a useful process. Some non-valuable minerals will fortuitously remain in any panned concentrate and it may be useful to have these identified since some may pass through to final processing units in the gold room.

Hand picking

Coarse gold may be picked out by hand using a wooden toothpick or sharpened match. The end is wetted and used to pick up each grain separately. This procedure has the advantage over amalgamation in that the gold is recovered in the same state that it existed in the ground, thus yielding accurate information of size range and other physical parameters. Hand picking also allows the fineness of the gold to be determined without the uncertainty attached to losses of soluble metals such as silver and copper during the parting operation. The process is more tedious than amalgamation only for large numbers of fine particles. For them the labour must be cheap if the samples are to be dressed by hand at reasonable cost.

Using a binocular microscope

An experienced observer can identify most commonly occurring detrital minerals including metallic and non-metallic varieties. It may be useful to pick out a few dozen grains (especially non-metallic) from under the microscope and store in plastic vials for future reference, e.g. in a reference library.

Using a petrological microscope

Thin sections of concentrate can be prepared by special techniques and studied under a petrological microscope using such observations as colour, pleochroism, cleavage, and extinction angle and polarisation colours. This should serve to identify non-opaque minerals but would not positively identify the gold and some other metallic grains.

Mineragraphic or polished section study

Small sections of concentrate are mounted in cold setting plastic and polished. Reflected light techniques including micro-hardness and reflectivity will identify most opaque and metallic minerals including gold. Grain counting will provide an estimate of the relative proportions of the minerals present.

Heavy liquid separation

It is often desirable to isolate specific mineral types for separate study. Heavy liquid separation utilises the density of mineral for separation. Grains are placed in heavy liquids for a ‘float’ or ‘sink’ test. Heavy liquids are available up to a density of 4.4 g/m^3. Any mineral that sinks at this density is worthy of further examination because it might be an economically valuable component. Most minerals that float at 4.4 are likely to be non-economic species; rutile, zircon and the gemstones are exceptions. It is often desirable to isolate specific mineral types for separate study. This may be accomplished by removing highly magnetic minerals such as magnetite using a hand magnet. For preparing heavy mineral fractions using a variety of heavy liquids for density classification and cleaning the individual fractions further by the removal of such valueless minerals as pyrite, marcassite and limonite in aqua regia, the separation procedure is as follows:

- Stir a small amount of the concentrate into the selected heavy liquid in a separation funnel or flask. The lights float to the surface and may be overflowed or scraped from the top. If a separating funnel is used, the ‘heavies’ are removed from the bottom after taking the ‘lights’ away.
- Wash the recovered heavies using a solvent such as methylated spirits or acetone and examine microscopically.
- Recover the heavy liquid residues for beneficiation and further use by passing the liquid through a filter funnel and filter paper to remove the solids.

A micro-panner may be used to scavenge any gold caught up in the heavy mineral fraction.

Using X-ray diffraction

Though somewhat time consuming, X-ray diffraction is the most definitive method of identification, every known species having its own X-ray pattern. A small amount (say 20 or 30 grains) of the unknown mineral is extracted beneath a binocular microscope by a wetted needle and powdered in a small mortar. The powder is picked up on a thin glass spindle coated with a veneer of inert adhesive. The spindle is then placed in an X-ray camera and the pattern on the exposed film measured for both ‘d’ spacings and line intensity ‘l’. Reference to the ASTM index will provide accurate identification for which the theoretical composition of the mineral can be given.

Using an electron microprobe

Using a polished section of mineral grains the major elements of a mineral can be obtained accurately by an electron microprobe. Quantitative analysis of minerals can also be made by means of a scanning electron microscope (SEM) which is also able to take three-dimensional photographs of minute grains.

Using an atomic absorption spectrometer (AAS)

A small amount of mineral is dissolved in aqua regia and a beam of light is passed through the solution. Each type of atom present will absorb certain wavelengths of the light; the amount of absorption being proportional to the concentration of the atoms present.

Amalgamation

Amalgamation is a commonly used technique but for health reasons should be avoided wherever possible. Where its use is mandatory for practical reasons (e.g., large numbers of very fine gold particles, which would be very difficult and time consuming to pick out by hand), extreme care must be taken not to inhale any mercury fumes from the parting process. Residues should be disposed of carefully away from any possible human contact.

