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GUIDELINES TO THE EVALUATION OF SELECTIVELY MINED, OPEN PIT GOLD DEPOSITS DURING THE EXPLORATION STAGE OF MINE CREATION.

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ABSTRACT

This dissertation studies the evaluation of selectively mined, open pit gold deposits during the exploration stage of the mine's life. Since 1970 a large number of selectively mined, open pit gold mines have come into operation. The most common deposits include epithermal vein, mesothermal lode and laterite gold deposits. In general the deposits are characterized by small tonnages (1-20 million tonnes), relatively high grades (2-10 grams per tonne gold), submicroscopic to coarse gold, inexpensive mining, and both free milling and refractory ores.

The key components that require evaluating during the exploration period are the deposit's geology, ore reserves, pit design, ore metallurgy and environmental impact. Feasibility studies are the main vehicle by which to report and guide the exploration programme. During the exploration period a company may undertake an initial (geological feasibility), second (preliminary mine feasibility) and third (final feasibility) delineation programme in order to gather sufficient data to justify a mine development decision. The responsibility of evaluating the mineral prospect lies primarily with the exploration geologist and mining engineer. Broad experience, a professional attitude, a thorough understanding of mining economics, and a high level of geological, engineering and technical skills are traits required by the evaluators.

In order for mining companies to make sound investment decisions the geographical, geological, mining, metallurgical, environmental, marketing, political and financial aspects affecting the economic potential of the venture must be integrated so that the likely costs, risks and returns of the investment alternative are quantified. Ultimately, it is the economic analysis of these three items that determine whether the mineral prospect is developed into a mine, delineated further, retained until economic circumstances improve, or abandoned. To assess the costs, risks and returns, extensive use of the risk analysis is advocated throughout the exploration period. When combined with intelligent judgement of the intangible risk elements, the probabilistic distribution of discounted cash flows are invaluable in making sound investment decisions. However, the economic analysis is only as good as the information on which it is founded. Accurate and representative field data is the most important prerequisite to successfully evaluating and developing a new mine.

CONTENTS

LIST OF TABLES

LIST OF FIGURES

LIST OF FIGURES CONTINUED

GLOSSARY

Accuracy. See precision

Agglomeration. Metallurgical treatment process for heap leaching of gold and silver ores whereby the fine ore particles are attached (agglomerated) to the coarse ore particles with lime and cement to form small balls. The process immobilizes the fines, eliminates channeling and allows the cyanide solutions to percolate uniformly through the heap.

Bench. A ledge, which, in open pit mines forms a single level of operation above which ore or waste materials are excavated from a continuous bank or bench face. Several benches may be in operation simultaneously in different parts of, and at different elevations in an open pit mine.

Capital. Is the sum of long-term debt and shareholders equity. The cost of capital is the sum of debt costs or interest and costs attributable to equity financing which a company must pay or earn to satisfy all investors, both on a short- and long-term.

Cash flow. Is the sum of net earnings. Cash flows, which are usually expressed annually, may be positive if revenues exceed costs or negative if costs exceed revenues.

Coefficient of variation. Is the ratio of the standard deviation to mean, and a useful measure of relative variability of the observations. A wide variability of sample values gives a higher coefficient of variation.

Cutoff grade. See ore.

Deposit. A body of rock, thought of as a geological entity, containing one or more minerals in sufficient concentration to make extraction potentially economically viable. One or more orebodies comprise a deposit.

Discounted cash flow (DCF). Is the value of future earnings discounted by a specified rate to the present. The calculation takes into account the time value of money.

Flowsheet. The flowsheet shows diagrammatically the sequence of operations in the ore treatment process. In its simplest form it is usually presented as a block diagram in which all the unit operations are shown.

Grade. Content of valuable mineral or elements expressed as a weight per unit mass, i.e. grams per metric tonne (g/t).

In situ or geological resources. Those reserves (incorporating both tonnes and grade) which are defined geologically.

Heap leaching. Term applied to dissolving and recovering gold and silver from a heap. The heap is irrigated with cyanide solvent, which percolates into and through the heap. Runoff from the bottom of the heap is collected and metal in solution is recovered by physio-chemical reaction.

Kriging. Is a geostatistical reserve estimation method used for obtaining the statistically best estimate of point or block grades (or thickness).

Leaching. The extraction of a mineral or metal from an ore by selectively dissolving it in a suitable solvent (e.g. cyanide, thiorea). See heap leaching.

Mean. The term "average" used in connection with a set of numbers refers to their arithmetic mean. The arithmetic mean is calculated by dividing the sum of the values of the observations within the population by the number of observations. The computation of the mean assumes all the observed values are composed of the same size of sampling unit.

Mineable reserves. Those reserves (incorporating both tonnes and grades) which will be mined and treated in the metallurgical plant or in the heaps during the production period of the mine's life. This definition requires that mining, metallurgical, environmental, economic, marketing, social and governmental factors be incorporated into the calculation of mineable reserves.

Mineral prospect. A mineral prospect exists once there are indications of mineralisation (delineated through drilling or excavations) of potentially economic grades over mineable widths.

Open pit mine; opencast mine; opencut mine. A mine working or excavation open to the surface.

Ore. Ore is "rock that may be, is hoped to be, will be, is or has been mined; and from which something of value may be extracted" (Taylor, 1986).

Ores are usually defined by a cutoff grade; material with a metal/mineral content above the cutoff grade is ore, scheduled for treatment, other material is left or dumped as waste. Cutoff grades change according to the prevailing economic circumstances at the time of mining.

Ore zone. Synonymous with orebody.

Orebody. A well defined body of rock of sufficient mass and with a sufficient content of one or more minerals or metals so that their extraction is economically feasible under prevailing conditions.

Oreshoot. The spatial projection of the orebody, usually expressed with plunge direction and dip.

Overburden. Surface barren or low-grade material that must be removed to gain access to mineable grade material. Synonymous with cover.

Pit slope. The angle at which the wall of an open pit stands as measured along the imaginary plane extending along the crests of the benches.

Precision. In wet chemical analyses precision is the agreement between replicate assay determinations. Accuracy is the agreement between the determinations and the true value.

Reef. A geologically defined body of rock that contains sufficient mineralisation to be of economic interest. Not all reefs need necessarily be above the prevailing cutoff grade.

Recovery. Recovery is the the percentage of the total metal contained in the ore that is recovered from various treatment processes. For example a gold recovery of 90% by direct cyanidation of the ore means that 90 per cent of the metal is recovered in solution and 10 per cent is retained in the ore or lost to tailings.

Reserves: A tonnage or volume of rock whose grade, limits, mineralogy, and other characteristics are known with a qualified and explicit degree of knowledge, in relation to a defined sampling grid and other related information and tests.

Selective mining. That process which attempts to optimize the extraction of profitable material from the ore zone by differentiating between ore and internal waste. In contrast by bulk mining all the material within in the ore zone is mined.

Selective mining unit (SMU). The smallest block of ore that can be flagged from blasthole of other sampling data. For example a 5m by 5m by 2.5m block, if blastholes are drilled on a 5m grid and the bench height is 2.5m.

Support. In statistical terms, point or block samples with the same support means that they represent the same type of ore, have the same volume and were sampled and assayed in the same way. When changing from point values to block values one is referring to the volume support component only.

Stripping. The removal of barren or low-grade earthy or rock material required to expose and permit the mineable grade of ore to be mined.

Waste. The barren or low-grade material removed to gain access to the ore.

1. INTRODUCTION

The purpose of this dissertation is to study the evaluation of selectively mined, open pit gold deposits during the exploration period of the mine's life, that is, from the moment significant gold mineralisation is discovered to the start of mine optimization. The key components that require evaluating during this period are the deposit's geology, ore reserves, pit design, ore metallurgy and environmental impact. Introductory guidelines to the evaluation of these components are presented together with methods of reporting the results, guiding the delineation programme and determining the economic potential of the mineral prospect.

Since 1970, as a result of higher gold prices and improved metallurgical techniques, there has been an increase in open pit gold mining. In particular many of these deposits are characterized by small, irregular ore zones that require highly selective mining. Selective mining attempts to maximize the profitability of the operation by recovering less ore at a higher grade than the mean grade which would be obtained by bulk mining (Carras, 1986). In practice a combination of bulk and selective mining may be employed. Porphyry copper-gold and disseminated epithermal (Carlin-type) gold deposits favour bulk mining where disseminated mineralisation allows for high-tonnage, low-grade mining. These deposits are not described in this report but the principles of evaluation are equally applied. In contrast selectively mined (hereafter abbreviated to selective), open pit gold mines treat relatively low-tonnage, high-grade ore. The best known examples are the epithermal vein silver-gold deposits of the south-west Pacific islands, and the mesothermal lode and laterite gold deposits of western Australia.

Selective open pit gold deposits are one of the most difficult classes of deposits to evaluate. Nevertheless, the mines are low-cost gold producers and with the worldwide trend to find more cost competitive operations, their future potential seems assured.

This report is principally addressed to exploration geologists and mining engineers whose work is to carefully evaluate these gold deposits before allowing their companies to commit large sums of money to developing new mines. In the first part of the report the concepts of mineral prospect evaluation are introduced. This recognizes the complex interplay of the many risks and uncertainties associated with the mining business. The need for accurate and continuous evaluation, and reevaluation of the venture, combined with a cautious business-like approach to mining investments is stressed. Feasibility studies are presented as the best means of guiding the delineation programme and integrating all the complex issues into a single set of conclusions.

In the next five chapters attention is then focussed on the geology of selective open pit gold deposits; the methods and pitfalls of ore reserve estimation and grade control; the collection and interpretation of geotechnical and hydrological data for mine design; the mineralogical investigation and metallurgical testing of gold ores; the common gold ore treatment processes; the environmental problems associated with open pit gold mining; and finally some recommendations on how best to assess and approach environmental issues.

The last chapter introduces the common methods for determining the economic potential of the gold prospect and discusses the investment criteria needed to make a sound investment decision. This is one of the most important, yet highly subjective, parts of the delineation programme. Cash flow simulations using risk analysis are shown to assist decision making by providing a quantitative assessment of the likely costs, risks and returns of the mining investment.

A glossary of terms to help with any unfamiliar words is provided at the start of the report. In addition four appendices are attached to provide more details on feasibility studies, the role of geology in evaluation, mineral resource and ore reserve classifications, and environmental impact assessments.

2. OUTLINE TO MINERAL PROSPECT EVALUATION

Mineral prospect evaluation is the most important preliminary requirement for sound investment decisions in the minerals industry, for without it, individuals and companies would stand a high chance of financial ruin. Without presenting any radical new ideas, this chapter hopes to offer a suitable framework for the evaluation of selective open pit gold deposits. The framework, which could be applicable to the evaluation of any mineral prospect, forms the basis for further discussions in this report. Most of the ideas were presented in an earlier report (Pelly 1990); other concepts have been obtained from Mackenzie (1981), Cobb (1988), and Anderson and Tingley (1988).

2.1. GENERAL CONCEPTS OF EVALUATION

There are two ultimate reasons for evaluating a mineral prospect, namely:

1) to estimate the money making (economic) potential of the property; and

2) optimize the profitability of the mine. Optimization (which is not discussed in detail in this report) is normally performed just prior to mine development and throughout the mine production period.

In order to estimate the economic potential of the mineral prospect, the three essential items requiring investigation are the likely costs, risks and returns of the investment alternative. The costs include all operating and capital costs; the risks relate to the uncertainties affecting the economic outcome of the venture; and the returns are the financial gains or losses incurred whilst exploiting the mineral resource. Under most circumstances the resource is exploited for a profit. Sometimes, for strategic purposes or when the mine is facing short-term economic problems the resource may be exploited for a loss. Whatever the situation, mineral prospect evaluation is equally valid.

In order to assess the costs, risks and returns of the investment alternative, all the geographical, geological, mining, metallurgical, environmental, marketing, political and financial aspects affecting the economic outcome of the venture need to be integrated and evaluated. This is the basis of mineral prospect evaluation. It is a complex, multifaceted subject, too broad to be covered by a single publication and so varied that generalizations are unavoidable. These issues result from the fact that each mineral prospect is unique and must be evaluated as an entity. Differences in the geology, rock conditions, ore mineralogy, location, climate, topography, society and government, mean that every mineral prospect must be evaluated using site specific information. The variety of deposits also means that many aspects of evaluation can only be learnt through experience.

Creating a new gold mine essentially involves three main stages: exploration, development and production (Fig. 2.1). The exploration period is critical as it is during this time that the many risks are evaluated. During the development period construction of the mine occurs. This stage is marked by a rapid increase in capital expenditure and the main risk is financial. Finally, during the production period, the economic potential of the property is realized through the sale of gold. At this stage the success of the property will be largely dependent on the accuracies, or inaccuracies of the feasibility studies conducted during the exploration period.

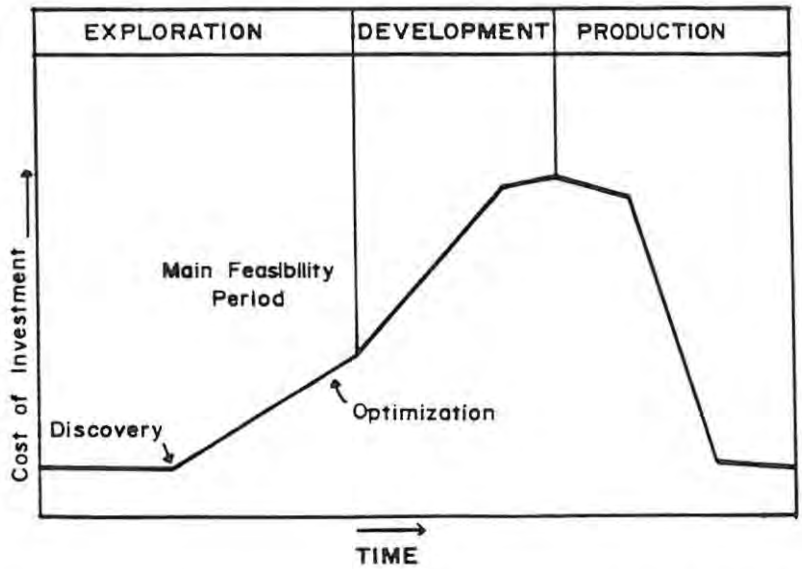


FIGURE 2.1. Main stages involved in the creation of a new mining venture.

During the exploration period, the evaluation process involves a sequential series of data gathering and data analysis steps (Fig. 2.2). Once a mineral prospect is discovered or acquired, an initial delineation programme can be undertaken to obtain information on the deposit's geology, mining, metallurgical and environmental aspects. Thereafter the prospect is subject to a series of decision (evaluation) checkpoints to determine the future course of the prospect. At each checkpoint the economic potential of the prospect is determined and a decision is taken whether to abandon the prospect, postpone it, develop and mine it, or continue with gathering more data. If the company continues to gather more data then another evaluation and decision checkpoint are required. This process may continue many times until the company finally decides to abandon, postpone or mine the prospect. For the prospect to be mined all the geological, mining, metallurgical, environmental and other aspects must be economically acceptable to the company.

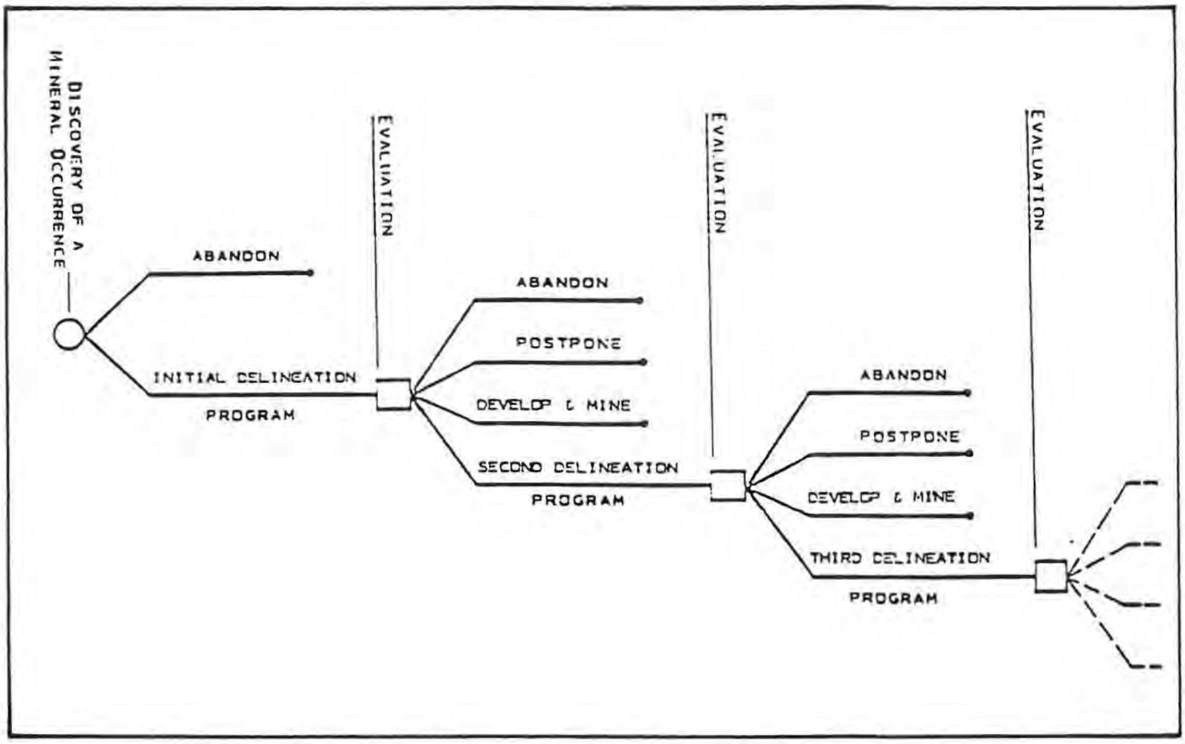


FIGURE 2.2. Sequential evaluation and design process in the development of a mineral prospect (from Mackenzie, 1981).

If the prospect is uneconomic it is abandoned or postponed. A prospect may be postponed indefinitely, typically in hopes of improved metal prices, but this will normally incur further holding costs, such as mineral rights or option payments. Prospects which are abandoned will probably never be mined in the short- to medium-term. This may be due to poor grades, insufficient reserves, remoteness, political factors or any other reason that makes the project uneconomic. In general a prospect that goes through two delineation programmes is less likely to be abandoned.

Four important points emerge from this sequential process. Firstly, the evaluation process is continuous. Delineation programmes that run for long spells without some intermediary checkpoints can be dangerous. There are dangers of loosing sight of the objectives, assessing non-risk items, over spending, collecting irrelevant data, and even collecting too much data. Secondly, the evaluation process is iterative. Additional information on the geological, mining, metallurgical and environmental aspects is collected several times in order to further constrain their associated uncertainties. Thirdly, uneconomic mineral prospects must also be evaluated. Holding mineral rights can be expensive and it may be necessary to relinquish these rights in order to focus more attention on the better mineral prospects. Lastly, economic circumstances change with time so it is important to continuously reevaluate properties.

