SMALL-SCALE GOLD MINING IN SOUTHERN AFRICA

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Dissertation submitted in
partial fulfilment of the
requirements for the
degree of Master of Science
(Mineral Exploration) at
Rhodes University, Grahamstown

January 1992

ABSTRACT

The general characteristics of gold deposits are reviewed, and a classification of gold deposits based on mineability is proposed.

Evaluation, mining and beneficiation methods are briefly discussed. It is concluded that the most viable targets for small-scale companies comprise deposits that require the least pre-production time and expense.

Great potential exists for the small-scale reclamation of gold from tailings dumps and abandoned mines in Southern Africa. There is also potential for developing new small-scale gold mines in the Archaean greenstone terranes of the Zimbabwean and Kaapvaal cratons.

CONTENTS

INTRODUCTION

Although South Africa remains the world's leading producer of gold, recent times have shown a dramatic decrease in the earnings of the large-scale Witwatersrand gold mines. The reasons for this are multiple, intricately interrelated and fall beyond the scope of this dissertation. This phenomenon has however re-established interest in the small-scale approach to gold mining, an approach perhaps all too much neglected, especially in South Africa, since the advent of the mega-scale mines of the Witwatersrand.

Barua et. al. (1987) maintain that by definition there is no agreement on what exactly constitutes a small-scale gold mine. They quote annual production (less than 50 000 tons of ore per year), daily production (less than 200 tons per day), limited investment on infrastructure and heavy reliance on manual labour as some of the defining criteria. In the writer's opinion all of the abovementioned form an integral part of the definition, with one important additional criterion: a small-scale gold mining operation will have a very limited contingent of managerial staff.

This added criterion immediately emphasises the capabilities that would be required of such managerial staff. A sound and fundamental understanding of the product and technical aspects of production form perhaps the most important requirement. Other requirements would include experience in finance and financial control, legal aspects, labour relations and environmental control, to name but a few.

In short, the small-scale gold miner would have to be a good "all rounder", equally at home in negotiating prospecting rights and overhauling a dump truck. He would have to be highly self-motivated, with a strong entrepr eneurial spirit and "can do" attitude. A personal view is that the exploration geologist with experience of running an exploration programme is the single person in possession of most, if not all, of the abovementioned skills and requirements.

The main objective of this dissertation is to examine some of the more relevant aspects pertaining to small-scale gold mining in Southern Africa. An attempt has been made to arrange the sections in this paper in the most logical sequence: Firstly, the small-scale miner should have a sound understanding of the geological environment of gold mineralization, hence the first section on gold mineralization and the geology of gold deposits, which concludes with a classification of gold deposits applicable to the small-scale operation. Secondly, the small-scale operator should be able to accurately assess the viability, geologically and economically, of the prospect on hand, and therefore the second section on evaluation summarizes some of the methods used in such an assessment. Thirdly, the small-scale miner should know how to extract the product, therefore the third section provides a summary of the mining and metallurgical methods applicable to small-scale mining. Section four provides a brief review of gold deposits in Southern Africa and discusses the viability of small-scale gold mining in the region.

The space and time available does not permit exhaustive coverage of each subject, and where necessary the reader will be referred to other works for more detail.

SECTION A

GOLD MINERALIZATION AND THE GEOLOGY OF GOLD DEPOSITS

Crocket (1991) provides a comprehensive discussion on the abundance of gold in rock-forming minerals and in the different rock types. Tables A-1, A-2 and A-3 summarise the abundance of gold in igneous, sedimentary rocks and metamorphic rocks respectively. From these data it follows that concentration factors of two to three orders of magnitude are needed to form economically viable ore deposits under present levels of technology.

TABLE A-1 AVERAGE GOLD CONTENT OF IGNEOUS ROCK GROUPS (PPB) (CROCKET 1991)

Rock group	Comment	$\bar{X}$	*R*	*N*	Ir/Au
Rocks from non-orogenic environments					
MORB	MAR, 98; EPR, 112	1.2	0.04–15	210	0.03
Intraplate basalt	Mainly Hawaii	2.0	0.2–6.6	69	0.38
Flood basalt	3 Provinces	3.5	0.5–11	49	0.02
Initial magma; layered gabbroic complex	Anakita, Bushveld, Jimberlana, Skaergaard	4.6	2.8–8.0	4	0.08[1]
Kimberlites and mantle xenoliths					
Kimberlites	Siberia; 50 pipes	3.1	0.8–9.1	55	
Garnet peridotite	Siberia	2.7	0.6–8.1	47	
Garnet peridotite[2]	Lesotho	0.85	0.08–2.7	10	7.3
Eclogite	Siberia	3.4	0.8–9.1	25	
Spinel lherzolites	Alkali basalt host	0.5	0.1–1.1	27	7.0
Alkaline plutons		2.8	0.64–4.5	291/21	
Rocks from orogenic environment					
Ophiolite harzburgite		2.8	0.3–6.4	138/14	3.2[3]
Mafic volcanics, convergent margins		2.2	0.5–5.6	315/10	
Felsic volcanics, convergent margins		1.55[4]	0.56–4.2	305/7	
Granitic plutons		2.6	0.5–6.9	969/66	
Igneous rocks of Precambrian greenstone belts					
Perioditic komatiite		4.2[5]	0.49–13.5	156/10	1.5
Komatiitic basalt		12.4	1.0–36	44/6	0.03[6]
Tholeiitic basalt		5.7	1.3–37	323/13	
Granitic plutons		1.5	1.1–2.3	232/45	

TABLE A-2 AVERAGE GOLD IN SEDIMENTS AND SEDIMENTARY ROCKS (PPB) (CROCKET 1991)

Group	$\bar{X}$	*N* samples	$\bar{X}$	*N* areas
Sediments				
Clay-bearing deep-sea sediments	3.0	50	3.6	7
Globigerina and siliceous oozes	1.5	33	1.4	3
Terrigenous sediments, <500m water depth	3.2	75	3.5	6
Sedimentary rocks				
Conglomerate, sandstone, siltstone	8.1	1212	6.5	13
Shale	2.3	288	2.4	9
Carbonaceous shale	6.7	553	7.4	9
Carbonate rocks, associated evaporites	1.9	251		
Precambrian iron formation				
Archean, Canadian Shield			38	21
Proterozoic, Canadian Shield			19	7

TABLE A-3 AVERAGE GOLD CONTENT OF REGIONALLY METAMORPHOSED PELITES AND GRANULITES (PPB) (CROCKET 1991)

Facies	Average gold weighted by: Number of samples ( )	Number of localities ( )[1]	Refs[2]
Greenschist metapelites	7.1 (284)	6.3 (6) / 1.8–9.6	1, 4, 6, 8, 13, 15
Amphibolites	5.9 (276)	6.4 (9) / 1.9–10.4	1, 6, 8, 13
Granulites	2.2 (531)	3.2 (6) / 1.5–7.4	2, 6, 8, 11, 14

Gold is most commonly found in the free state (native gold) or as one of the tellurium compound minerals, or tellurides (Boyle, 1979). Gold is often alloyed with silver, but can also be alloyed with copper, platinum, palladium, rhodium and iridium (Bache, 1987). Gold is not lithophile, and is more siderophile than chalcophile in character (Bache, 1987).

Boyle (1979) indicates several distinct metallogenetic epochs of gold mineralization throughout the Earth's history. The greatest epoch of gold mineralization took place in the Precambrian (Woodall, 1979). Figure A-1 summarizes gold mineralization in geological time.

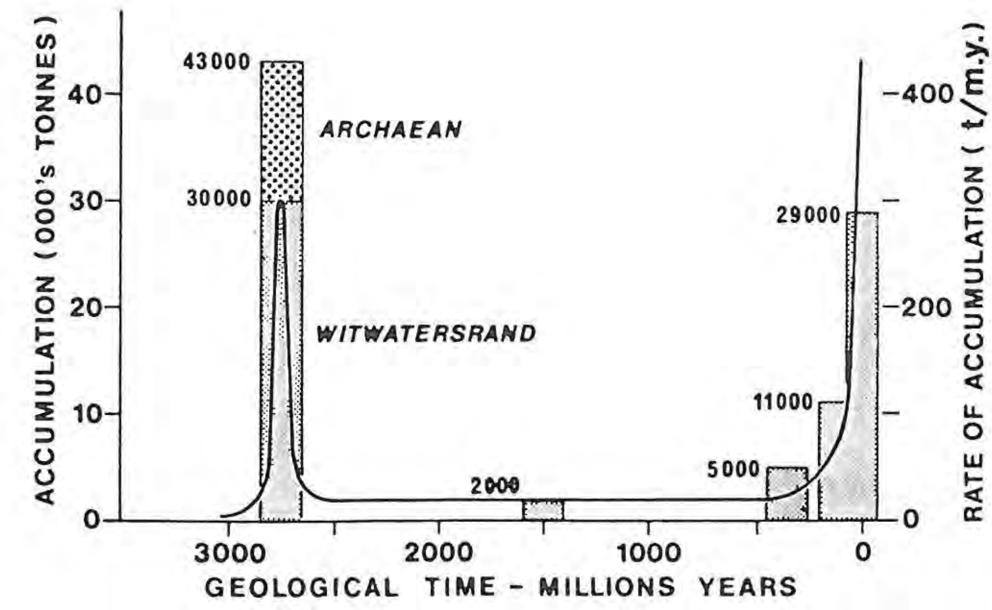


FIG A-1 GOLD MINERALIZATION THROUGH GEOLOGICAL TIME (WOODALL 1979)

GOLD DEPOSIT TYPES

For the purposes of this discussion gold deposit types have been divided into five broad categories:

. Primary magmatic deposits.
. Hydrothermal deposits.
. Supergene deposits.
. Placer type deposits.
. Miscellaneous sources.

1.PRIMARY MAGMATIC GOLD DEPOSITS

Primary magmatic gold deposits are considered those in which gold mineralization occurred syngenetically within igineous intrusive bodies. Until recently gold was produced only as a by-product from some Ni-Cu deposits associated with mafic intrusive rocks. Boyle (1979) reports an average gold content for pyrrhotite, chalcopyrite, pentlandite and magnetite from the Sudbury Complex of 0,0142, 0,0198, 0,1566 and 0,0056 ppm. respectively, and an average of 0,08 ppm Au in general from the Sudbury ore. Vakhrushev et al. (1972; in Boyle 1979) found great differences in the gold contents of the various sulphides in the Noril'sk ores, and reported up to 9 ppm from pyrrhotite, chalcopyrite and pentlandite in these ores.

In 1988 gold mineralization in sufficient grade and tonnage to constitute ore was reported from the Skaergaard layered mafic intrustion of south eastern Greenland. (Nielson, 1989). The obvious similarities between the Skaergaard and other layered mafic intrusions such as the Bushveld Igneous Complex of South Africa make this a very significant find indeed. In the Skaergaard intrustion gold mineralization occurs in a sequence of layers near the base of the Triple Group horizon of the Middle Zone (Bird et. al., 1991), and up to 20 ppm Au has been reported locally from the uppermost of these layers (Bird, op. cit.). The gold generally occurs with Cu -Fe sulphides and as inclusions in the rims of cumulus plagioclase and pyroxene.

Wager et. al. (1952) concluded that sulphur was concentrated by the exclusion of S from the crystallizing cumulus phases, and this led to the formation of immiscible droplets of Cu-rich sulphide liquid. Turner (1986, in Bird et al. 1991) identified the onset of suphide immiscibility near the top of the Middle Zone of the intrusion. Bird et al. (1991) indicate the possible importance of the abundant gabbroic and anorthositic blocks, collapsed from the solidified roof of the magma chamber and now residing in the lower portion of the Middle Zone, in creating the required physico- chemical conditions for the formation of immiscible liquids. Sinking and entrapment of these heavy immiscible liquids occurred within leuco -, melano - and mesocratic layers producing the distinct, but locally discontinuous horizons of precious metals. Concentration and entrapment may have been more efficient in the thicker and slower cooling portions, i.e. towards the central parts of the intrusion (Bird et al. 1991). (See figure A-2).

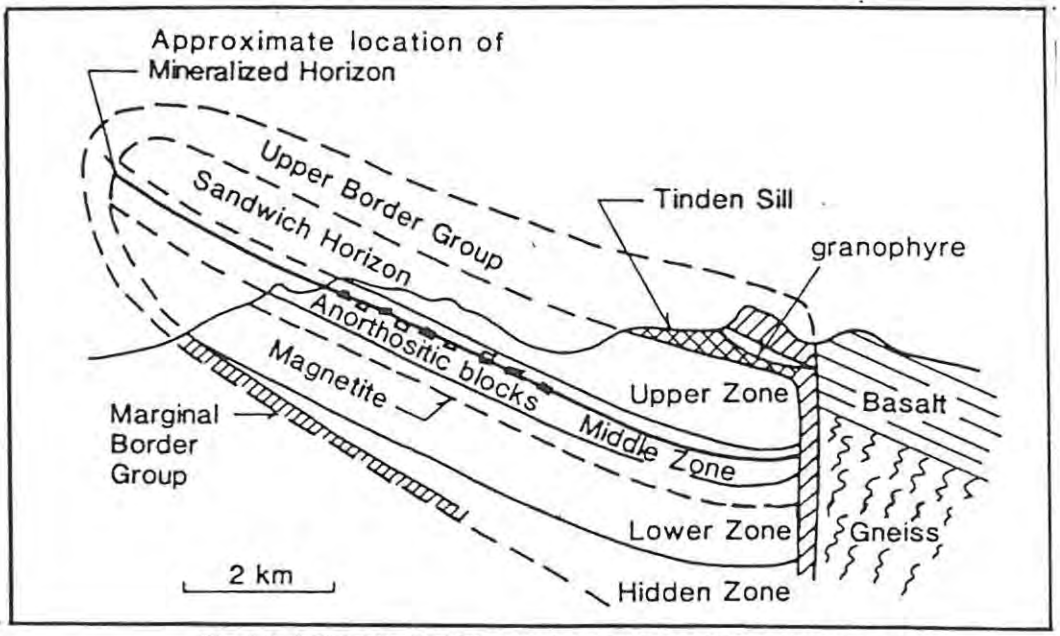


FIG A-2 SCHEMATIC CROSS SECTION THROUGH THE SKAERGAARD INTRUSION SHOWING THE APPROXIMATE LOCATION OF GOLD MINERALIZATION (NIELSEN 1989)

Nielsen (1989) quotes the mineralized layer to be 2 to 5m. thick, with a strike extension of over 8 km. Mineralization increases in grade and thickness towards the centre of the intrusion, and grades of up to 5g/t Au and 3g/t Pd have been recorded. The orebody morphology can thus be described as tabular and stratabound, with two dimensions much larger than the third, i.e. essentially a two dimensional orebody, horizontal or gently dipping.

Gold is also produced as by-product of copper mining in alkaline complexes eg. the Phalaborwa Complex. Boyle (1979) indicates an average of 0,0005 ppm Au for various rocks in carbonatite complexes. Woolley et al. (1989) however, report values as high as 12 ppm Au for ferrocarbonatite, although admittedly this is from two analyses only.

2.HYDROTHERMAL GOLD DEPOSITS

For the purposes of this discussion these deposits include those which show clear evidence of remobilization by hydrothermal mechanisms, and epigenetic deposition with respect to their host rocks.

Although this group includes a wide and varied range of deposits, thay all have one aspect in common, namely the gold was leached and transported from an original site of deposition, and reprecipitated in its current locality. It is thus evident that some understanding of the solubility and transport of gold would be essential in understanding these deposit types.

Figure A-3 summarizes the cycle of gold inter-conversions under natural conditions. Gold is usually found in one of three oxidation states: 0 (native); +1 (aurous); +3 (auric) (Boyle 1979). Zerovalent gold is generally chemically immobile. Seward (1979) has demonstrated the activities of uncomplexed free Au^+ and Au^{3+} to be very low, and concluded that very little significant transport of gold as uncomplexed free ions is possible, even at elevated temperatures. Hydrothermal fluids contain ubiquitous chloride, reduced sulphur and ammonia, all of which may form stable complexes with aurous gold (Seward 1979). Seward (op. cit.) does not consider auric gold to play any meaningful role in hydrothermal gold transport. He further indicates that hydrosulphide and chloro-complexes of Au^+ play the

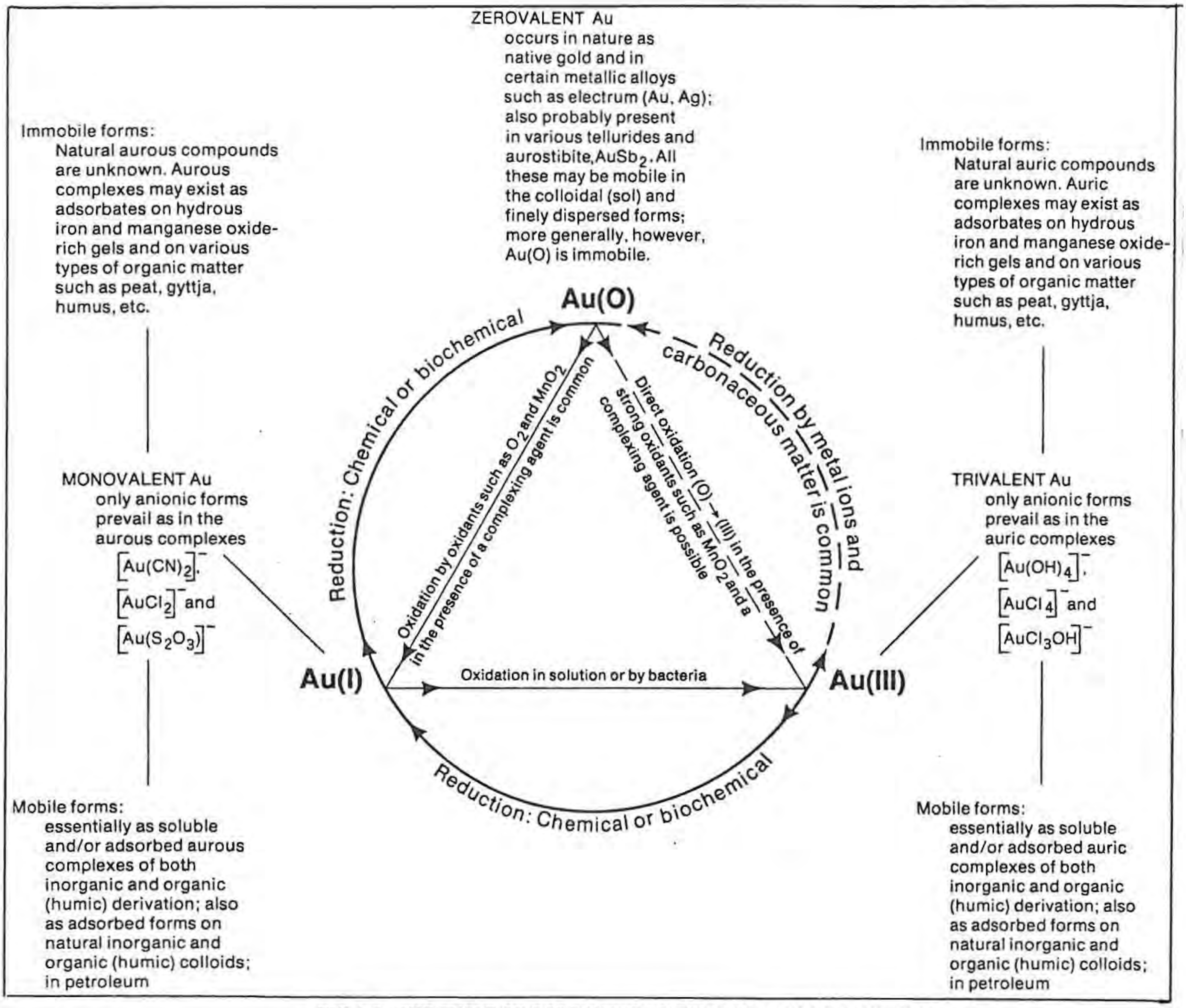


FIG A-3 THE CYCLE OF GOLD INTERCONVERSIONS UNDER NATURAL CONDITIONS (BOYLE 1979)

most important part in the transportation of gold, and of these two the hydrosulphide or so called thio-complexes offer the most plausible mechanism of transport (Seward 1979). Depending on the composition, temperature and pressure of the ore-forming fluids, chloro-complexes may contribute to, or in some cases comprise the total gold in solution (Seward 1991). Gold in solution is precipitated by changes in termperature, pH, oxidation potential or activity of reduced sulphur. For example, a decrease in the activity of reduced sulphur may be accomplished by oxidation of sulphide to sulphate, precipitation of sulphide minerals, or by boiling of the hydrothermal fluid (Seward 1979).

In conclusion thus: gold is leached and transported as thio and chlorocomplexes in hydrothermal fluids, and precipitated when these fluids encounter any physico-chemical changes.

A further feature common to these deposits is the presence of alteration assemblages in the wall rocks surrounding the area of mineralization. Alteration in the wallrocks results from the tendency of the hot, circulating hydrothermal fluids to equilibrate with the rocks flanking the conduits, causing both fluids and rocks to adjust (Guilbert and Park 1986). Alteration patterns are always progressively, predictably and repetitively arranged around areas of mineralization, a feature referred to as alteration zonation. The recognition and interpretation of alteration zonation is of obvious importance to the explorationist and miner.

Guilbert and Park, (1986, pp177-180) provide a detailed discussion of the chemical reactions involved in wallrock alteration. Alteration assemblages may be summarised as follows:

<u>Potassic</u>: Introduced or recrystallized K-feldspar with or without biotite and sericite, commonly with traces of accessory anhydrite, apatite, fluorite, calcite or siderite and sheelite. Sulphides include chalcopyrite, molybdenite, pyrite, magnetite or hematite. Replacement of hornblende or chlorite by biotite, and plagioclase by K-feldspar are also considered potassic.

<u>Propylitic</u>: Involves epidote, chlorite and calcite replacing plagioclase and hornblende - biotite.

<u>Phyllic or sericitic</u>: The presence of sericite dominates. All primary rock-forming silicates such as feldspars, micas and mafic minerals are converted to sericite and quartz. Accessories include minor to major pyrite and chlorite if magnesium is present.

<u>Argillic</u>: Clay minerals dominate, with kaolinite replacing plagioclase, montmorillonite after amphiboles and plagioclase, and amorphous allophane after both.

Greisen: Quartz, muscovite and topaz dominate with accessory tourmaline, fluorite, rutile, cassiterite, wolframite and magnetite.

Skarn An association of calcium bearing, usually iron-rich silicates, including amphibole, pyroxene, garnet, epidote-zoisite and pyroxenoids that have replaced limestone or dolomite with the introduction of large amounts of silicon, aluminium, iron and magnesium.

In order to discuss the myriad deposits which fall under this sub-heading, the following further subdivision is proposed:

- Primary epigenetic deposits.
- Epigenetic deposits.
  Metamorphic types.
  Epithermal types.
- Syngenetic deposits.

2.1PRIMARY EPIGENETIC DEPOSITS

These deposits occur in igenous intrusive bodies. The source of the gold is considered to be the intrusive bodies themselves (i.e. primary magmatic), however mineralization occured epigenetically with the remobilization and concentration of the gold and associated ore minerals by hydrothermal fluids.

2.1.1GOLD-RICH PORPHYRY COPPER DEPOSITS.

The term "porphyry copper" applies to large, relatively low grade, epigenetic, intrusion-related, open-pit mineable copper deposits. Generally these deposits produce copper, copper and molybdenum, or copper and gold. These deposits are spatially and genetically related to igneous intrusions which are generally felsic, but can range widely in composition. The intrusions are mesozonal, invariably contain porphyritic members and multiple intrusions, dyke swarms and intrusive breccias are characteristic. Mineralization and alteration is widespread and exhibits lateral zoning (Macmillan et al. 1980). Figure A-4 presents a schematic cross sectional view of an idealized porphyry copper deposit.