The amalgamating process simply requires adding a few globules of mercury to the concentrates in a shallow dish, and gently moving it around to allow all of the gold to make contact with the mercury. When no more gold is visible, wash the amalgum with water and place in a small crucible. Add a small quantity of 50% HNO_3 and warm on a hot plate. The mercury dissolves forming Hg $(NO_3)^2$ leaving a spongy residue of gold in the bottom of the crucible. Dry and add a drop of acetone and anneal at high heat. Acetone prevents spattering if any moisture is retained in the spongy mass after drying. Weigh the gold, using a high-sensitivity balance and store in a plastic vial.

Size-shape analyses

Various methods of sizing are available but the most useful technique for a field laboratory is sieving, using a nest of sieves and a mechanical shaker. BS and Tyler screens are amongst the most common types and have within their range the screen sizes shown in Table 4.2. Microscopic measurements are made approximately using a glass slide or plate that has been engraved with a scale divided accurately into, say, 0.25 mm divisions. With experience, sizes much less than 0.25 mm can be estimated by interpolation. Glass slides with a 1 mm line divided into 100 sub-divisions can be obtained for use with binocular or petrological microscopes.

During this procedure, grain shapes (e.g., flaky, irregular, rounded, crystalline, wires) can be noted for the various size fractions. Such information can be of considerable use where unforeseen losses occur, e.g. where there is a marked discrepancy between initial assay results and recovered values. Tailing losses, for example, may be due to the lack of recovery of ultra-fine gold but larger gold grains of a particular shape may also be represented with the finer fractions. With gold, there is quite a range of different shapes.

Sub-sieve materials (e.g., silts and clays) may be separated in a measuring cylinder by shaking the sediment vigorously with water and allowing it to settle. The silts settle out rapidly but clay remains in suspension in the water column for several hours and will then settle only to a certain level. Clay particles are often measured in terms of the settling time to reach a state of equilibrium in the cylinder.

Colour coding

The sample column technique for visual display represents an extension of colour code chart techniques used in the oil drilling industry. Lengths of glass or clear plastic tubing about 3 cm diameter are cut into 10 cm to 20 cm lengths and sealed at one end using tightly fitting corks and latex rubber glue. A sample of alluvium sufficient to fill about two-thirds of the tube is placed inside and the tube is almost filled with water. The tube is shaken vigorously and the sediment is allowed to settle. Water above the sediment is drawn off by pipette leaving a permanent record of the sediment arranged on the basis of sediment size. Mud colour differences may be used as a diagnostic tool to correlate sediment layers intercepted in widely spaced boreholes.

For permanent storage, the tubes are allowed to dry out until most of the water has evaporated, the remaining small amount keeps the sample layering in place if handled carefully. The tubes are sealed and filed away in a vertical position in specifically made wooden holders. They are then available for comparative studies when required.

Colour indexing of alluvium

The basis of this procedure is quite simple and takes notice of the old adage used in sediment lithology ‘the closer you look the less you see’. The author of this adage went on to say ‘the colour, composition and texture of cuttings can often be determined better by megascopic examination than in any other ways’. Be this as it may, the fact remains that mass colour of sediment can be an important element in stratigraphy. This extremely useful technique is used consistently in many geological pursuits to index sediments by colour coding. It is important to view the colour changes in a broad way when using a colour-coding chart; the samples should not be viewed in any minute or detailed form.

Preparation of colour code charts using actual sediment

Most colour coding uses a range of artificial colours as the colour comparison medium. The actual sediments must be used to convey colour variations and it is stressed that some experimentation will be required to determine which screen size will give the optimum colour variation. Usually, a somewhat finer fraction than the -1 mm size range will give a much better and more uniform result.

The sediment is glued in stratigraphic sequence onto a piece of stiff white cardboard with appropriate details appended to each sample. This is best done with a piece of heavy metal (or similar material) with a 3–4 cm square or circular gap cut through it to form a rim. When this is in place and held down firmly, latex rubber glue is painted onto the cardboard through a hole in the guide rim. Care in the thickness of the glue is important as it is necessary to have a uniform thickness of sediment throughout; too much glue will allow too great a thickness of sediment to be affixed. With the recommended very fine fractions, a thin layer of glue will give the required results without too much pebbly material.