Fig. 2.3 illustrates the main parties of people involved in mineral prospect evaluation. These may be divided into evaluators, investors and third parties. The evaluators comprise a team of mining experts such as geologists, engineers, statisticians, surveyors, metallurgists, accountants, market analysts and technical consultants. Their work is to co-ordinate, collect and interpret data for the purpose of evaluating the mineral property. The evaluators main concern is that the data is accurate and representative of the components being evaluated.

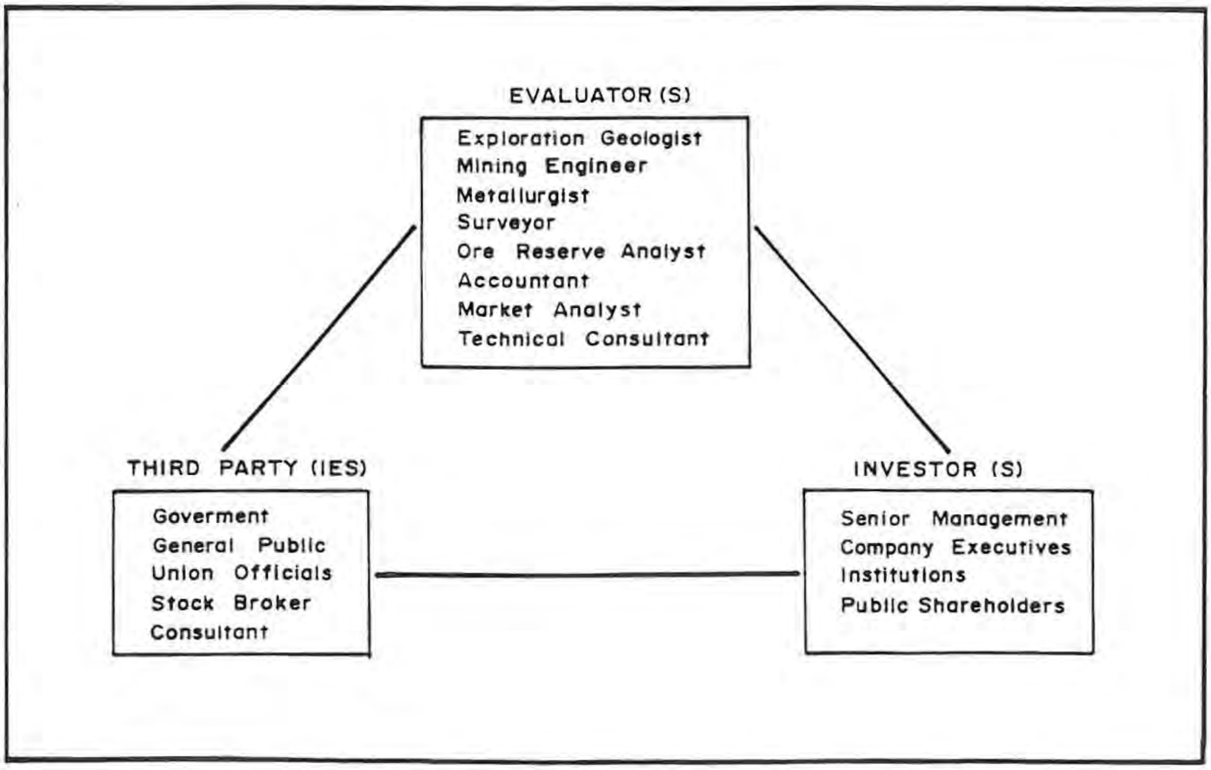


FIGURE 2.3. Main parties of people involved in the evaluation of a mineral prospect.

The investors may comprise senior company managers, company executives, institutions or the investing public. Their concern is to see that money is wisely invested for the purpose of profit and growth. The evaluators and investors normally work very closely as their goals coincide. The third parties comprise government agents, independent consultants, the general public, stock brokers, bankers and unions. Their concerns are with legal, environmental, promotional and labour related matters.

During the exploration period, most companies will undertake three main consecutive delineation programmes in an attempt to gather sufficient data to justify an eventual mine development decision. In this report, the three main delineation programmes are introduced as the geological feasibility, preliminary mine feasibility and final feasibility stages. Fig. 2.4 shows how these three feasibility stages are associated with the development of a typical selective open pit gold mine. With each successive delineation programme, the difference between the estimated and actual value of the property decreases. This is logical, for as the amount of data gathering increases the accuracy of the estimate must improve. At the geological feasibility stage, a low level of accuracy of about 45 per cent is likely as only limited information is available. The delineation costs incurred at this stage will have been relatively low and further expenditure is justified as there is always the possibility that a bonanza deposit will be found. At the preliminary mine feasibility stage, the estimated value may differ by only 25 per cent from the actual value as sufficient data will be available to design a mine. At the final feasibility stage, the estimated value should differ by less than 15 per cent from the actual value. By this stage, the potential value of the prospect needs to be known with a fair degree of certainty as capital expenditure increases exponentially during the mine development stage.

The level of accuracy required to make a mine development decision will probably vary according to the company's investment policy. In general large companies are willing to accept greater financial risks (or lower levels of accuracy) as they have the financial "muscle" to overcome a possible failure. On the other hand, small companies with limited funds must be sure of their investment as one failure could mean the company is liquidated. Some companies may even decide to forego another round of data gathering if the potential profits of the prospect can be spent trying to justify the investment decision. This is certainly true for very small mining ventures where the project's cash flows become more sensitive to capital expenditure.

To determine the economic potential of the mineral prospect, the evaluator

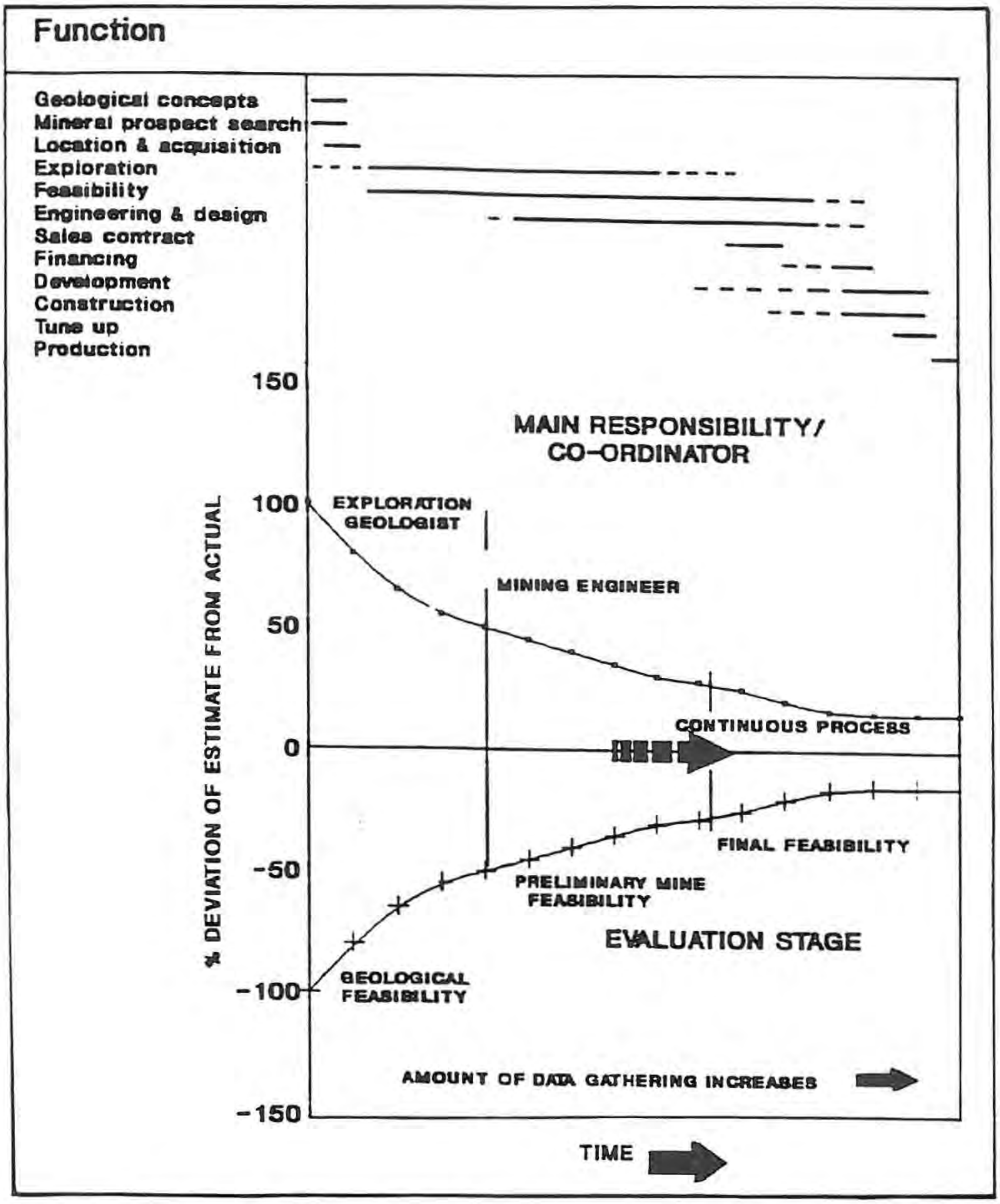


FIGURE 2.4. The evaluation and functional stages associated with the creation of a hypothetical open pit gold mine.

needs basic information concerning mineable reserves and grade, stripping ratios, mining and milling rates, metallurgical recoveries, infrastructure requirements, capital and operating costs, tax rates and gold price forecasts. With this information annual costs and revenues can be estimated for use in generating the property's cash flows. Cash flow iterations generated from a risk analysis programme are ideal for determining the costs, risks and returns associated with the mineral prospect. When combined with sound judgement, the results of the risk analysis are invaluable for making investment decisions.

The main responsibility of evaluating the mineral prospect lies with the exploration geologist and mining engineer. During the geological feasibility stage, the exploration geologist is responsible for co-ordinating the evaluation process as he or she is best equipped to make investment decisions concerning the delineation of the orebody. However, in order to make sensible decisions it is imperative that the geologist be aware of the economic, technological and financial aspects of mining investments. During the preliminary mine and final feasibility stages the main co-ordinator must be the mining engineer as it is during these stages that the actual mine design starts. In this instance the exploration geologist acts in an advisory capacity.

2.2. FEASIBILITY STUDIES DURING EXPLORATION

Where the evaluator often experiences most difficulty is not with the data gathering, but in the presentation of the results into a meaningful conclusion. Feasibility studies are generally accepted as the best vehicle for recording information and interpreting the results of the delineation programme. The studies integrate all the salient aspects into a set of conclusions which will help the investor make a sound investment decision. Traditionally, feasibility studies have been conducted prior to mine development. In this report their use throughout the exploration period is recommended, not only to present the results, but also to successfully guide the delineation programme. To fulfill these requirements some basic requirements of feasibility studies need to be mentioned. These are listed below.

1) All aspects affecting the exploitation of the resource need to be evaluated. Feasibility studies must therefore be <u>comprehensive</u> studies.

2) Feasibility studies are important documents and considerable time, effort and expense go into compiling them. If the project is successful, the studies will form the foundation of any promotional listing on the stock exchange. Accordingly, <u>professional</u> studies of the highest quality are always required.

3) Usually there are several mineral prospects competing for funds. The feasibility studies must therefore allow for <u>comparison</u> between the investment alternatives.

4) Since economic circumstances change with time, the mineral prospects may require reevaluation at a later date. The results of the economic analysis must therefore be reproducible by other evaluators.

Table 2.1. lists the subjects that are commonly included in feasibility studies. Many of these factors will have been assessed prior to the discovery or acquisition of the mineral prospect. Important items such as selecting a favourable country for exploration, avoiding remote areas, reviewing the country's attitude towards mining, judging the environmental factors, forecasting gold prices and purchasing mineral rights are also matters conducted during the reconnaissance stage of exploration. Nevertheless, reevaluation of these factors is necessary for the feasibility studies. For example another check of the mineral rights should always be made just to be sure! Government laws and regulations concerning conservation, water use, pollution, land reclamation, taxes, royalty payments, import duties, work permits and labour conditions can also change with time and need to be checked.

The basic format for all feasibility studies can be the same, the only difference is with the contents and detail of the subject matter which will change subtly during the course of exploration. These changes follow a logical sequence and start with the most important items being evaluated first. Thus the geological feasibility stage is mainly concerned with defining the orebody. The preliminary mine feasibility stage is mainly concerned with mining, metallurgical, personnel and environmental aspects, and production volumes, and capital and operating costs. The final feasibility stage is mainly concerned with the competitive assessment, market analysis and financing arrangements. This concept is illustrated in Fig. 2.5 which divides mineral prospect evaluation into six, vertically stacked

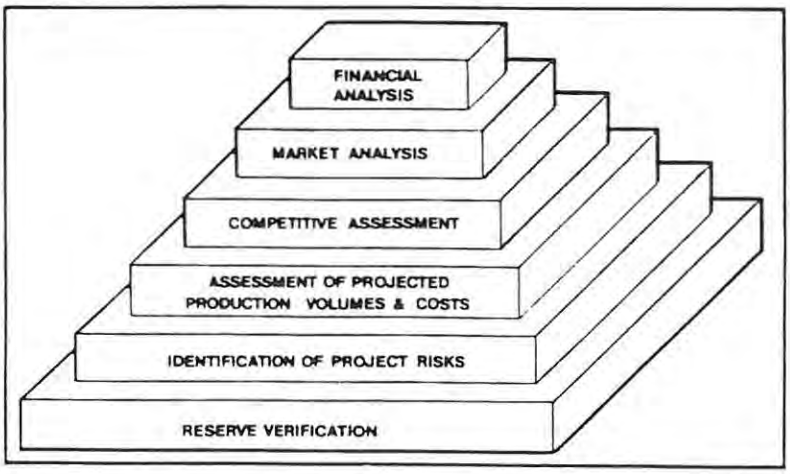


FIGURE 2.5. The building blocks in mineral prospect evaluation (from Anderson and Tingley, 1988).

TABLE 2.1. Principle contents of feasibility studies.

Geography	- location, access, topography, climate, power supply, towns.
Holdings, titles and permits.	- ownership of mineral, water, and surface rights, land and water use restrictions, and owner's terms of agreement, mining permits.
Historical, political and sociological factors	- past production and profitability, political situation, threat of nationalization, general stability, cultures, working attitude, levels of skills, unemployment levels.
Geology	- regional and local geology, stratigraphy, structure, alteration, weathering, types of mineralisation, ore genesis, geometric shape and attitude of ore body.
Ore reserves	- in situ and mineable tonnage and grade, selected areas, cutoff grades.
Mining conditions	- mine design, stripping ratios, mining methods, grade control, waste rock dumps.
Geotechnics	- pit slope, geomechanics/hydrological factors
Metallurgy	- milling, treating and smelting requirements and design, flowsheets, tailings disposal.
Infrastructure	- buildings, dams, roads, vehicles, housing, recreation facilities.
Personnel	- manpower needs, policies, safety, security, labour relations, health codes, wages, unions, training, pension requirements.
Environment	- environmental assessment, rehabilitation, integrated environmental management.
Competitive assessment	- strengths, weaknesses, threats and opportunities (SWOT analysis).
Market analysis	- gold price forecasts, rand/dollar exchange rates, inflation rate forecasts.
Capital costs	- exploration, plants, mining, pre-production construction, miscellaneous.
Operating costs	- labour, administration, consultancy, contractors, power, equipment, metallurgical reagents, transport, royalties, ownership.
Financial and Economic analysis	- financing means: retained earnings, bank/gold loans, equity; cash flow projections, sensitivity and risk analyses.

blocks or subject matters. The importance of the subject matter is directly proportional to the size of the block and the stability of each block is dependent on the underlying blocks. Ultimately the stability of all the blocks is dependent on the base block, represented by the ore reserves.

To assist the evaluator with reporting his results Appendix 1 provides a feasibility study format that is suitable for selective open pit gold deposits. This format can be modified to suit individual tastes for any feasibility study compiled during the exploration period. In addition Table 2.2, summarizes the general features, objectives and requirements of each feasibility stage. The remainder of this subsection discusses these stages in more detail.

Geological Feasibility. At this stage, only preliminary information is available on the possible size and grade of the orebody, the mineability of the deposit, ore metallurgy and environmental aspects. An initial mine plan, however, will indicate that the operation is potentially economic. The objective of the geological feasibility will be to expand upon and confirm this information.

The first priority of the geological feasibility is to understand the geology of the mineral deposit. This is essential for directing the delineation programme and evaluating the ore reserves, mining, metallurgical and environmental aspects.

As illustrated in Fig. 2.5, ore reserves are the prime concern at this stage of exploration. Previous estimates will only have delineated inferred in situ resources. These will be inaccurate and the first goal will be to produce a more accurate indicated resource estimate. Widely spaced drilling, augering and backhoe trenching may be used to verify the earlier estimates. The controls on mineralisation are particularly important. Since open pit gold deposits are exposed to surface, both secondary (weathered) and primary gold mineralisation will probably be present. Both types will need to be differentiated during the delineation programme.

With regards to mining, metallurgical and environmental aspects, a comprehensive data bank will be required to more closely define the associated risk elements. The major concerns affecting mining are going to be the likely mining methods (soft or hard rock), the expected depth and

TABLE 2.2. Objectives and scope of feasibility studies used for the progressive evaluation of a typical mineral prospect.

ITEM	GEOLOGICAL FEASIBILITY	PRELIMINARY MINE FEASIBILITY	FINAL FEASIBILITY
Responsibility/Accuracy	Exploration Geologist 35 to 60%	Mining Engineer 20 to 25%	Mining Engineer 10 to 15%
Exploration Stage	Mineral prospect discovery and early exploration.	Middle & late exploration	Late to post exploration.
Main Objectives	Confirm/expand upon geology and metallurgy with most cost efficient exploration methods.	Identify and solve critical risk items related to geology, orebody, mining, metallurgy, environment.	Optimize mining method, mine plan, scale of operation and cutoff grade policy.
Information Available	Preliminary estimate grade/tonnage, environmental and political situation governmental legislation, marketing.	Geology, good estimate of ore reserves. All mineralogical data. Good definition of orebody.	Geology, ore reserves, mine design, recovery plant/process, personnel, infrastructure, and environmental issues.
Information Required	Preliminary information on geology, exploration methods, metallurgy & mining methods, Capex & Opex.	Site specific. Collect data on ore reserves, mineralogy, mine plan, metallurgical flowsheet & environment.	Actual detailed development costs. All capital phasing and engineering plans.
Sources of Information	Company files, conduct literature researcy on similar deposits, other personnel. Site specific.	Visit other mines, compare and discuss matters with experienced mine operators. Site specific.	Direct quotes/costs from manufacturers/ suppliers and contractors. Site specific.
Uses of Feasibility Study	Setting up the exploration programme within economically acceptable limits.	Monitoring the delineation programme continually. Identifying critical items.	Optimizing the operation. Monitoring trial mining and pilot-scale testwork.
Detailed Reports Generated	Geological; Interim ore reserves.	Ore reserves, Metallurgical, Mine design, Engineering, Manpower, Environmental.	Engineering, Manpower, Legal, Finance, Environmental, Marketing, Competitive.
Major Cost Items	Drilling Metallurgical testwork.	Drilling, metallurgical testwork, computer mine design, mine visits.	Engineering plans/drawings New technological developments

size of the pit, the pit slope angle, the amount of stripping needed to expose the ore, ground conditions, and dangers of flooding. To constrain these risks, information is required on the depth of weathering, rock hardness, overburden thickness, topography, climate, shape and attitude of the orebody, water fissuring, and geological structures. Metallurgy will require more information on the gold recoveries. Bottleroll testwork and mineralogical investigations can be performed to determine the amenability of the material to direct cyanidation. Environmental concerns are with pollution control and possible disturbance of the land. Monitoring the situation by analysing soil and stream samples for pollutants may be all that is necessary at this stage. However, an environmentally concerned attitude and opening of communication channels with government agents and land owners will serve to formulate a plan of action which is mutually acceptable to all parties.