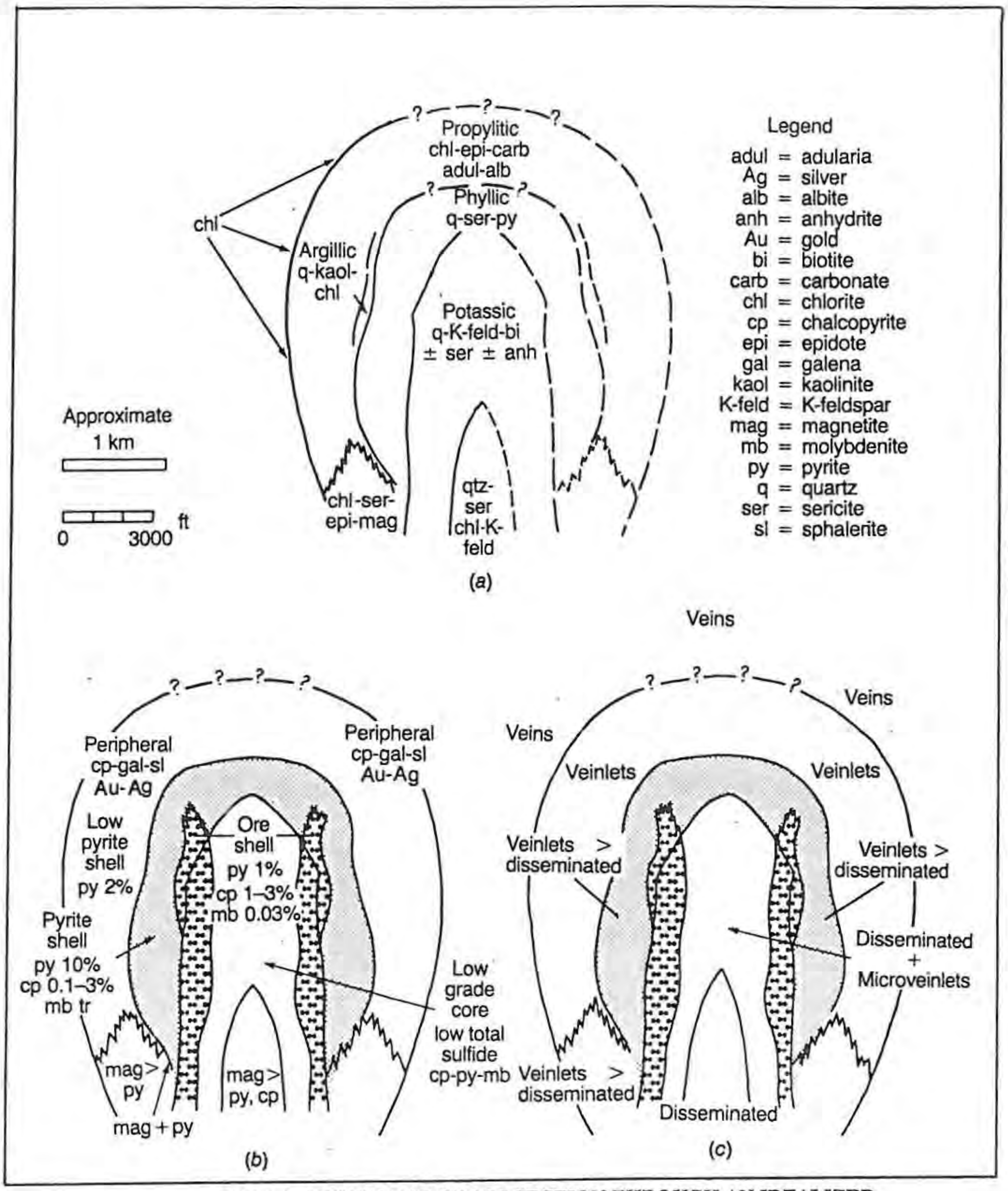


FIG A-4 SCHEMATIC CROSS SECTION THROUGH AN IDEALIZED PORPHYRY COPPER DEPOSIT SHOWING A) ALTERATION ZONES B) MINERALIZATION ZONES AND C) OCCURRENCE OF SULPHIDES (GUILBERT ET AL. 1986)

Hydrothermal fluids are derived from both magmatic and groundwater sources (Macmillan, 1980) Hydrothermal alteration commonly includes propylitic, argillic, phyllic and potassic assemblages laterally zoned around a core of potassic alteration and grading outwards through phyllic, argillic and propylitic zones into unaltered country rock. (See figure A-4). Mineralization in these deposits consists of disseminations, fracture fillings and quartz veinlets containing varying amounts of pyrite, chalcopyrite, bornite, molybdenite and trace gold (Macmillan 1980). Classic porphyry copper deposits exhibit a well-developed lateral metal zonation as indicated in figure A-4.

Detailed descriptions of gold-rich porphyry - type deposits are given in Sillitoe (1979; 1987; 1991) Flemming et al. (1986); Kesler (1973); Titley (1978); Bamford (1972); Ford (1978); Fountain (1972); and Howell et al. (1978).

Sillitoe (1979) summarises the features which are characteristic of gold-rich porphyry-type deposits as follows:

- The high gold values are mainly associated with potass ic - silicate alteration i.e. biotite and K-feldspar, and where present the sericitic alteration carries substantially lower gold values,
- The gold-rich deposits tend to be poor in molybdenum.

  Kesler (1973) divides porphyry copper deposits into two categories, depending on whether they carry by-product gold or by-product molybdenum,
- Gold is present in the native state, closely associated with chalcopyrite bornite, and in quantities proportional to the copper grade,
- Many of the gold-rich porphyries contain an abundance of magnetite, commonly accompanied by replacement quartz in the gold-bearing potassic zones,
- Abundant replacement, or hydrothermal quartz is present in all the deposits.

Table A-4 provides a summary of the characteristics of gold-rich porphyry-type deposits.

TABLE A-4 CHARACTERISTICS OF SELECTED GOLD AND COPPER-GOLD PORPHYRY DEPOSITS (SILLITOE 1991)

Deposit	Status	Contained Au, tonnes	Age, Ma	Progenitor intrusion	Associated mineralization	Host rocks	Metal association	Key hypogene alteration products
Lepanto, Philippines	Feasibility stage (underground flotation)	440	3.5	Quartz porphyry stock	Acid-sulphate type enargite-Au deposit	Late Tertiary andesitic volcanics,	Cu–Au–(Mo)	Biotite, anhydrite, magnetie
Santo Tomas II, Philippines	Mine: block caving, flotation	200	1.5	Diorite and quartz diorite porphyry stock	None	Mainly Late Cretaceous-Palaeogene andesitic volcanics	Cu–Au	Biotite, actinolite, anhydrite, magnetite
Dizon, Philippines	Mine: open pit, flotation	130	2.7	Quartz diorite porphyry	None	Late Tertiary andesitic volcanics	Cu–Au–(Mo)	Sericite, clay, chlorite martitized magnetite
Ok Tedi, Papua New Guinea	Mine: open pit, flotation (formerly CIP)	368	1.2	Monzonite porphyry stock	Cu–Au skarn deposits	Late Cretaceous siltstone, Eocene-mid-Miocene limestone	Cu–Mo–Au	Biotite, K-feldspar
Cuervo (Sullivan), Nevada, USA	Feasibility stage	12.4	Jurassic (?)	Microdiorite sill	None	Triassic metavolcanics	Cu–Au	Clay
Marte, Chile	Mine: open pit, heap leach	53.5	13.3	Quartz diorite porphyry stock	None	Miocene andesitic volcanics	Au–(Cu)	Sericite, clay, chlorite, gypsum (after anhydrite), hematite (partly after magnetite)
Young-Davidson, Ontario, Canada	Abandoned mines, under exploration	19	*ca.*2700	Syenite stock	Au veins, porphyry Cu–Mo–Au protore	Archaean metavolcanics and metasediments	Au–(Cu–W)	K-feldspar, hematite, tourmaline
Boddington, Western Australia	Mine: open pit, heap leach	120	2650-2670	Quartz microdiorite bodies	None	Archaean andesitic lava	Au–Cu–Mo–W	Biotite, K-feldspar, actinolite, epidote

In general the genetic model for porphyry type deposits involves the concentration of magmatic fluids and ore elements during crystallization of the magma.

Sillitoe (1991) indicates that K-feldspar-silicate alteration and variable quantities of Au, Cu and/or Mo are the hallmarks of magmatic - hydrothermal brine activity. The release of the brines is achieved through retrograde boiling, which is capable of dissipating sufficient mechanical energy to induce rock fragmentation, resulting in the development of porphyry type stockworks, breccia pipes and of tons) low grade (generally less than 1ppm Au) orebodies. Mineralization occurs in veins and stockworks, and as dissiminations in an orebody which is essentially three dimensional in morphology.

2.1.2GOLD IN FELSIC INTRUSIONS

Essentially this subclass forms part of subclass 2.1.1. There are however important differences, especially in the morphology of the orebodies, that warrant a separate classification.

According to Mann (1984) approximately 11% of the total gold mines of Zimbabwe are located in Archaean granitic rocks. In the Abitibi greenstone belt of Canada more than 25% of host rocks of gold mineralization comprise post-volcanic intra-belt felsic plutons (Colvine et al., 1988). Numerous similar felsic hosted deposits are reported from South Africa and Swaziland.

As in the porphyry - type deposits these deposits are generally hosted in felsic intrusions of which the composition varies between granodiorite and quartz monzonite (Colvine et al., 1988) The geometries of the ore zones appear more varied than those in porphyry - type deposits (Sillitoe, 1991).. Generally these deposits lack the multidirectional stockworks of porphyry-type deposits, and mineralization appears in veins and stockworks confined to relatively narrow zones, possibly reflecting structural control. Alteration is in the form of mainly sericitization and carbonation confined to areas immediately adjacent to the veins and stockworks and is generally not as extensive as in the porphyry type deposits. Mann (1984) notes the importance of introduced mafic minerals in the form of semidigested xenoliths or greenstone contacts to ore formation.

The morphology of these deposits can be summarised as essentially two dimensional, discontinuous, steeply dipping to vertical.

hydrothermal breccia with associated mineralization. It is evident that the circulating magmatic brines would at some stage mix with the groundwater (meteoric, connate or seawater) and this is thought to give rise to argillic alteration (Sillitoe 1991). Sillitoe (op.cit.) further indicates that the early deposited porhyry-type ore could undergo partial remobilization and reconcentration by the circulating meteroic fluids.

Figure A-5 summarizes the interrelationships between porphyry-type gold mineralization and other types of gold deposits, as suggested by Sillitoe (1991). In this paper these other types of deposits are discussed in sections 2.2.1 and 2.2.2.

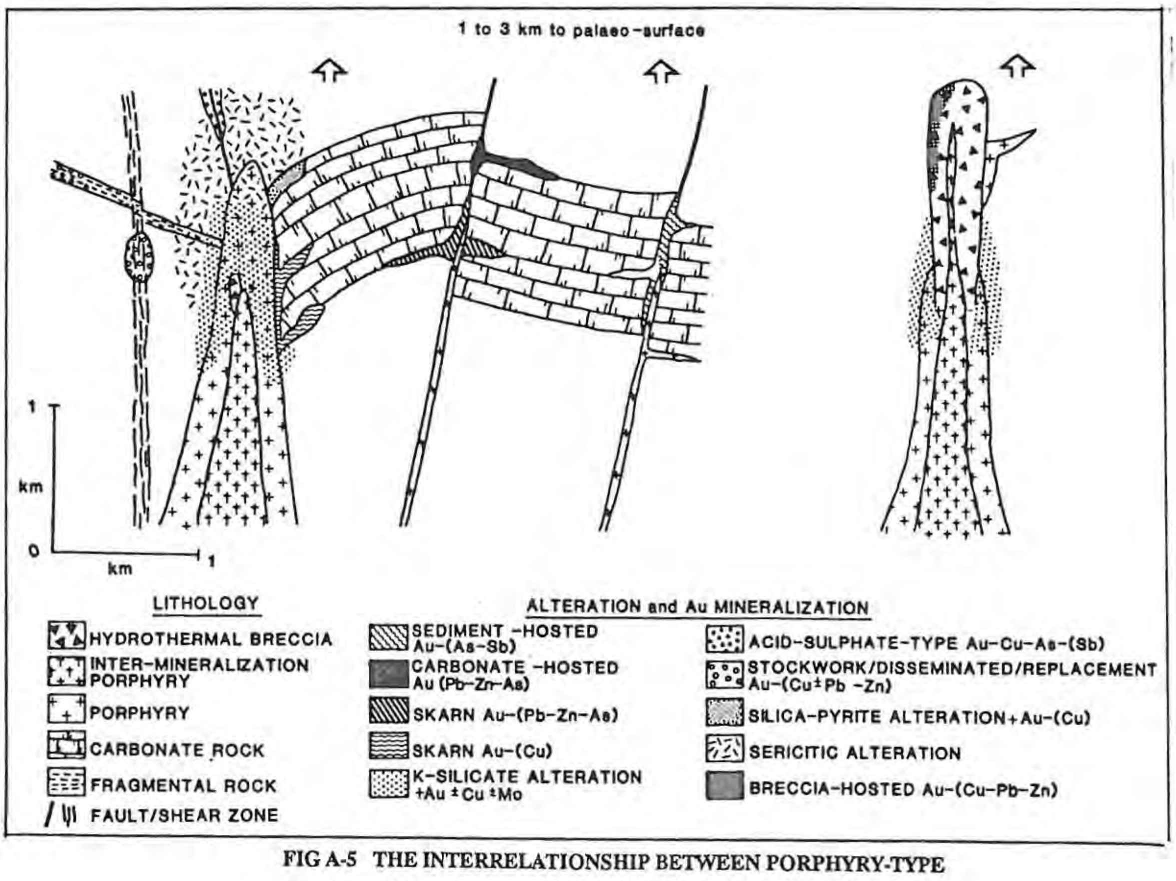


FIG A-5 THE INTERRELATIONSHIP BETWEEN PORPHYRY-TYPE DEPOSITS AND OTHER TYPES OF GOLD DEPOSITS (SILLITOE 1991)

In conclusion, these deposits represent large (generally in excess of 10's of millions of tons, but more often in excess of 100's of millions

2.2EPIGENETIC GOLD DEPOSITS

This subclass comprises those deposits formed epigenetically, and hosted in a variety of rock types. These deposits may be (and often are) associated with igneous intrusions. The exact relationship between the deposits and the intrusions however remains contentious.

2.2.1METAMORPHIC DEPOSIT TYPES

The deposits in this subclass are considered to have formed through metamorphic processes.

REGIONAL METAMORPHIC DEPOSITS

The hydrothermal fluids responsible for these deposits are thought to have been derived from regional metamorphic devolatilization reactions, although this may still be a point of contention. These deposits include Archaean lode gold and Phanerozoic "turbidite-hosted" gold deposits. Nesbitt (1991) comments on the close similarity between Archaean and Phanerozoic mesothermal deposits, and suggests a general category of mesothermal lode gold deposits encompassing mesothermal type ores ranging from the Archaean to Tertiary in age.

The deposit types are well documented in the literature. The reader is referred to Groves et al. (1991), Nesbitt (1991), Roberts (1987), Hutchinson (1987) and Colvine et al. (1988) for review type papers on the subject.

Archaean lode gold deposits form the most important group in this category. They form one of the most characteristic features of Archaean greenstone belts, with major deposits situated in most major cratonic areas (Groves et al., 1991).

Mineralization is generally in the form of veins (open-space filling) and disseminated in altered wall rocks (Roberts, 1982). Ore mineralogy is generally simple with free gold and/or gold sited in pyrite, pyrrhotite, arsenopyrite, with a heterogeneous distribution of accessory scheelite, tellurides, stibnite, galena, sphalerite, chalcopyrite, magnetite, hematite and anhydrite (Groves et al., 1991). Multiple phases of mineralization are common.

Wallrock alteration normally involves the introduction of CO_2, K_2O, S and H_2O with either the introduction or redistribution of SiO_2. Alteration assemblages are dependent upon host rock lithology and metamorphic grade (Colvine et al., 1988). Table A-5 provides a summary of the alteration assemblages associated with different host rocks in greenschist-facies domains (see also table A-6).

Groves et al. (1991) maintain that structure is the single most important factor controlling the distribution of these gold deposits and the morphology of the orebodies. Major gold deposits are commonly sited adjacent to transcratonic shear-zones, and more specifically on related shorter strike length, smaller scale structures (Groves et al. 1991). Different structural styles of mineralization result from variations in the orientation of the regional stress field and the strength of the host rocks, or contrasts in strength between adjacent host rocks. Thus, mineralization may occur in cross-cutting or layer-parallel shear zones, strike-extensive laminated quartz veins, extensional quartz-vein arrays and/or breccias. Table A-6 summarizes the characteristics of some of the larger Archaen lode-gold deposits.

Foster et al. (1986) suggest the following classification for Archaean gold deposits in Zimbabwe:

- Stratabound deposits
  - In banded iron formation
  - In banded sulphides
  - Clastic - hosted

TABLE A-5 TYPICAL WALLROCK ALTERATION IN AND ADJACENT TO LODE GOLD DEPOSITS IN GREENSCHIST FACIES DOMAINS (GROVES ET AL. 1991)

Host rock	Main ore zone	Proximal alteration	Distal alteration
Basalt and dolerite	Quartz–carbonate veining; silicification; ankerite + biotite or sericite, some albite; rare green mica; abundant Fe-sulphides ± arsenopyrite; stibnite locally abundant	Ankerite-dolomite + biotite or sericite ± high-Mg chlorite; some Fe-sulphides	Calcite + chlorite + amphibole + albite; minor Fe-sulphides
Komatiites and ultramafic sills	Quartz–carbonate veining; silicification; ankerite-magnesites + phlogopite or green (V or Cr) mica; abundant Fe-sulphides and arsenopyrite; stibnite locally abundant	Dolomite-ankerite-magnesite + talc ± phlogopite; minor Fe-sulphides	Chlorite + amphibole + talc-dolomite- calcite; serpentine ± chlorite ± talc
Banded iron formation	Quartz–carbonate veining; silicification; sulphidation of magnetite or siderite to Fe-sulphides ± arsenopyrite	Partial sulphidation of magnetite or siderite to Fe-sulphides; some chlorite	No obvious alteration
Granitoids and felsic porphyries	Quartz veining; intense silicification; biotite or sericite + calcite; K-feldspar or albite; Fe-sulphides ± arsenopyrite; chalcopyrite, sphalerite, galena, molybdenite and tourmaline locally abundant	Sericite ± K-feldspar ± albite; calcite + titanite ± tourmaline; minor Fe-sulphides	Sericite + calcite; commonly limited alteration

TABLE A-6 CHARACTERISTICS OF LARGER ARCHAEAN LODE GOLD DEPOSITS (GROVES ET AL. 1991)

Region/Deposit	Gold production	Host rock	Lode type	Alteration in ore zone	Ore minerals
Yilgarn, Australia					
Golden Mile, Kalgoorlie	~1200 t	Tholeiitic dolerite sill	Steeply dipping brittle-ductile shear zones and brittle fracture sets; some breccias	Muscovite + ankerite + pyrite; some silicification and quartz veining	Gold + pyrite + minor scheelite, arsenopyrite and anhydrite; late tellurides
Sons of Gwalia, Leonora	>90 t	Tholeiitic to high Mg basalts	Steeply dipping major shear zone with local boudinaged quartz veins	Muscovite + biotite + ankerite + pyrite; quartz veins	Gold + pyrite + arsenopyrite ± pyrrhotite; minor chalcopyrite
Mararoa-Crown, Norseman	>70 t	Tholeiitic basalts and ultramafic intrusions (?)	Folded laminated quartz veins	Biotite + amphibole ± chlorite ± ankerite – dolomite	Gold + pyrite ± galena ± tellurides ± scheelite
Mt Charlotte, Kalgoorlie	>70 t	Granophyric unit in tholeiitic dolerite sill	Quartz vein set in hydraulic fractures	Sericite + ankerite + pyrite ± albite; some silicification	Gold ± pyrite ± scheelite
Abitibi, Canada					
Hollinger, Timmins	>600 t	Mafic flows; minor felsic flows and pyroclastics; quartz-feldspar porphyries	Quartz veins and stockworks; steeply dipping shear zones	Sericite + ankerite ± chlorite ± calcite; quartz and albite	Gold + pyrite + pyrrhotite ± galena ± sphalerite
Kerr Addison, Larder Lake	>320 t	High-Mg basalts, tholeiitic basalts; felsic porphyry to syenite dykes; clastic sediments	Stockworks ladder veins	Ankerite/dolerite–albite–muscovite	Gold + pyrite + scheelite + arsenopyrite
Sigma, Val d' Or	>110 t	Intrusive dolerite plug with andesite flows; minor feldspar porphyry dykes	Subvertical veins and breccias in brittle-ductile shear zones; flat veins	Calcite + white mica	Gold + pyrite + pyrrhotite; minor chalcopyrite, sphalerite, galena, tellurides, scheelite
Zimbabwe					
Cam and Motor, Kadoma	>145 t	Tholeiitic basalt/andesite; high-Mg basalts; dolerite intrusions; minor clastic sediments	Steeply dipping veins, shear-zone vein arrays, and stockworks	Quartz + ankerite; serpentine in high-Mg rocks	Gold + pyrite + arsenopyrite + stibnite ± sphalerite ± scheelite
Phoenix, Kwekwe	>105 t	Dunite-peridotite intrusive complex	Quartz veins; minor stockworks and silicification	Magnesite ± fuchsite ± talc	Gold + pyrite + stibnite + arsenopyrite
Dalny, Kadoma	>50 t	Tholeiitic basalt flows	Steeply dipping brittle-ductile shear zone	Ankerite + white mica + chlorite + albite	Gold + pyrite + arsenopyrite ± chalcopyrite ± galena ± tetrahedrite ± scheelite ± sphalerite

- Non-stratabound
    - Veins
    - Mineralized shear zones

It is tempting to classify the gold mineralization in banded iron formation (BIF) and banded sulphides as syngenetic deposits. However Groves et al. (1987) have shown a clear epigenetic source for gold mineralization in BIF of the Yilgarn Black, Western Australia. The banded sulphide deposits have been compared to Kuroko-type volcanogenic massive sulphide deposits (Foster et al. 1986).

The most common host rock to Phanerozoic mesothermal lode gold deposits is a greywacke shale assemblage formed in continental margin, turbidite settings (Boyle 1986). Other hosts include volcano-sedimentary sequences similar to Archaean greenstone lithologies, granitic to intermediate intrusions and intermediate- to high-grade gneisses (Nesbitt 1991). Nesbitt (op. cit.) indicates a close relationship between these deposit types and metamorphic grade, with mineralization generally present in rocks of sub-to upper-greenschist facies. In regions where higher metamorphic grades are present, mineralization is restricted to greenschist-facies units (Nesbitt 1991).

The dominant style of mineralization is in the form of thick, vertically extensive quartz veins with discontinuous mineralization producing a typical ore shoot structure consisting of alternating ore grade and sub-ore grade vein material (Nesbitt 1991). The quartz veins are typically milky with a laminated or banded texture. There is often evidence for several generations of quartz veining, with the gold introduced during the late veining events. Stockwork and disseminated mineralization constitute important mineralization styles in some districts (Nesbitt 1991).

Wallrock alteration of Phanerozoic mesothermal lode-gold deposits varies according to the host lithology: Arenacious wackes generally display minimal alteration with minor disseminated sulphides, sericite and carbonate. Alteration in more feldspathic to pelitic units may be more pronounced with extensive albitization. Intermediate to mafic volcanic rocks and ultramafic rocks display intensive and pervasive alteration, with carbonates, chlorite, pyrite, sericite, graphite and talc being the most prominent products. Fuchsite is common to ultramafic units. Felsic plutonic rocks display minor to pervasive albitization, silicification, chloritization and carbonation (Nesbitt 1991).

Table A-7 provides a summary of the general characteristics of Phanerozoic mesothermal types of deposit.

TABLE A-7 GENERAL GEOLOGICAL AND GEOCHEMICAL CHARACTERISTICS OF PHANEROZOIC MESOTHERMAL DEPOSITS (NESBITT 1991)

Tectonics	Typically in accreted, deformed and metamorphosed continental margin or island arc terranes.
Size and grade	Generally several hundred thousand to a few million tonnes, typically 5–25 g/t
Host lithology	Widely variable; greywackes-pelites, chemical sedimentary units, volcanics, plutons, ultramafics
Metamorphism	Typically sub- to upper-greenschist; occasionally host terranes were metamorphosed to higher grade prior to mineralization
Relations to plutons	Variable; some areas close spatial and probable genetic link; other districts no evidence of plutonic activity
Structure	Varies from fold to fault control; where fault controlled, mineralization is generally confined to second-order faults related to major structures
Timing	Late in orogenic sequence; subsequent to principal deformation and metamorphism
Ore morphology and textures	Thick quartz veins, typically banded, occasionally vuggy with high-grade ore shoots; vertically continuous mineralized zones; occasional stockwork and disseminated mineralization
Mineralogy and paragenesis	Early phases: quartz, Ca–Mg–Fe carbonates, arsenopyrite, pyrite, albite, sericite, chlorite, scheelite, stibnite, pyrrhotite, tetrahedrite, chalcopyrite, tourmaline. Late phases: gold, galena, sphalerite, tellurides
Hydrothermal alteration	Carbonatization, albitization, sericitization, silicification, sulphidation, chloritization; listwanite development
Zoning and elemental geochemistry	Au:Ag typically >1; associated elements: Ag, Sb, As, W, Hg, Bi, Mo, Pb, Zn, Cu, Ba. Zoned from high-temperature Au ± Ag, As, Mo, W to Sb ± Au, Hg, W to Hg ± Sb
Fluid inclusions	H_2O–CO_2 inclusions, typical XCO_2 0.05 to 0.2; <5 equiv.wt.% NaCl; *T* (homogenization) 250-350°C, *P*> 1000 bars
Stable isotopes	Typical values: $\delta^{18}O$ (Qz) = + 11 to + 18‰; $\delta^{13}C$ (Cb) = − 25 to −3‰; δD (fluid inclusions) = − 160 to − 30‰; $\delta^{34}S$ −10 to + 10‰
Radiogenic isotopes	Indicate heterogeneous crustal sources for Sr, Pb and Nd

In view of the preceding general descriptions of Archaean lode gold and Phanerozoic mesothermal lode gold deposits it is evident that, apart from age of actual mineralization and host rocks there is very little difference between these deposit types. One could therefore expect them to have been farmed by broadly similar genetic processes.