When the sediment is poured into the hole in the guide rim the rim may be removed and the sediment adhering pressed down onto the cardboard firmly. Latex glue dries very quickly and there should be no delay in applying the sediment to it. Latex glue is the preferred glue type, as it does not disturb the colour of the sediment and can be rolled or folded without disturbing the adhering sediment. Finally, any surplus sediment that has not penetrated into the glue may be blown or lightly brushed away.

For any form of colour indexing to be of use any changes that are sought should be absolutely unambiguous and definitive. Colour differences should thus be sufficiently apparent that several observers are able to identify the colours independently. Equally important is the matter of reproduceability, e.g. three or four separate sievings from, say, a 1 m section should give the same colour.

Appendix II

Variogram structural analysis

The variogram is a fundamental tool used to study a regionalised variable such as the concentration of gold within a placer deposit. It is constructed by identifying pairs of values that are separated by multiples of a given amount of distance, e.g. 0–20 m, 20–40 m, 40–60 m, etc. Within each distance grouping the difference between each pair of sample values is squared, the squares are summed and then divided by the number of pairs. The resulting value, usually called 'gamma' is a measure of the variability of samples within that distance group. The variogram display consists of a graph of the gamma values plotted against separation distance. Variograms often take the form illustrated in Fig. 6.36. Statisticians in sampling design and ore reserve estimation commonly use this form. The parameters that define this form are known as nugget value, sill value and range of influence.

Gamma values for separations greater than the range of influence are quite constant, and indicate the variability of sample pairs that have no correlation between values. The nugget value is the variability projected for zero separation. The sill value is the amount that variability increases until the distance limit of sample correlation is reached. A variogram showing true nugget effect has no sill value and no range of influence. A sampling programme is said to be statistically efficient if every point in the sampled body is spaced within the range of influence of at least one other sample.

Design of alluvial gold processing plant is based primarily upon the results of borehole sample dressing at the drill site followed by bench and laboratory-scale investigations. Small-scale plant is then used to test the ore materials under simulated prototype conditions to provide data for physical relationships that can be scaled up to any desired dimensions. The task is to examine those factors relating to the prediction of particle stratification in shear flow (e.g., spirals, cones, sluices and tables), and under the pulsatory conditions of the jig bed.

Appendix III

Sitework testing

Initial concepts are developed for dealing with the removal and replacement of topsoil and overburden, and strategies are developed for treating the ore, recovering the gold and safely disposing of the tailing and slime. Soil testing is carried out over the area of the proposed pit and its immediate surrounds. In addition to pit studies the work is extended to cover all of the areas selected for mining and ancillary purposes such as treatment plant sites, processing dam sites, slimes dam sites, buildings and infrastructure, power reticulation and access. Ground studies normally comprise:

- close drilling around the boundaries of the area selected for opening-up operations
- soil tests to examine the stability of each of the various lithologies represented in the pit walls
- hydrology studies, including falling-head permeability tests at selected borehole locations during both wet and dry seasons of the year
- tests to determine the amounts and type of surfacing materials for haulage roadways.

Bulk density measurements

The bulk density of rock material differs according to the density of the solids, degree of compaction, percentage of voids in the mass of particles and the extent to which the voids are filled with water, air or gas. Most samples are taken in a disturbed state and their properties of bulk density and swell can be measured only indirectly using physical processes of compaction, including vibration and compression, to bring the samples back to a semblance of their original state. Swell factors are calculated by comparing the volume of the cylinder of material excavated with its volume when broken down in a measuring box. However, although calibrated boxes with vertical walls braced firmly for strength and rigidity are used for measurement, it is very difficult to reconstitute the material evenly. Regardless of the care taken and how well the techniques are standardised, the results are typically variable and must be averaged statistically for prototype design. Indeed, with long size range sediments, it is almost impossible to return any one sample to even a rough approximation of its original state. A statistically proven number of samples must be tested before accepting an average value.

Performance data from within the industry are similarly unreliable. Sand and gravel merchants usually adopt an average bulking factor of 1 m^3 (bank) = 1.3 m^3 (loose). This relationship is assumed to provide for a range of operational contingencies such as differences in lithology, moisture content, etc., but is probably biased in favour of the contractor. Other authorities differ widely in their assessment of bulking factors for similar type materials.