To conduct an economic analysis, preliminary estimates of production rates, capital and operating costs, and revenues are required. The information will be uncertain at this stage but estimates may be available from company files, through literature research of similar operations or from other personnel.

Preliminary Mine Feasibility. The preliminary mine feasibility stage takes place as a continuing process throughout the middle to late stages of exploration. As more drill holes are completed, as grades and tonnages firm up and as mining, metallurgical, environmental and other data are received, the accuracy of the estimate will improve.

The objectives at this stage are to: 1) direct the development funds towards constraining critical risk elements; and 2) obtain site specific information on the production rates, and capital and operating costs. Besides ore reserves, mining, metallurgical and environmental aspects, other risk elements may be related to infrastructure, water supply, access, logistics and personnel. Extensive use of the sensitivity analysis will help identify these risk elements and rank them according to their economic importance. This will ensure that delineation funds are wisely spent on solving the risk elements that significantly affect the economic potential of the property.

At this stage infill drilling and bulk sampling will be needed to raise the in situ resource estimate into the probable reserve category. Mining aspects may require consultants to interpret the geotechnical data so that pit slope angles can be recommended. Bench-scale metallurgical testwork will be needed to provide details of gold recoveries and other ore treatment requirements. Environmental baseline studies will be required to assess the impact of the proposed mine on the environment. Also at this stage a limited number of basic engineering drawings of mine infrastructure, metallurgical plant and mine layout will be required to estimate the capital cost. Production cost information may be obtained from visits to other mines or from discussions with mine personnel who have had first hand experience in similar operations.

Before proceeding to the final feasibility stage, the ore reserves, mine design, ore treatment process, personnel requirements, infrastructure and environmental issues will be known with a fair degree of certainty. Capital and operating costs will have been obtained mainly from site specific information. Provided the cash flow analyses continue to indicate a potentially economic project then the delineation programme proceeds to the final feasibility stage.

Final Feasibility. The final feasibility will take place prior to mine development. Exploration may still be continuing but sufficient mineable ore reserves will have been delineated to warrant starting a mine. It may even be justified at this stage to expose the deposit by trial mining. This will provide direct visual access of the orebody and at the same time help define the optimum grade control and mining methods. Pilot-scale testwork may also be considered at this stage. If the ore is difficult to process or if the company has had no previous experience, pilot-scale metallurgical plants are recommended.

The objectives of the final feasibility are to 1) optimize the operation; 2) provide detailed construction plans for use in developing the mine; 3) provide a definitive economic, financial, technical and operational assessment of the mining venture which can be presented to the board of directors; and 4) if necessary, provide details of the investment which can be used by promoters to raise equity for a stock exchange listing.

Optimization selects the best mining, sampling, milling and processing methods; the best materials and equipment; the best cutoff grade, ore exposure rate, mining plan, mine capacity and milling rates; the best managerial, personnel, safety, security and environmental policies; and the best capital phasing, legal, financial, tax and marketing structure. Optimization is a complex task involving many interdependent factors which must be manipulated together until the optimum economic, financial, technical and operational package is formulated. Nowadays the task is made a lot easier using specially designed computer software. Gibbs (1989) provides a comprehensive directory of computer programmes that will be suitable for most mining applications.

At the final feasibility stage a competitive assessment, market analysis and detailed financial analysis will be necessary. The competitive assessment involves a study of the strengths, weaknesses, opportunities and threats associated with the venture. Also referred to as the SWOT analysis, it affords a good opportunity to list some of the intangible risk elements that may affect the project's economic potential. The market analysis involves forecasting likely gold prices, supply and demand factors, and inflation and exchange rates. The financial analysis involves selecting the most appropriate means of financing mine development. Funding may be from retained earnings, a share issue, gold loans, a bank loan, or a combination of these sources.

By the final feasibility stage all capital and operating cost estimates will be based on detailed site specific engineering information and costing of all engineering components. This involves drafting construction plans, extracting itemized costs from equipment suppliers and obtaining quotes from contractors.

The board of directors decision will depend on whether the costs, risks and returns of the investment alternative are acceptable to the company's profit expectations and growth strategies. If a decision to proceed with mine development is made then stock exchange listings and press announcements will naturally follow.

2.3. FORMULATING AN APPROACH

To conclude this chapter it is appropriate to examine the traits needed by the evaluators. In the preceding text, successive feasibility studies have been offered as the best vehicle for evaluating the mineral prospect. Whilst this framework seems suitable in theory, the delineation programme rarely runs smoothly in practice. Invariably there are complications which relate to physical, technical, natural and human failings. One problem in particular is being able to maintain the right approach. No person is infallible and sometimes the wrong decisions are made. In an article aptly titled "Due diligence in mining investments", by Anderson and Tingley (1988), some useful approach criteria are offered to the mineral investor. These are equally applicable to the evaluators and are listed below.

1) Caution. All aspects of mineral deposit evaluation require a cautious approach. No decision must be made without thoroughly investigating all the relevant facts.

2) Business. A business-like review is required. Mining ventures need to be investigated like any other business. Under most circumstances the venture must make a profit and perform favourably with competing investments.

3) Breadth. Evaluation is a multi-disciplinary exercise. It draws on the expertise of a team of people who must integrate all their knowledge and experience.

4) Realism. However valuable modern techniques of geophysics, core drilling, computer programmes and geostatistics, they can also create a false sense of security. A modest amount of scepticism may sometimes help to offset the euphoria that surrounds new opportunities.

5) Objectivity. The evaluator must guard against undue optimism and pessimism in his conclusions and those of others. An over optimistic estimate deceives the investor whilst a conservative estimate could lead to rejection of a profitable venture.

6) Experience. Successful prospect evaluation requires experienced evaluators. Evaluation is not a young man's game.

7) Prudence. Mining ventures are generally long-term investments. Foresight and carefulness are traits required by successful evaluators and investors alike.

3. GEOLOGY OF PRINCIPLE TYPES OF SELECTIVE OPEN PIT GOLD DEPOSITS

The purpose of this chapter is to briefly describe the geological characteristics of the principle types of selective open pit gold deposits. Geological knowledge plays a critical role throughout the entire evaluation process and it is the responsibility of the exploration geologist to obtain a good understanding of the gold deposit as any errors or omissions concerning the geology can have major effects on the magnitude and duration of future profits (Mackenzie, 1981). The main evaluation components that require geological input are the design of the sampling programme, ore reserve estimation, grade control, the design of the open pit, ore treatment, environmental concerns and optimization. These items are discussed in more detail by Grace (1986), and Bolin and Rozelle (1987). Appendix 2 provides a checklist of the important geological features that must be recorded during the exploration period.

3.1. SOME GENERAL OBSERVATIONS

Selective open pit gold deposits are characterized by their extensive worldwide distribution and geological heterogeneity. Examples of such deposits occur in all the major western-world gold producing countries - South Africa, USA, Canada and Australia; and many emerging gold producers such as Brazil, Chile, the south-west Pacific islands and some European countries. The variety of deposits can be attributed to the varied tectonic settings and climatic conditions under which the deposits formed. Obviously, irrespective of origin, any gold deposit near surface can be selectively mined provided the economic conditions are favourable. However, for the purposes of this report three classes of gold deposits are discussed, namely 1) epithermal vein silver-gold deposits; 2) mesothermal lode gold deposits; and 3) laterite gold deposits. Their principle geological characteristics are compared in Table 3.1.

These three classes represent the most common types of selective open pit gold deposits that are presently being mined. Other less common types that deserve mentioning are eluvial gold deposits, various strataform and stratabound gold deposits, and gold skarn deposits. Eluvial gold deposits are common in Brazil where they are mined by the garimpeiros (Brazillian gold prospectors). Strataform deposits like the Witwatersrand palaeo--placer have been mined on a very limited scale by some South African companies, for example at Rand Leases gold mine. Stratabound deposits like the large Telfer gold mine in western Australia are an uncommon type, but nevertheless an important gold producer. Skarn deposits are of varied origin and the most common types are those associated with porphyry copper intrusions. Keays et al. (1989) and the contained articles provide a wealth of up-to-date information on the geology, nature, structure, tectonic setting and genesis of these deposits and many more.

TABLE 3.1. Principal characteristics of epithermal vein silver-gold, mesothermal lode gold and laterite gold deposits (from Rossiter, 1984; Colvine et al. 1988; Groves and Phillips, 1987; and Butt, 1989).

PRINCIPLE CHARACTERISTICS	EPITHERMAL VEIN SILVER-GOLD DEPOSITS	MESOTHERMAL LODE GOLD DEPOSITS	LATERITE GOLD DEPOSITS
TECTONIC SETTING	Continental regions with high heat flow, volcanism, tectonic activity and active geothermal field. Convergent plate boundaries are most favourable.	Mainly granite-greenstone terraines; possibly analogous to modern island ard-interarc basins, accretionary wedges.	Variable; stable continental setting with no strong erosional elements preservation.
AGE	Mostly Tertiary to Recent; some Palaeozoic.	Mostly Archaean to early Proterozoic; some Palaeozoic.	Probably all Tertiary to Recent.
NATURE OF DEPOSIT	Thin to wide veins, stockworks and disseminations. Ore zone has extensive strike but restricted vertical extent. Locales of Au mineralisation in tension fractures, and extensional faults and hot springs. Vertically zoned.	Thin to wide veins, stockworks, shear zones and disseminations. Ore zone has limited strike but extnesive down-plunge vertical continuity. General lack of systematic vertical zonation.	Sheet-like parallel to palaeo-water table. Typically ore zone is T-shaped overlying primary vein/lode.
ASSOCIATED ROCKS	Any; mostly calc-alkaline volcanic sequence containing andesites, rhyolites and dacites.	Any; mostly mafic and felsic intrustions and chemical sediments (BIF's).	Weathered laterite horizons: soil, mottled zone and saprolite.
GOLD-BEARING MINERALS	Electrum, native gold, tellurides.	Native gold, electrum, submicroscopic gold in pyrite/arsenopyrite.	Native gold.
GANGUE MINERALS	Pyrite, quartz, calcite, adularia argentite, silver, +/- galena, sphalerite, cinnabar, stibnite, tetrahedrite.	Pyrite, arsenopyrite, pyrrhotite, quartz, carbonate, sericite +/- talc, galena, sphalerite, chalcopyrite, stibnite, scheelite, tourmaline.	Goethite, kaolinite +/- hematite, magnetite and manganese oxides.
ALTERATION	Propylitic, argillic, silicification, adularization, albitization, sulphidation.	Sericitization, carbonation, sulphidation, silicification, albitization.	Kaolinitization, ferruginization.
TEMPERATURE AND DEPTH OF FORMATION	50 - 200 C; surface to 1km.	200 - 400 C; 2 - 10km.	<25 C; surface.
MINERALISING FLUID	Near neutral (pH 6 - 7), CO2 bearing, weakly saline solution dominantly meteoric with possible magmatic component.	Near neutral, weakly saline, moderate to high CO2 solutions of possible magmatic, metamorphic and meteoric origins.	Acidic chloride-rich or alkaline sulphate rich meteoric ground waters.

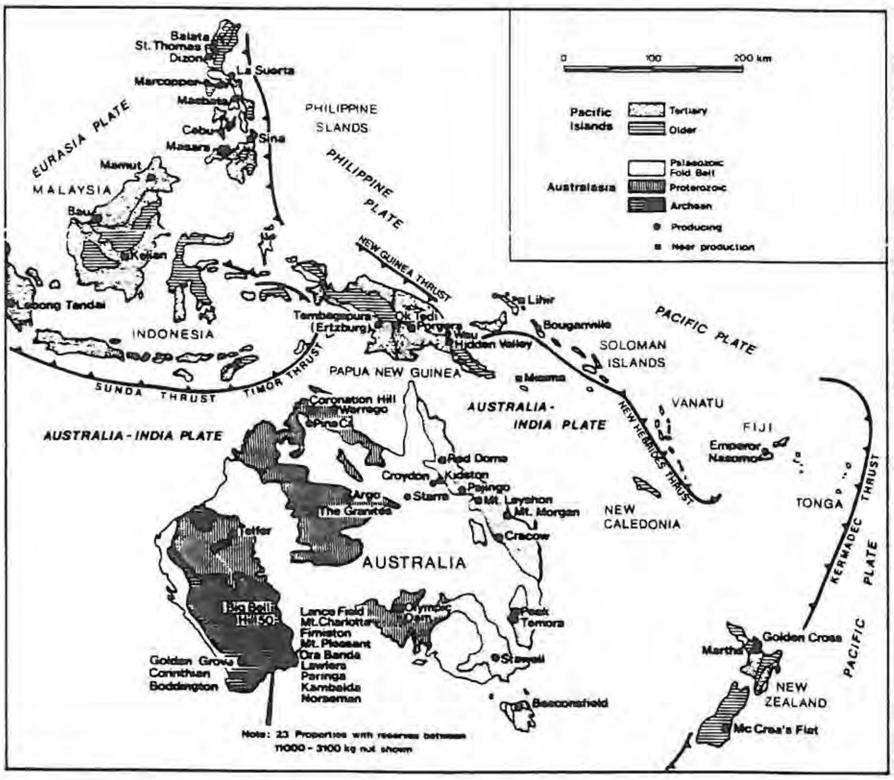
By far the most prolific and suitable region for looking at selective open pit gold deposits is the south-west Pacific. Gold exploration and mining developments in this region have recently been reviewed by Bradshaw and Cox (1988) who relate the gold mineralisation to two present-day tectonic settings (Fig. 3.1), namely:

1) the Pacific "Ring of Fire", including New Zealand, Fiji, Indonesia, Papua New Guinea and the Philippines, where gold is associated with Tertiary and younger volcanic hydrothermal activity, related to converging plate boundaries. Epithermal vein silver-gold deposits (with porphyry and

skarn deposits) are developed along the plate boundaries; and

2) <u>the stable Australian plate</u> where gold is associated mainly with Precambrian granite-greenstone belts (the Yilgarn and Pilbara Blocks) in western Australia and the Palaeozoic rocks of the Tasman geosyncline in eastern Australia. The former are host to the mesothermal lode gold deposits - possible Precambrian equivalents of modern-day island-arc back-arc couples; whilst the latter are host to some Palaeozoic epithermal vein silver-gold deposits - possible convergent plate tectonic setting.

Laterite gold deposits may occur throughout the south-west Pacific. The best known example is the deposit at Boddington in south-west Australia.



<u>FIGURE 3.1</u>. Simplified geological map of south-west Pacific showing distribution of major gold deposits and tectonic plate boundaries (from Bradshaw and Cox, 1988).

On the basis of ore reserves quoted by Bradshaw and Cox (op. cit.), epithermal vein silver-gold deposits are the largest in terms of contained gold and also have some of the highest grades (Fig. 3.2A). Porgera Zone VII and Lihir, are both Tertiary vein deposits situated in Papua New Guinea with quoted reserves containing over 132 and 356 tonnes of gold - by no means small even compared to a large Witwatersrand gold mine. Some of the smaller deposits are also suited to open pit mining. Pajingo, in north-east Australia has a quoted reserve of 1.4 million tonnes at 12.5

grams per tonne. Here gold is present in a narrow, steeply dipping vein and is planned to be mined by opencast methods with waste to ore ratios eventually reaching 20:1.

By comparison, the mesothermal lode gold deposits are generally small, yet, they are by far the most prolific producers in terms of operating mines. Quoted ore reserves are generally from 1 to 20 million tonnes at grades from 1.5 to 15 grams per tonne. An exception is the large Golden Mile deposit of 100 million tonnes at over 10 grams per tonne (Fig. 3.2B).

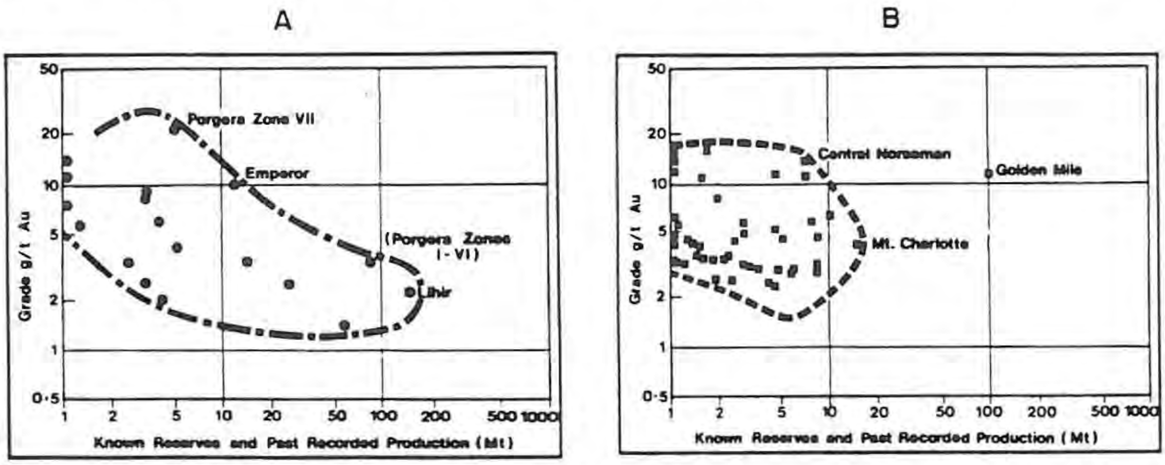


FIGURE 3.2. Reserves for epithermal vein silver-gold deposits (A) and Precambrian lode gold deposits (B) with gold reserves over 3.1 tonnes from the S-W Pacific. Note that tonnage and grade estimates include open pit and underground reserves (from Bradshaw and Cox, 1988).

The laterite gold deposits are usually characterized by very small ore reserves. Boddington, with quoted reserves of 45 million tonnes at 1.8 grams per tonne seems to be exceptionally large and certainly not typical of all laterite gold deposits. Butt (1989) reports that most individual mines exploiting gold in laterites have mineable reserves in the range 0.5 to 1.5 million tonnes at grades of 1.5 to 5.0 grams per tonne. Although small by comparison to epithermal and mesothermal gold deposits, they can be exploited because of their relative ease of mining and ore treatment.