Groves et al. (1991) and Nesbitt (1991) provide a comprehensive discussion of the various models that have been proposed for the genesis of these deposits. The most favoured model currently involves hydrothermal fluids derived from devolatilization reactions during prograde metamorphism. Nesbitt (1991) suggests deep circulation of meteoric waters for the Phanerozoic deposits.· The close spatial association of many of these deposits with felsic intrusions cannot be ignored, and many workers suggest a primary magmatic source for these fluids, even to the extent that where there is no physical evidence of an intrusion on surface they postulate one at depth.

Movement of the hydrothermal fluids occurred along regional, crustal-scale structures, and gold was precipitated where the required physico-chemical conditions were encountered in structural traps and compositionally favourable host rocks.

Being predominantly structurally controlled, the morphologies of these orebodies are highly variable. The scale of mineralization may vary from a single vein-associated orebody (i.e. two dimensional) to a wider, vein-system associated ore body (i.e. a noticeable third dimension). Selective replacement mineralization in BIF's may result in stratabound orebodies. Given the complex structural histories of the host rocks, these orebodies are unvariably steeply dipping to vertical, and may show structural modification and deformation. Grade usually varies between 2 and 50ppm Au (Groves et al., 1991) and is often erratic and discontinuous.

CONTACT METAMORPHIC DEPOSITS

The most important group of deposits in this sub-class are those related to the development of skarn. Skarnification occurs in carbonate rocks around intrusive masses (Gilbert et al., 1986) and refers to the gangue material in these deposit types - a coarse grained, generally iron-rich mixture of Ca - Mg-Fe-Al silicates formed by metasomatic processes (Einaudi et al. 1982). Meinert (1989) provides a detailed discussion of gold skarn deposits.

These deposits occur within the thermal aureoles of felsic intrusions, and gold skarns may be related to either unmineralized intrusions with porphyritic or equigranular textures, or mineralized porphyry stocks (Sillitoe, 1991). Skarn development can be subdivided into an early, anhydrous prograde phase, and a late hydrous retograde phase. Sillitoe (1991) indicates that proximal skarns (where the skarn is in contact with the progenitor intrusion) display an association between gold precipitation and retrograde events, whereas with distal skarns gold precipitation is associated with prograde alteration. Most auriferous skarn is present in the exoskarn environment and is normally developed in limestone and/or calcareous siltstone as stratabound ore bodies. Gold skarn is rich in sulphides, and total sulphide content commonly exceeds 10% volume, of mainly pyrite and/or pyrrhotite, and locally also magnetite (Sillitoe 1991). Table A-8 presents a summary of the characteristics of some skarn gold deposits.

Sillitoe (1991) proposes a magmatic source for the hydrothermal fluids in these deposits types, and relates these deposits to a variety of other deposits found in association with felsic intrusions (see figure A-5).

The orebody morphology can be summarized as stratabound and either horizontal or steeply dipping depending on the degree of deformation. Grades are generally relatively low (less than 5g/t), and concentrated in "pods" or lenses.

TABLE A-8 CHARACTERISTICS OF SKARN GOLD DEPOSITS (SILLITOE 1991)

Deposit	Status	Contained Au, tonnes	Age, Ma	Ore-related intrusion	Associated mineralization	Host rocks	Metal Association	Key hypogene alteration products
Fortitude, Nevada, USA	Mine: open pit, CIP	96	37.2	Porphyritic granodiorite stock and dykes	Porphyry Cu–Mo protore, Cu–Au skarns, Pb–Zn veins	Carboniferous-Permian limestone and calcareous siltstone	Au–Ag–Cu–Zn–Pb–As–(Bi-Te)	Hedenbergite–andradite, actinolite–chlorite–calcite-prehnite–K-feldspar
McCoy, Nevada, USA	Mine: open pit, heap leach	30	39.7	Porphyritic granodiorite stock and dykes	Minor fracture-controlled Au in stock, Cove Au–Ag deposit (Table 6.4)	Triassic limestone	Au–Ag–(Cu)	Garnet, diopside, calcite, orthoclase, chlorite
Nickel Plate (Hedley), British Columbia, Canada	Mine: open pit, CIP	83.5	Early Jurassic (>200<225)	Diorite and quartz diorite porphyry sills and dykes	None	Triassic calcareous and tuffaceous siltstone	Au–Cu–As–Zn–(Co–Ni–Bi–Mo–Te)	Clinopyroxene-garnet, epidote-chlorite–clinozoisite–prehnite
Red Dome, Queensland, Australia	Mine: open pit, heap leach	39	Carboniferous	Rhyolite porphyry dykes	Porphyry Mo protore, Cu–Pb–Ag skarn deposits	Silurian limestone	Au–Cu–As–Zn–(W–Sn–Bi–Te)	Hedenbergite, andradite, wollastonite-andradite-ferrobustamite, chlorite-calcite
Thanksgiving, Philippines	Mine: underground, flotation	13	5.5	Diorite porphyry stock and dykes	Kennon porphyry Cu–Au deposit, Au veins	Miocene limestone	Au–Ag–Zn–(Pb–Cu–As–Te)	Garnet, clinozoisite, chlorite, calcite
Suan district, North Korea	Mines: underground, flotation	>100	Jurassic	Granite pluton	Au veins	Late Proterozoic-Cambrian limestone and dolomite	Cu–Au–(Zn–Pb–As–Bi)	Garnet-diopside, actinolite, phlogopite, ludwigite, chlorite, talc, tremolite, wollastonite
Navachab, Namibia	Mine development: open pit, heap leach	22	Cambro-Ordovician	Leucogranite dykes	None	Late Proterozoic dolomitic marble	Au–(Cu–Pb–Zn–W–Bi–Te–As–Mo)	Garnet-diopside, biotite-tremolite

2.2.2EPITHERMAL DEPOSIT TYPES

Henley (1991) defines epithermal deposits as the products of large-scale hydrothermal convective systems driven by magmatic heat in the upper 5-10 km. of the earth's crust. This definition is perhaps too broad for use here, as it includes other deposit types already discussed in previous sections. (Compare figure A-5). For the purpose of this discussion epithermal deposits are considered to be those which were formed by hyrothermal action in surface or near surface continental environments, implying temperatures of 50° -350° C. These deposit types are generally further divided into two broad groups ie. volcanic-hosted and sediment-hosted disseminated deposits.

The reader is referred to Berger et al. (1985) for detailed discussion on all aspects of epithermal systems, and to Henley (1991) and Berger et al. (1991) for more review-type papers.

VOLCANIC-HOSTED EPITHERMAL DEPOSITS

These deposit types all show a close temporal and spatial association to volcanism. Two types of volcanic-hosted epithermal deposits are distinguished mainly on the basis of vein and alteration assemblages, namely adularia- sericite type and the acid - sulphate type (Hayba, 1985). Henly (1991) prefers the term alunite - kaolinite to acid-sulphate. Other workers refer to "hot-spring" type and "open vein" type mineralization (Rossiter, 1984); these types may however represent subclasses of the adularia -sericite type.

The key features of these two styles of mineralization are summarized in table A-9.

In general the genetic model for these deposit types involves the circulation of predominantly meteoric hydrothermal fluids, with structures such as faults and fractures acting as primary conduits. Magmatic intrusions within the higher crustal levels act as heat sources. Mineralization occurs in response to boiling and consequent physical and chemical changes experienced by the fluids. Berger et al. (1989) relate the difference in alteration products of the two mineralization styles to direct magmatic influence: in the adularia-sericite types hydrothermal activity is driven by volcanic heat energy, and alteration and deposition occurs in reaction to boiling and extensive interaction with surficial waters. In the alunite - kaolnite types degassing of high-level intrusions results in an acid

TABLE A-9 CHARACTERISTICS OF ADULARIA-SERICITE AND ALUNITE-KAOLINITE EPITHERMAL DEPOSITS (HENLY 1991)

Characteristic	Adularia–sericite	Alunite-kaolinite
Structural setting	Structurally complex volcanic environments, commonly in calderas	Intrusive centers, 4 out of the 5 studied related to the margins of calderas
Size length : width ratio	Variable; some very large usually 3 : 1 or greater	Relatively small equidimensional
Host rocks	Silicic to intermediate and alkalic volcanics	Rhyodacite typical
Timing of ore and host	Similar ages of host and ore	Similar ages of host and ore (< 0.5 m.y.)
Mineralogy	Argentite, tetrahedrite, tennantite, native silver and gold, and base-metal sulphides. Chlorite common, selenides present, Mn gangue present, no bismuthinite	Enargite, pyrite, native gold, electrum, and base-metal sulphides. Chlorite rare, no selenides, Mn minerals rare, sometimes bismuthinite
Production data	Both gold- and silver-rich deposits variable base-metal production	Both gold- and silver-rich deposits, noteworthy Cu production
Alteration	Propylitic to argillic Supergene alunite, occasional kaolinite, abundant adularia	Advanced argillic to argillic (± sericitic) Extensive hypogene alunite, major hypogene kaolinite, no adularia
Temperature	100–300°C	200–300°C[1]
Salinity	0–13 wt.% Nacl equiv.[3]	1–24 wt.% NaCl equiv.[2]
Source of fluids	Dominantly meteoric	Dominantly meteoric, possibly significant magmatic component
Source of sulphide sulphur	Deep-seated magmatic or derived by leaching wallrocks deep in system	Deep-seated, probably magmatic
Source of lead	Precambrian or Phanerozoic rocks under volcanics	Volcanic rocks or magmatic fluids

condensate which causes the alteration, and gold mineralization occurs at a later stage involving low-salinity, higher pH fluids.

Figure A-6 presents a schematic section through an epithermal system, indicating the main zones of alteration and mineralization.

Mineralization in these deposits is mainly in the form of fracture fillings or veins, and disseminated in stockworks in and below the silicified cap (Rossiter, 1984). Grades are variable, with relatively

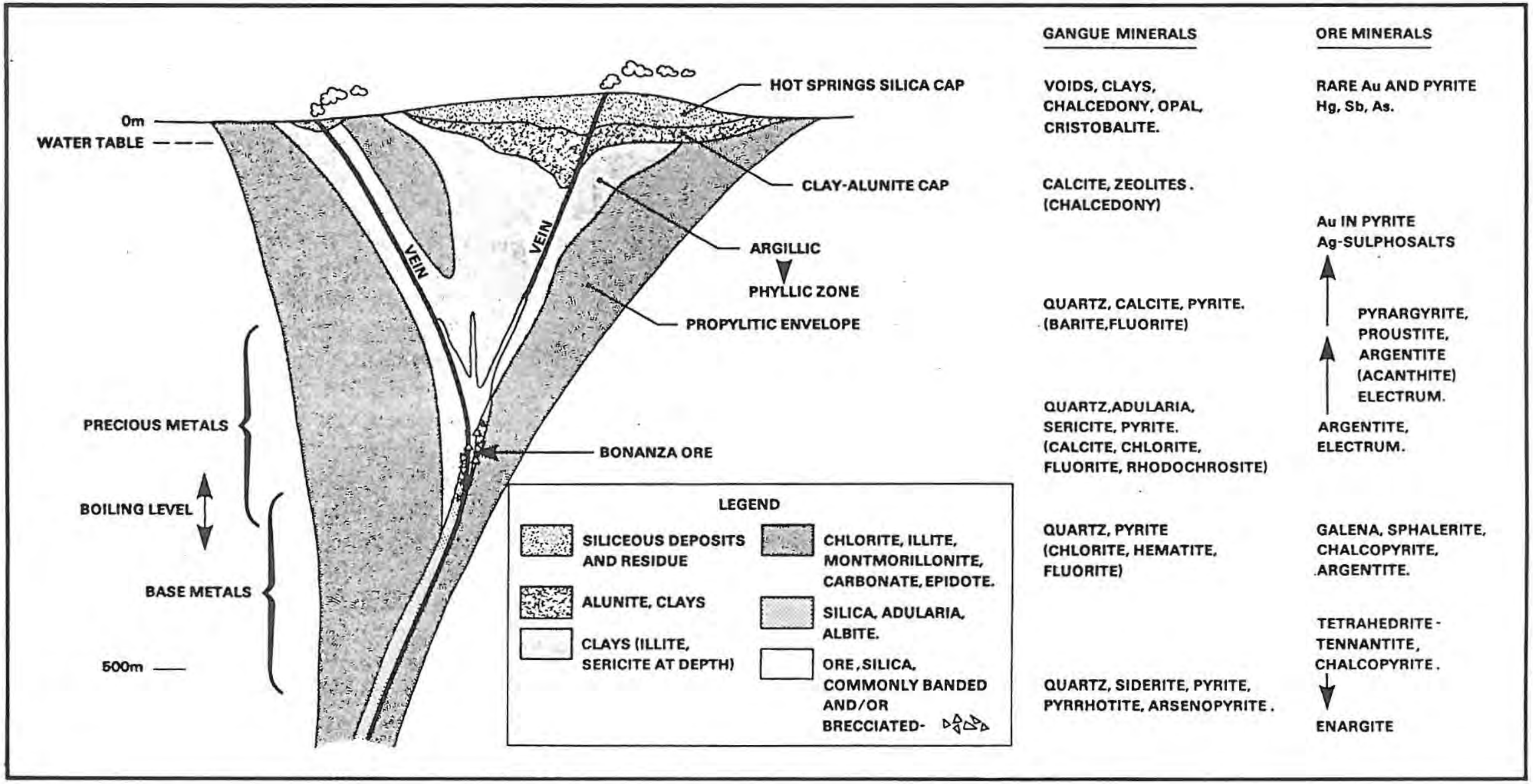


FIG A-6 SCHEMATIC SECTION THROUGH AN IDEALIZED EPITHERMAL SYSTEM (PANTELEYEV 1986) lower grades (5g/t) associated with the upper, disseminated parts of the orebody, and spectacularly high grades associated with the deeper lying "Bonanza" type veins. The ore body morphology is given in table A-9.

SEDIMENT-HOSTED DISSEMINATED DEPOSITS

Gold mineralization in these deposit types is present as very fine (submicroscopic) grains in predominantly carbonate host rocks. The Carlin deposit of Nevada represents the best known of these deposits, consequently they are also referred to as Carlin-type gold deposits.

The reader is referred to Bagby et al. (1985), Berger et al. (1991) and Romberger (1986) for detailed discussions of some of these deposit types. Tooker (1985) provides a comprehensive list covering the most important features of fifteen disseminated type deposits.

These deposit types invariably occur in carbonate host rocks. Romberger (1986) claims that disseminated deposits may also occur in other rock types, e.g. Round Mountain and Borealis, where mineralization is present in volcanic rocks. A review of the more salient features of these deposits in Tooker (1985) however indicates that they are better described as adularia-sericite type deposits.

The genetic model for Carlin-type deposits involves an early stage of "ground preparation", whereby the carbonate host rocks are extensively leached by heated, chloride-bearing fluids. Berger et al. (1991) propose a connate sedimentary origin for these fluids. The geothermal gradient (Romberger 1986) or magmatic intrusions (Berger et. al. 1991) act as heating mechanism. This stage results in the selective replacement of sedimentary beds with silica to form the pre-ore jasperoids seen at most Carlin-type deposits (Berger et al. 1991).

Tabel A-10 provides a summary of some of the characteristics of Carlin-type deposits.

TABLE A-10 CHARACTERISTICS OF SEDIMENT HOSTED DISSEMINATED GOLD DEPOSITS (BAGBY ET AL. 1985)

Jasperoidal, quartz-veinlet type	Disseminated, pod-like type
Quartz veins common	Quartz veins are uncommon
Main ore type is in silicified rock	Main ore type is not silicified
Ore primarily restricted to fault zones	Pod-like ore bodies extend away from faults
Several different silicification stages	Jasperoid may be present
Both gold- and silver-rich varieties	Gold-rich variety most common
Siliceous rocks common	Calcareous rocks common

Ore deposition takes place during the second stage, by deep circulating meteoric - magmatic, gold-bearing fluids. Structures such as faults provide important conduits for the fluids, and gold precipitation takes place in response to mixing of two chemically distinct fluids (Berger et al. 1991). Berger et al. (op. cit.) do not consider boiling to be the predominant ore-depositing process in these deposit types. Figure A-7 provides a schematic model of this sequence of events.

The orebody morphology can be summarised as stratabound, essentially two-dimensional. Berger et al. (1991) mention the considerable vertical extent of some of these deposits, indicating the possibility of a three-dimensional orebody. Grades are generally low, and vary between 1 and 10g/t.

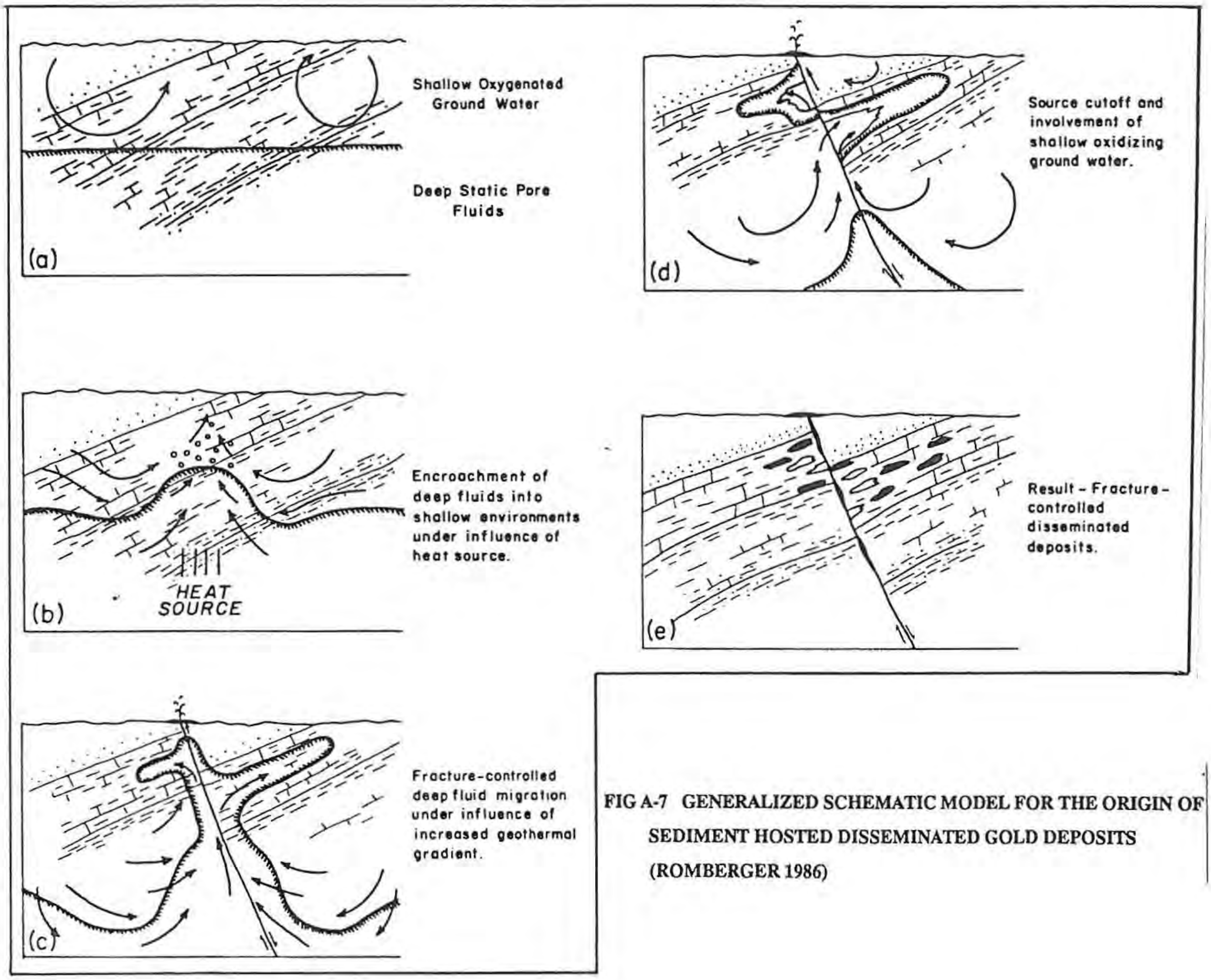


FIG A-7 GENERALIZED SCHEMATIC MODEL FOR THE ORIGIN OF SEDIMENT HOSTED DISSEMINATED GOLD DEPOSITS (ROMBERGER 1986)

2.3 SYNGENETIC DEPOSITS

The deposits included in this section were formed (and are presently being formed) by hydrothermal mechanisms on the sea floor, and ore formation occurred syngenetically with respect to the host rocks.

The discovery of gold-rich sulphide deposits actively forming at hydrothermal vents on the modern seafloor has confirmed the existence of gold-bearing fluids in submarine hot springs and supports a seafloor hydrothermal origin for gold in many preserved deposits now on land (Hannington et al. 1991). Rona (1988) provides a comprehensive discussion on modern seafloor mineralization.

Lydon (1984, 1988) provides review type papers of volcanogenic massive sulphide mineralization in general. Gold enrichment in this deposit type occurs primarily in the late stages of mineralization and almost exclusively in pyritic assemblages formed at temperatures below 300° C (Hannington et al. 1991). Huston et al. (1989) indicate that gold has two distinct modes of occurrence in these deposits : a gold-zinc-lead-silver association which is commonly found in the upper part of zinc-rich massive sulphide lenses, and a gold-copper association which is commonly present in the stringers and lower portions of copper-rich deposits. These two types of gold mineralization do not occur together. Gold may also be present in the more distal parts of the deposits, e.g. with baryte or distal pyrite (Huston et al. 1989).

Huston et al. (1989) provide a detailed discussion on the genetic model for gold mineralization in volcanogenic massive sulphide deposits. Briefly, this model involves an initial stage of gold transport in relatively cool hydrothermal fluids as thio-complexes, and deposition with silver, lead and zinc due to oxidation of the hydrothermal fluid caused by mixing and precipitation of metallic sulphides. As the sulphide mound grows gold is remobilized from the base of the mound and transported to, and deposited near, the top of the mound. During the second higher temperature stage of massive sulphide deposition when pyrite-chalcopyrite ore begins to replace sphalerite-galena-pyrite ore, gold is transported as chloride-complexes and deposited with the copper sulphides due to pH increase, and temperature and fO_2 decrease caused by the replacement. Precipitated gold may be remobilized as thio-complexes by the cooling fluids migrating up through the massive sulphide mound, and redeposited at the top of the mound, or as AuS(n) (n>2) complexes and deposited in the distal parts of the deposit. This model is summarized in figure A-8.

Table A-11 summarizes the distribution and mineralogy of gold mineralization from selected volcanogenic massive sulphide deposits.

Besides the well-known deposits of this type such as the Kuroko deposits of Japañ and the ophiolite hosted deposits of Cyprus, Oman and Newfoundland, Foster et al. (1986), Robert (1991) and Hannington (1991) suggest a similar genesis for massive sulphide deposits hosted in Archaean greenstone belts. Hannington (1991) further indicates the similarities between hydrothermal mineralization on the seafloor and gold occurrences in chemical and clastic sediments of Archaean greenstone belts.

Gold grades in volcanogenic massive sulphide deposits typically range between 0,2 and 2g/t, but some higher grade deposits average between 2 and 20g/t (Huston et.al. 1989). the orebodies are typically stratiform and lensoid in shape, but may show some structural modification.

3. GOLD DEPOSITS FORMED BY WEATHERING PROCESSES

The discovery of gold in laterites at Boddington emphasises the potential for gold mineralization and accumulation in the weathered profile. The deposit type discussed in this section is considered to have formed essentially in situ, and although some transportation of gold is implied in the genetic model, this is not to the scale of that implied for placer-type deposits.