Direct bulk density measurements can be made only in good standing ground above the water table. In suitable conditions, a cylinder of the material may be bored out, thus creating a regular cavity with smooth sides. The cavity is filled with a measurable volume of very viscous oil or a quick-setting substance such as plaster of Paris. Water may be used in areas of permafrost. The bulk density of a sample is computed by dividing the dry weight of the excavated material by the volume of filler used. Swell factors are calculated by comparing the volume of the cylinder of material excavated with its volume when broken down in a measuring box. One difficulty in using this method is the variability of individual samples. In compacted ground the cavity may be measured physically as at Royal Hill, Suriname where samples of gold-bearing laterite were chipped out of carefully measured channels in the ground. The bulk densities of pulverised and dried lateritic samples cut from four trenches gave dry weight measurements showing bulk density variations ranging from 1.842 to 2.176, a differential of almost 20% as shown in Table AIII.1. Individual tests of different combinations of the materials *in situ* yielded average bulk densities of kaolin 1.5, ferruginous claystone 1.6 and alluvium 1.2.

Whether a deposit is to be mined wet or dry is of major consequence when determining the type of swell factor to be used for transport and storage. The

Table AIII.1 Bulk densities of 6 pulverised and dried lateritic samples from 4 trenches at Royal Hill, Suriname

	Trench No.	Volume cm^3	g	Density
1	1	127,000	277,460	2.176
2	2	211,360	390,350	1.847
3	3	40,000	72,800	1.820
4	3	40,000	82,200	2.055
5	4	24,000	48,700	2,029
6	4	40,000	77,100	1.928
Totals		482,860	948,610	1.965

water/ore ratio may range up to 40–50% in very wet ground but commonly averages between 5 and 10%. The ratio is important because any increase in water content lowers the bulk density of the ore and reduces the weight of dry solids fed to the plant. Dredge bucket fill factors are estimated using saturated swell factors. Dry swell factors are used in studies involving dry mining transport and overburden disposal.

Index testing

Tests provide information on present water content, unit masses and Atterberg limits, unconfined compression tests, consolidation tests, vane shear tests and CBR (California Bearing Ratio) tests on remoulded, soaked samples. Bores are to be logged to their full depth. Data include:

- visual classification, ground water level and comments on boring progress, sample recovery and other pertinent details
- disturbed samples taken in different strata
- undisturbed samples (say 50 mm tube) taken in differing strata
- vane shear tests taken in differing clay strata to be replaced in granular material, typically at 1.5 m intervals.

Plant site and services (e.g., workshops, powerhouse)

Bores or back hoe pits to be sunk to a depth of 2 to 3 m, or 1.5 times the width of major (heavy load) foundations below the founding level of the plant, building and equipment.

For slimes dam footings, bores to be sunk to a depth of 8 to 10 m along the embankment at spacings of typically 200 m centres. If warranted by site conditions, spacing is varied to ensure adequate definition of variation in foundation conditions. Bores and back hoe pits in borrow areas should be sufficient to define variations in material properties throughout the extent and depth of prepared areas.

Laboratory tests

Index tests for classification of unit masses from disturbed and undisturbed samples:

- compaction tests (standard) on representative samples from borrow areas
- unconfined compression tests or preferably drained tri-axial tests of remoulded and compacted disturbed samples of representative borrow area materials
- shrinkage/swelling tests on remoulded and compacted disturbed samples of representative borrow area materials

- consolidated tests of undisturbed samples representative of various soil types in material beneath foundations
- grain size distribution analyses for granular materials; these should be standard procedures (e.g., AS1289, ASTM or AASHTO).

Note that information should be sought on:

- local experience and practice
- source and quality of embankment materials
- source and quantity of pavement materials
- seasonal climatic data – rainfall, temperature, humidity and liability to flooding.

Appendix IV
Gold economics

The present system of controlling the value of gold was established to bring a better level of order to the global gold market than had previously existed. The yearly average fine gold prices in the period 1834 to 1867 were constant at \$20.67/oz. Prices fluctuated up to an average maximum of \$47.02/oz. during the years 1862 to 1875 when demand exceeded supply during the final stages of the 'Great Gold Rushes'. The price fell to its previous level of \$20.67/oz. and remained steady at that figure until a worldwide depression in the early 1930s once again fostered gold exploration. In 1934 the price rose to around \$35/oz. where it remained until it was agreed between the nations to adopt some formal system of gold valuation.