3.2. EPITHERMAL VEIN SILVER-GOLD DEPOSITS

These deposits are formed in a hydrothermal system at shallow depths and under moderate temperatures. Gold precipitation normally takes place within about 1 kilometre of the surface in the temperature range 200 to 300^{o}C, or even as low as 50^{o}C. It is now generally accepted that epithermal deposits represent fossil geothermal systems and considerable understanding of the ore forming processes has been gained from studying active geothermal systems (Henley 1985). The essential requirements needed for the formation of epithermal deposits are: 1) an active tectonic environment which induces permeability and allows hydrothermal fluids to flow and be recharged; 2) a source of precious metals and complexing agents (ligands); 3) suitable structural traps for precipitation of the precious metals e.g. tension fractures and extensional faults; 4) a heat source such as an intrusion or regional heat flow to drive the geothermal system; and 5) upward moving hot fluids and marginal convective downward moving cold recharge water (mainly meteoric) to transport the gold. The principle elements depicting this situation are shown in Fig. 3.3.

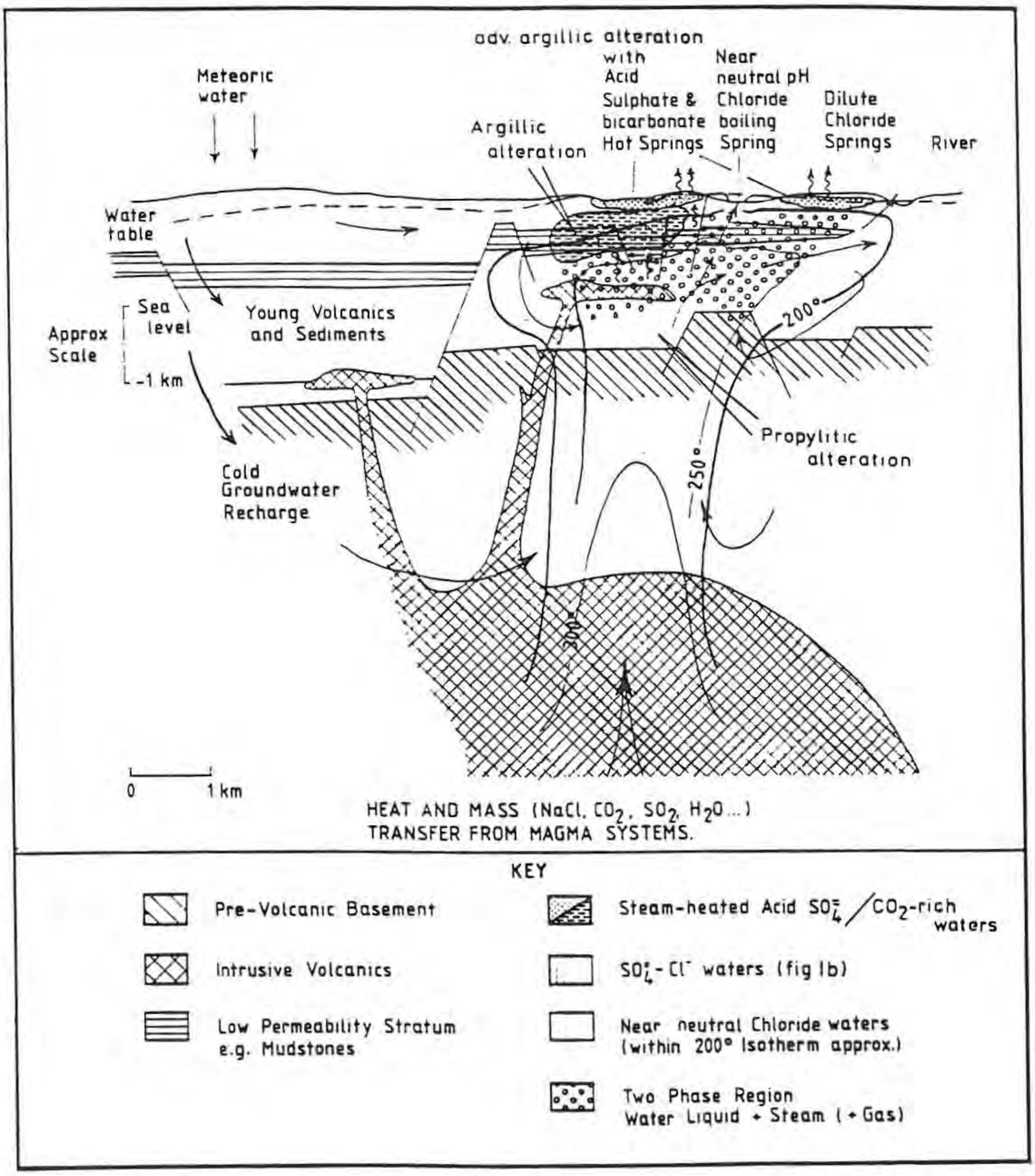


FIGURE 3.3. Generalized structure of a typical geothermal system in silicic-volcanic terrane. Notice the overall size of the system relative to the discharge features - hot springs and fumaroles (from Henley, 1985).

Extensive literature is available on epithermal deposits and selected review articles include Berger and Bethke (1985), Panteleyev (1985), Rossiter (1984), and Berger and Henley (1989). As yet, no single deposit model explains all the common features of epithermal deposits. The deposits display considerable diversity and Silberman and Berger (1985) emphasize that there exists a continuum of types between certain end members. This is ascribed to the very dynamic, evolving, three-dimensional interplay of heat sources, fluid flow, chemical reactions, and structural changes which have combined to produce the deposit. The two types most commonly distinguished are the relatively shallow hot-spring-type where ore is precipitated in stockworks and breccias up to 300 metres below surface; and the bonanza-type, where ore is precipitated in relatively deep veins and fissures. The principle elements of these two types are shown in Figs. 3.4 to 3.6.

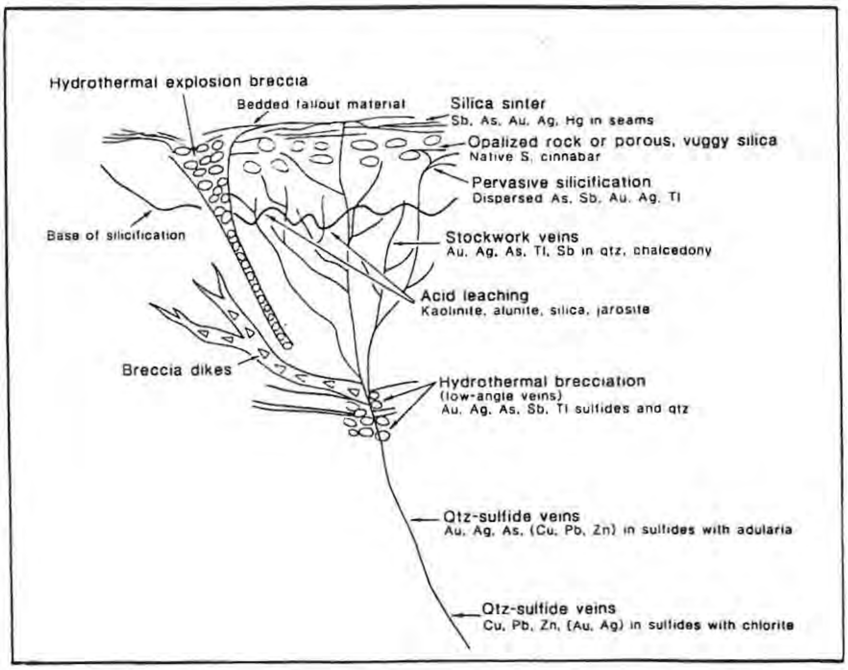


FIGURE 3.4. Schematic cross section of the quartz-adularia Hot--Spring-type epithermal ore deposit showing alteration mineralogy, generalized geochemical associations and other structural and mineralogical features (from Silberman and Berger, 1985).

The common features displayed by epithermal vein deposits include their association with regional and mine-scale structures, extensive wallrock alteration, dominant recharge by meteoric waters, and geochemically anomalous Au, Ag, As, Sb and possibly Cu, Pb, Zn. Silver and gold are the principle economic metals and the silver to gold ratios are commonly greater than 1:1 (Hayba et al., 1985). Gold usually occurs in the native state or as electrum, but may be intimately associated with pyrite such that recovery requires total destruction of the pyrite (Bradshaw and Cox, 1988).

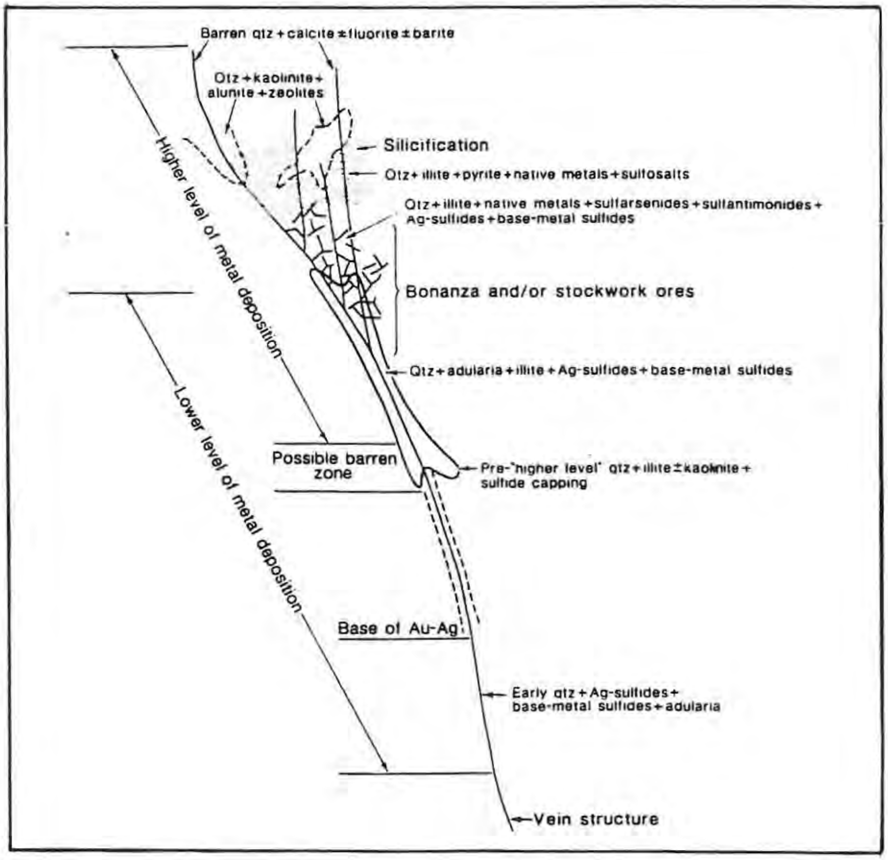


FIGURE 3.5. Schematic cross sections of quartz-adularia Bonanza--type epithermal ore deposit showing alteration mineralogy, two possible zones of mineralisation and other structural and mineralogical features (from Silberman and Berger, 1985).

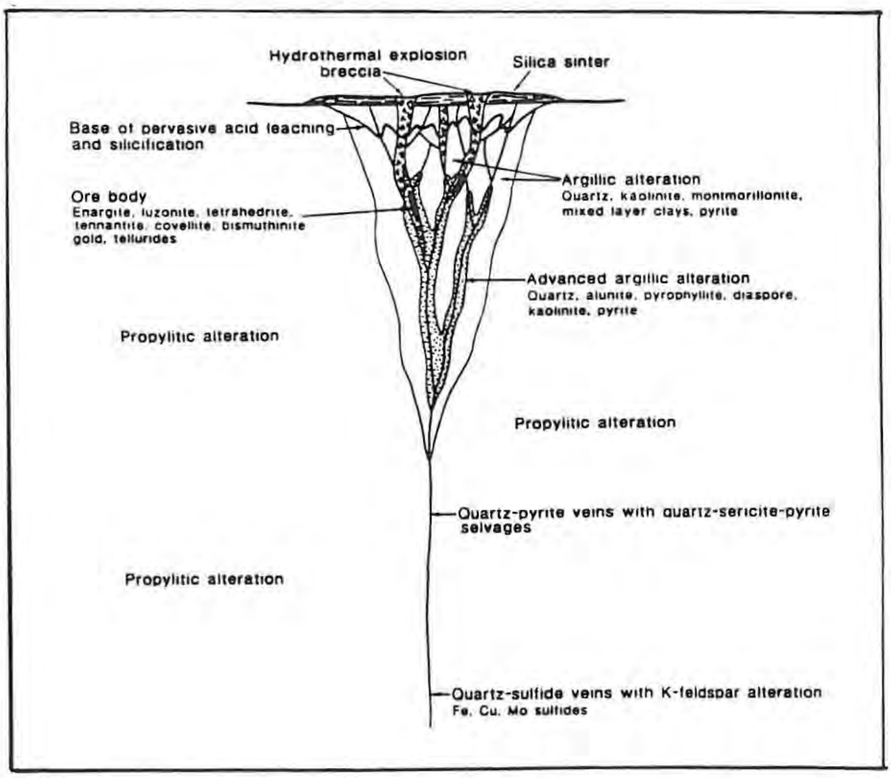


FIGURE 3.6. Schematic cross sections of acid sulphate Bonanza-type epithermal ore deposit showing alteration mineralogy, possible ore zones and other structural and mineralogical features (from Silberman and Berger, 1985).

3.3. MESOTHERMAL LODE GOLD DEPOSITS

These deposits are formed in a hydrothermal system at medium depths and under moderate temperatures. The genesis of mesothermal lode gold deposits is not as well understood as epithermal gold deposits, principally because the deposits have been strongly altered by post-ore forming processes and no present-day analogues can be observed. Students of mesothermal lode gold deposits recognize that most deposits are epigenetic. Gold precipitation probably takes place from 2 to 10 kilometres below surface and in the temperature range 250 to 350^{o}C. The essential requirements needed for the formation of mesothermal lode gold deposits are similar to epithermal vein deposits. These are: 1) large volumes of hydrothermal fluid derived from a deeper, external source; 2) an active tectonic environment which creates permeable channels to focus the hydrothermal fluids; 3) a source of precious metals and ligands; and 4) suitable structural and chemical traps to precipitate the gold.

Considerable controversy surrounds the source of the gold - whether it is derived from specialized magma (porphyry or lamprophyre) or leached from the country rocks during prograde metamorphic devolitization; and the source of the ore-bearing fluids - whether they are magmatic, metamorphic, meteoric or a combination of these sources. Although these issues are interesting, they do not alter the practical evaluation of the deposit and will therefore not be discussed further. For the latest contrasting views on the subject the interested reader is referred to the articles in Keays et al. (1989).

Almost all the known mesothermal lode gold deposits are Archaean (3.2-2.7 Ga.). These deposits are well documented and extensively mined in Canada, Australia, southern Africa, Brazil, as well as in other parts of the world. Only a very small number of mesothermal lode gold deposits are Phanerozoic, for example the Mesozoic Mother Lode deposit in California (Allgood, 1990). Since these younger deposits share similar features with their Archaean equivalents, only the latter needs to be considered. The two areas where the most detailed work has been conducted on Archaean lode gold deposits are the Yilgarn block of western Australia and the Superior Province of Canada. Ho and Groves (1987) and Colvine et al. (1988), together with the contained articles, provide a wealth of useful information on the geology, structure, alteration, fluid composition and genesis of the deposits from these two areas. The following account of Archaean lode gold deposits mainly summarizes observations from their work.

Archaean lode gold deposits occur within or immediately adjacent to greenstone belts of granite-greenstone terrains. The greenstone belts consist predominantly of mafic to ultramafic and felsic volcanic rocks interlayered with clastic and chemical sediments. These supracrustal rocks have been subjected to polyphase deformation so that most of the strata are folded and now steeply dipping. The gold deposits may be hosted by any rock type. However, Groves and Phillips (1987) note that almost all the gold mineralisation in the Yilgarn block occurs in mafic rocks and BIFs, whilst in the Abitibi greenstone belt, Colvine et al. (1988) show that gold mineralisation is preferentially found in post-volcanic felsic plutons, felsic volcanics and chemical sediments.

One prominent characteristic of Archaean lode gold deposits is their occurrence within large deformation zones comprising composite linear shear systems. These deformation zones may be several kilometres in width and up to hundreds of kilometres in length. In detail (Fig. 3.7) they commonly consist of major faults or ductile shear zones of regional extent

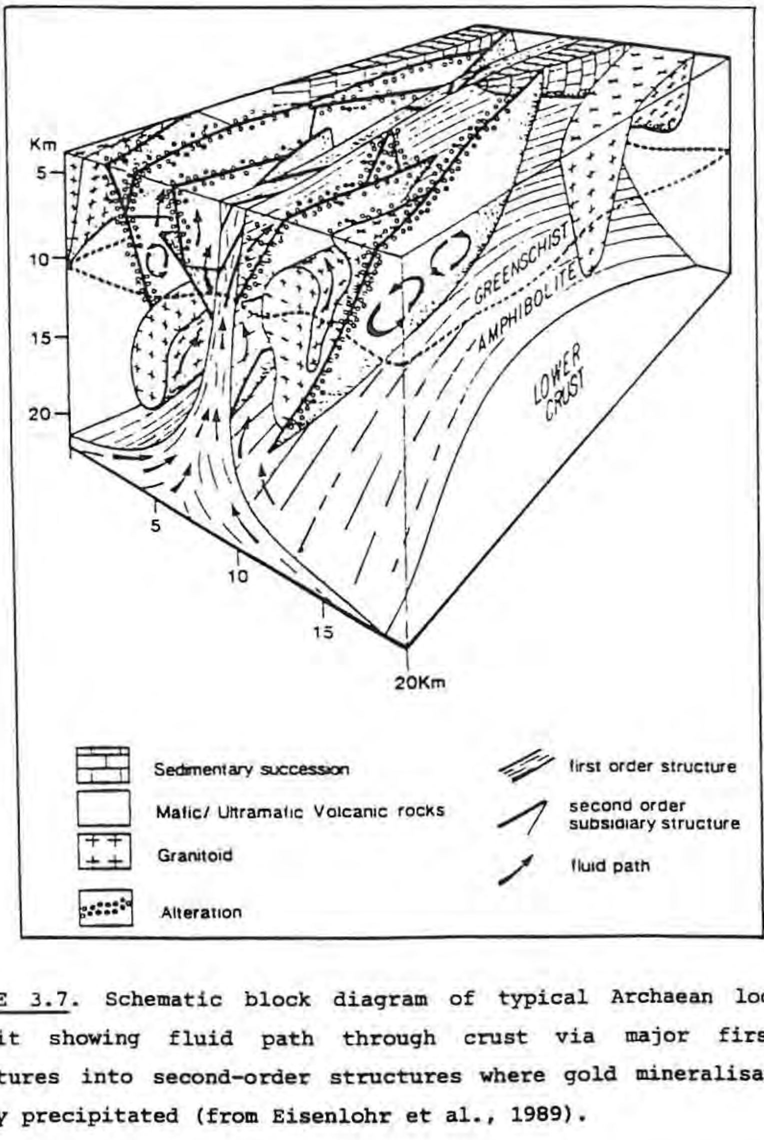


FIGURE 3.7. Schematic block diagram of typical Archaean lode gold deposit showing fluid path through crust via major first-order structures into second-order structures where gold mineralisation is mostly precipitated (from Eisenlohr et al., 1989).