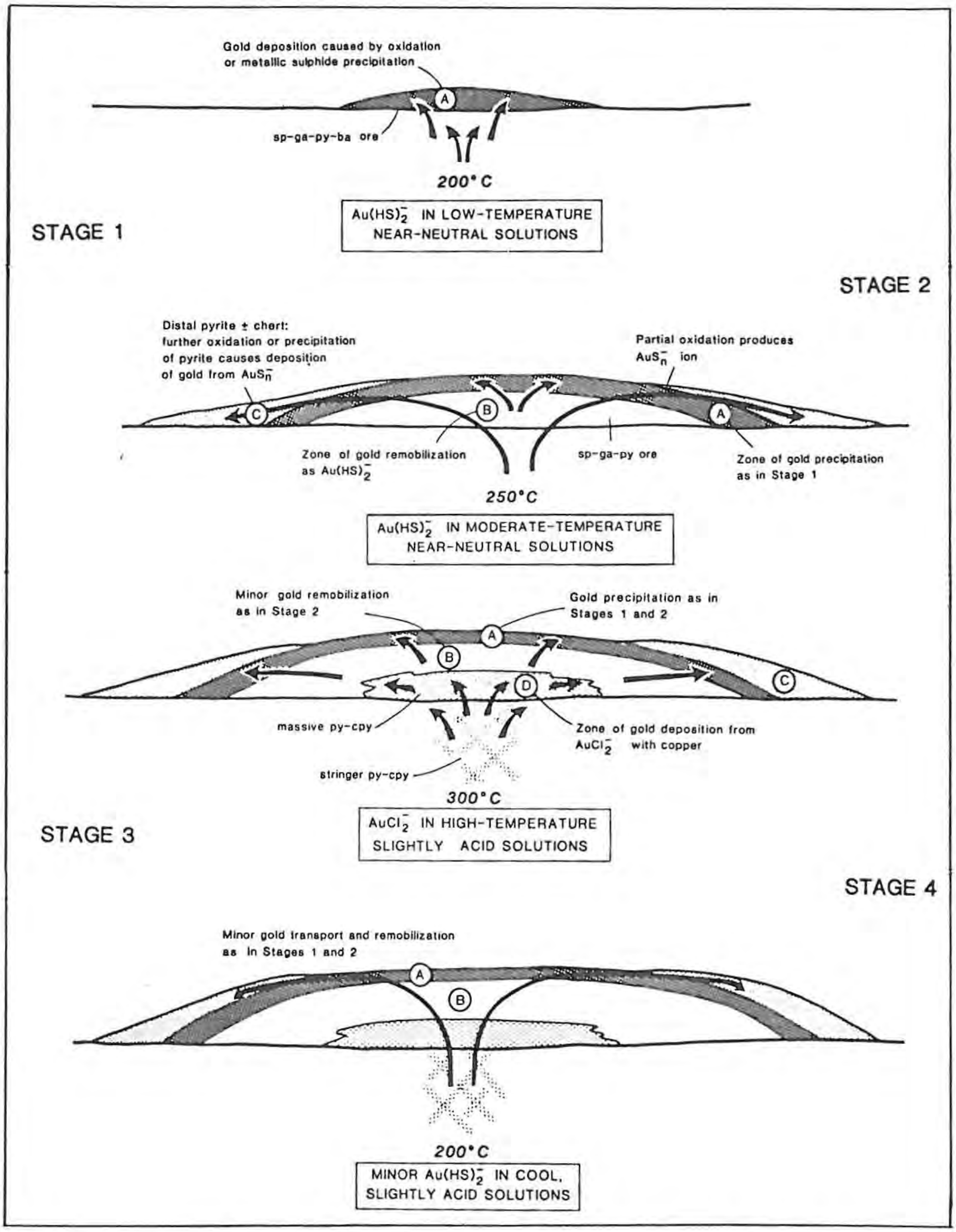


FIG A-8 SCHEMATIC MODEL OF GOLD TRANSPORT AND DEPOSITION IN VOLCANOGENIC MASSIVE SULPHIDE DEPOSITS (HUSTON ET AL. 1989)

TABLE A-11 THE DISTRIBUTION AND MINERALOGY FROM SELECTED VOLCANOGENIC MASSIVE SULPHIDE DEPOSITS (HUSTON ET AL. 1989)

Zinc-gold association		
Deposit	Grade and tonnage	Gold mineralogy and distribution
Corbet, Quebec, Canada	2.7 m.t. at 2.9% Cu, 2.0% Zn, 20.5 g/t Ag and 0.93 g/t Au.	Gold has a similar distribution to zinc and occurs in the pyritic fringes of the deposit as fine inclusions in pyrite.
Arizona Proterozoic deposits, U.S.A.	Variable.	Gold is enriched in the zinc-rich stratiform ores and chert cappings. It has average grades in copper-rich stratiform ores, and it is depleted copper-rich stringer ores.
Garpenberg, Sweden	0.4% Cu, 5.0% Zn, 130 g/t Ag and 0.9 g/t Au.	Gold has a higher concentration in zinc-lead ores than in copper ores.
Rosebery, Tasmania, Australia	19.4 m.t. at 0.74% Cu, 16.2% Zn, 5.0% Pb, 155 g/t Ag and 2.9 g/t Au.	Gold occurs dominantly as inclusions of native gold and electrum in pyrite and tetrahedrite. Gold concentrates in zinc-lead ore and massive barite mineralisation.
Hellyer, Tasmania, Australia	19 m.t. at 0.4% Cu, 13% Zn, 7% Pb, 160 g/t Ag and 2.3 g/t Au.	Gold occurs in arsenopyrite and pyrite as either submicroscopic inclusions or in solid solution. Gold is enriched in the upper parts of zinc-rich stratiform ore and in a pyritic chert that caps the stratiform ore.
Que River, Tasmania, Australia	2.6 m.t. at 0.42% Cu, 12.9% Zn, 7.4% Pb, 204 g/t Ag and 3.5 g/t Au.	Gold occurs as native gold or electrum associated with galena or as inclusions in pyrite and sphalerite. Arsenopyrite and pyrite also carry gold. Gold concentrates towards the top of zinc-rich stratiform ore and shows an exponential increase with zinc grades.
Tilt Cove, Newfoundland, Canada	Not available.	Gold occurs as native gold which is more strongly associated with sphalerite than chalcopyrite or pyrite.
Kuroko district, Japan	Variable.	Gold occurs as electrum with silver-rich rims. Gold contents of the electrum vary between 52% and 85%. Electrum in the oko (copper-rich) zone has a gold content of 95% to 96% in the Hisaka deposit (Watanabe, 1971; in Shimazaki, 1974). Gold concentrates in the upper part of the kuroko (zinc-lead) ore.
Shakanai, Japan	8.73 m.t. at 1.4% Cu, 3.4% Zn, 1.0% Pb, 44 g/t Ag and 0.3 g/t Au.	Gold occurs as electrum generally associated with galena.
Gold-copper association		
Millenbach, Quebec, Canada	3.0 m.t. at 3.5% Cu, 4.8% Zn, 48 g/t Ag and 0.86 g/t Au.	Gold occurs with copper in the pyrrhotitic core of massive ore. Gold has an antipathetic relationship to zinc.
Stekkenjokk, Sweden	27.9 m.t. at 1.35% Cu, 2.43% Zn, 0.29% Pb, 46 g/t Ag and 0.27 g/t Au.	Gold is enriched with copper, bismuth, molybdenum and tin at the base of the massive sulphide lens and in the stringer zone.
Mt. Lyell, Tasmania, Australia	98.6 m.t. at 1.17% Cu, 7.15 g/t Ag and 0.39 g/t Au.	Gold occurs as very fine inclusions in or along grain boundaries between chalcopyrite, bornite and pyrite. Coarse grained gold may occur in tension quartz veins. Gold correlates well with copper.
Heath Steele B-1 Orebody, New Brunswick, Canada	40.4 m.t. at 1.1% Cu, 4.7% Zn, 1.6% Pb, 64 g/t Ag and 0.9 g/t Au.	Gold occurs as a "discrete mineral phase." It is weakly associated with copper and decreases in grade towards the stratigraphic top of the orebody.
Mt. Chalmers, Queensland, Australia	4.3 m.t. at 1.6% Cu and 2.3 g/t Au.	Gold occurs as native gold associated with copper in the core of the massive sulphide and the upper part of the stringer zone.
Mt. Morgan, Queensland, Australia	67 m.t. at 0.70% Cu and 4.87 g/t Au.	Gold occurs as native gold, gold tellurides and in pyrite. All occurrences have an association with pyrite and chalcopyrite.
Cyprus deposits	Variable.	Gold is erratically distributed through the ores. Pyritic ores of the Mathaitis area have higher gold grades than copper-rich bodies at Mavrovouni and Skouriotissa. The highest gold (4.2 g/t) came from the small (0.3 m.t.) copper-rich (2.9% Cu) body at Kinoussa. In the Kilrou area high gold grades occur at the top of stockwork mineralisation near the contact with massive sulphide.
Mavrovouni, Cyprus	17 m.t. at 4.5% Cu, 0.5% Zn, 10 g/t Ag and 0.8 g/t Au.	Gold is associated with chalcopyrite rather than pyrite. Gold occurs in copper-rich ores.
Kalavosos, Cyprus	1.0 m.t. at 0.5-2.5% Cu and 0.7% Zn.	Gold occurs in the copper-rich ores.
Nurukawa, Japan	Not available.	Gold occurs in paragenetically early quartz-chalcopyrite veins in siliceous ore and in the footwall stringer zone. Mineralogically gold electrum with a fineness between 500 and 700. This is a particularly gold-rich (15-100 g/t Au) deposit.

Manti (1987), Mann (1984) and Butt (1989) provide comprehensive review-type papers on gold in the laterites of Western Australia and the mobility of gold in the weathered profile, in general.

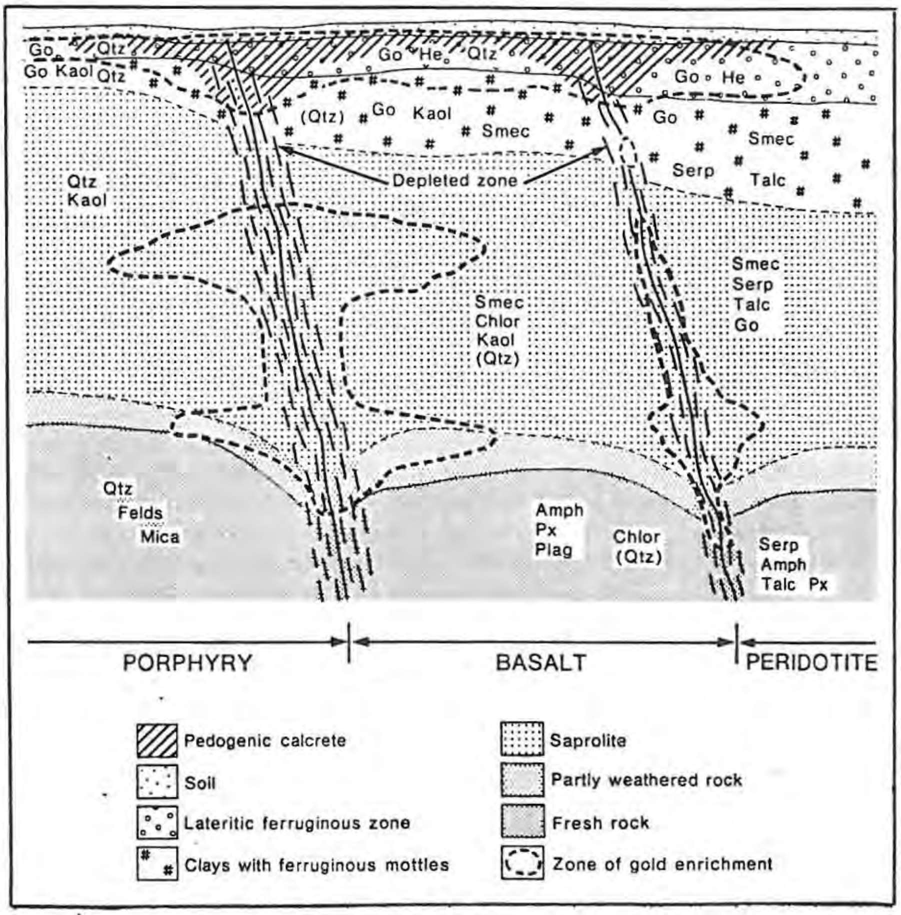
Butt (1989) has shown that gold distribution in the deeply weathered lateritic regolith of Western Australia is primarily concentrated in the upper ferruginous horizon and the deeper seated saprolitic horizon. These two gold-rich horizons are usually separated by a 5 to 15 m thick zone which is barren of gold. Figure A-9 illustrates the distribution of gold in the weathered profile.

The upper ore zone is characterized by fine-grained gold of high fineness (Ag<0,5%). Coarser grained primary and secondary gold may be present as small primary nuggets enclosed in pisoliths or ferricrete, and euhedral secondary crystals developed in association with iron oxide segregations (Butt 1989).

In the saprolitic horizon supergene enrichment of gold is either confined to the source unit or laterally dispersed in the wall rocks. Secondary gold is of high fineness and may occur in a variety of forms (Butt op.cit.).

Butt (op.cit.) relates the distribution pattern and mineralization in this deposit-type directly to the history of weathering under changing climatic conditions.

Humid tropical climates result in a deeply weathered profile. Oxidation at the weathering front produces neutral to acid conditions, and gold associated with tellurides or held in the lattice of sulphides is released, but the free metal remains largely immobile due to the absence of suitable complexing ligands. Butt (op.cit.) indicates that some dispersion is possible during this stage due to complexing



Amph = amphibole, Chlor = chlorite, Go = goethite, He = hematite, Felds = feldspar, Kaol = kaolinite, Serp = serpentine, Smec = smectite, Plag = plagioclase, Px = pyroxene, Qtz = quartz.

FIG A-9 SCHEMATIC SECTION SHOWING THE DISTRIBUTION OF GOLD IN THE WEATHERED PROFILE OF WESTERN AUSTRALIA (BUTT 1989)

with organic ligands, and if carbonate is present in the primary mineralization, the oxidation of pyrite may produce thiosulphate ions and gold transport as thio complexes. Mechanical processes during this stage cause lateral dispersion in the upper parts of the lateritic profile.

Uplift and a change in climate to more arid conditions result in a lowering of the water table so that the upper horizons become progressively unsaturated. Calcite, dolomite, gypsum, halite and other salts are precipitated in the unsaturated zone and the ground waters become saline; this permits the formation of soluble gold and silver halide complexes (Butt op.cit.). Reversals in climatic conditions to more humid climates during the dry cycle have an important influence on the formation of these deposits, and it is during these periods that significant redistribution of gold takes place. Gold is mobilized and reprecipitated due to reduction either by organic material or oxidation of ferrous iron at the water table (Butt op.cit.). Successive humid periods during the general lowering of the water table may result in the formation of two or three subhorizontal zones of supergene gold enrichment.

Figure A-10 presents a schematic summary of the formation of supergene gold deposits.

These deposit types are generally stratiform to three dimensional in shape. They are mostly small (<1,5 million tons) and of low grade (1,5 - 5g/t) (Butt op.cit.).

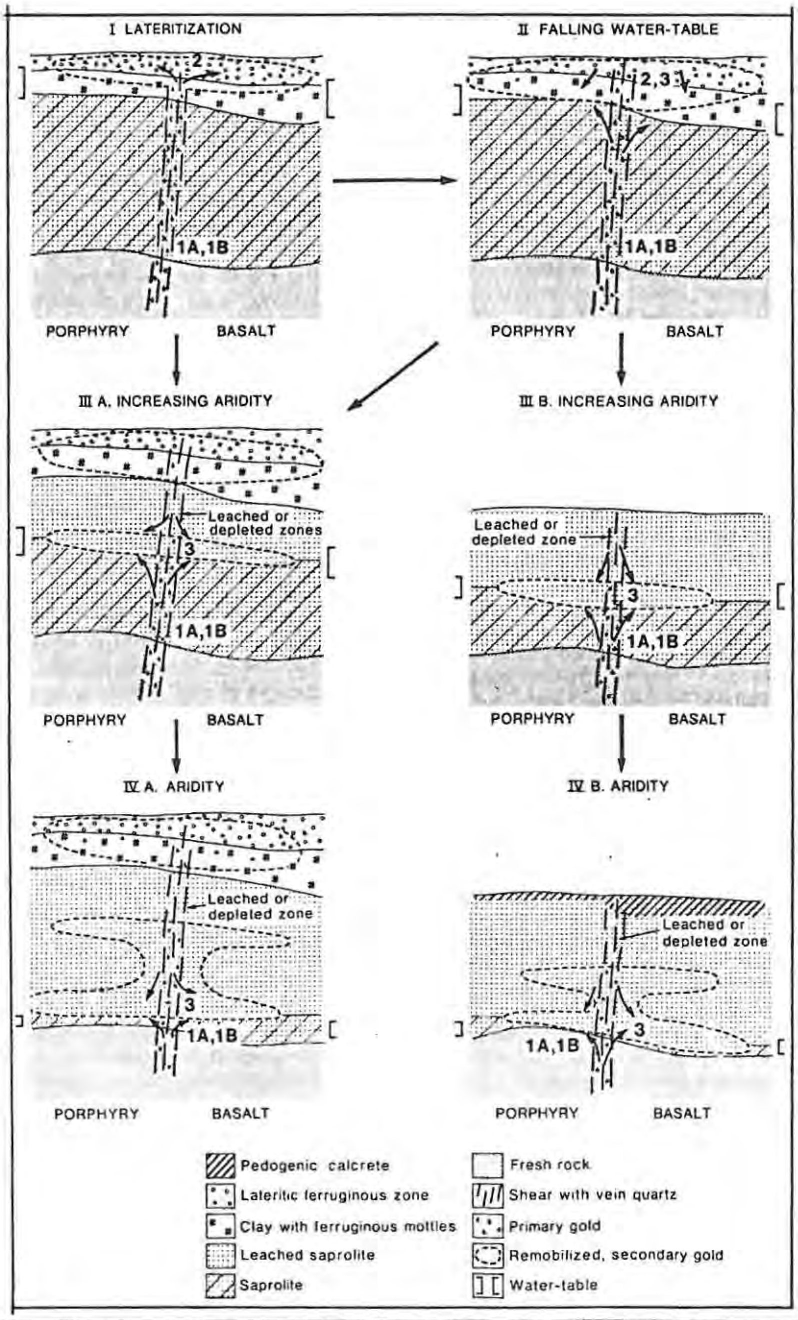
4. PLACER TYPE GOLD DEPOSITS

The term "placer" is derived from Spanish, and refers to those deposits where gold was accumulated by alluvial processes.

4.1 PALAEOPLACER DEPOSITS

The most obvious and best known examples of this deposit type are found in the Witwatersrand Supergroup of South Africa. Similar deposits occur in Central Africa and Brazil (Minter 1991).

In the Witwatersrand Supergroup, the placers are associated with conglomerates and quartz arenites. The conglomerates comprise clasts of mainly vein quartz, chert and quartz arenite with minor silicified shale and quartz porphyry,



	Complex	Dissolution conditions	Precipitation conditions	Product
1A	Thiosulfate (carbonate-rich ore)	Alkaline, mildly oxidizing	Dilution, acidification, oxidation, reduction	Electrum
1B	Thiosulfate (carbonate-poor ore)	Alkaline-acid, mildly oxidizing	Dilution, acidification, oxidation, reduction	Electrum, reduction in grain size
2	Organic	Neutral-acid, mildly reducing-oxidizing	Reduction	Fine-grained gold, high fineness
3	Chloride	Acid, oxidizing	Dilution increasing pH, reduction	Gold, high fineness

FIG A-10 DEVELOPMENT OF SUPERGENE GOLD DEPOSITS (BUTT 1989)

which range from pebbles to cobbles in size. The matrix consists of mostly silica with a small proportion of phyllosilicates (Minter 1991). The placers are generally tabular and sheet-like, and average 0,5 m in total thickness (Minter op.cit.). Minter (op.cit.) indicates that the environment of deposition of the placer sediments appears to have been of a fluvial nature in which unimodal transport by shallow transient streams prevailed.

Gold and associated heavy minerals is concentrated in the placers on scour surfaces, within clast-supported conglomerates and on foreset and planar bedding surfaces in the sand facies. Pebbly bottom and top surfaces of gravel increments in the placer units are generally better mineralized (Minter op.cit.).

Although an epigenetic hydrothermal origin for these deposits has been proposed (Phillips et al., 1989) most evidence appears to favour a syngenetic, detrital origin. Table A-12 summarizes the common characteristics of ancient gold-bearing placers.

TABLE A-12 THE COMMON CHARACTERISTICS OF ANCIENT GOLD BEARING PLACERS (MINTER 1991)

• Ancient gold placers overlie Archaean granite-greenstone basement representing an auriferous stable cratonic source.
• The relief of the palaeotopography is low, indicating a mature landsurface. This is supported by the lithic clast assemblage of the sedimentary host rock, which is composed predominantly of quartz and chert, with a noticeable absence of granitic and volcanic components. Small proportions of felsic porphyry may occur.
• They may lie stratigraphically beneath Superior-type iron formations. This reflects an age of greater than 2.2 Ga, a cratonic margin setting, and evidence of preservation.
• They occupy quartz–arenite-filled channels on unconformities that are greater than 200 km^2 in extent. This indicates widespread erosion and sediment maturity.
• They occur in stacked tectonic/sedimentary units. This demonstrates repetition of degradation/aggradation cycles but does not preclude single clastic packages having a gold potential.
• They are associated with onlapping marginal unconformities, and palaeocurrent evidence indicates flow perpendicular to the subcrop strike of the unconformities. This indicates crustal warping to form highlands and adjacent basins.
• The palaeocurrent systems are unimodal and the deposits reflect shallow gravel-bar and channel morphology, indicating fluvial transport by braided streams on fans or braid deltas.

Although the Witwatersrand probably represents the worlds largest single source of gold, the foregoing discussion has been kept intentionally brief. For more detail on these deposits the reader is referred to numerous and voluminous papers included in Anhaeusser et al. (1986) and Minter (1991).

4.2 RECENT PLACER DEPOSITS.

Boyle (1979) notes four prerequisites for the formation of placer deposits :

- The occurrence of gold in deposits, i.e. a gold source. The source of the gold may be in the form of any of the deposit types previously discussed.
- A long period of deep chemical and mechanical weathering on a surface of submature to mature topography, during which time the gold is set free from the deposits.
- Concentration of gold by some agency, generally water.
- Absence of extensive glaciation.

Boyle (op.cit.) distinguishes between eluvial and alluvial placer deposits, the primary distinction being that eluvial deposits form more or less in situ. In this paper the eluvial deposits of Boyle are better classified as lateritic deposits discussed in section 3.

The initial ore-forming process in placer deposits is weathering. During weathering one of three things may happen to the gold in the primary deposits : The gangue minerals may be disintegrated and leached away, and the gold may remain in situ or passed into alluvial placers. The gold may be dissolved and carried away without precipitation. The dissolved gold may be wholly or partly reprecipitated. Placers are therefore the result of both chemical and mechanical processes. Boyle (1979) indicates that the size of primary gold particles is of major importance, and quotes some districts where the primary gold is very fine grained

(<50 microns) or is a microscopic or lattice constituent of other ore minerals, that have not formed alluvial deposits even though other conditions were favourable. There are, however, examples of where the reverse is true, and Boyle (op.cit.) relates this to the purity and the surface characteristics of the fine-grained gold.

Since alluvial gold is invariably found on or near bedrock surface, the nature of the bedrock or "false bottoms" are decisive in the formation of alluvial placers. The most favourable bedrocks are those that form natural riffles perpendicular to the stream or river course. Nests of boulders, and pitted limestone bedrocks are of obvious importance, as are sheared and fractured bedrocks. "False bottoms" formed by indurated clay and hardpan layers appear to be particularly favourable (Boyle 1979).

The importance of stream and river dynamics in the formation of placers is evident. The controlling factors involved in sedimentation in static water are the specific gravity, size and shape of particles. Factors which influence the movement of particles in streams and rivers include the velocity of the water (essentially a function of the gradient), the degree of turbulent flow, the specific gravity of the particles and the nature of the stream or river bed. Boyle (op.cit.) indicates that placer formation is usually associated with the "mature" stage of river evolution.

Figure A-11 presents a schematic model of the distribution of material in an experimental stream due to differences in specific gravity.

Gold in placers occurs in a variety of forms ranging from extremely fine-grained "flour" or "float" gold to nuggets, the largest of which, the "Welcome Stranger" found near Ballarat Australia, weighed 2516 oz. troy (78 kg). In general however placer gold occurs in small flattened scales

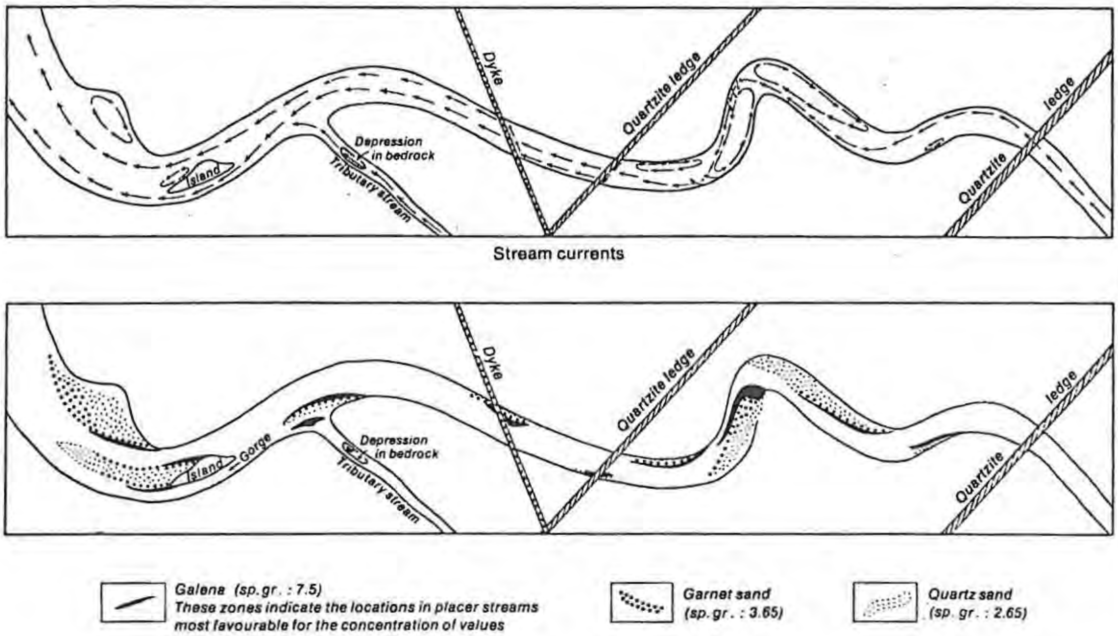


FIG A-11 DISTRIBUTION OF MATERIAL DUE TO SPECIFIC GRAVITY (BOYLE 1979)

or grains averaging a few millimetres in diameter, or as fine-grained particles known as "dust", averaging a few tenths of a millimetre in diameter (Boyle,1979). Boyle (op.cit.) further indicates that most placer gold is above 850 fine. Other elements in placer gold include mainly silver, copper and iron. Placer gold usually has a higher fineness than the source gold.