Under the 'Bretton Woods' International Monetary System, the official price of gold was fixed at US\$35 per fine ounce. The US Treasury stood ready to buy gold from, or sell gold to, foreign banks and official institutions. The free gold price fluctuated around this price for a few years, but the system began to break down towards the end of the 1960s and on 15 August 1971, President Nixon unilaterally severed the link between the dollar and gold. After the Bretton Woods International Monetary System collapsed in 1971, the price of gold rose steadily, reaching a peak in the beginning of 1975. The deep world recession through 1974 and early 1975 led to a moderation of consumer price inflation throughout the world in the following two to three years.

Subsequently, world nations decided to entirely remove gold from the international monetary system. Accordingly, in November 1975, Article IV of the IMF was revised to prevent nations fixing par values of their currencies in terms of gold. Furthermore, it was decided that the IMF would gradually dispose of 150 million oz. of gold, which it held as part of the quota subscriptions of members accumulated since the inception of the Bretton Woods system. Following this decision, the IMF auctioned 25 million oz. of gold to the public between June 1976 and May 1980, and returned another 25 million oz. to the various countries that had previously subscribed to it as part of their original quotas. In addition, the USA sold 17 million oz. of gold from its official reserves between January 1975 and November 1979, in support of its stated policy of gold demonetisation. Although US auctions were discontinued, the Treasury indicated that it would resume sales whenever it wished to do so.

Nevertheless, despite these moves towards demonetising of gold, countries continued to hold gold in their reserves and, in fact, purchased substantial amounts at the IMF and US Treasury auctions. By 1984, the central banks of world nations held reserves of almost one billion oz. of gold, against the 100,000 oz. still remaining in the Treasury coffers. The European Common Market also set up its own European monetary system fund, under which all member countries are required to subscribe a certain proportion of their quotas in gold.

Other forces were also involved. The Opec oil cartel more than trebled the price of oil between 1970 and 1973, plunging the world into double-digit inflation. Gold prices soared rapidly and when the Russian armies invaded Afghanistan in January 1980, the price of gold rose to its highest recorded peak (US$850/oz.) a few days later. This led to a great deal of unloading and selling of gold by speculators. Tight monetary policies and recessionary trends led to a further decline in demand and the price moderated to around the US$400/oz. level.

High gold prices combined with a volatile stock market triggered a fresh wave of speculative ventures in the 1980s headed largely by inexperienced operators who tended to underestimate costs and overestimate possible rewards. Indeed, in many cases, companies were set up with no real intention to establish profitable operations, but simply to make short-term gains on the share market. This period ended with almost complete disillusionment in the late 1980s; one project finance banker (Tinsley, 1990) stating at the time that, in the past 15 years, he did not know of a single alluvial gold project financing that had not gone into serious default. Adding to current problems, lower yields from placer operations appear to be compensated for by increased production from newly discovered epithermal orebodies (some very rich) and low cost recovery processes (e.g., heap leaching and CIP processes) used to treat previously rejected tailings and lower grade primary ores.

By 1989, investment demand had absorbed large forward sales and important signs were emerging of a downward trend in gold prices to their lowest levels in 15 years. As of January 1990, gold was priced at $410/fine troy oz. and it was originally thought that the price would level out between $350 and $300/oz. However, a sharp fall to below $300 in 1997 led to a decline approaching $260/oz. at the turn of the century. By May the price had risen to 286.40/fine troy oz., and the fundamental requirement of producers to maintain operating margins of at least 15% in real terms led to gold again breaking the $300/oz. price barrier in 2001. The price has steadily advanced up to its current levels of around US$600 to $650/fine oz. in July 2006. The risk of falling prices has been partly overcome by forward selling contracts and other methods of managing the market risk. Time intervals between sudden price rises and subsequent price modification have invariably been much shorter than time intervals between obtaining finance for a new gold venture and the sale of the metal product.

The current London gold price fixing, which commenced at Rothschild in St Swithen's Lane, London, on 12 September 1919, is performed twice each day in an office at R.N. Rothschild & Sons Ltd. International gold trading is a twenty-four hour a day operation, with the London Market overlapping those of the Far East in the morning and New York in the afternoon. Bullion traders can make deals from abroad daily from between 7:15 a.m. and 7:15 p.m., but the representatives must meet face to face at Rothschild to trade gold for physical settlement. The Gold Information Network is described by Courtesy of William Hanley (U.K. Market Eye) http://goldinfo.net/londongldfix.html.

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