(first-order structures), associated with numerous small scale shear zones and faults (second-order structures). Gold mineralisation is mainly concentrated in dilation zones within the second-order structures where host rock lithology has been important both for its control on physical response to deformation and its chemical response to the ore fluids. In southern Africa and western Australia the gold is often spatially related to discrete brittle-ductile shear zones which commonly form anastomosing zones of high strain separated by more extensive low strain zones (Vearncombe et al., 1989). The first-order structures, despite evidence that they have channelled large quantities of fluids, are generally not mineralised.

Fig. 3.8 presents a composite, idealized, vertical profile through an Archaean lode gold deposit. The figure, which is based on empirical data after Colvine et al. (1988), shows the main structural ore traps as a function of depth of burial below palaeosurface. The nature of the orebody changes with depth, though at any one mine or prospect, these changes are unlikely to be seen as only a small vertical section through the deposit will be exposed below the present erosion surface. In fact a remarkable feature of most deposits are their extensive vertical continuity relative to strike length and width (cf. epithermal vein silver-gold deposits, Table 3.1). A composite vertical range of greater than 5 kilometres but less than 10 kilometres is implied.

According to Colvine et al. (op. cit.) the ambient pressure and temperature during mineralisation exerts the strongest overall control on the style of deformation, mineralisation and alteration. Other factors such as lithology and fluid pressure exert only local controls. Thus at shallow depths under brittle deformation conditions, the orebodies fill narrow, discrete faults, tension fractures and breccia zones. At intermediate depths the deformation becomes increasingly ductile and the orebodies fill shear zones (especially R, R', P and D Riedel shear zones), saddle reefs and en-echelon tension gash arrays. At greater depths, the deformation becomes entirely ductile and wider shear zones containing replacement (disseminated) and boudin vein type orebodies occur. These changes with depth will no doubt be complicated by other factors. For instance the host rock control is not depicted but as a general rule, competent units such as felsic intrusions retain a brittle response to deformation at deeper levels, whereas schists deform in a ductile manner even at higher levels. The effect of hydrothermal fluids is more complex and may alter with time as the alteration mineral assemblage changes. For example the addition of fluid can promote ductile deformation, but over pressure can result in local brittle response in a ductile regime. Also an early phase of sericitic alteration may mechanically weaken the rock and make it susceptible to shearing during a later deformation phase.

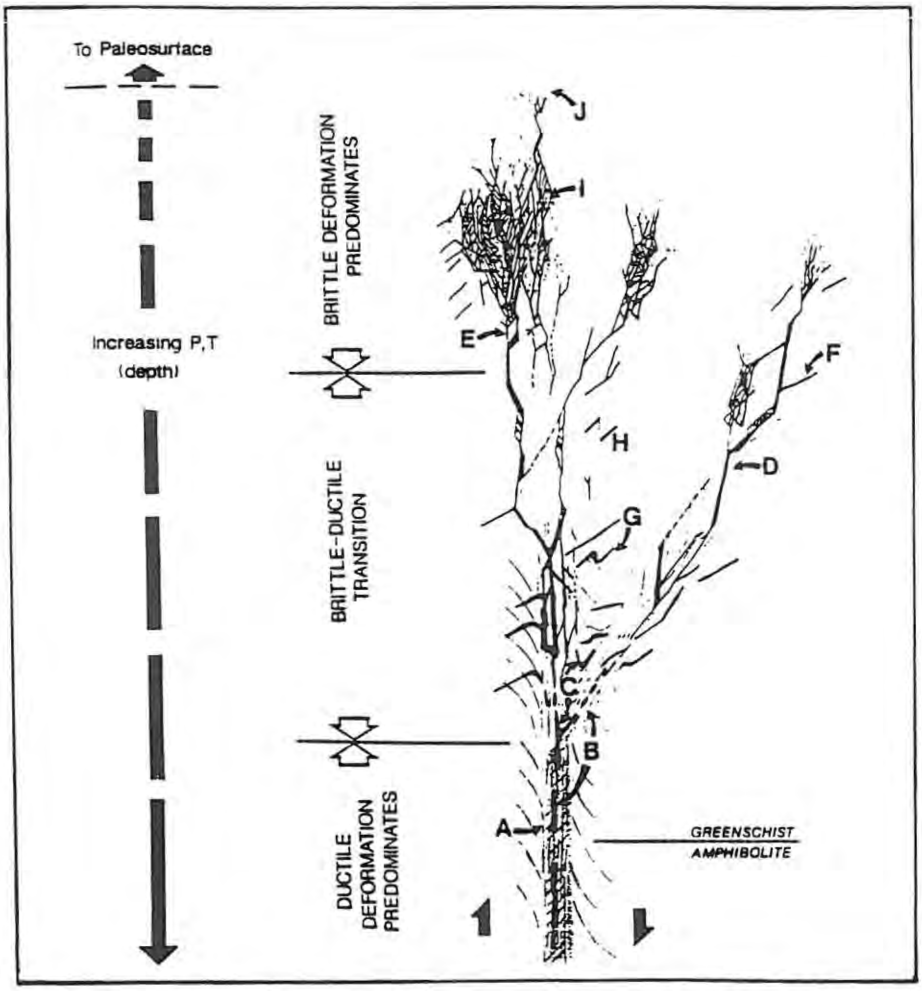


FIGURE 3.8. Schematic depositional model for Archaean lode gold deposits, showing progressive change in structure with depth below paleosurface. The structures illustrated include: (A) replacement veins, (B) boudinaged veins, (C) saddle reef veins, (D) Riedel shear veins, (E) "break ore" veins, (F) undeformed, and (G) deformed tension veins, (I) breccia veins, and (J) pervasive silicification (from Colvine et al., 1988).

The ore mineralogy of Archaean lode gold deposits is usually simple. Pyrite and/or arsenopyrite and/or pyrrhotite are the predominant sulphide minerals with pyrrhotite more common in deeper (higher metamorphic grade) deposits (Colvine et al., 1988; Groves et al., 1989). Submicroscopic gold usually occurs in the native state or as electrum; often intimately associated with the sulphide minerals. Other minor minerals include scheelite, base metal sulphide minerals (chalcopyrite, galena, sphalerite), stibnite, tourmaline, carbon, magnetite, hematite and anhydrite. The geochemical association is normally Au, Ag, As, W +/- Sb, B, and Te with low Cu, Pb and Zn.

In both epithermal and mesothermal gold deposits the interpretation of the structural features on both a regional and local scale is vital to evaluating the deposit. Numerous examples show that the size, attitude and continuity of the orebody depends largely on the history and style of deformation. At first glance the pattern of shear zones, faults, foliation and veins may appear haphazard. However, a careful analysis of the kinematic indicators - especially the movement sense and direction on shear zones and faults - is usually able to show that the gold is located in dilational structures that have a consistent and predictable orientation. These may be modelled according to strike-slip, dip-slip or oblique-slip movement senses. Some caution is always required as more than one deformation and gold mineralising event may have occurred. Pre-existing structures can be reactivated and mineralised, and later movement may rotate, fold and displace the orebody. A full appraisal of the structures associated with epithermal and mesothermal gold deposits is beyond the scope of this report. Hancock (1985), Mclay (1987) and Hodgson (1989) offer excellent interpretations of the structures that may be found at the gold prospects.

3.4. LATERITE GOLD DEPOSITS

These deposits are formed near surface by lateritization - a supergene weathering process which dissolves, mobilizes and reprecipitates gold within the surface bedrock and soil. Until recently the nature of gold laterite deposits had been poorly documented. However, as the mechanisms of their genesis are becoming understood, an increasing number of deposits are being described. The best known examples are from the Yilgarn Block, western Australia (Mann, 1984 a and b; Butt 1989), but other examples are known from Fiji (Lawrence, 1984) and South America (Michel, 1987).

Laterite is a generalized term used to describe any surface or near surface material composed largely of iron oxides and hydroxides derived by the upward and lateral movement of ground waters. Not all laterites contain gold. In fact laterite gold deposits are the surface expression of primary mineralisation. A separate classification is, however, justified as the character of the secondary mineralisation is quite different from the primary mineralisation. These differences include the overall mineral assemblage and geochemical associations of the ore, the composition of the gold, the conditions of gold mobilization and reprecipitation, and the nature of the transporting fluids (Butt, op. cit.).

Research indicates that the essential requirements needed to create a laterite gold deposit are: 1) primary gold mineralisation of any origin that out crops at surface; 2) a source of iron, derived either from the decomposition of pyrite from the primary orebody or magnetite from basic rocks; 3) a hot and humid tropical climate, either recent or palaeo; 4) a gold transporting agent such as an organic, chloride or thiosulphate complex; 5) a gold fixing agent such as ferrous iron; and 6) a stable tectonic environment with low relief and a long subaerial history devoid of any glaciation or other strong erosional force.

It is beyond the scope of this report to discuss how the gold is dissolved, mobilized and reprecipitated. The controls make for interesting reading and more discussion on the subject is presented by Krauskopf (1951), Lakin et al. (1974), Boyle (1979), Webster and Mann (1984), Mann (1984 a, b) and Butt (1989).

The laterite profile is normally divided up into three zones which differ in their mineralogy and physical properties. These zones together with the gold distribution that may be encountered over a typical lode gold deposit from western Australia are systematically illustrated in Fig. 3.9.

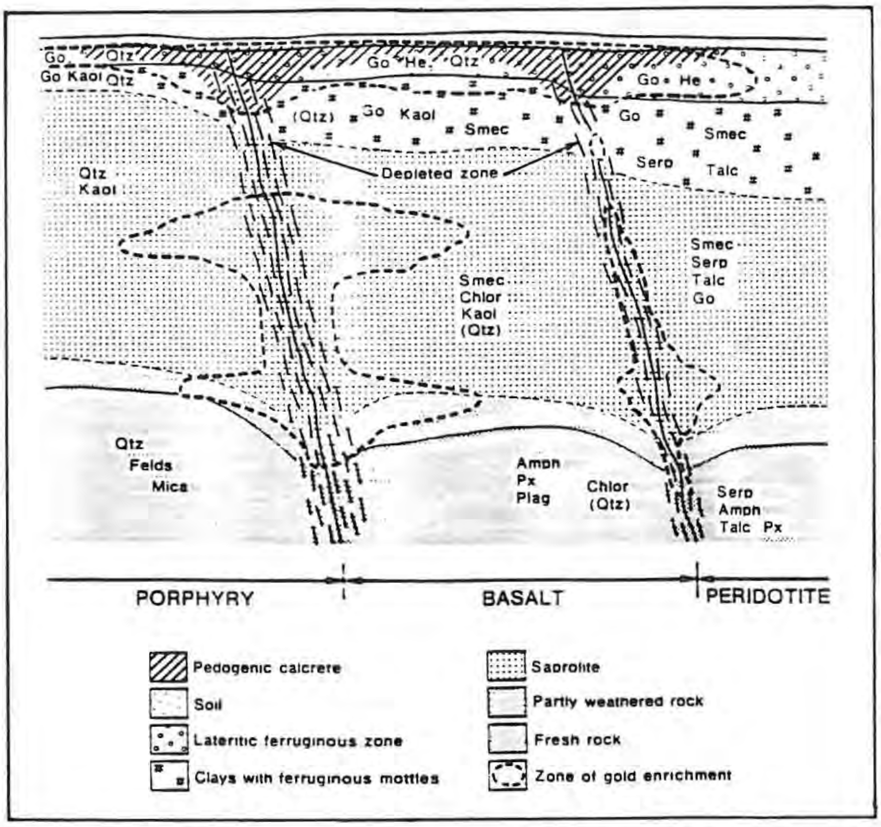


FIGURE 3.9. Schematic cross section of the typical laterite gold deposit above a primary epigenetic gold vein from western Australia. Pedogenic calcrete is only present in present-day arid regions (from Butt, 1989).

From top to bottom the laterite horizons are soil, laterite and saprolite.

1) Soil. The soil horizon need not be present, but where developed it is usually less than 1 metre thick.

2) Laterite. The laterite or iron-rich horizon may comprise an upper pisolitic layer or ferricrete consisting of coarse ironstone pebbles cemented by iron-rich material and a lower mottled zone consisting of iron-rich concretions and weathered bedrock. The dominant colour of this zone is dark to pale reddish-brown. Minerals present include mainly goethite, gibbsite and hematite with subordinate amounts of maghemite and quartz. Kaolinite and minor smectite become abundant in the mottled zone. At the time of lateritization this zone was water saturated with the water table at the zone of maximum ferruginization. The laterite zone is typically 1 to 10 metres thick.

3) Saprolite. The saprolite is typically pallid and consists mainly of kaolinite with minor quartz, mica, gibbsite and hematite. This zone is residual in origin and all primary textures are still preserved. The saprolite is typically 5 to 40 metres thick and extends gradually downwards into fresh bedrock.

Gold may be present as native gold and/or electrum and can occur in all three horizons. According to Mann (1984a, b), a large portion of the gold occurring in the oxidized vein is residual in origin, i.e. it has retained the physical and chemical characteristics of its high temperature precipitation. However, the original sulphide minerals have been decomposed to form new iron oxides and hydroxides. The gold adjacent to the vein is clearly not residual and has been moved laterally near the palaeo water table during lateritization. This "secondary" gold produces the characteristic T-shape orebody which may extend tens of metres on either side of the vein.

In some cases erosion of the land surface may result in the mechanical enrichment of gold on surface, but where chemical mobilization is the sole transporter, the grade is often less than the primary vein or lode. Within the laterite profile, gold commonly shows an increase in grade and purity in the mottled zone where it is erratically distributed and concentrated with iron-rich concretions. Large nuggets, many in excess of 1 kilogram, have also been reported from the laterite horizons. These nuggets are enigmatic. Whilst some are residual, others appear to be related to the lateritization process. The latter appear to have grown by additional precipitation of gold and iron oxides on pre-existing gold grains, as evidenced by the higher purity (low silver content) of the outer gold coatings (Mann, 1984a; Wilson, 1984). The erratic gold distribution and varied size of gold particles may be a major problem in obtaining an accurate grade estimate.

In detail the laterite profiles and gold distribution pattern will vary according to the local geomorphology, topographic relief, climatic conditions, underlying host rock, gold dispersion mechanism and attitude of the primary deposit. For instance economic concentrations of gold generally form flat-lying enrichment zones contiguous with the pisolitic and mottled zones. However, complicated ore zones can arise where the weathering process changes. Butt (1989) reports that with uplift and subsequent lowering of the water table, many of the Australian laterites developed a second or third T-shaped zone of gold enrichment lower down in the saprolite horizon. Leaching of gold from the vein may also result in a depleted zone near surface. Alternatively removal of the surface gold enrichment by erosional processes can produce blind laterite gold deposits which are obviously difficult to discover and evaluate.

4. ORE RESERVE ESTIMATION

The purpose of this chapter is to: 1) examine the factors that need to be assessed in order to make an accurate ore reserve estimate; 2) discuss the reasons why the predicted reserve estimates sometimes differ radically from the realized reserves; and 3) briefly examine the requirements of a grade control programme. In this regard a holistic approach is adopted by examining the importance of geology, sampling, assaying, calculation method, mining dilution, and ore and gold loss factors. I hope to show that ore reserve estimation is not just a calculation. In fact the calculation is one of the least important parts of estimating the ore reserves.

4.1. DEFINITION OF RESOURCE AND RESERVE

Before examining the factors affecting ore reserve estimation it is first necessary to define what is meant by resource and reserve. These terms are always a subject of confusion and the terminology is more complex that might be imagined. As yet there is no worldwide accepted code of reserve classification. Rather it seems that each country has adopted its own classification, which, instead of clarifying the situation has simply added to the confusion. It is beyond the scope of this report to discuss the various classifications and contexts in which the terms are used. Articles by Taylor (1972, 1986, 1990), Esterhuizen (1983), and Vallee (1986) provide useful definitions and further discussion on terms such as ore, reserves, mineral inventory, resources and grades. For the purposes of this report the 1988 classification adopted by the Australian Institute of Mining and Metallurgy is the most practical and will be used (Fig. 4.1). The following definitions apply.

Resource is an in situ (meaning as it occurs on surface or underground) mineral occurrence quantified on the basis of geological data and a geological cutoff grade only. Such in situ or geological resources may be defined further into inferred, indicated and measured categories depending on the increasing degree of geological knowledge and confidence.

Reserve refers to that part of the measured or indicated resource, which has been classified into mineable tonnes or volume and grade using technical and economic criteria. Only proven and probable categories are quoted. Possible reserves are eliminated from the code.

Thus mineable ore reserves are those reserves which will be delivered to and treated by the metallurgical plant during production. This definition requires that the ore and gold loss, and dilution gain caused by the physical process of mining and ore treatment be incorporated into the calculation of reserves. Only mineable reserves are used in the economic analysis of the mineral prospect. Appendix 3 elaborates on the subcategories of resource and reserve used in the code.

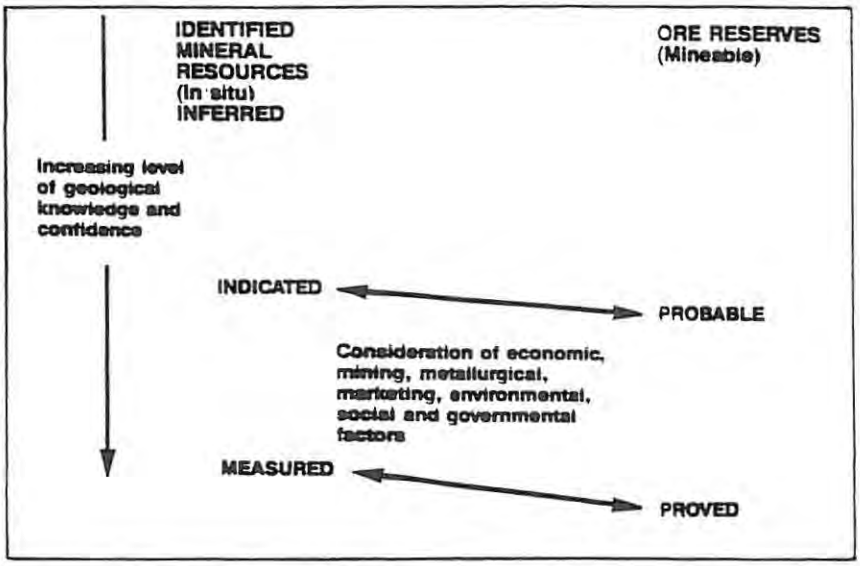


FIGURE 4.1. Resource and reserve reporting terminology adopted by the Australian Institute of Mining and Metallurgy (1988).