The tenor of alluvial placers is quoted as value (grams) per cubic meter. Boyle (1979) indicates that placers with a gold content as low as 0.1 ppm can be worked economically. Boyle (op.cit.) and Bache (1982) summarise the general characteristics of placers as follows :

. coarser gold is generally deposited in the upper reaches, and finer gold in the lower reaches of a placer water course,
. The richest and coarsest gold and nuggets are deposited in layers with comparatively coarse sediment, and the finer grained gold is commonly deposited with the finer,

sandy fractions of the sediment,

- Gold normally occurs in streaks with other heavy minerals such as magnetite, scheelite, barite etc.,
- Coarser gold is deposited on a bottom with a steep gradient, and in narrow gulches and gorges,
- Little gold is deposited in places where scouring action is marked,
- Potholes generally do not form effective traps due to the centrifugal grinding and ejection of fine gold from such sites,
- Gold generally collects on the downstream side of natural riffles, and zones of slow water flow are favourable sites,
- Glaciation does not preclude the occurrence of placers.

5. MISCELLANEOUS SOURCES OF GOLD

In countries with extended histories of gold mining such as South Africa, Zimbabwe, Swaziland and Botswana additional sources of gold are represented by the extensive reserves in waste tailings dumps evident throughout the major gold producing areas.

Although low grade, these "orebodies" offer substantial advantages. In contrast to conventional mining operations, the entire orebody can be proven and both tonnages and grades established to a high degree of confidence. This fact, coupled with low production costs, make these targets ideal for the small-scale operator.

In general the younger slimes dumps of the Witwatersrand report lower grades than the older dumps of the Barberton and Pilgrim's Rest area, averaging 0,5 - 1g/t and 2g/t respectively. Gold, uranium and pyrite are recovered from the Witwatersrand dumps, and gold and pyrite from the Pilgrim's Rest dumps.

6. A CLASSIFICATION OF GOLD DEPOSITS FOR SMALL SCALE OPERATIONS

A variety of classification schemes based on different criteria has been suggested by various workers in the past (see Bache, 1982; Boyle, 1979; Guilbert and Park, 1986; Lindgren, 1928). The classification proposed here is based on practical aspects of mining rather than theoretical aspects of mineralization.

Primarily, gold deposits are divided into open-pit mineable and underground mineable, with reference to the amenable mining method. It is important to note that underground mineable deposits could effectively be mined by opencast methods depending on depth and degree of selectivity of the orebody. However, open-pit mineable deposits by virtue of their nature would generally not be amenable to underground mining methods.

Secondly, the deposit types in each class are arranged in order of their amenability to small scale mining. This may be a point of contention, as it could correctly be argued that any deposit type could be mined on a small scale. However, certain deposit-types, by virtue of their nature, are more amenable to small scale mining than others. Reference to section B will provide more clarity on this point.

CLASS A : OPEN-PIT MINEABLE GOLD DEPOSITS

1 Tailings dumps
2. Lateritic gold deposits
3. Recent placers
4. Disseminated sediment-hosted gold deposits
5. Skarn gold deposits
6. Porphyry type deposits
7. Volcanic-hosted epithermal deposits (upper parts)

8. Felsic-hosted gold deposits
9. Volcanogenic massive sulphide deposits

CLASS B : UNDERGROUND MINEABLE GOLD DEPOSITS

1. Lode gold deposits
2. Volcanic hosted epithermal deposits (lower parts)
3. Paleoplacer deposits
4. Primary magmatic deposits

SECTION B

THE EVALUATION OF SMALL SCALE GOLD DEPOSITS

Possibly the primary objective of any exploration programme, small or large scale, is a clear and as accurate as possible indication of the in situ size and value of an area of mineralization, i.e. an ore reserve estimation. In large companies this often forms the final stage of involvement for the exploration geologist, wherafter all manner of "experts" take over and the geologist packed off to the next prospect. In the small scale operation, however, the geologist will find himself the only expert available.

Mackenzie (1980) states that the evaluation of an orebody comprises two distinct dimensions, namely the geological and the economic dimension. The parameters are initially estimated in terms of contents within the mineral deposit, i.e. the in situ ore reserve estimation. For use in economic evaluation such estimates must be converted to a "recoverable" basis to assess the tonnages and grades which will actually be mined and recovered. Economic parameters can then be applied to the estimates in order to establish the economic feasibility of the project.

1 THE GEOLOGICAL DIMENSION

During the delineation stage of the evaluation process information is gathered by either drilling or trenching in order to make reliable estimates of the physical characteristics of the orebody.

The techniques used to process and analyse this information may be subdivided into conventional, statistical and geostatistical techniques.

1.1 CONVENTIONAL ANALYSIS TECHNIQUES

All conventional methods use basically the same data. After the orebody has been adequately delineated, it is subdivided into blocks of various shapes depending on the overall shape and attitude of the orebody, the drilling pattern used for delineation and the method of estimation chosen. Each block is approximated by a geometric figure. It is given the average grade of its samples. A volume is computed based on the geometric relationships and converted to a tonnage by dividing by the estimated tonnage factor. Block values are combined to assess the average tonnage and grade of the deposit. Figure B-1 provides a summary of the geometric patterns used in ore reserve calculations.

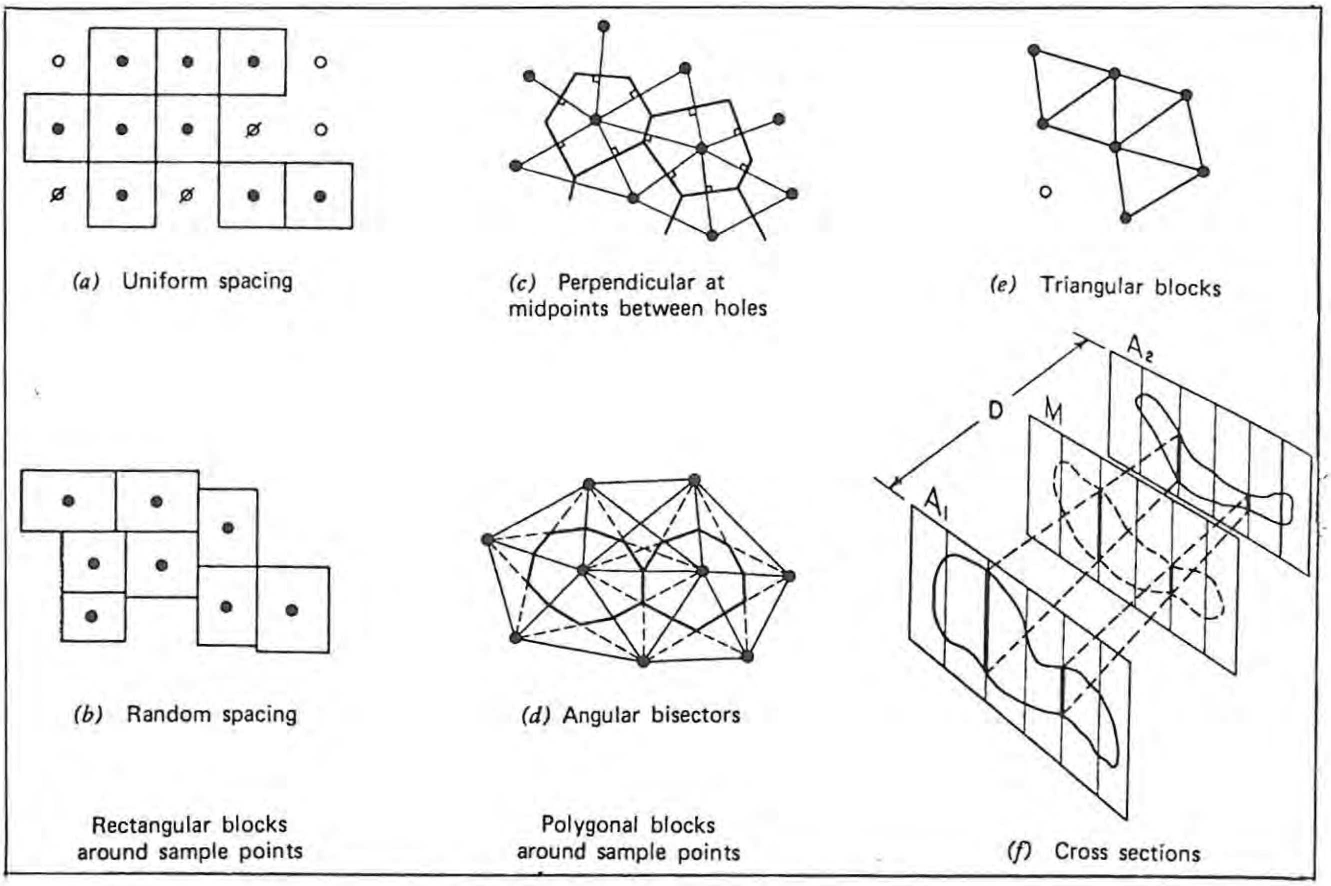


FIG B-1 GEOMETRIC PATTERNS USED FOR ORE RESERVE CALCULATIONS (PETERS 1987)

All conventional methods require the following assumptions :

i) The basic elements of the deposit (e.g. grade or width of interception) observed at any point change or extend to the adjoining point of observation according to a chosen principle of interpretation. The two most common principles of interpretation are the rule of gradual change and the rule of nearest point.

According to the rule of gradual change the elements of a deposit change linearly and continuously between two adjacent observations (see figure B-2).

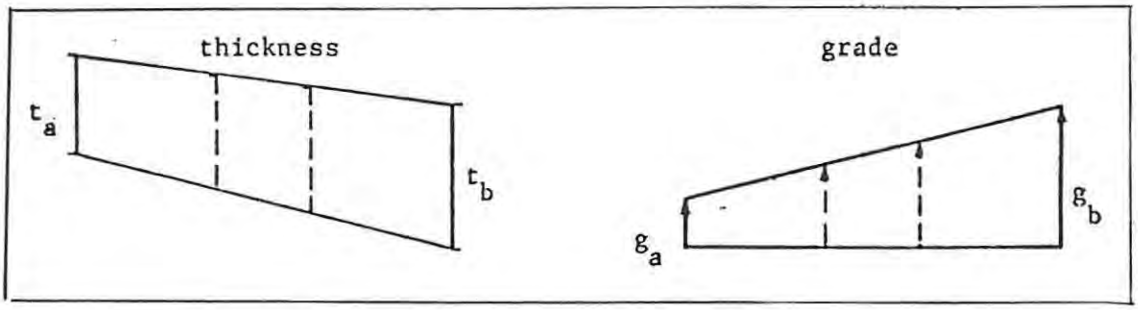


FIG B-2 THE RULE OF GRADUAL CHANGE (MACKENZIE 1980)

According to the rule of nearest point the value between two adjacent observations is considered constant and equal to the value of the nearest observation (see figure B-3).

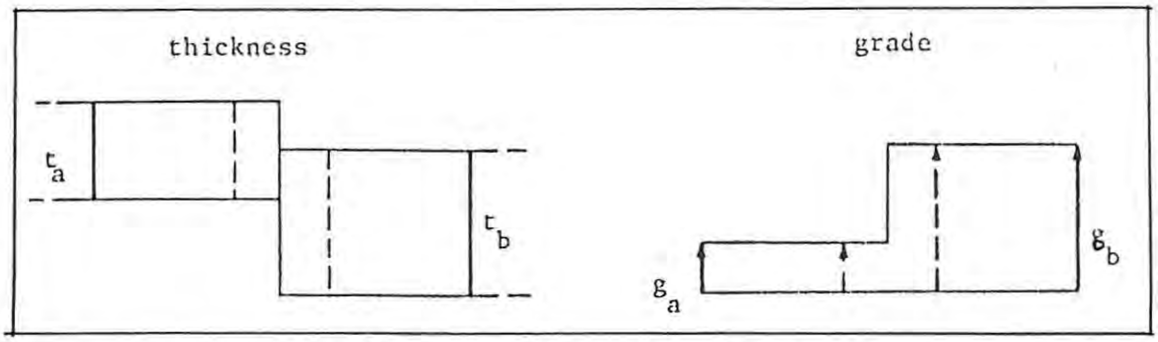


FIG B-3 THE RULE OF NEAREST POINT (MACKENZIE 1980)

ii) The mineral deposit has been adequately delineated by the chosen method of exploration (i.e. drilling or trenching) and that this method proves the continuity of the deposit between observation points.

iii) The actual form of the orebody can be approximated with reasonable accuracy by the combination of several hypothetical geometric shapes.

iv) A rule of thumb is used to extend the deposit beyond the last observation point (e.g. half the distance between the last two observation points).

1.1.1 THE CROSS-SECTIONAL METHOD

With the cross-sectional method the orebody is divided into blocks by constructing vertical or horizontal cross-sections at certain intervals. These intervals must be equal and are usually made to coincide with the sampling points (i.e. drill holes). The rule of gradual change is usually used to draw the cross-sections, and a rule of thumb is applied to the ends of each section. Each drill hole intersection is assigned an area of influence by the rule of nearest point. The average grade and area of the section is then computed, and the section values are then extended to blocks. The block volumes are computed by assuming either the rule of gradual change or the rule of nearest point and a rule of thumb is again applied beyond the end sections. A tonnage factor based on the specific gravity of the rock type is computed, and applied to convert the volumes to tonnages. Tonnages and average grades for the blocks are combined to obtain the tonnage and average grade of the deposit.

1.1.2 THE LONGITUDINAL SECTION METHOD

With this method the section is a projection of the orebody on to a plane parallel to the orebody. The section is then divided into triangular blocks by connecting all drill hole intersections with straight lines. Each triangle on the plane is assumed to represent a prism with edges equal to the thicknesses of the individual drill hole intersections. The volume is computed using the formula for a truncated triangular prism, and the average grade is estimated as the weighted average of the drill hole intersections. Individual block values are combined to derive overall deposit tonnage and grade estimates. The longitudinal section may also be divided into polygonal blocks by constructing medians to the lines joining the drill hole projections. Each block assumes the grade and thickness of the central drill hole according to the nearest point rule. Block values are combined to determine the tonnage and average grade of the deposit.

1.1.3 THE MOVING AVERAGE METHOD

This method is most applicable to larger, low grade deposits where it may be required to obtain block values for open pit mining. The deposit is divided into blocks of uniform size and the average grade of each block is computed from the weighted sum of observations near the block. The assigned weights may be based on various assumptions :

. Inverse of the distance.
. Inverse squared distance.
. Inverse cubic distance.

Ore reserves are determined by combining all blocks with average grades above a predetermined cutoff grade. The reader is referred to Mackenzie (1980) for a more detailed discussion of the conventional analysis techniques.

Mackenzie (op.cit.) quotes two major deficiencies with conventional methods :

. The area of influence given to individual drill holes generally far exceeds their actual area of influence.

- The conventional methods do not provide a measure of confidence (or a measure of the uncertainty) in the overall grade and tonnage estimates of the deposit.

1.2 STATISTICAL METHODS OF ORE RESERVE ESTIMATION

A measure of the uncertainty associated with the deposit characteristic estimates can be obtained when using statistical analysis techniques. These techniques are based on the assumption that observed variations are due to random fluctuations.

Statistical techniques involve presenting the sample data for any specific deposit characteristic (e.g. grade) as a probability distribution of its population, i.e. a frequency distribution histogram, whereby the sample values are divided into class intervals and the number of occurences (frequency) in each class interval plotted.

A number of statistical equations have been developed by which certain parameters are calculated. The theory of statistics and the derivation of these equations falls beyond the scope of this paper (and indeed beyond the ken of most geologists!). However, the application of these equations is a fairly simple matter. For more clarity on the applications the reader is referred to David (1977) and Rendu (1981).

If the sample population displays a normal or Gaussian probability distribution the arithmetic mean may be calculated by simply dividing the sum of the values by the number of samples. Thus :

Sample mean $\bar{x} = 1/n \Sigma x_i$
where n = number of samples
x_i = sample value

This provides an estimate of the mean value. The sample variance (S^2) provides an indication of how far the sample values deviate from the estimated mean, and is calculated by :

$$S^2 = 1/n\text{-}1 \ (\Sigma x_i^{\ 2} - (\Sigma x_i)^2/n)$$

and the standard deviation is simply the square root of the variance.

Confidence limits (where n > 25) are calculated as follows :

Central 68% : $\bar{x}$ - S ; $\bar{x}$ + S.
Central 95% : $\bar{x}$ - 1,96 S ; $\bar{x}$ + 1,96 S

When n is smaller than 25, confidence limits are expressed in terms of percentage confidence intervals using Student's t tables.

It is evident that the reliability of the mean value estimate is a function of both the degree of variability of the deposit characteristic and the sample size. The distribution of actual mean values about the mean value estimate can now be expressed in terms of the variance of the samples. Thus :
Variance of mean : $Sn^2 = S^2/n$
and the standard deviation of the mean : $Sn = S/\sqrt{n}$
and confidence limits for n > 25 :

Central 68% : $\bar{x}$ - $S/\sqrt{n}$; $\bar{x}$ + $S/\sqrt{n}$
Central 95% : $\bar{x}$ - 1,96 $S/\sqrt{n}$; $\bar{x}$ + 1,96 $S/\sqrt{n}$

The cumulative frequency percentages of a normal population will plot as a straight line on log probability graph paper, and this provides a very useful method of analysing the data. The geometric mean is determined directly form the graph at 50% cumulative frequency. From the above equation for the central 68% confidence limits it follows that the standard deviation and variance may also be obtained from the graph:

The sample value at 84% cumulative frequency minus the sample value at 16% cumulative frequency will give 2S, thus :

$S = a-b/2$

where a = value at 84% cumulative frequency

b = value at 16% cumulative frequency

and variance = S^2

Further formulae refer specifically to pay limit calculations and include :

. Percentage above a certain grade (i.e. cutoff grade)

$\%(+Xc) = \phi(Zc)$

where $Zc = (Xc-\mu)/\sigma$

Xc = cutoff grade

μ = mean

σ = standard deviation

ϕ refers to the area under the curve as related to the z - score (Zc) and is obtained from tables.

. Average recovered grade above a certain cutoff :

$\mu(+xc) = \mu + (\sigma/\sqrt{2\pi}).(e - (Zc)^2/\phi(Xc))$

Although the above equations refer to a normal probability distribution, ore deposit grades generally display a positive skewness in the distribuiton. This is referred to as a log-normal distribution, the premise being that if the logarithms of the values rather than the actual values themselves were plotted a normal distribution about the natural log of the mean would result. In such a distribution the arithmetic mean is very unreliable because of the influence of the few very high outliers. The geometric mean (G.M.) is however much less sensitive to outliers, and the G.M. is then used to estimate the mean (μ) of the population.

The cumulative frequency percentages of the sample values are plotted on 2 cycle log probability paper. Such a plot often displays a drooping tail end, and this is corrected by adding a constant β to the sample values. This constant is arbitrarily chosen, and several trial runs are usually necessary before the β factor is found which produces the best fit straight line.

The resulting graph is referred to as a three parameter log-normal distribution, and all applications valid for the normal distribution may now be applied to the log-normal distribution.

The G.M. is determined from the graph at 50% cumulative frequency, and the mean (μ) is calculated by the following formula :

$$\mu = \text{G.M.} \times e^{\sigma^2/2} - \beta$$

where G.M. = geometric mean

σ^2 = population variance

The population variance is determined from the graph in the same manner as with a normal distribution, thus :

$$\sigma^2 = \left(\frac{\ln A - \ln B}{2}\right)^2$$

where A = value at 84% cumulative frequency
B = value at 16% cumulative frequency

These equations hold for large sample sizes (n > 1000). If the number of samples is less than 1000 then :

$$\mu = \text{G.M.} \times (\Psi n(V)) -$$

$\Psi n(V)$ has been tabulated by Sischel, as have various confidence limits.

Pay limit calculations include :

. Percentage above cutoff grade

$$\%(+Xc) = \phi\left\{\left(1/\sigma \ \ln \frac{Xc + \beta}{\mu + \beta}\right) + \frac{\sigma}{2}\right\}$$

where Xc = cutoff grade
ϕ = obtained from tables

. Average value above cutoff :

$$\mu(+Xc) = \frac{\phi\left[\frac{1}{\sigma} \ \ln \frac{Xc + \beta}{\mu + \beta} - \frac{\sigma}{2}\right]}{\phi\left[\frac{1}{\sigma} \ \ln \frac{Xc + \beta}{\mu + \beta} + \frac{\sigma}{2}\right]}$$

These equations are used to compile grade-tonnage curves.

Although statistical methods are widely used there are some potentially serious deficiencies to these methods. The first and most obvious is the assumption of randomness. A number of tests exist which may be applied to sample data in order to determine whether the randomness assumption is met. However, grade variations within mineral deposits are non-random and the data is said to contain "structure" i.e. the grade changes either abruptly or gradually from uneconomic mineralization to high grade ore. Usually at an early stage of exploration/delineation the data are far enough apart to be treated as "random".

It further follows that any specific deposit characteristic will be influenced by its position within the deposit. The second important deficiency of classical statistical techniques is that they fail to take into account this spatial relationship.

The primary method of dealing with non-random data is by geostatistics.

1.3 GEOSTATISTICAL ANALYSIS TECHNIQUES

A report of this nature cannot include more than a very basic and general discussion on geostatistics, and the reader is referred to Clarke (1987) and David (1977) for more detailed coverage of the subject.

Clarke (1979) indicates that a geostatistical ore reserve estimation can be divided into two parts. The first is the investigation and modeling of the physical and statistical structure of the orebody. Semivariograms, which reflect the continuity and structure of the orebody are constructed during this stage. The second stage involves the estimation itself by "Kriging". This procedure depends entirely on the semivariograms constructed in the first stage.

The semivariogram is the basic tool of geostatistics (just as the histogram is for classical statistics). It is a graph (and/or formula) which examines the correlation between samples as a function of distance, in a specific orientation :

$$\overrightarrow{\gamma h} = 1/2n \sum (xi - x(i + h)^2$$

where $\overrightarrow{\gamma h}$ = semivariogram

n = number of samples

xi = value of sample at point i

x(i + h) = value of sample at distance h from i

An idealized semivariogram is depicted in figure B-6.

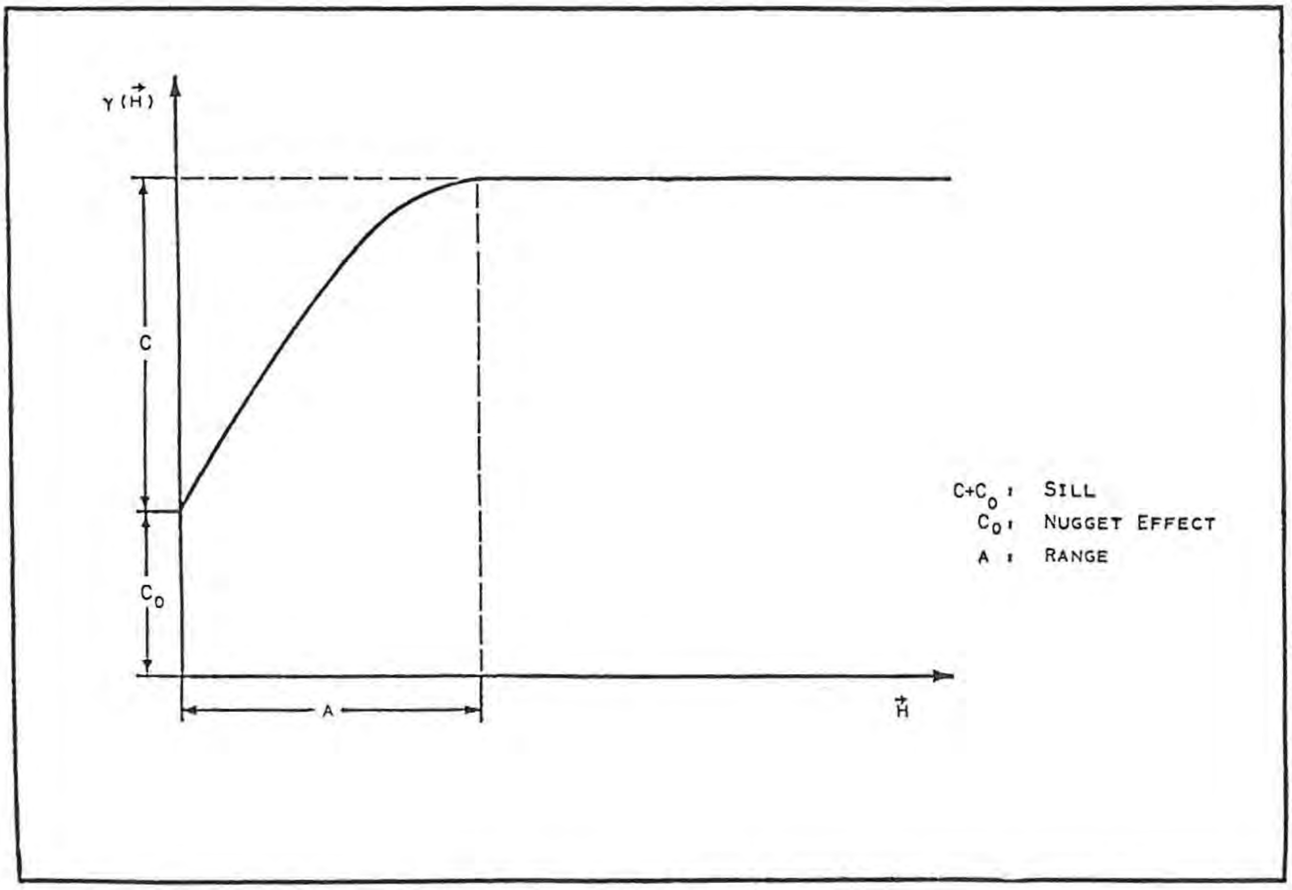


FIG B-4 THE SEMIVARIOGRAM (CLARK 1987)

The semivariogram is defined by four parameters:

- The shape and type of curve fitted to the points.
- The sill (C + Co) which depicts the variance of the population.
- The nugget effect (Co) which represents small scale structural variation.
- The range, A, which indicates the distance of influence of one sample on another.

the procedure of calculating and plotting a semivariogram is described in detail by Clark (1987) and David (1977). Basically the procedure involves computing the average square of the differences in an orebody parameter (e.g. grade, thickness, accumulation etc.) between all samples at a given distance apart, in a specified direction. The computed value is then plotted against the distance used on a graph. The procedure is repeated for different distances and a spread or configuration of points, termed the calculated or experimental semivariogram, is obtained. The final stage involves fitting one, or a combination, of the theoretical models to the experimental semivariogram.

Geostatistical ore reserve estimation, or Kriging, is basically a linear estimation method which develops optimal weights to be applied to each sample in the vicinity of the block being estimated. It uses both the position of the samples with respect to the block and the continuity of mineralization in different directions as portrayed by the selected semivariograms. Kriging is therefore not directly, and only, linked with the distance as are distance weighting methods. It involves the optimal selection of weights in such a manner that the estimation variance (i.e. the variance of the estimated grades about the true grade) is minimized, and the sum of the weights equals 1, thereby meeting the conditions set for the best linear unbiased estimator or "BLUE".

An additional important application of geostatistics is as applied to the volume-variance relationship. In mining terminology the volume is determined by the selective mining unit, or SMU. The SMU reflects the smallest volume of ore that can be selectively mined, and it is evident that the SMU will depend upon the mining method used, which is in turn dependent upon the nature and structure of the mineralization. It follows that the larger the SMU the smaller the variance between the SMU's, and the larger the variance within each SMU, or in other words : the larger the SMU, the more similar in grade they are to each other, but the more different the grades of the samples within each SMU.

The volume-variance relationship is such that the variance of samples within the whole volume (i.e. orebody) is equal to the variance of the SMU's plus the variance of samples within each SMU. Thus :

$$\sigma^2_{(o/v)} = \sigma^2_{(u/v)} + \sigma^2_{(o/u)}$$

- $\sigma^2_{(o/v)}$ is the variance of the samples within the whole volume, and this is given by the sill of the semivariogram.
- $\sigma^2_{(o/u)}$ is the variance of samples within each SMU. These values have been calculated for the various semivariogram models and different block sizes (i.e.SMU's), and are obtainable from tables. For example Clark (1987 pp 58-59) presents tables for a spherical model.

The variance of the SMU's may thus be calculated from the above equation, and this value is used to compile a grade-tonnage curve based on the variance of the SMU's.

2. DISCUSSION

The preceding sections have largely focussed on computational aspects, with reference to the grade and tonnage aspects of ore reserve estimation.

Bujtor et al. (1983) and Carras (1984) draw attention to the development in recent years whereby the concept of ore-reserve estimation has been broadened from its traditional computational basis in an effort to cover all aspects which may influence the actual amount of ore recovered. The amount of ore recovered, and consequently the metal recovered, is the single most important aspect in determining the eventual economic feasibility of the project, and the importance of this approach is therefore evident. In this regard Bujtor et al. (1983) list the major components of an ore-reserve estimate as follows :

- Sampling
- Geology
- Estimation methods
- Mining factors
- Metallurgical factors
- Environmental factors

Each of these components impact to varying degrees on the recoverable metal of a new mining project.

Correction factors are applied in virtually all established gold mines in an attempt to reconcile prediction and production, and this may indicate the complexity of the problem in converting global ore reserve estimates to recoverable estimates.

2.1 SAMPLING

Sampling is the primary means whereby the dimensions and values of an orebody are measured, and any errors in these data will carry through to all subsequent stages of evaluation. Bujtor et al. (1983 p 24) make an important, but often ignored statement in this regard : "No amount of sophisticated mathematical manipulation can compensate for poor quality data".

Carras (1984) discusses the most common problems which may lead to sampling errors. Briefly these include the following :

- Physical problems leading to sample bias
- Size of the sample, as it relates to the volume-variance relationship
- Representativity of the sample
- Accuracy of analytical methods

Carras (op.cit.) indicates the usefulness of the coefficient of variation (simply the standard deviation divided by the mean) in determining the extent to which data should be composited to yield a meaningful sample length. From the volume-variance relationship it is evident that the coefficient will be dependent upon the volume (or core length). Carras (op.cit.) indicates that data should be composited until the coefficient of variation is about 1.00. At that sample length the samples have some meaning.

2.2 GEOLOGY

Geological interpretation of the orebody is possibly the single most important aspect that may influence the ore-reserve estimate. Invariably interpretation is based on widely spaced surface drill holes. In complex orebodies this usually results in oversimplified conclusions regarding the continuity and orebody outline. Infill drilling may improve the reliability of the interpretation; however, this has to be measured against the additional cost and the concept of diminishing returns.

Bujtor et al. (1983) list the most important geological factors which need to be considered in ore reserve estimation as follows :

- The size, shape and attitude of the orebody
- The structural geology and rock types comprising the orebody and country rocks
- The economic minerals together with their relationship and distribution with other minerals and rock types
- Oxidation, leaching and enrichment
- Sizes and shapes of crystals and grains and their relationship to the mineral distribution
- Assay populations, means, grades and thicknesses, specific gravities.

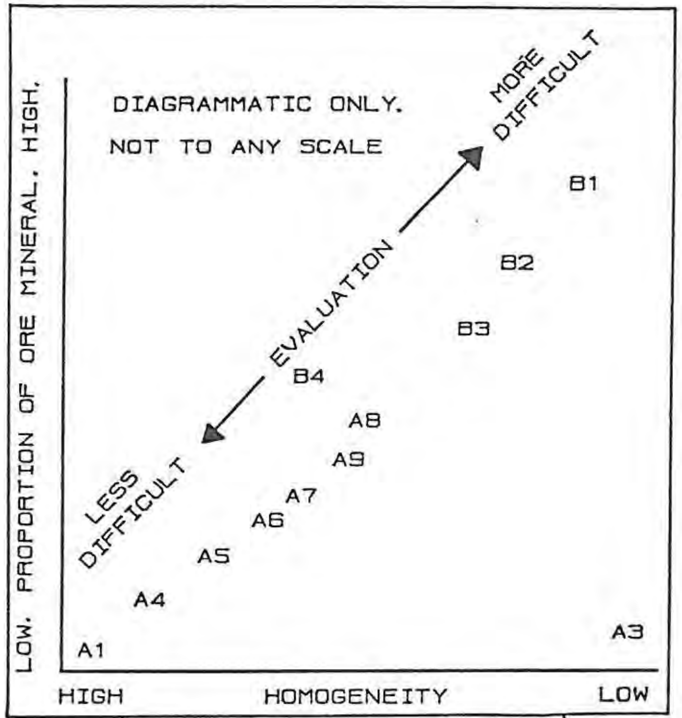
Carras (1984) indicates the importance of establishing a conceptual orebody model before any ore reserve calculations are made. An incorrect conceptual orebody model will result in a wrong ore reserve, regardless of the method used to calculate reserves. The only way to meaningfully change an ore reserve estimate, is to change the conceptual orebody model based on geological information and not mathematics.

Figures B-5 and B-6 provide an indication of the degree of difficulty associated with the evaluation of different gold deposit-types.

2.3 ESTIMATION METHODS

From the discussions in section B 1.1, 1.2 and 1.3 it is evident that no particular computational method can be regarded as the "best" method. All ore reserve estimation techniques have their limitations, and an understanding of the capabilites and limitations of the various methods is essential in order to decide upon the most applicable method.

Carras (1984) provides a comparison of the various



A1 TAILINGS DUMPS
A2 LATERITIC GOLD DEPOSITS
A3 RECENT PLACERS
A4 DISSEMINATED SEDIMENT-HOSTED GOLD DEPOSITS
A5 SKARN GOLD DEPOSITS
A6 PORPHYRY-TYPE DEPOSITS
A7 VOLCANIC HOSTED EPITHERMAL DEPOSITS (UPPER PARTS)
A8 FELSIC-HOSTED GOLD DEPOSITS
A9 VOLCANOGENIC MASSIVE SULPHIDE DEPOSITS
B1 LODE GOLD DEPOSITS
B2 VOLCANIC-HOSTED EPITHERMAL DEPOSITS
B3 PALAEOPLACER DEPOSITS
B4 PRIMARY MAGMATIC DEPOSITS

FIG B-5 PROPORTION OF ORE MINERALS VS. HOMOGENEITY OF GOLD DEPOSITS (ADAPTED AFTER BUJTOR ET AL. 1983)

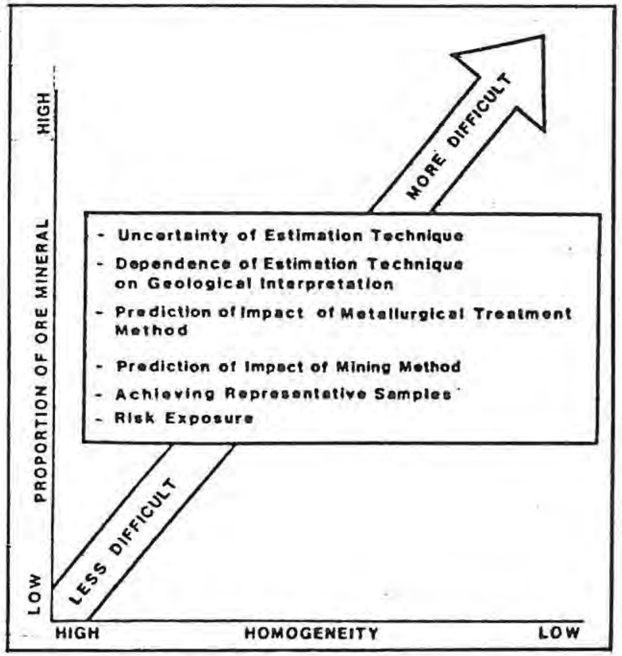


FIG B-6 GENERAL TREND OF INCREASING DIFFICULTY IN DERIVING ORE RESERVE ESTIMATES (ADAPTED AFTER BUJTOR ET AL. 1983)

techniques, and proposes a practical approach to ore reserve estimation, which incorporates aspects of conventional, statistical and geostatistical techniques.

Mackenzie (1980) proposes that conventional methods combined with classical statistical analysis be used in the early stages of an exploration programme when few data are available. Geostatistical analysis will be required for more closely spaced samples.

Carras (1984) makes an important point in that there is no mathematical method to produce confidence limits for an ore reserve estimate. Confidence limits are produced for sets of data only, and the question is therefore : how valid are the data? He concludes that the application of geostatistical methods to complex orebodies produces just as many problems (if not more) than any other method, and that for all data where the coefficient of variation is greater than 2.0 traditional ore reserve methods usually provide the best estimates.

2.4 MINING FACTORS

Dilution of the ore is one of the main causes of discrepancies between prediction and production. The extent of dilution is implicit to the mining method used, and an understanding of the applicable mining methods is therefore important in estimating this factor. Figure B-7 provides an indication of the degree of dilution associated with different mining methods.

Bujtor et al. (1983) distinguish between internal and external dilution. Internal dilution results from oversimplistic interpretations regarding the orebody outlines and homogeneity of the mineralization. External dilution may result from the following sources :

. Blasting practice used

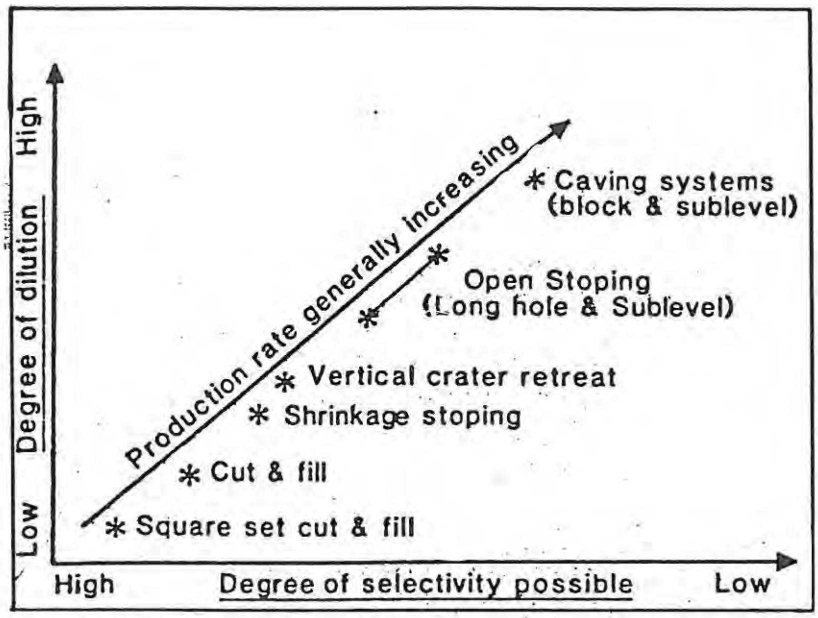


FIG B-7 THE DEGREE OF DILUTION ASSOCIATED WITH VARIOUS MINING METHODS (BUJTOR ET AL.1983)

- Dilution through fill systems
- Development waste ending up in ore passes
- Road surface "sheeting" in open pits
- Overdigging of areas
- Incorrect routing of waste and ore
- Weak hanging walls and/or footwalls

2.5 METALLURGICAL FACTORS

Bujtor et al. (1983) summarise the main metallurgical factors as follows :

- Metallurgical recovery
- The physical characteristics of the ore, i.e. hardness, grindability, abrasiveness, sliming characteristics, degree of oxidation, clay content etc.
- Mineralogy of the ore
- Petrology of the ore
- The gangue minerals and their potential effect on the metallurgical process
- The waste material and its potential effect on the metallurgical process

Estimates of metallurgical recovery are usually based on laboratory test work, and these samples may not be representative of the ore that will eventually be fed to the treatment plant.

2.6 ENVIRONMENTAL FACTORS

The obligation of a mining operation towards the environment, and the ever increasing leverage of the "green" lobby cannot be ignored, and many examples exist where mining is prohibited or discontinued due to environmental factors.

According to Thatcher et al. (1987) an environmental assessment typically addresses the geological-, hydrological-soil-, vegetation-, wildlife-, aquatic-, and cultural (historical and archeological) resources of the proposed project area. Other parameters such as air quality, noise, visual resources (aesthetics), land use and socioeconomics are usually considered as well. Table B-1 presents a summary of the types of data usually collected for preparation of an environmental assessment.

Failure to recognize and assess the environmental aspects early in an operation may lead to serious complications in the later phases.

3. THE ECONOMIC DIMENSION

The economic feasibility assessment of any mining venture involves formulating a detailed schedule of development and production expenditures, and a cash flow projection for a specified number of time periods.

Cash flow is the difference between revenues and costs for a specified time period. An annual period is usually used for evaluation purposes.

TABLE B-1 DATA COMMONLY COLLECTED DURING AN ENVIRONMENTAL ASSESSMENT (THATCHER 1987)

Physical Factors

1. Location
2. Geomorphic/physiographic
    a. Geologic hazards
    b. Unique land forms
3. Climate
4. Soils
    a. Productivity
    b. Capability
    c. Hazard
        (1) Erosion characteristics
        (2) Mass failure
5. Minerals and energy resources
    a. Locatable minerals
    b. Leasable minerals
    c. Energy sources
6. Visual resources
7. Cultural resources
    a. Archaeological
    b. Historical
    c. Architectural
8. Wilderness resources
9. Wild and scenic rivers
10. Water resources
    a. Water quality
    b. Streamflow regimes
    c. Flood plains
    d. Wetlands
    e. Ground water recharge areas
11. Air quality
12. Noise
13. Fire
    a. Potential wildfire hazard
    b. Role of fire in the ecosystem

Biological Factors

1. Vegetation
    a. Forest, including diversity of tree species
    b. Rangeland, including conditions and trends
    c. Other major vegetation types
    d. Threatened or endangered plants
    e. Research natural area (RNA) potentials
    f. Unique ecosystems (other than RNA's
    g. Diversity of plant communities
    h. Noxious weeds
2. Wildlife
    a. Habitat
    b. Populations
    c. Threatened or endangered species
    d. Diversity of animal communities
    e. Animal damage control
3. Fish
    a. Habitat
    b. Poplulations
    c. Threatened or endangered species, including State-listed species
4. Recreation resources (usually a combination of physical and biological factors)
5. Insects and diseases
6. Exotic organisms; for example Russian thistle, Siberian ibex

14. Land use including prime farm, timber, and rangeland
15. Infrastructure improvements
    a. Roads
    b. Trails
    c. Utility corridors and distribution
    d. Water collection, storage, and distribution
    e. Communications system
    f. Solid waste collection
    g. Sanitary waste collection

16. Infrastructure improvements

1. Population dynamics
    a. Size (growth, stability, decline)
    b. Composition (age, sex, minority)
    c. Distribution and density
    d. Mobility
    e. Military
    f. Religious
    g. Recreation/leisure
2. Special concerns
    a. Minority (civil rights)
    b. Historic/archaeological/cultural

3. Ways of life-defined by:
    a. Subcultural variation
    b. Leisure and cultural opportunities
    c. Personal security
    d. Stability and change
    e. Basic values
    f. Symbolic meaning
    g. Cohesion and conflict
    h. Community identity
    i. Health and safety
4. Land tenure and land use
5. Legal considerations

Revenue is obtained from product sales, salvage value and return of working capital. In the small-scale operation product sales form the most important aspect, and this is simply the product of the recovered metal (or estimate of recovered metal) and the prevailing metal price.

Costs are distinguished as capital expenditure, operating costs and taxation payments.

Cash flows are usually expressed in constant money terms.

This means that neither prices nor costs are escalated over the cash flow periods, but are held constant at a specified year value, usually the present.

Various criteria are applied to the cash flows in order to measure the economic potential of a venture, and the reader is referred to Mackenzie (1980) for a detailed discussion of these. Briefly, these include the following :

. Payback period : defined as the length of time required to recover initial investment capital. Mackenzie (1980) refers to several weaknesses with this method, the most serious of which is that it fails to take into account the time value of money

. Discounted cash flow (DCF) criteria : DCF methods incorporate the time value of money in their calculations. This is done by compoundly discounting successive cash flows by some specified discount rate. The result is that successive cash flows are expressed in present value (i.e. their would be value at the time of inception). The nett present value (NPV) is the sum of all the discounted cash flows. The internal rate of return (IRR) is defined as the discount rate which results in a zero NPV. this discount rate is also referred to as the hurdle rate, and represents the threshold below which investment would be unattractive.

It is important to note that these criteria represent analytical techniques applied to the cash flow. It is thus the variables contained within the cash flow calculations that have the most important influence on the economic viability of a prospect, i.e. those factors which influence the revenue and cost elements.

The need for accurately predicting all factors which may influence either revenues or costs is self evident. The most important of these factors are discussed in section B-2.

4. DISCUSSION

The time value of money concept forms the basis to understanding the philosophy and approach needed for small-scale operations. It is evident that the small-scale operation will have only limited resources available and the need to control costs is obvious; however, certain investment costs remain inevitable (e.g. mine development costs etc.). An important conclusion of the time value concept is that the time over which the initial investment is made should be kept at an absolute minimum. This aspect has far-reaching implications for the small scale operator.

Starbuck (1987) indicates that most successful small-mine ventures achieve positive cash flow as quickly as possible by seeking out those deposits which require the least time and expense on activities leading up to production. To do this they look for deposits that are the most accessible, easily investigated, mined and processed with the cheapest fundamental methods.

Small operators should confine themselves to known districts which favour the occurrence of the required deposit types rather than launching extensive exploration programmes.

Old, defunct mines are a prime target. Advances in technology such as higher grade recoveries, compact mobile plants and leaching methods, and changed market conditions may give new life to previously uneconomic deposits. The potential for reprocessing mill tailings, heap and in-situ leaching of dumps and mine backfill is obvious.

Although the small scale operator is described as a good all rounder in the introduction to this paper, it is clear that certain aspects would require specific expertise beyond his abilities. All manner of expertise is available to the small-scale operator on a consultancy or contract basis. These specialists provide the small-scale company with the facilities, materials and technology that traditionally only major companies could afford, and includes expertise in the following disciplines :

. Geology specific expertise, e.g. structure, hydrology etc.
. Mining expertise, e.g. mining engineers, mine contractors, safety specialists, metallurgists
. Legal aspects, e.g. mineral lawyers, mine accountants
. Engineering expertise, e.g. electrical, mechanical, civil engineers and surveyors

Starbuck (1987) concludes that a well planned venture can earn back the cost of good advice many times over.

In general it may be concluded that the small-scale company cannot afford the luxury of extended and exhaustive feasibility studies. Production may have to be initiated on limited "proven" ore reserves, and continued viability will depend greatly upon "probable" reserves. This greatly increases the risk involved in these ventures, however, it is maintained that the experienced exploration geologist is the only specialist able to fully comprehend all aspects of the orebody, mineralizing processes and controls, and therefore the only person qualified to assess "probable" and "possible" reserves, and the risks involved.

The multitude of varied and complicated aspects which need to be considered in the evaluation of a prospect emphasises the need for a systematic approach to the problem. Many examples of failures exist where key issues were either not recognized or their influence on the overall viability not fully appreciated.

One such a systematic approach is provided by risk analysis. A detailed discussion of the implementation of a risk analysis system falls beyond the scope of this paper, and the reader is referred to Mackenzie (1980) and Mallinson (1987) for such discussions.

In risk analysis subjective models are set up as discrete or continuous probability distributions for each quantifiable factor. Successive estimates of profitability are then made on the basis of mathematical procedures, the most common of which is a Monte Carlo simulation technique using randomly selected numerical values from each of the probability models. Results obtained from risk analysis are probabilistic rather than deterministic, and therefore provide a measure of the uncertainty, or risk involved.

Mallinson (1987) has shown the relative ease with which a risk analysis template may be set up on a PC using one of the spreadsheet software packages available. It is important to note that the input variables in such a system would be unique to small scale operations, and therefore would require careful consideration. Mallinson (op.cit.) further shows how such a system may be adapted to yield a sensitivity analysis, thus providing an indication of the most sensitive determining factors.

SECTION C

MINING AND BENEFICIATION METHODS

The philosophy of small-scale mining as discussed in section B-4 has important implications for the choice of mining method. This section is not intended as a review of existing mining methods. For this the reader is referred to Crawford et al. (1979), Hustralid (1982) and Maturana Bascopé (1983). Rather it is intended as a general discussion of some of the basic principles applicable to small-scale mining. Beneficiation processes remain basically the same, albeit on a different scale, for small- and large-scale operations. Certain beneficiation processes are however more amenable to small-scale application, and section C-2 provides a general review of these processes.

As each orebody is unique, each would require its own method or combination of methods, best suited or adapted to its own specific set of circumstances.

1.MINING METHODS

In general, mining methods have evolved in order to utilize the economies-of-scale principle whereby production costs are offset by increased production rates. This has, however, led of the fact that certain high productivity mining methods have developed beyond the means of the small-scale operator, i.e., they require an initial capital investment far exceeding that which would be available to the scale of company envisaged. This is especially true of certain underground mining methods.

With respect to the basic principles employed, mining methods may be divided into the following categories:

- Surface mining methods, including open pit and dredging operations,
- Naturally supported stopes, including room-and-pillar mining and sublevel stoping,
- Artificially supported stopes, including shrinkage stoping, cut-and-fill mining, square-set mining and longwall mining,
- Caving methods, including sublevel caving and block caving.