4.2. OBJECTIVES AND SCOPE OF ORE RESERVE ESTIMATION

The objective of estimating the ore reserves is to quantify in probabilistic terms what mineable reserves can be expected within a given mining volume - the open pit envelope; and estimate these reserves according to their grades, tonnage, spatial distribution and types.

Grade is often the most important factor as it is very sensitive to the cash flows of the mining venture. The three basic grade calculations required are: 1) the overall mean grade with confidence limits; 2) the overall cumulative distribution function (cdf) of the selected mining units (SMUs); and 3) the estimated grade of each SMU with confidence limits. Items (1) and (2) comprise the global ore reserve estimate whilst item (3) would normally be needed to perform open pit optimization. The tonnage affects the size of the operation and is therefore more sensitive to the capital cost of the mining venture. The spatial distribution of the mineable reserves is vital to optimizing the design of the pit which in turn affects the short- and long-term profitability of the mining venture.

Obviously the most profitable mines will have high-grade ore near surface. If high-grade ore is only located at the bottom of the pit then an underground operation may prove more economic. The types of reserves affect the treatment and rock handling processes. It is not uncommon today to find open pit gold mines treating and handling rock classified as waste, low-grade waste, run-of-mine (ROM) heap leach ore, agglomeration heap leach ore, and milling ore. The different types of reserves and materials are therefore critical to mine planning.

In chapter 2 (Fig. 2.5), ore reserves were shown as forming the foundation block to the whole mining venture. This needs to be stressed, as, unlike the design of the open pit or choice of mill, the orebody is unalterable. Gross errors in the reserve estimate will have greater adverse consequences than say a wrongly selected bench height. This does not mean that a conservative estimate of the ore reserves is required. The investor only wants an accurate estimate of the probability of obtaining the specific grades, tonnages, distributions and types of reserves. A conservative estimate may prevent investment in an otherwise profitable venture, whilst an optimistic estimate may result in investment in an otherwise unprofitable venture.

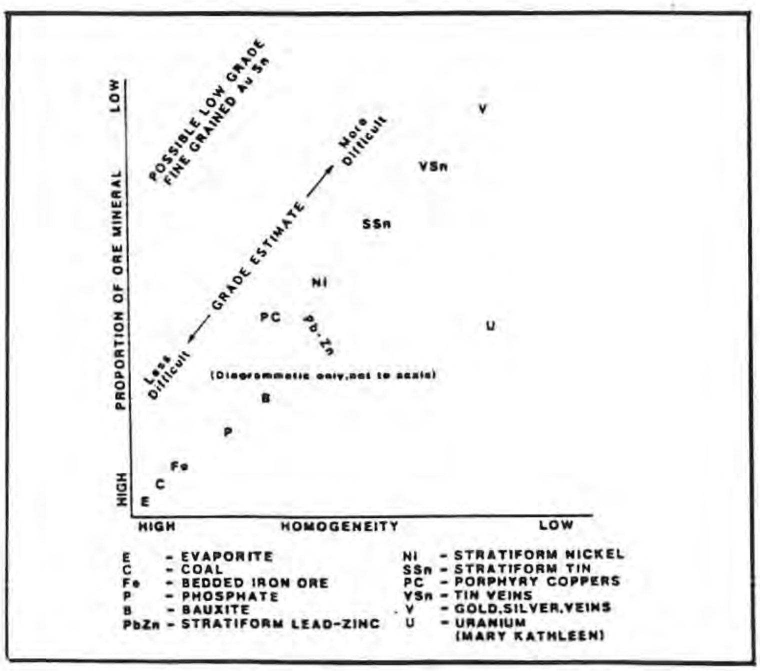
Also to be considered is the possibility of failure. Invariably when a new mining venture fails, the causes can be traced to the predicted grades and gold recoveries not being realized. In selective open pit gold deposits this is relevant for two reasons. Firstly, the grades may already be marginal so that no error of misjudgment can be absorbed. Secondly, additional reserves are unlikely to be found as the surface may already have been fully explored. Gross errors in estimating the reserves must therefore be avoided, not only because it demonstrates that the evaluation team is incompetent, but more important because investors will lose confidence in the mining company. The repercussions resulting from lack of confidence can persist for several years and will adversely affect equity funding for other investments.

Conversely it is important that the investors appreciate that the reserve is only an estimate. It is not a precise prediction - though the precision by which reserves are calculated may imply otherwise. One always needs to stress confidence limits when ore reserves are quoted in the feasibility studies.

Despite the importance of ore reserves, the orebody is the most difficult item to evaluate. Two main reasons account for this difficulty:

1) there is always a limited amount of sampling data, both with respect to the size of the sample (samples are too small) and the sample spacing (spacing too great). Although selective open pit gold deposits enjoy obvious advantages over underground deposits by being close to surface, the amount of material sampled is unlikely to represent more than 0.1 per cent of the entire orebody, i.e. statistically a <u>very</u> small amount. To make the correct prediction of the ore reserves from such limited data is impossible, no matter how sound the reserve estimation procedure may be; and

2) there are difficulties caused by the high variability (low homogeneity) and low concentration of gold. It is not uncommon to be sampling gold concentrations of less than 1 part per million (ppm). This problem is highlighted in Figs. 4.2 and 4.3, which show that vein and lode gold deposits are not only the most difficult category of deposit to evaluate in terms of grade, but also the most difficult to define in terms of estimation techniques, metallurgy, mining and risk exposure. Clearly, the exploration geologist, mining engineer and statistician have an unenviable task of predicting the ore reserves and it is hardly surprising that the estimates are quoted in probabilistic terms.



<u>FIGURE 4.2.</u> Proportion of ore mineral versus homogeneity for equal size samples. Note the difficulty in evaluating gold deposits compared to other common classes of deposits (modified from Bujtor and McMahon, 1983).

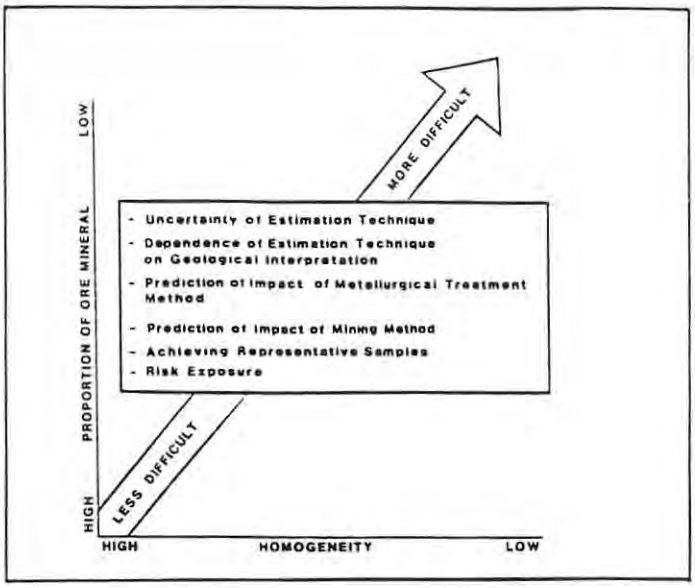


FIGURE 4.3. General trend of increasing difficulty in deriving ore reserve estimates (from Bujtor and McMahon, 1983).

The main factors needed to define and estimate ore reserves are geology, sampling and assaying, resource/reserve calculation, and economic, mining, metallurgical, environmental and other factors. Fig. 4.4 shows how these factors interrelate during the ore reserve estimation procedure. During the geological and preliminary mine feasibility stages a global reserve estimate is required. However, during the final feasibility stage a local reserve estimate - giving the estimated block distribution within the open pit - is required. This latter task is performed during the pit optimization stage and is not discussed further. Nevertheless it is important to realize that pit optimization should be performed and the data must be suitably computerized from an early stage. Note that the cutoff grade is a function of the cash flow expectations. These will change according to the prevailing economic conditions, mining plan, tonnage to be milled and treatment methods. To further illustrate the interrelationship of these factors, the geology, sampling and assaying, and in situ resource and mineable reserve calculations are discussed below.

4.3. THE IMPORTANCE OF GEOLOGY

Geology is the most significant factor affecting all ore reserve estimates. This statement cannot be over emphasized and it may be added that an ore reserve estimate should not be accepted until the geology is understood. Bujtor and McMahon (1983) illustrate this point with a section

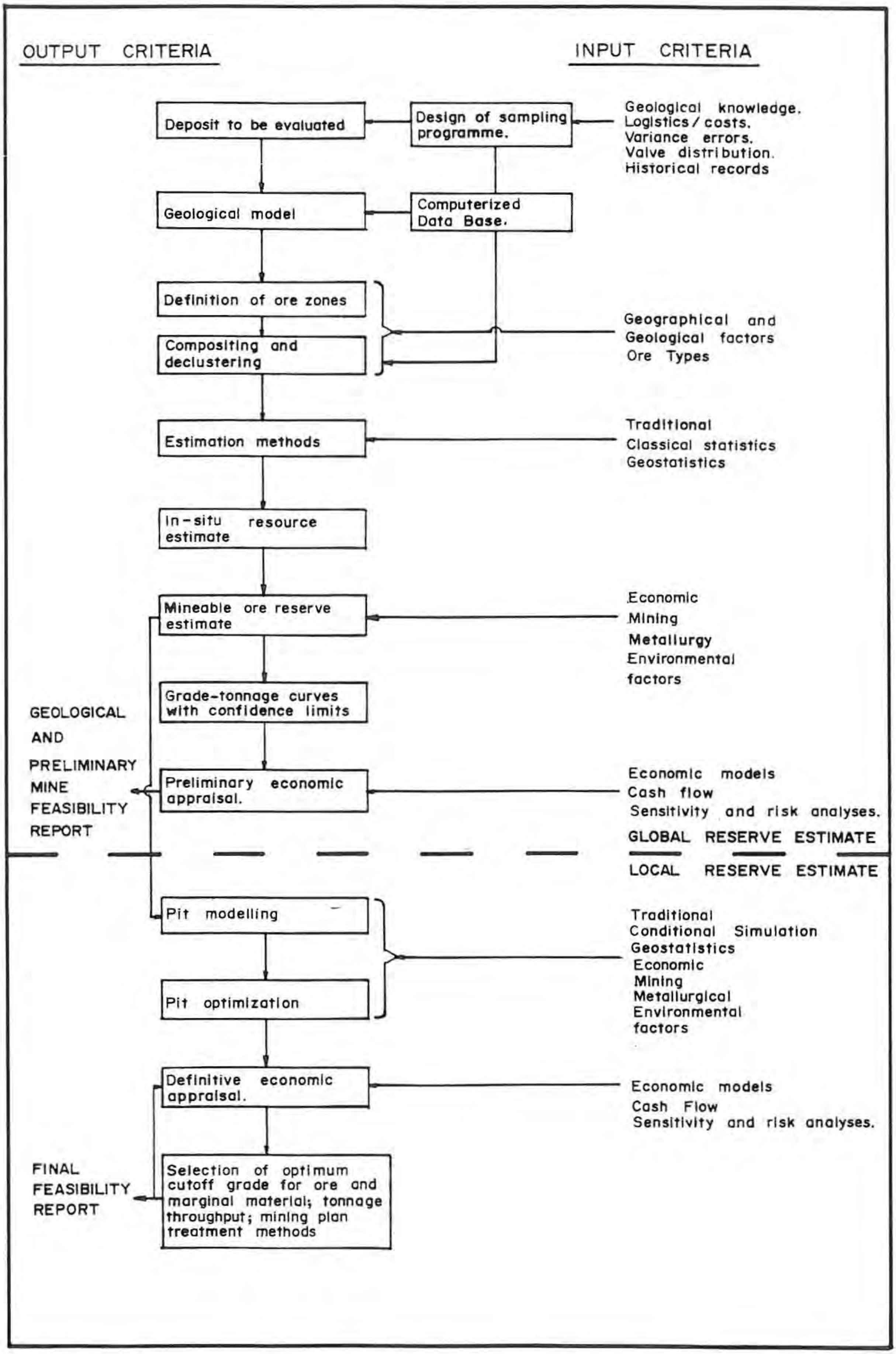


FIGURE 4.4. Interrelationship of main components required during ore reserve estimation procedure.

through part of the Stekenjokk Mine in Sweden (Fig. 4.5).

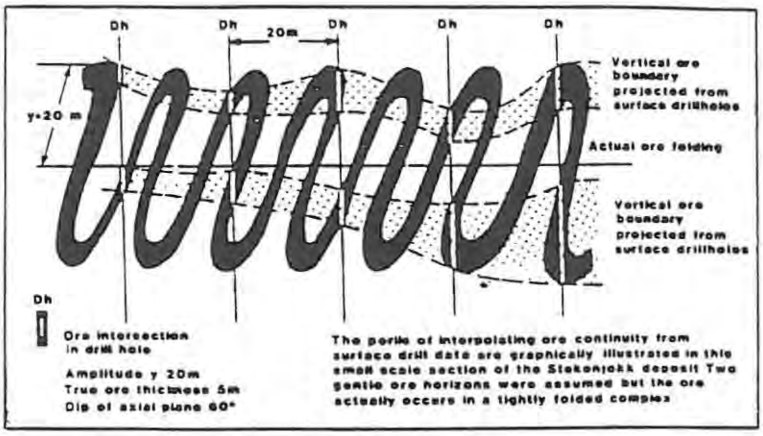


FIGURE 4.5. Geological section of the Stekenjokk Mine (from Bujtor and McMahon, 1983).

Stekenjokk is not a selective open pit gold mine, but the figure clearly shows how the initial geological interpretation of the orebody - as based on boreholes - differed radically from the actual folded shape of the orebody. Although this misinterpretation is inexcusable - for instance the ore zone intersection angles and the lower ore intercept in the second drill hole from the left should alert any geologist - no investor would be pleased to learn that his potential profits had vanished because the shape of the orebody had been wrongly interpreted. The answer is clear. If the orebody cannot be sufficiently well defined from drill holes then it should be opened up and exposed, either by underground or surface excavations. Of course, knowing the geology does not imply that the reserve estimate is accurate. During the early stages of exploration the estimate is poorly constrained but as more geological information becomes available, the calculations are likely to become more sophisticated with a concomitant improvement of the confidence in the reserve estimate.

4.4. SAMPLING PROGRAMME DESIGN

Reliable and representative sampling is the primary means by which the dimensions and grade of the orebody are estimated. Besides geology, getting sampling right is the second most important part of the estimation programme, as any errors will carry through and invalidate all subsequent stages of the ore reserve estimation. The phrase "garbage in - garbage out" is as true here as anywhere. No mathematical manipulation of the data can account for poor quality sampling (Bujtor and McMahon, 1983).

The initial concern in designing the sampling programme is to minimize errors. The errors in the sample values may be systematic and/or random, and can be expressed in terms of variance S^2. Thus:

$$S^2 = S^2_{ntr} + S^2_{smpl} + S^2_{prc} + S^2_{anal}.$$

where:

S^2_{ntr}—natural or regional variance of the element content in the sample medium;
S^2_{smpl}—variance due to sampling;
S^2_{prc}—variance due to sample processing;
S^2_{anal}—variance due to analyses. (From Nichol, 1986).

For reliable sampling, the variances generated by sample collection, processing and analyses must be minimized as these reduce the accuracy of the reserve estimate. Ideally only the natural (S2ntr) variability is required. However, some errors are inevitable. Of the two types, systematic errors are the most serious as they introduce a bias in the reserve estimate, which, if undetected cannot be corrected (Grant, 1981). Random errors are less serious as they can be modelled in the variogram.

Sampling is also the most costly and difficult part of the delineation programme to repeat. Plenty of care and planning should go into the design and execution of the sampling programme. Increasing costs mean that the exploration geologist needs to design a cost effective programme, which balances expenses in terms of money and time with the fundamental sampling requirements. The fundamental requirements are to ensure that an adequate number of reliable and representative samples are obtained over the whole orebody. Eventually the aim of the sampling programme is to build up a comprehensive data bank from which to model the gold deposit's geology and estimate the mineable reserves.

In this subsection, sample representativeness, drill programme design (sample collection), sample processing and size, and analysis techniques are discussed. Selected articles dealing with these topics include Neuhoff (1980), Grant (1981), Nichol (1986), Rendu (1986) and Shaddrick (1987). Since the reserve estimate will largely rely on drill hole information, only this evaluating technique is considered, though it is recognized that augering and backhoe trenching provide valuable sampling near surface.

Sample representativeness. A major problem encountered whilst evaluating vein and lode gold deposits is obtaining a representative sample. This affects the reserve estimates and mining method. Consider for example Fig.

4.6 which shows a vertical drill hole intercept through a steeply dipping vein. In this situation the rock adjacent to the vein is not covered by the drill hole so that mining dilution cannot be predicted. Also, unless the core and vein intersection angles are known, there is a strong chance of overestimating the grade of ore within the vein.

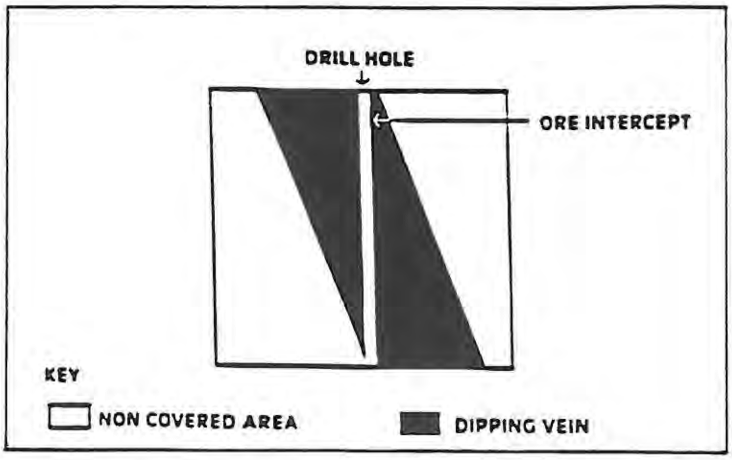


FIGURE 4.6. Representative drilling and vein dip (from Carras, 1986).

Ideally the borehole intersection should be perpendicular to the vein in order to obtain a representative sample. This may not always be possible where several veins with different attitudes are present. In addition, where the mineralised structures dip steeply (greater than 50 degrees), it becomes more difficult to collar the drill hole at angles shallow enough to obtain a representative sample.

Drill programme design. In virtually all cases today drilling is by diamond core and reverse circulation (RC) percussion drilling. Conventional percussion drilling is seldom used because of the more severe contamination problems encountered with this technique. The choice of drilling technique depends largely on four conditions: rock quality, openness, groundwater conditions and mode of gold occurrence (Shaddrick, 1987). Rock quality refers to the ability of the drill to penetrate the rock cleanly and effectively, and to return a good sample. This may be hampered by fracturing, shearing or alteration which can cause blocking of bits, binding of rods and hole caving or swelling. Openness reflects the extent to which fluid circulation (water, mud and air) can be maintained during drilling. Fluid circulation can be lost via cavities, faults, vugs, and underground workings. Groundwater refers mainly to water in fissures. Water severely hampers RC percussion drilling by "clogging" the hammer. The mode of gold occurrence refers to whether gold is firmly bound in the rock and therefore easily retrieved with the sample, or is loose and likely to be lost. For instance gold occurring in laterites is loosely bound and best drilled with RC percussion drilling; core drilling will tend to dilute and wash the fine gold away.