Nilsson (1982) divides ore deposits into four categories with respect to applicable mining methods:

- Deposits suitable for surface mining only
- Deposits suitable for initial open pit mining, followed by underground mining
- Deposits that are too deep to be considered for open pit mining
- Deposits that have previously been worked by underground methods, and due to technology/price changes are now suitable for open pit mining.

It is evident that the category in which any specific deposit resides would have a major influence on the overall planning of the mining operation.

Table C-1 summarizes the most important factors which will influence the choice of mining method. The final choice of mining method will depend upon the economics of the project, and therefore detailed cash flow analyses of all available options are required before this choice is made.

The time-value of money concept dictates that production levels be achieved as soon as possible, and this aspect will obviously have a major influence on the planning and design of a small-scale operation.

TABLE C-1 FACTORS IN THE CHOICE OF MINING METHODS (PETERS 1981)

Physical	
Geometry	Size, shape, continuity, and depth of the orebody or group of orebodies to be mined together
	Range and pattern of ore grade
Geology	Physical characteristics of ore, rock, and soil
	Structural conditions
	Geothermal conditions
	Hydrologic conditions
Geography	Topography
	Climate
Technologic	
Safety	Identification of hazards
Human resources	Availability of skilled labor
Flexibility	Selectivity in product and tonnage
Experimental aspects	Existing or new technology
Time aspects	Requirements for keeping various workings open during mining
Energy	Availability of power
Water requirements	
Surface area requirements	
Environment	Means of protecting the surface, water resources, and other mineral resources
Economic	
Cost limits	
Optimum life of mine	
Length of tenure	Prospects of long-term rights to mine

The most viable targets would therefore comprise those that require the least pre-production development.

Rieber (1987) indicates that the cost of the capital, or more specifically the high real interest rates, in less developed countries may offset the technical economies of scale, and where initial capital investment is high relative to production costs, smaller scale operations requiring less investment will tend towards lower costs and higher profits. He further suggests that new "capital-using" techniques are required for small scale mines. These techniques include correct mechanization for optimum productivity and maximum cost effectiveness, and combining development with production by driving drifts in the orebody where this is feasible. All the traditional underground mining methods may be modified for in-the-orebody development (Rieber, op. cit.)

A small-scale operator may consider combining open pit and underground methods in orebodies so suited. The use of lower cost open-pit methods in the initial stages of a mining operation may reduce the required capital investment, and may have the same effect as early "high-grading" in order to shorten the period of negative cash flows. This practice may however present some problems: In order to utilize the advantages of economies of scale, the open pit operation needs to be of a specific optimal size, and the size of operation is in turn determined by the capacity of the processing facilities.

Open pit operations are especially amenable to contract mining, thereby reducing the required initial capital investment and enabling the small operation to utilize the economies-of-scale principle. However, plant capacities are not easily (or cheaply) changed. This may be overcome in part by using different initial processing methods, for example heap leaching. Heap leach processing may not result in optimal gold recoveries, but it does represent a low cost means of treating large tonnages of ore, and the ore remains available for re-treatment at a later stage (if economically feasible), once positive cash flows have been attained. It is evident that the use of contractors in small-scale operations has great potential. It is however important that all options be carefully considered, and the economic feasibility of each option accurately calculated and compared.

Old worked-out mines represent a prime target for the small-scale operator. Old mines comprise established infrastructure of shafts, drives, drifts etc. The mining methods employed by the large companies which originally mined them invariably accentuated tonnage rather than grade which often results in considerable amounts of ore being left behind. Krige (1981) discusses the methods employed in the reclamation of gold from worked-out Witwatersrand mines by the Boshoff group of mines. This small-scale company has been highly successful, and in the ten years preceding 1981 produced gold to the value of R65 million from abandoned

Witwatersrand mines. The method basically consists of physically removing every last piece of ore left in the hanging wall, footwall and pillars by hand. Old stopes, panels and drives are carefully swept to recover all fines and crushed ore that may have been left behind. The ore is hand sorted underground and only ore is brought to surface, the waste being packed in abandoned workings underground. Equipment and old mine timbers are also treated for the gold they may contain. The method is labour intensive but also highly selective, and its potential, especially where a visual grade control can be applied is obvious.

2.BENEFICIATION METHODS

The following characteristics of gold affect its recovery:

- Very high specific gravity. (15,5-19,32 depending on the amount of alloyed metal admixed).
- Mercury readily wets gold in the presence of water (amalgamation).
- Gold is soluble in dilute aqueous solutions of alkaline cyanide.
- It responds to flotation collectors.

A major factor in determining the recovery of gold is the nature or habit of the gold mineralization, i.e. whether the gold is present as free gold and readily liberated by milling, or whether it is present as small inclusions in sulphidic ores, referred to as free milling and refractory respectively.

Historically and prehistorically gold has been recovered by gravity separation methods. The earliest methods employed on the South African mines include combinations of gravity and amalgamation methods.

In the latter part of the 19th century it was discovered that gold could be dissolved in a cyanide solution and reprecipitated on zinc. This process was implemented on the Witwatersrand gold mines towards the turn of the century, and probably represents the greatest revolution in recovery techniques of gold. More recently it was found that gold dissolved in cyanide solutions is readily adsorbed on carbon The carbon is then treated by a hot caustic - cyanide solution and the gold recovered either by precipitation on zinc or electrowinning processes. This resulted in the carbon-in-pulp (CIP) and carbon-in-leach (CIL) processes. A further application of the cyandiation process is the treatment of ores by heap leaching.

Refractory gold ores require initial flotation to produce a sulphide concentrate, which is then treated by roasting in order to liberate the gold. The gold is then recovered by the usual cyanidation process. The roasting process is complex and costly and is therefore considered as a last option for small scale operations.

2.1GRAVITY CONCENTRATION METHODS

Gravity concentration methods have always been the primary conventional method of gold concentration from alluvial placer deposits. In the past these methods have also been applied to a more limited extent to free milling ores.

It is important to note that despite the high SG of gold, the effects of particle shape and hydrophobicity of fine gold particles can significantly reduce the SG advantage for gold separation systems.

The most widely used gravity equipment has been jigs, riffled tables, shaking tables and sluices. These methods are successful to varying degrees, but all have one serious deficiency: they fail to recover very fine-grained gold.

Richards et al. (1984) indicate that recent developments in gravity concentration methods show significant improvements in fine particle benefication. These include cone concentrators, spiral separators and centrifugal gravity systems.

The Reichert cone concentrator is a high-capacity low-cost gravity concentrator designed for the treatment of material finer than 2mm in size, and to recover fine heavy mineral particles down to 38 microns in size. Figure C-1 shows a sectional view of the Reichert cone concentrator. The separation mechanism involves a combination of hindered settling and intergranular trickling, resulting in a stratified flowing bed in which the heavy and fine particles are concentrated at the bottom of the pulp stream.

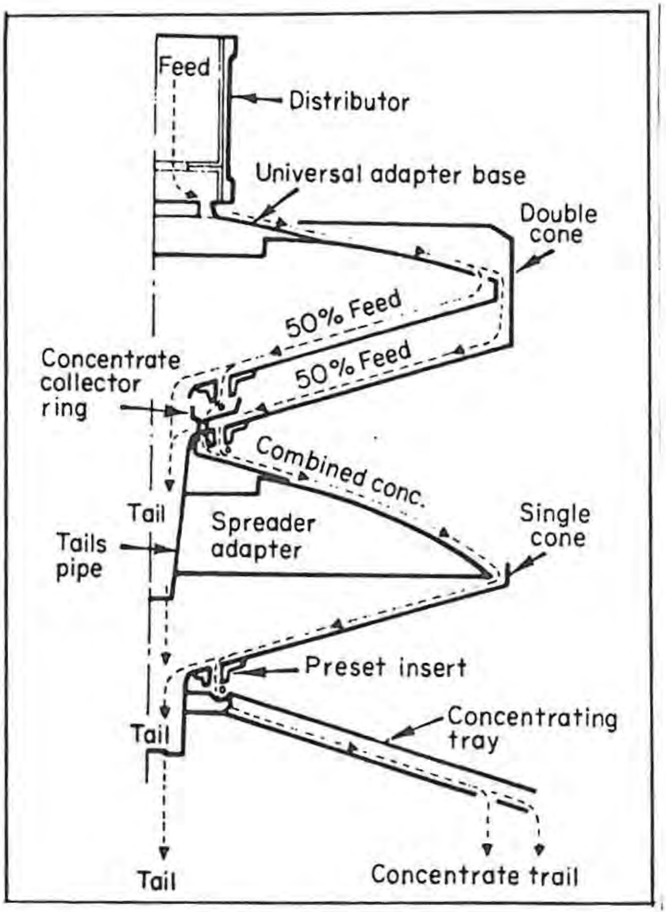


FIG C-1 CROSS SECTION THROUGH REICHERT CONE CONCENTRATOR SYSTEM. (WILLS 1979)

In the spiral concentration, pulp is fed in at the top of the spiral, and as it flows spirally downwards the coarsest and heaviest particles concentrate in a band along the inner side of the stream (see figure C-2). Ports for the removal of the higher specific gravity particles are located at the lowest points in the cross section. The more recent wash-waterless spirals have significantly increased the performances attainable.

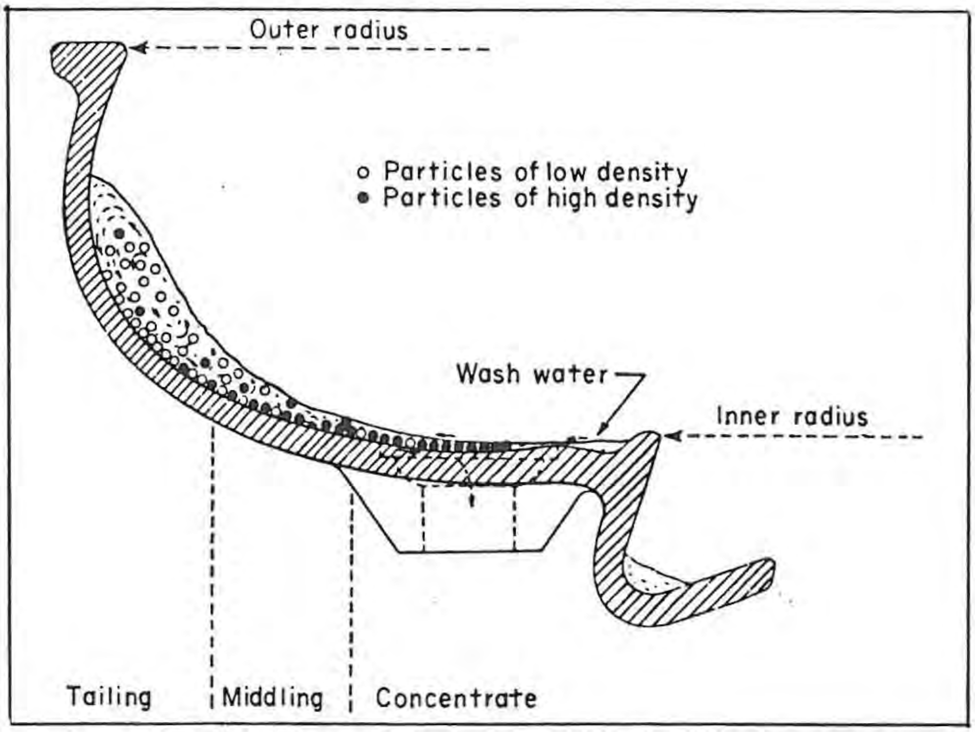


FIG C-2 CROSS SECTION OF A SPIRAL STREAM (WILLS 1979)

The Knelson concentrator is a centrifugal gold concentrator which utilizes the combination of high gravitational forces and a unique fluidizing action to recover free gold particles from minus 6mm to fine micron gold in both alluvial and hard rock applications (Harris, 1984) Figure C-3 shows a schematic cross section of the Knelson concentrator. The fluidizing action involves injection of water through holes in the inner bowl of the concentrator. This prevents compaction of the concentrate bed, thereby allowing even "flour" gold to penetrate the bed under the influence of high "G" forces.

Figures C-4, C-5 and C-6 show the implementation of gravity concentration systems in processing circuits.

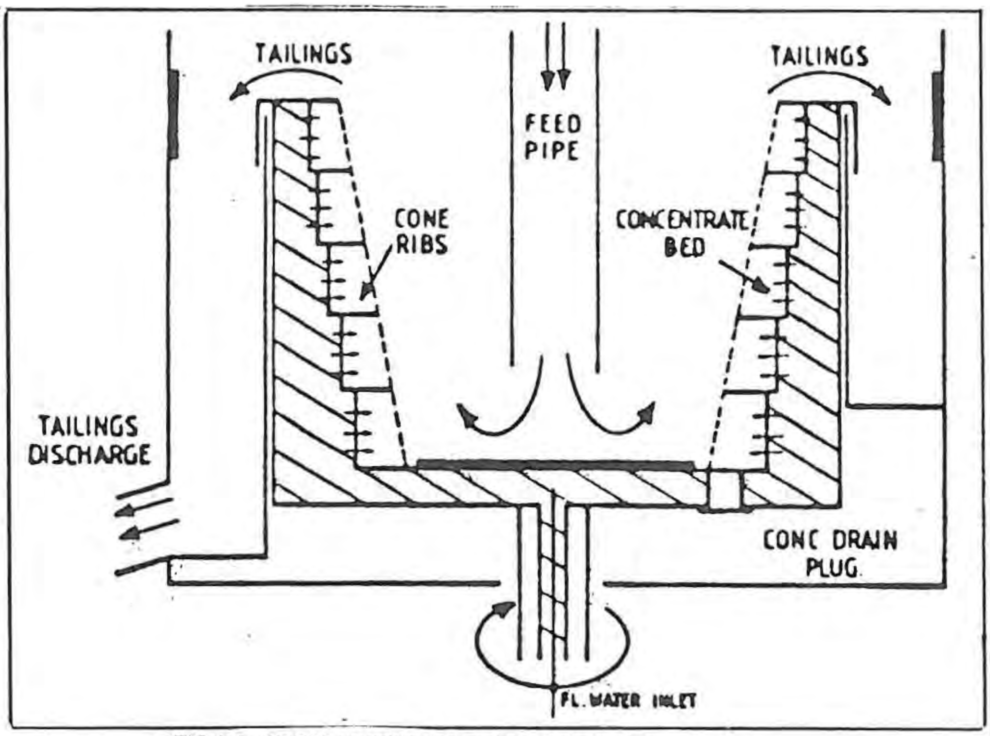


FIG C-3 CROSS SECTION OF A KNELSON CONCENTRATOR (HARRIS 1989)

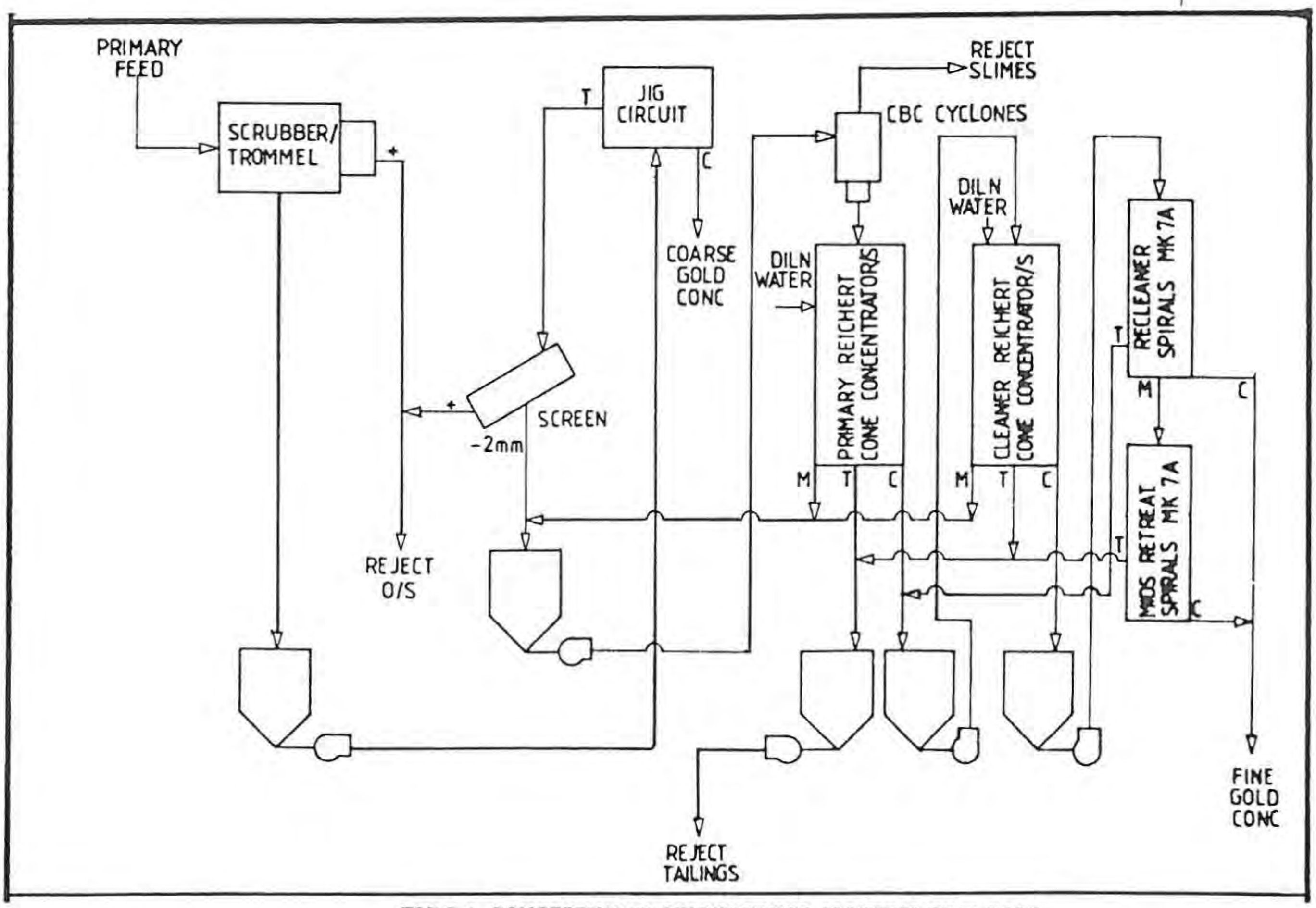


FIG C-4 CONCEPTUAL FLOW SHEET FOR ALLUVIAL FINE GOLD RECOVERY (RICHARDS ET AL. 1984)

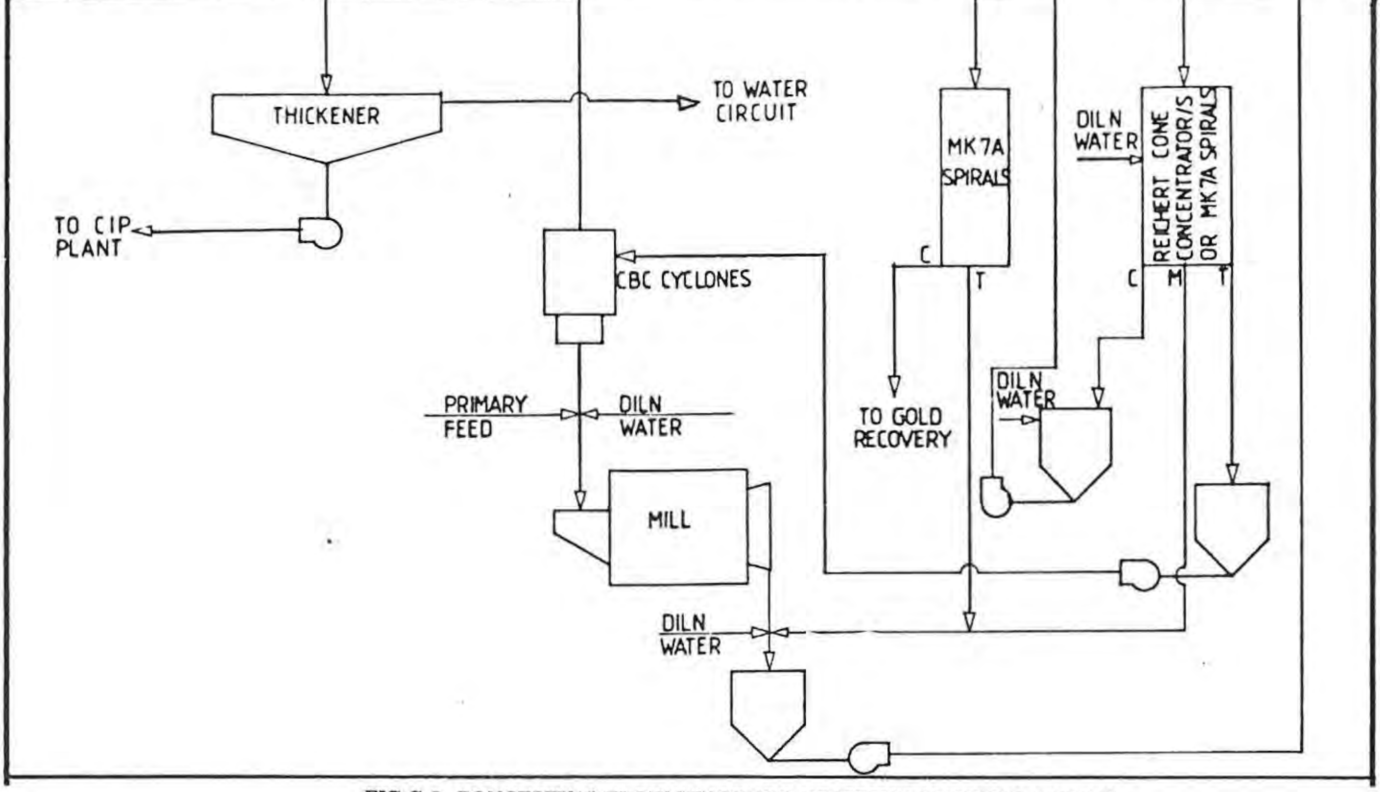


FIG C-5 CONCEPTUAL FLOW SHEET FOR GRAVITY CONCENTRATION IN A GOLD GRINDING CIRCUIT (RICHARDS ET AL. 1984)

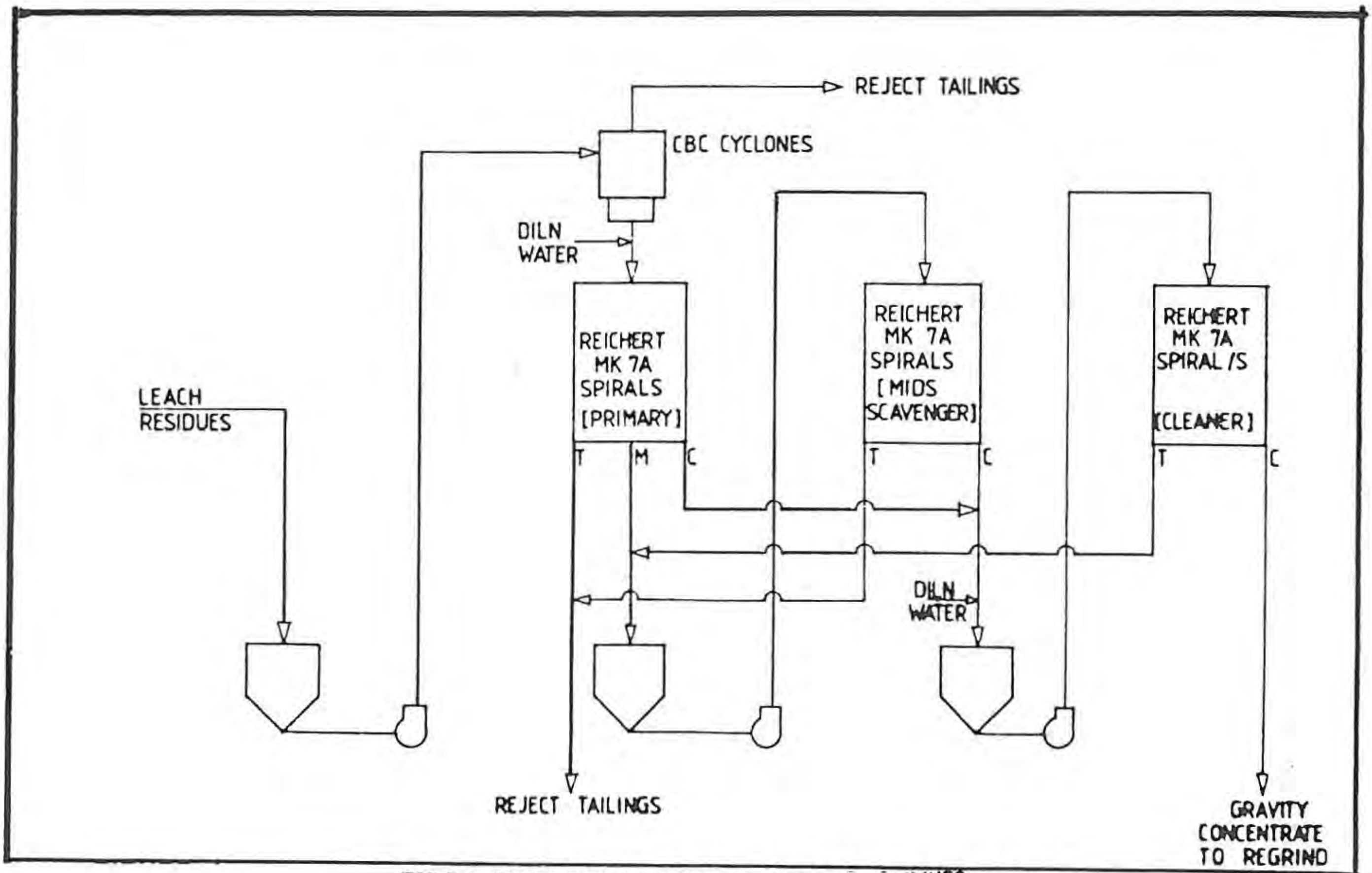


FIG C-6 CONCEPTUAL FLOW SHEET FOR GOLD LEACH PLANT TAILINGS SCAVENGING BY GRAVITY CONCENTRATION (RICHARDS ET AL. 1984)

2.2CYANIDATION PROCESSES

Gold and silver, and their tellurides are soluble in aerated dilute solutions of NaCN or KCN. This takes place by the following electro-chemical corrosion process.

$$4\ Au + 8NaCN + O_2 + H_2O = 4NaAu(CN)_2 + 4NaOH$$

In practice, the finely ground ore and water mixture is pumped into a tank to which 0,02 - 0,08% (volume) of the cyanide solution is added. The pulp is continually agitated by bubbling air through from below. The pulp is kept alkaline during the process to prevent the production of HCN gas. The process is continued for 6-72 hours, depending on the nature of the ore. The rate of dissolution of the gold depends on the following factors:

- Particle size; under ideal conditions a 45 micron grain will dissolve completely in 13 hours. A 150 micron grain will take approximately 44 hours,
- Degree of liberation; the rate at which gold will dissolve is directly proportional to the exposed surface area,
- Silver content.