In practice both core and RC percussion drilling may be used since each method offers different advantages in terms of geological, engineering and metallurgical requirements. However, once selected the technique should not be changed. Statistical problems with treating more than one sample population are created if the drilling technique and method of sampling and assaying is changed halfway through the drilling programme, i.e. all samples must have the same support.

The advantages (+), and disadvantages (-), of the two drilling methods are set out below.

Core drilling.

(-) More expensive and time consuming method.
(-) Poor gold recovery where gold is loosely bound.
(-) Cannot obtain good core recovery within highly broken, open, vuggy, poorly cemented or weathered rock, even with the aid of a triple core barrel.
(-) May experience difficulties when the rock formation is alternatively hard and soft.
(+) Provides solid core for observation of structure, lithological relationships, vein intercept angles, ore and waste boundaries etc.
(+) More appropriate for mineralogical and geotechnical studies.
(+) Can drill shallow inclined holes as well as vertical holes.
(+) Only method not significantly hampered by excess water or loss of drill fluid circulation.

Because of the substantially higher costs, core drilling is favoured where geological definition of the orebody is required or where geotechnical conditions need to be assessed.

Reverse circulation percussion drilling.

(-) Only provides rock chips for logging. However, careful observation with a binocular microscope can reveal enough relevant information on the degree of oxidation, mineralogy, veining and lithology.
(-) No detailed geotechnical logging can be performed.
(-) Cannot drill where groundwaters are present.
(-) Cannot drill effectively at angles of less than 60 degrees from the horizontal.

- (-) Cannot drill effectively at vertical depths greater than 80 metres.
- (-) Cannot drill quickly through very hard rock.
- (-) May not be able to recover coarse gold from the base of the hole.
- (+) Significantly less expensive and faster than core drilling. RC is about half the price per metre of core and up to four times as fast as core drilling in soft ground (personal observations).
- (+) Continuous uncontaminated samples can now be obtained from surface using specially designed hammers.
- (+) Recovery is better than core in broken and weathered formation.
- (+) Able to drill through small cavities and stopes.
- (+) Best method for recovering loosely bound gold particles.
- (+) Less affected by drilling through alternatively hard and soft rock formations.
- (+) Provides large sized samples (about 30 kilogram mass), ideal for bottleroll metallurgical testwork.

Because of the substantially reduced costs in terms of time and money, RC percussion drilling is ideally suited to infill drilling, provided of course that ore zone continuity has been established by core drilling.

In recent years the design of the percussion hammer has improved and a point has now been reached where RC percussion drilling compares favourably with core drilling in terms of sample return. This was demonstrated by Prochnau (1990) at the South Mercur gold project, Nevada, where three holes were drilled adjacent to one another using core, crossover RC percussion and the new Samplex RC percussion techniques (Fig. 4.7). The findings demonstrate the very close correlation between individual Samplex and core assays but suggest probable down-hole contamination of gold values with the crossover system.

The optimum drill hole spacing and down-hole sampling interval depends on the variability of the gold distribution and the delineation funds available. The gold variability can be determined by constructing directional variograms. Ideally samples should be taken in the strike direction, in the dip direction and perpendicular to the dip direction of the ore zone. If prior information is not available, then some closely spaced drill holes or wedges drilled early in the delineation programme, may be necessary to determine the grade variability. Separate variograms can then be constructed in these three directions and the range of influence used to determine the minimum spacing between holes and down--hole sampling intervals. Ideally what is required are evenly spaced drill

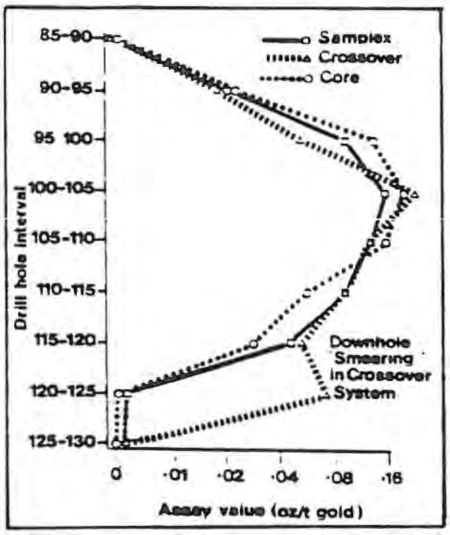


FIGURE 4.7. Correlation of gold values in Samplex, crossover and core drill systems, South Mercur project, Nevada, (from Prochnau, 1990).

holes which provide enough coverage of both the high- and low-grade areas to ensure a confident reserve estimate. Very often, due to limited funds or a narrow range of influence, the desired drill hole spacing is never achieved. In this situation the best approach is to start with widely spaced drill holes and then conduct infill drilling as funds permit. One point that must be stressed, however, is that the individual sample lengths should not be larger than the SMU as it is fundamentally impossible to estimate the reserve of a block which is smaller than the sample size; i.e. a lot of small, say 25 centimetre, equal support samples would be preferred for a 5 metre bench height. Prior knowledge of the selected mining method is therefore useful.

Sample processing and sample size. Sample processing is the process by which samples collected in the field are reduced to smaller sizes for assaying. This commonly involves splitting the core or riffling of percussion chips in the field, followed by crushing, pulverizing and further splitting of the sample into sub-samples in the laboratory. The aim of sample reduction is to ensure that the portion chosen for analysis represents the metal content in the whole of the original sample. Significant errors can be generated in sub-sampling due to the heterogeneity of gold in the original sample and the inhomogeneity in the crushed sample.

Heterogeneity occurs where the gold is erratically distributed. Nichol (1986) illustrates this situation in Fig. 4.8, where two samples of 100 grams each have the same gold content. In the heterogeneous sample, A, all the gold is contained in a single grain, thus nine of the ten 10 gram sub-samples taken for analysis would contain no gold and one sub-sample would report ten times the true value. In sample B, the gold is distributed evenly and every 10 gram sub-sample will give a representative analysis.

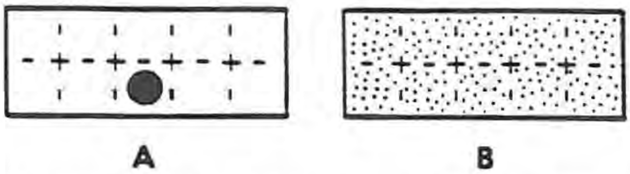


FIGURE 4.8. Variations of representivity of sub-samples with particle size of gold grains for a constant gold content (from Nichol, 1986).

In cases where the gold is known to be erratically distributed, the size of the sample must take into account the distribution and diameter of the gold particles. Nichol (1986) reviews two theoretical studies regarding sample size originally presented by Gy (1954) and Clifton et al. (1969). Clifton et al. (op. cit.) present the easiest method to follow and according to their findings, a sample must contain at least twenty gold particles of a given size to be representative. Fig. 4.9 graphically presents the required sample weight containing twenty gold particles as a function of gold particle size. The results assume that all gold particles are randomly distributed and of uniform size. Although this is rarely the case in nature, the graph can be used to determine the minimum size of the sub-sample needed to provide the necessary level of accuracy during analysis. For example, assuming gold occurs as spheres of 62 microns a sub-sample containing 1 ppm will contain 20 gold particles per 60 gram sample, and a sub-sample containing 250 ppb will contain 20 gold particles per 200 gram sample. If one assumes that the gold is coarser than 62 microns (as illustrated in Fig. 4.8, sample A), and the level of interest is above 250 ppb, then the analysis of say a 50 gram sub-sample will theoretically be unreliable. To be safe, the sub-samples must be large enough to ensure that the most severe heterogeneity is accommodated.

To reduce errors caused by inhomogeneous sub-samples, the samples submitted to the laboratory must always be pulverized (Grant, 1981). This will ensure that homogenous sub-samples are split from the sample prior to analysis. Crushing by itself or crushing of the sample followed by splitting and pulverizing of the sub-sample is inadequate and can lead to erroneous grade determinations.

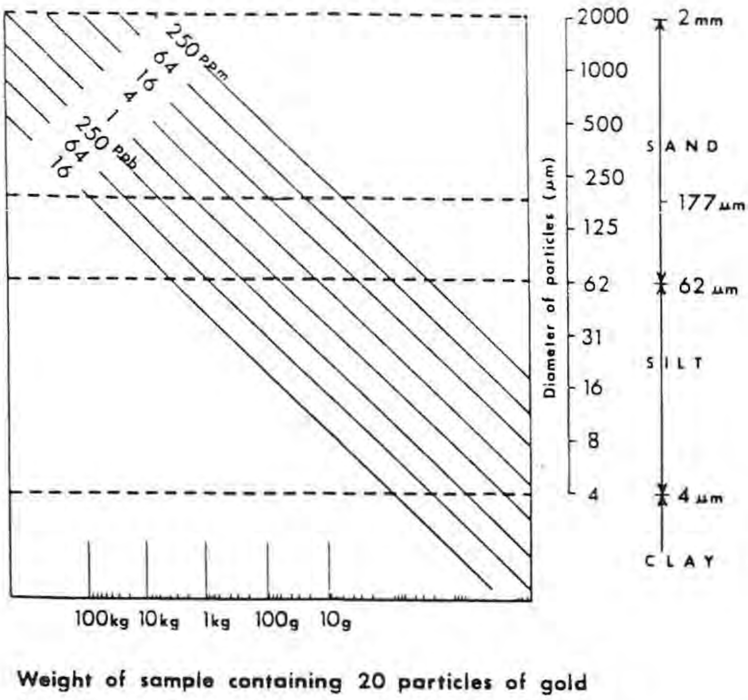


FIGURE 4.9. Weight of sample containing twenty particles of gold as a function of gold particle size and content, assuming all gold particles are randomly distributed and of uniform size (after Clifton et al., 1969; from Nichol, 1986).

Sample analysis. This can be a further source of error. However, today's instruments are very precise and provided the correct analysis techniques are employed the errors should be minimal. Fire assay and bulk leach are the two analysis techniques most commonly used. Fire assay has proved historically successful and is still preferred as it allows large sub-samples of about 100 grams to be analysed. In situations where the gold is known to be non-refractory and erratic (as in gold laterite deposits), then bulk leach may be more suitable. Bulk leach dissolves gold using cyanide solvent and commonly uses 5 to 10 kilogram samples. These large samples may help reduce problems caused by heterogeneity. The accuracy of the analyses should be checked using duplicate and blank samples of known gold content. The precision of the techniques should be checked by submitting returned pulps to a second laboratory (Grant, 1981).

4.5. IN SITU RESOURCE ESTIMATION

To estimate the in situ resource it is necessary to perform four main tasks: 1) define the ore zone; 2) composite the samples; 3) decluster the samples; and 4) calculate the tonnage and grade of the resource. These items are discussed in turn below.

Definition of ore zone. On the basis of the results obtained from the sampling programme, one of the first tasks is to define a three dimensional block model of the orebody. There are likely to be numerous individual orebodies within the open pit, each one requiring definition and resource estimation. Typical features that need to be built into the three dimensional model are, after Rendu (1986):

- topography;
- location of mined out and caved areas if the deposit has already been partially exploited;
- bedrock and overburden interface;
- water table position;
- depth of weathering;
- location of leached areas and secondary enrichment;
- oxide to sulphide interface;
- location of faults;
- rock types or the interface between rock types;
- limits of dykes or other pre- and post-mineralisation intrusions;
- limits of continuous ore zones and different types of ore;
- limits of ore zones between which significant differences in anisotropies are expected; and
- limits between ore zones within which reasonably constant grades are expected.

Compositing. The purpose of compositing sample values is to provide a sample representative of a particular unit or mining unit that can be applied through some extension function to estimate the grade or value of a much larger volume of the same unit (Barnes, 1980). The key to compositing is to provide samples with equal support and use the smallest common sample size. It must be remembered that the composite length should not exceed the proposed bench height. Some prior knowledge of the mining methods is therefore required.

Declustering. Depending on which calculation method is used, declustering may be required. Declustering is performed in areas where the samples are too closely spaced relative to the sample spacing over the rest of the orebody. This is often the case in high-grade areas which are inevitably drilled more intensely than low-grade areas. If the high-grade values are not declustered then the global mean grade estimate will overestimate the true mean grade when classical statistical methods are used. Kriging and geometrical estimators automatically decluster.

Resource calculation. The resource calculation methods are extension functions, used to extend sample values to estimate the value of the surrounding blocks. There are numerous methods used in industry to calculate the resource and it would be impractical to try and discuss every type. The intention here, is to examine the applications and failings of the most common methods. These include traditional methods, such as inverse distance weighting, and polygonal and sectional (geometrical) estimators; classical statistical methods, such as the arithmetic mean and Sichel's t estimators; and geostatistical methods which include various kriging estimators. The actual calculation procedures are not described as these are covered in a number of publications. Relevant articles dealing with the calculation procedures include Barnes (1980) for computer assisted resource estimation; Rendu (1978) for classical statistical methods; Kwa and Mousset-Jones (1988) for traditional methods; and David (1977), and Clark (1979) for geostatistical methods.

During pit optimization, more advanced disjunctive kriging or conditional simulation methods may be employed. Conditional simulation arguably offers the best prediction of the actual grade distribution within the pit. It is a complex procedure and until recently required a mainframe computer. However, more powerful personal computers are now changing this situation and conditional simulation can be applied. The method is particularly useful where there is limited data, as is commonly the case during the critical feasibility stage. Further information on conditional simulation is presented by David (1977, p. 323) and Journel (1979).

Before describing the methods it is worth discussing what the extension functions must do in order to give an accurate resource estimate. The main problem is the variance. The concept of variance, whether it be related to grade, ore zone thickness or specific gravity is a key element in understanding the behaviour and accuracy of the estimators. In nearly all cases grade is the most critical and difficult to estimate because gold is usually erratically distributed. Here, two fundamental concerns exist: the concept of extension and the concept of error of estimation (David, 1977). In other words, knowing the sample grade, what is the block grade and how confident are we of the accuracy of that estimate.

Gold variance between samples can be split into spatial and random components as determined by the variogram (Royle, 1978, 1979; Fig. 4.10). The spatial variance (C) represents differences between sample values taken at points separated by increasingly greater distances. The range of

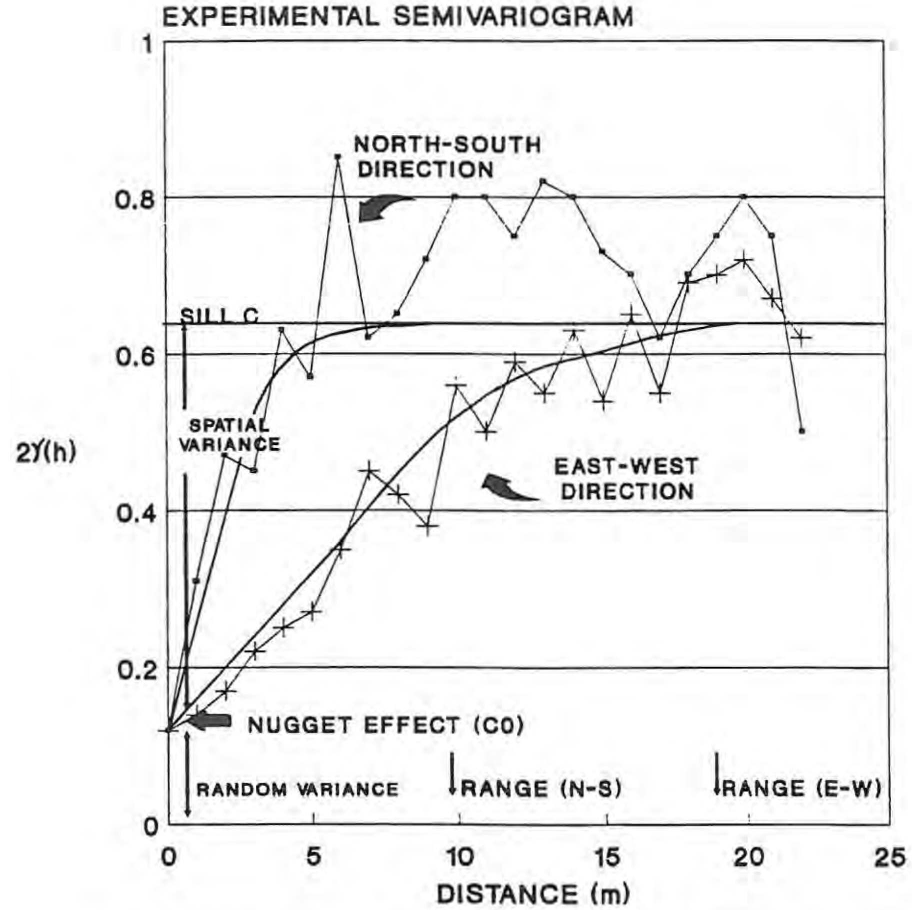


FIGURE 4.10. Major elements of the spherical model experimental variogram showing east-west and north-south anisotropy.

influence represents the distance up to which values between sample points can be correlated. The nugget effect (Co) represents very short range variances, as, for example when two halves of the same drill core produce different assays. Co also contains the random errors of sampling and assaying. The relative nugget effect, E, is the ratio of the random and spatial variances. Thus:

$$E = \frac{\text{Random component of the variance}}{\text{Spatial component of the variance}} = \frac{Co}{C}$$

Therefore ore zones with a large random component of variance have a high nugget effect. Since the nugget effect cannot be predicted beyond the drill core, any estimator which does not recognize the influence of the nugget effect will be suspect. Anisotropy is also a problem. Anisotropy exists if the two components of variance change with the direction in which the samples were taken. For example, in Fig 4.10 the range of influence is far greater in the east-west direction than the north-south direction. Clearly any estimator which fails to take into account anisotropy will also be suspect.

One consequence of the high gold variability is the "regression effect" (Fig. 4.11). The regression effect is often forgotten in reserve calculations yet it is very important to understanding why the realized reserves may fail to meet the estimated reserves. No matter which estimator is used, the estimated block value will only coincide by chance with the actual block values. In practice an elliptical cloud of estimated values of the blocks is produced.