Conventionally the gold-bearing solution is retrieved by thickening and filtration of the pulp. The gold is precipitated by adding zinc dust to the solution in a process known as the Crowe-Merrill procedure. This process has been developed to the point where the gold content of the barren solution is less than 0,01 g/t (Loxen et al. 1979).

Although the adsorption power of charcoal for dissolved gold has been recognized since the late 18th century, it was not until the 1950's that interest in the application of activated carbon was revived when a procedure was developed for the elution of gold and silver adsorbed on carbon granules (Habashi, 1987).

In the carbon adsorption process activated carbon is mixed with the pulp in a series of tanks or adsorption contactors, hence the name carbon-in-pulp (CIP). The carbon is made to flow in a counter-current to the cyanide pulp in order to achieve optimal mixing. Figure C-7 shows a conceptual flowsheet for a CIP circuit. Elution refers to the process whereby the adsorbed gold is stripped from the carbon by hot caustic cyanide. The gold is recovered by either precipitation with zinc dust or electrowinning, where gold is collected on steel-wool cathodes (Laxen et al. 1979).

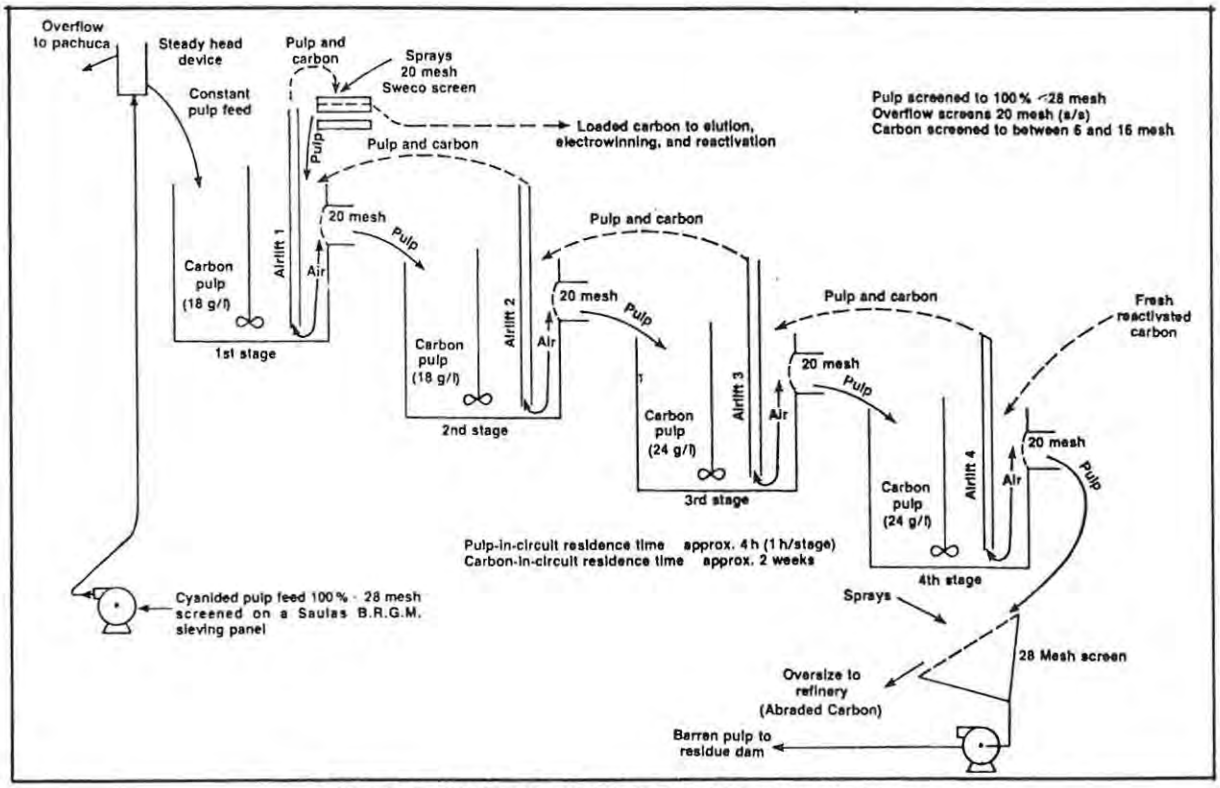


FIG C-7 CONCEPTUAL FLOW SHEET FOR A CIP CIRCUIT (LAXEN ET AL. 1979)

The carbon is reactivated by heating to a temperature of 650°C for about 30 minutes in the absence of air, and is then allowed to cool in air.

The most important aspect which affects the rate of extraction of gold is the mixing efficiency within a CIP contactor. Other factors include the mean particle size of the carbon granules, and the possible presence of organic or inorganic compounds which may lower the adsorption efficiency of the carbon (Fleming et al. 1984).

Because the carbon-in-pulp process eliminates the necessity for filtration and thickening, it has an advantage when applied to any ore where these two processes present difficulties. The CIP process also has an important advantage over the conventional zinc precipitation process where the ore contains soluble nickel and copper minerals. These elements can interfere with the zinc precipitation but do not affect the adsorption of gold by carbon.

Most processing circuits incorporate both gravity methods and cyanidation processes.

2.2.1HEAP LEACHING

Heap leaching provides a low cost option for the extraction of gold. Basically the process involves applying alkaline cyanide to the ore in bulk, and processing the effluent to extract the gold.

Menne (1984) indicates several options of heap design, namely:

- Normal heap design, where a free-standing heap is built to a height dictated by the available materials handling system,
- Flooded heap (vat) design, where the heap is contained within impermeable walls allowing fully flooded operations. This is generally necessary in cases where the ore lacks cohesiveness on wetting,
- Thin-layer design, where only a relatively thin advancing ore face us leached. This is used where ore constituents rapidly decrease the capability of the leachate to dissolve the gold.

In ore other than fine-grained tailings a certain degree of comminution is required. Menne (1984) suggests a firkin roll test in order to determine the optimum feed ore size. Conversely, some fine grained or clay rich ores may seriously inhibit the penetration of the leachate. These ores have to be pre-treated with substances which bind the particles, referred to as pozzolonas (Menne op. cit.) in a process described as agglomeration.

Heap-leach processing requires liners in order to contain the leaching solutions for both economic and environmental incentives. These liners may be of natural (clay) or synthetic materials. Cadwallader (1987) provides a detailed discussion and comparison of leach pad liners.

Important factors which may influence the recovery of gold by heap-leach processes include the amount of cyanide solution per unit of ore passed through the heap, the distribution of the leachate, and water quality. The reader is referred to Menne (1984) for a detailed discussion of the influences of these factors.

The main gold precipitation process applied in heap leaching is electrowinning, either direct or following prior concentration by carbon. In this case carbon is mixed with the pregnant solution in a process referred to as carbon-in-leach (CIL).

3.DISCUSSION

Most gold recovery circuits generally include both gravity concentration processes and cyanidation processes.
The ultimate design of the processing facility will obviously depend on the nature of the orebody. There are however important factors that need to be considered by the small- scale operator. Small-scale mines are by implication relatively short lived, and this feature should reflect strongly in the design of the processing plant. Mobile and/or movable equipment is essential. Barua et al. (1987) estimate the cost of 75 mtpd caravan mill mounted on trailers and incorporating a complete CIP - cyanidation circuit at approximately U.S.$ 2.5 million (1987 dollars), compared to a similar nonportable facility at approximately U.S. $ 0.8 million. However, the primary advantage of the portable facility is that it can be moved to a new location within days and a new operation started more or less immediately with instant generation of cash flow.

A further option for the small-scale operator is the initial use of a cheaper processing method (e.g. heap leaching) in order to either minimize initial capital investment or to generate capital for more detailed exploration and delineation of the orebody. This, however, will depend upon the nature of the mineralization.

SECTION D

SMALL SCALE GOLD MINING IN SOUTHERN AFRICA

The constraints discussed in the preceding sections adequately define the small-scale operator's approach to gold mining in general, and the deposit types most suited to small-scale mining.

To briefly recap: the small-scale company cannot involve itself in extended and extensive exploration activities, and the most suitable deposits are those that require the least pre-production time and effort.

A further major concern for the small-scale company is the mining and taxation laws of the country in which operations are intended.

South African law vests the ownership of mineral rights in the individual. Moreover, mineral rights ownership can be alienated from land ownership. In the case of precious metals, diamonds, uranium and oil the mineral rights owner must obtain the right to exploit these commodities from the state. A lease payment or royalty is associated with the right to mine (Beukes, 1986). This law is advantagous to the large-scale company which has the available resources to procure ownership of mineral rights of large tracts of land thereby securing large portfolios of potential economically viable ore deposits and effectively cutting out the small-scale company.

In the newly independent developing countries of Southern Africa, mineral rights ownership is generally vested in the government or state, and this may improve the circumstances for small-scale companies. However, bureaucratic approval to explore or mine may be time consuming and conducive to corruptive practices. Coupled to this is the fact that these governments generally impose some restrictions regarding equity, management, import and export, and repatriation of dividends (Beukes op. cit.).

Extensive reserves in the form of tailings dumps are located in the historical mining areas of Zimbabwe and South Africa. The dumps of the Witwatersrand may be more suitable for reclamation because of the nature of the primary mineralization being more free milling than refractory. Tailings of mines located in Archaean greenstones may contain significant amounts of refractory gold which may complicate their reclamation, although the degree of oxidation of the sulphide ores, and hence the age of the dump, may prove to be the most decisive factor in selection.

The age of the dump further determines the level of extraction technology prevalent at the time of mining. It is therefore evident that the older dumps represent more attractive targets.

It has been shown that the reclamation of gold from old, abandoned mine workings represents a very viable target for the small-scale company. There are however certain requirements for such an orebody, the most important of which is that mineralization should be relatively high grade, continuous and should allow for visual grade control. The defunct mines of the Witwatersrand offer the most viable targets for such exploitation. Similar deposits in the Pongola group may have some potential but the average grade of less than 5g/ton (Saager et al. 1986a) appears too low. Archaean greenstone-hosted gold deposits represent additional problems to such reclamation methods in that these orebodies are generally discontinuous, and the mineralization itself is not as visible. It is, however, important that each "orebody" or defunct mine be assessed on its own specific characteristics.

Section B-4 emphasized that small scale companies should confine themselves to known mineral districts or metallotects. In this regard the Archaean granite - greenstone terranes of Southern Africa represent the greatest potential for targets amenable to small-scale exploitation. Figure D-1 illustrates the distribution and extent of exposed Archaean granite - greenstone terranes in the Zimbabwean and Kaapvaal cratons.

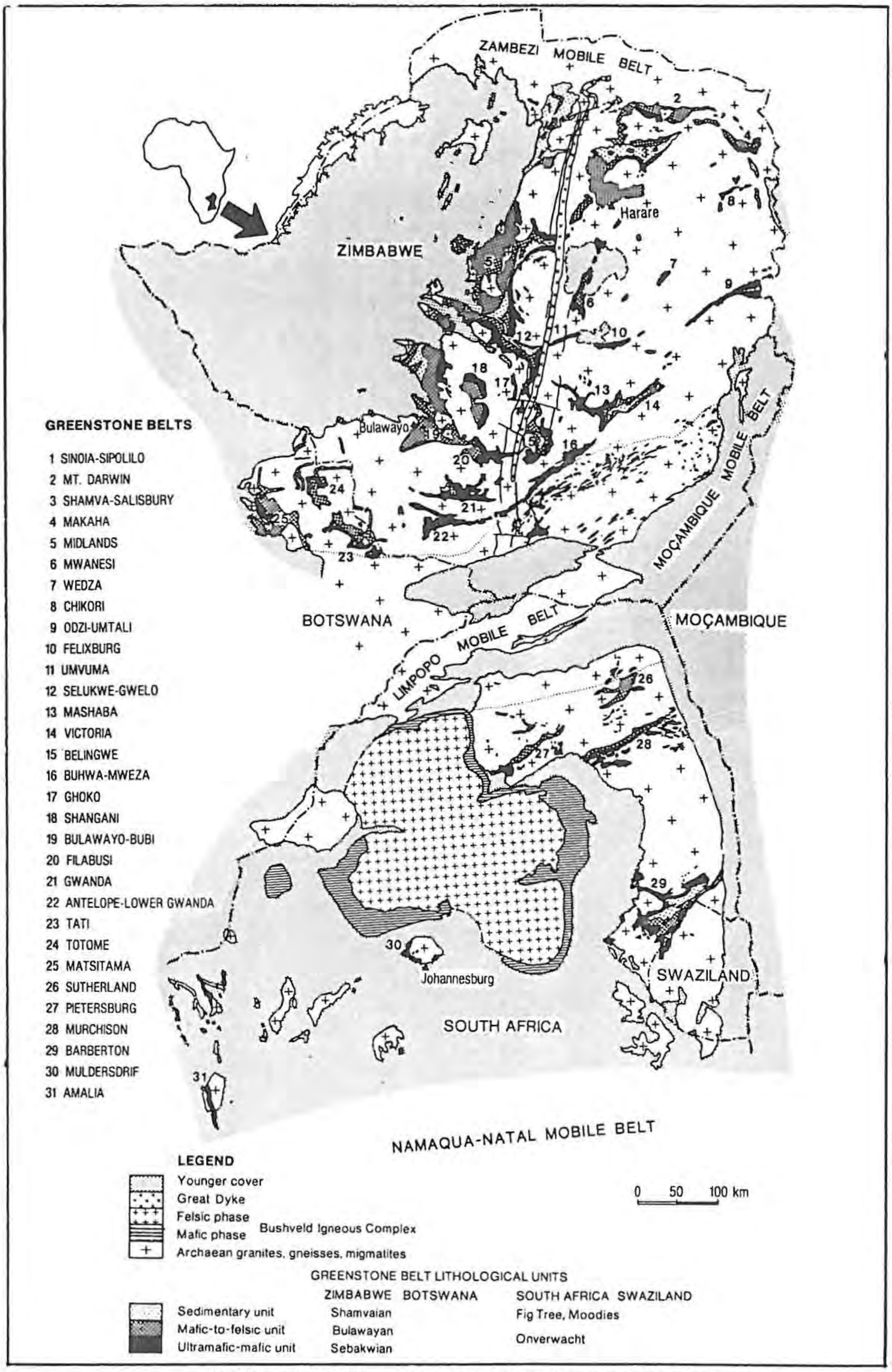


FIG D-1 THE EXPOSED GRANITE-GREENSTONE TERRANE OF THE ZIMBABWEAN AND KAAPVAAL CRATONS (ANHAEUSSER ET AL. 1986)

The Archaean terrane of Zimbabwe represents the most productive of its kind in the world in terms of gold yield per square kilometer (Bartholomew, 1990). Bartholomew (op. cit.) states that approximately 6000 mines have declared some production since the turn of the century, most of these are very small deposits. Most of the gold production comes from vein and shear-zone hosted deposits. Bartholomew (op. cit.) describes the vein-hosted deposits as tabular or lenticular bodies of quartz with minor amounts of carbonate minerals and small quantities of sulphide minerals. Shear zone deposits are tabular orebodies of numerous quartz-carbonate veinlets, with more abundant sulphides and a greater proportion of refractory ore. Table D-1 gives an indication of the general tenor of vein and shear zone-hosted gold deposits of Zimbabwe.

	Vein Deposits (g/t)			Shear Zone (g/t)		
Host Rock	No	$\bar{x}$	S	No	$\bar{x}$	S
All deposits	185	12,0	5,8	67	9,4	5,8
Granitic	44	10,4	4,4	15	7,5	5,5
Sedimentary	17	14,9	7,9	6	11,2	7,6
Felsic	7	8,8	2,6	8	12,0	9,2
Mafic	107	12,4	5,6	35	9,5	4,6
Ultramafic	10	12,2	8,3	3	3,0	3,4

Based on data from mines which have produced more than 311 kg gold.

TABLE D-1 TENOR OF ARCHAEAN NON-STRATABOUND GOLD DEPOSITS OF ZIMBABWE (FOSTER ET AL. 1986)

A significant proportion of Zimbabwean gold production has been from "stratabound" deposits, where the gold is associated with particular rock units. These include mainly BIF associated orebodies, and deposits in felsic pyroclastic sediments (Bartholomew, 1990). Foster et al. (1986) further include banded sulphide and clastic hosted deposits in this group. The tenor of these deposits ranges from 2g/t to 17g/t, with the BIF associated deposits generally above 5g/t. Bartholomew (1990) provides a list of 700 gold deposits in Zimbabwe, together with their localities, tenor, host rock, mode of occurrence and production data.

Gold has been mined for over a century in the Barberton Mountain land, and production has been recorded from over 350 gold occurrences in this area (Anhaeusser, 1986). More than 95% of the gold has been recovered from the north-west flank of the Barberton greenstone belt, with 2,7% mined in Swaziland. Anhaeusser (op. cit.) distinguishes three types of gold mineralization in the Barberton greenstone belt ie. complex refractory ores, gold-bearing quartz veins and oxidized ores. The proportion of complex refractory ores ranges from approximately 70% to 14% and therefore has a major influence on the method of beneficiation. Gold bearing quartz veins contain free milling gold and occur in almost all of the gold deposits in the Barbertone greenstone belt. There is thus no distinction in the occurrence of these two types of mineralization, and most deposits contain both refractory ore and free milling quartz-vein associated gold. The third mineralization type represents supergene enrichment in the upper oxidized parts of deposits.

Anhaeusser (1986) provides a list of all the known gold occurrences in the Barberton Mountain Land, together with their localities, stratigraphic position, host-rock setting, ore minerals and gold and silver production data.

Gold is also recovered from other Archaean remnants in the Kaapvaal craton. The Eersteling goldfield in the Pietersburg greenstone belt (see figure D-1) is the site of the first lode-gold discovery in South Africa in 1871. Saager et al. (1986b) distinguish four types of gold occurrence in the area: gold-quartz veins; impregnated sheared schists; sulphidic banded iron formations and alluvial deposits. The gold quartz veins carry mostly free milling gold, whereas the gold-bearing schists have a larger proportion of refractory gold. A significant proportion of the past gold production has been from supergene enriched eluvial deposits (Saager et. al. 1986b). Saager et al. (1986c) further report a small low-grade poleoplacer deposit in conglomerates of the Uitkyk formation within the Pietersburg greenstone belt.

Potgieter et al. (1986) report on the gold occurrence in the Sutherland greenstone belt (see figure D-1). Most of this gold is refractory, and 74% is either included in or attached to arsenopyrite.

Gold occurs in association with antimony in the Murchinson greenstone belt (see figure D-1). Pearton et al. (1986) distinguish four types of auriferous reefs in the area. The highest gold values are found in irregular antimonial quartz-carbonate veins along the Antimony Line. Most of the gold is refractory.

Gold was discovered in the Tati greenstone belt of Botswana as early as 1866, making this the first goldfield ever worked by european miners in Southern Africa (Keys 1976). Keys (op. cit.) reports that gold occurs in five types of deposits:

- Quartz reefs in granitoid rocks adjacent to the Tati schists,
- Quartz reefs along granite/schist contacts,
- Quartz reefs within the schists,
- Fissure vein systems consisting of numerous en echelon veins,
- Eluvial deposits.

Molyneux (1971, in Keys, 1976) records grades of between 30 and 120 g/t in the upper 15 metres (presumably the supergene enriched zone), and 5 to 20g/t in the oxidized zone. Keys (op. cit.) notes that the old workings are mainly in the oxidized zone, and rarely extend below the water table. During the period between 1933 and 1955 the Tati gold field produced 6 265 111g of gold from 944 700 tons of ore, indicating an average grade of 6,63 g/t. Keys (op. cit.) provides a summary of the relevant features of sixteen of the most important mines in the area.

Apart from the potential for reclaiming gold from tailings dumps and old mines in the Archaean terrains of Zimbabwe, Botswana and South Africa, there is also the possibility of extending existing defunct mines or establishing new mines, on a small scale. Given the extended history of gold mining in the region, one would imagine the chances of finding supergene enriched eluvial or lateritic deposit types very small. There is however a major potential for recovering alluvial gold from the major river systems draining the gold-rich areas.

Relatively few gold occurrences are reported from Namibia. These include occurrences at Otjivasandu in the Kamanjab inlier; Kobos, where gold is associated with pyrite, arsenopyrite, chalcopyrite and bornite in chloritized shearzones located in volcano sedimentary layers of the Elim formation, Rehoboth inlier. Phanerozoic lode gold-type vein mineralization at Ondundu, and skarn-tye gold minealization at Navachab. (Killick 1986). Of these the vein type mineralization at Ondundu probably represents the greatest potential for small-scale exploitation.

CONCLUSION

Gold as a mining commodity offers some unique characteristics which make it especially amenable to small-scale production. It has a high value per unit mass ratio. It is relatively easily beneficiated and beneficiation yields an easily marketable product.

The small-scale mining company requires a different approach and perspective to that of large companies. By definition small-scale companies will mine small-scale deposits, and small, high-grade deposits constitute the most desirable targets. However, one may well question the availability of such deposits. The small-scale operator therefore requires a "think smart" attitude (Starbuck 1987) in order to mine lower grade deposits profitably.

To achieve this the small scale-operator cannot "reinvent the wheel". He must use proven exploration, mining and processing techniques, along with economic evaluation and experienced personnel, and concentrate on known areas of mineralization.

The main objective is profit, and therefore a comprehensive economic assessment forms the basis of all decisions. It is vital that all aspects which may influence costs in any way be recognized and assessed as early as possible. Geological factors usually have the greatest influence on costs, thus emphasizing the role of the geologist in a small scale company.

The targets of highest potential are those that require the least pre-production time and expense. In this regard reclamation of gold from tailings dumps and abandoned mines represent prime targets for the small-scale company.

Gold mining on a small scale in the Southern African region has great potential. The prolonged history of mining in Botswana, Zimbabwe, Swaziland and South Africa implies extensive reserves in the form of tailings dumps and abandoned mines, suitable for the small scale reclamation of gold.

Increasingly, large companies in South Africa are processing their own tailings dumps. Also, the reclamation of gold from dumps has become popular in South Africa, resulting in a number of small companies involved in the process. Furthermore, the mineral rights law at present does not particularly favour the establishment of small-scale gold mining in South Africa. In this regard the neighbouring states may represent more suitable environments. There are however potentially serious problems in these states in the form of bureaucratic intervention and legal restrictions which need to be considered.

Although it has been emphasized that the small-scale company cannot involve itself in extended exploration programmes, it is evident that some pre-production exploration is essential. It is important to note that substantial amounts of information in the form of historical mining records etc. should be available from the appropriate authorities. Mining operations may have to be launched on limited proven reserves, accentuating the role of the experienced geologist.

ACKNOWLEDGEMENTS

I wish to thank all the staff members of the Geology Department at Rhodes University who taught me during the time I spent in post-graduate studies. All I know of geology, I learnt here.

In particular I wish to thank Professor Franco Pirajno for encouraging me to do the M.Sc. course, and for his enthusiastic support during the course. Professor Roger Jacob is gratefully acknowledged for academic guidance, moral support and patience throughout both the Honours and M.Sc. courses. Clyde Mallinson is thanked for revealing a whole new dimension of geology to me, for friendship, and for endless discussions, both relevant and irrelevant.

I further wish to thank the management of Rio Tinto Exploration for affording me the time in which to complete this dissertation, and the management of Foskor for financial assistance.

Above all, I wish to thank my wife Elna for her support and endless encouragement, without which nothing would have been possible.

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