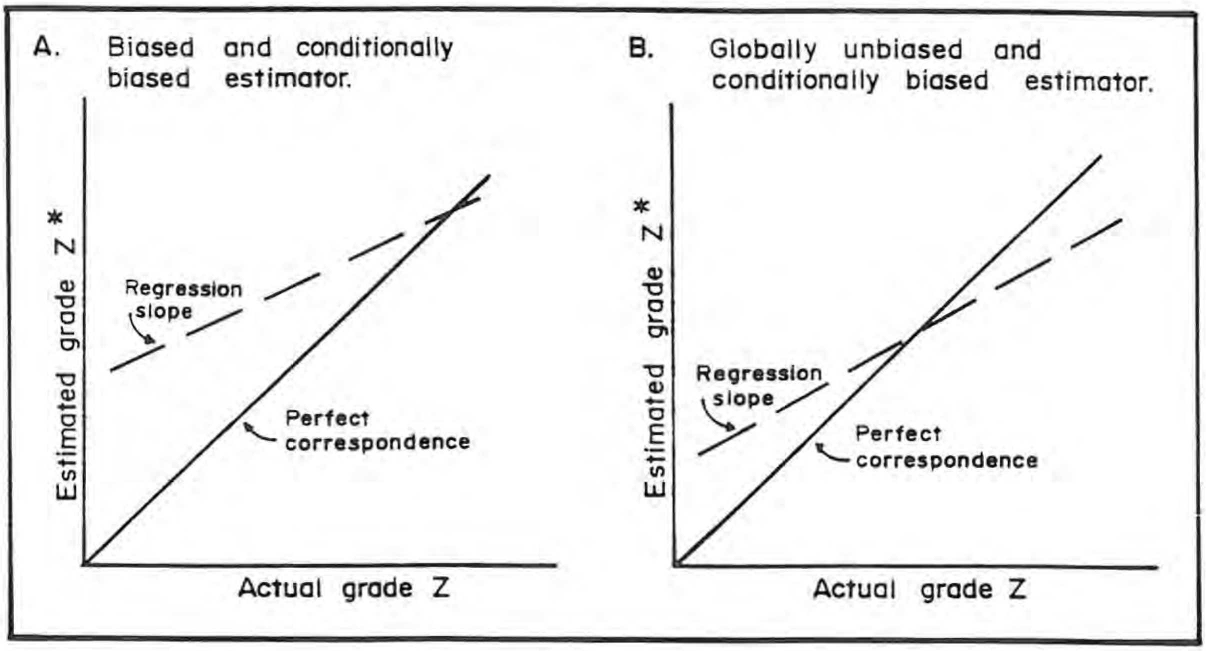


FIGURE 4.11. Correspondence diagrams of estimated value and actual value showing regression effects.

The line of regression passes through the means of the arrays of the estimates which correspond to each actual block value. If a perfect estimator were available, the regression line would be a straight line sloping at 45 degrees to each axis, and all points would lie on the line, i.e. the regression slope is unity (tan 45°; Royle 1979). Estimators may be compared by examining their respective regression lines. Royle (1978) defines a biased (poor) estimator as one whose regression line is significantly and persistently either above unity, or below it (Fig. 4.11A). Conversely in an unbiased (good) estimator there is no systematic error and the predicted global mean value of the estimate is equal to the actual global mean (Fig.4.11B). In addition all estimators will be conditionally biased to varying degrees. Where gold deposits are concerned a conditionally biased regression line essentially means that on average low-grade blocks are overestimated and high-grade blocks are underestimated. Overall, the two estimation errors tend to balance each other. However, as soon as a cutoff grade is applied, the problem becomes compounded, usually with the net effect of overestimating both tonnage and grade (Dagbert, 1987; see section 4.6). Another factor to note from Fig. 4.11 is that as the regression slope approaches unity the spread of values increases. Thus a good estimator maximizes the variance of the estimated block values and is unbiased.

The main advantages and disadvantages of the common resource estimation methods are discussed below. Useful papers which compare the merits of the various methods include Royle (1978), and Kwa and Mousset-Jones (1988).

Inverse distance weighting. This method assigns a grade to a block by combining the values of surrounding samples according to their respective distance from the block. Various weighting powers ($1/d^2$, $1/d^3$) and elliptical search radii are used according to the geological assumptions of how grades vary between drill hole intersections. The combined weights used in estimating each block must each add up to unity. Essentially the method gives an estimate of a point value at the centre of the block (Fig. 4.12).

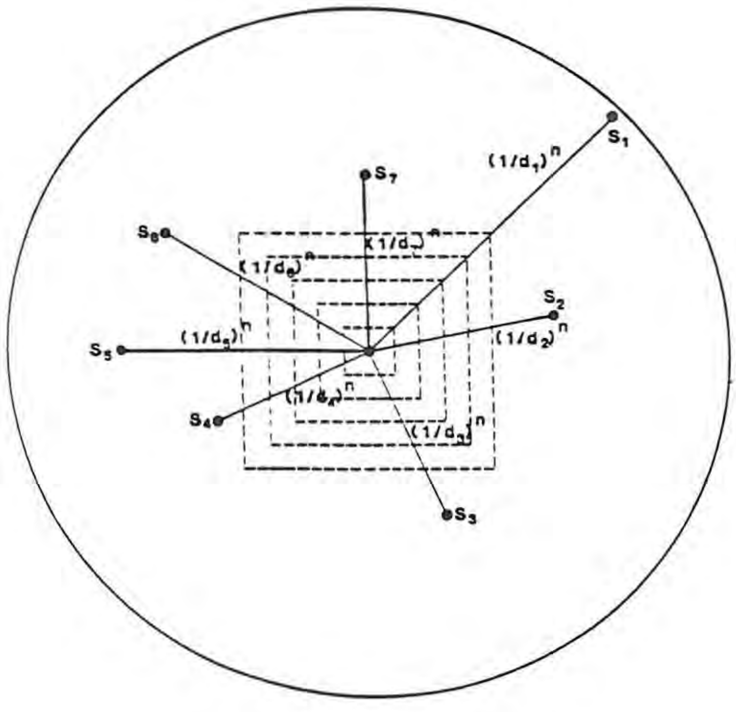


FIGURE 4.12. Inverse distance weighting estimator showing how different size blocks can have the same weights (from Kwa and Mousset-Jones, 1988).

Geometrical methods. These methods can be used to calculate a tonnage and grade. Fig. 4.13 illustrates the various geometrical patterns used in assigning areas of influence to drill hole samples. These range from rectangles, triangles and polygonal blocks to regularly spaced cross sections matching the outline of the orebody.

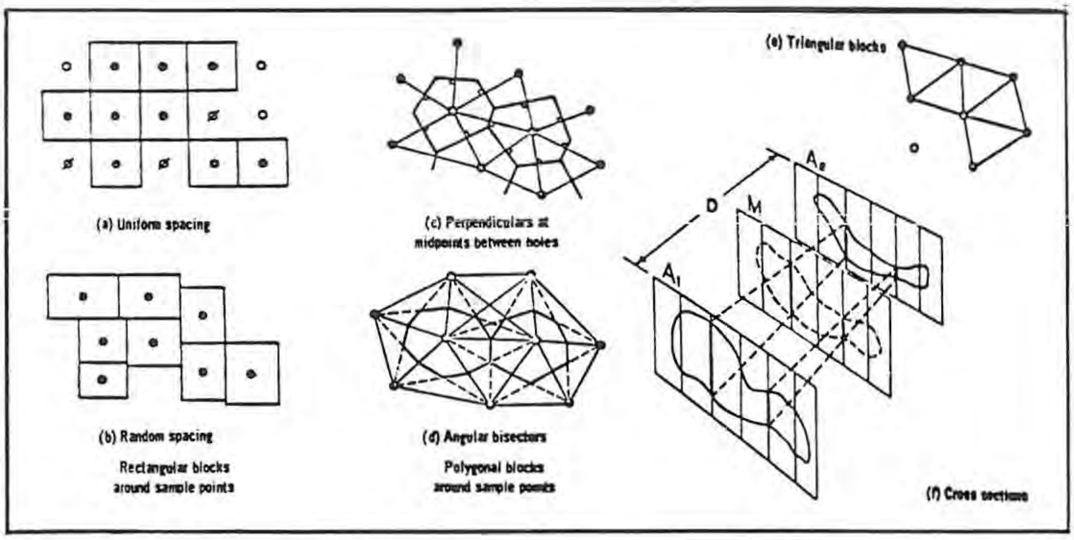


FIGURE 4.13. Geometric patterns used in assigning areas of influence to drill hole samples (from David, 1977).

Whichever shape is used the methods assume that there is continuity between drill holes; the form of the deposit can be described by a geometrical shape; and the tonnage and grade elements vary systematically between the drill holes (see Barnes, 1980 p. 52 for details on the systematic changes).

Both the inverse distance square and geometrical methods suffer from two major drawbacks. First, as soon as the nugget effect increases and range of influence decreases, the conditional bias increases and the methods give very inaccurate estimates of the real block values. Second, no measure of reliability (confidence) can be attached to each estimate. Only a single estimated block value is provided. There are also limitations pertaining to each method. In the geometrical methods drill holes are rarely spaced on a regular grid. Thus the area of influence assigned to each drill hole will be different. In addition polygons and triangles also fail to take into account anisotropy. With the inverse distance square method one further failing is that samples spaced close together are assigned similar weights unless declustering has been performed beforehand.

The main advantages of the geometrical methods are their practicality and ability to decluster. In comparison inverse distance square methods require relatively complex calculations which make them difficult to apply in practice. However, inverse distance square uses the surrounding samples to estimate the value of a block, which, depending on the spatial variance and range of influence may mean that a more accurate estimate is made.

Classical statistical methods. These methods can be used to: 1) help distinguish the different sample populations (secondary, oxide, bimodal); 2) compute the mean grade and variance for each population group; and 3) provide a measure of the reliability of the grade estimate. In addition statistics is very useful for just "looking at" the data and getting a "feel" for the grade distribution.

Whereas the traditional methods determined only the mean grade, classical statistical methods provide useful additional information. The most common parameters include: 1) the number of composite values; 2) the geometric and arithmetic means, median, variance, coefficient of variation, standard deviation and range of values; and 3) the histogram and cdf of the point values. The variance of the samples is relevant for reasons expressed earlier, however, in this case only the total variance is determined. The range provides an indication of the spread of sample values. The histogram and cdf indicate the distribution of values about the mean.

In general gold deposits have a positively skewed grade distribution (Fig. 4.14). This distribution has a long drawn out tail towards the higher values and the addition of a beta constant is normally required to transform the data into a 3 parameter model for lognormal statistics (Rendu, 1978). Lognormal statistics should consider the Sichel's t estimator of the average grade and the confidence limits for Sichel's t estimator.

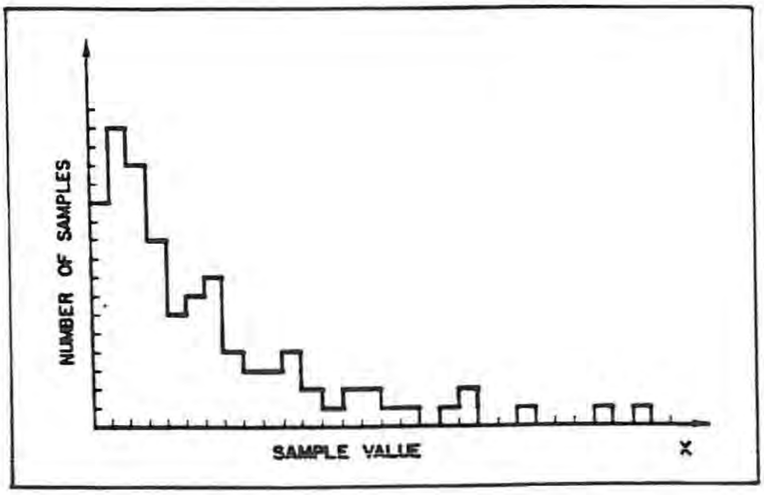


FIGURE 4.14. Typical value distribution of gold values (from Rendu 1978).

The grade distribution is particularly important as it determines how the estimators may behave. The most well known estimator is the arithmetic mean. The arithmetic mean is an unbiased estimator. However, experience on the Witwatersrand gold mines has shown that it sometimes gives very inaccurate grade predictions - in particular overestimating the true mean grade. This is not to say that the arithmetic mean is a poor estimator. One must just be aware of its limitations and the sample values used in its calculation. The problem with the arithmetic mean is that it is sensitive to outliers (high values in the tail). This means that if one samples a positively skewed population and too many of those samples are from the tail, then the arithmetic mean will severely overestimate the global mean. On the other hand the Sichel's t estimator is significantly less sensitive to outliers. Consequently it will give a more accurate estimate of the global mean when too many outliers are included (or rather thought to be included) in the estimate. In situations where the values are normally distributed, the Student's t estimator can be used to determine the global mean and confidence limits of the estimate. However, normal distributions in gold deposits are highly unlikely to exist.

A major limitation associated with estimating grade using classical statistics is that no account is taken of the spatial distribution of the samples within the orebody. The methods assume that the samples were taken randomly and that all values have an equal probability of being selected. As a consequence anisotropy is ignored. In reality this situation never exists. Selective open pit gold deposits have well defined oreshoots and samples are never taken randomly but rather located along lines of drill holes. For these reasons Rendu (1978) recommends that classical statistics should only be used in the early stages of exploration, when the number of samples available is too small and the distance between samples is too large to permit the use of geostatistics.

Geostatistical methods. These methods are used to: 1) estimate the range of expected mean grades; 2) provide a measure of reliability of the grade estimate; and 3) determine the tonnage.

Geostatistical methods are a logical extension of classical statistics. The methods estimate grade and ore zone thickness by taking into account the spatial relationship of the samples and their geometrical characteristics. The concept recognizes that variations in grade and thickness between samples are not random, but rather a function of specific physiochemical conditions present at the time of mineralisation. Since this is true in nature, geostatistics should provide an accurate estimate of the actual global grade and tonnage.

The geostatistical method of estimating resources is divided into two parts. The first is modelling the structure of the orebody by constructing experimental variograms. Concepts of continuity of the mineralisation, anisotropy and variance between sample positions are embodied in the variogram. The second part of the procedure is the actual estimation process itself, kriging, which is performed using quantified parameters derived from the modelled variograms constructed during the first stage (Clark, 1979). The most important prerequisite to a reliable estimate is that geological information is used to model the variogram. This means that the direction of anisotropy, nugget effect, spatial variance and range of influence - as defined by the variogram - must be in agreement with the observed oreshoot trends, ore zone continuity and nature of the mineralisation. If not, then the kriged reserve estimate will be unreliable.

Geostatistics has several advantages which sometimes make it preferred over the traditional and classical statistical methods. The greatest advantage of geostatistics is that kriging has the power to give different estimates according to the geological assumptions modelled in the variogram. In comparison, geometrical and classical statistical methods neglect the nugget effect, anisotropy and ore continuity. They are consequently "rigid" estimators and will always give the same estimate from a given set of values. Secondly, kriging is an unbiased estimator which minimizes the error (random) variance modelled from the experimental variogram. In this respect, many case histories have demonstrated that kriging gives a more accurate estimate than traditional methods (see for example papers in David et al., 1986). Thirdly, geostatistics attaches confidence limits to each estimate.

The main limitations with geostatistics are that the experimental variograms could be noisy or difficult to model. For instance with gold reefs of the Witwatersrand, it is common practice to first model the experimental variograms from similar reefs which have been extensively sampled and studied in producing mines; and then use this model to krig the borehole values from the new lease area. In selective open pit gold deposits there might not be enough closely spaced continuous sampling to obtain a good experimental variogram. Sampling data from another similar deposit may also not be available. In addition kriging normally requires a lot of samples (or bigger geological assumptions) in order to work effectively. Very often this extra sampling information is not available at the final feasibility stage when it is most needed.

Summary of resource calculation methods. From the foregoing pages it is apparent that calculating an in situ resource is one of the most problematic parts of the whole estimation exercise. All the various estimators have their limitations and no single method is always going to be suitable for every deposit. The choice depends on such factors as the stage of exploration, the estimation accuracy required, the density and pattern of drilling, the experience of the evaluator and the assumptions made in the geological model (Bujtor and McMahon 1983). Kwa and Mousset-Jones (1988) conducted a research questionnaire of mining and exploration companies worldwide and found that the sectional and polygonal methods were the most common estimators followed by geostatistics. This is interesting. Despite the more severe drawbacks of traditional methods when compared to geostatistical methods, the former are still preferred. A possible reason for this, is that exploration geologists and mining engineers find geostatistical methods more difficult to apply.

One problem facing the evaluator is which calculation method to select. Serious errors can arise if the person performing the resource estimate is unaware of the capabilities and limitations of each method. If in doubt, professional advice should be obtained. Also more than one method can be used. This helps verify the reserve estimate and gives the evaluator confidence in the accuracy of the estimate. Invariably, by using several methods, the limitations of one method are soon realized and it can be discarded.

For a global ore reserve estimate eight observations can be made.

1) All methods must be a function of the geological interpretation of the orebody.

2) Construction of several directional variograms to determine the spatial and random variance within the orebody should be performed early in the exploration programme. This will highlight potential problems arising from the high nugget effect, anisotropy and range of influence, and help with selecting a suitable resource estimation method.

3) No resource estimation method will be reliable if inaccurate sampling and assaying are practiced. Both systematic and random errors should therefore be minimized and any random error modelled in the nugget effect.

4) Geometrical methods are good at giving a declustered mean but are poor

estimators when the nugget effect increases or the range of influence decreases. They also provide no confidence limits to the estimate and fail to take into account anisotropy.

5) Inverse distance square methods take into account the spatial distribution of point values using various weighting factors. The methods, however, are poor estimators when the nugget effect increases or the range of influence decreases. In addition they cannot decluster or provide confidence limits to the estimate.

6) Classical statistics are ideal for "looking at" the data and providing a global mean estimate where insufficient sampling information is available to apply geostatistics. The Sichel's t estimator should be used in preference to the arithmetic mean when too many outliers from a skew distribution of point values are suspected.

7) Kriging provides an unbiased estimate which minimizes the error variance. Consequently it should give the most accurate estimate of all the methods. The main problem likely to be encountered with kriging is obtaining a modelled variogram that matches the structure of the deposit.

8) A probabilistic approach to estimating the global mean - using Sichel's t or kriging estimators - is preferred at the exploration stage as the variance of the global mean grade estimates can be quantified with confidence limits. Both the variance of the global mean estimate and confidence limits are important input parameters required for the project's cash flow simulations (see chapter 8).

4.6. MINEABLE RESERVE ESTIMATION

The final step in the estimation of ore reserves is to determine the proven and probable mineable reserves. It is essential that mineable reserves are used in the economic analysis as there is usually a substantial downward change in grade and possible increase in tonnage when converting from in situ resources to mineable reserves. Carras (1986) notes that there could be a 30 per cent grade difference between the in situ resource and mineable reserve estimates, as compared to say a 5 per cent difference between a sectional and kriged in situ resource estimate. This emphasizes the importance of estimating mineable reserves.

To estimate mineable reserves it is necessary to consider economic, mining, metallurgical, environmental and other factors (Fig. 4.1). A convenient way to discuss these factors is to consider the ore and gold loss, and dilution gain that is incurred during the various phases of the mining project. Ore loss refers to ore which is left in the ground or is sent to waste as a result of being undetected. Gold loss refers to gold which is not recovered during ore treatment and sent to the tailings or left in the heaps. Dilution gain refers to material delivered to the mill or leach pad which is below cutoff grade. Dilution may be divided into internal dilution which includes material within the predefined ore shape and external dilution which includes material surrounding the orebody. The external dilution is the most important criteria to be considered whilst estimating mineable reserves. Internal dilution will probably have been incorporated in the in situ resource estimate. The concepts of ore and dilution are illustrated in Fig. 4.15.

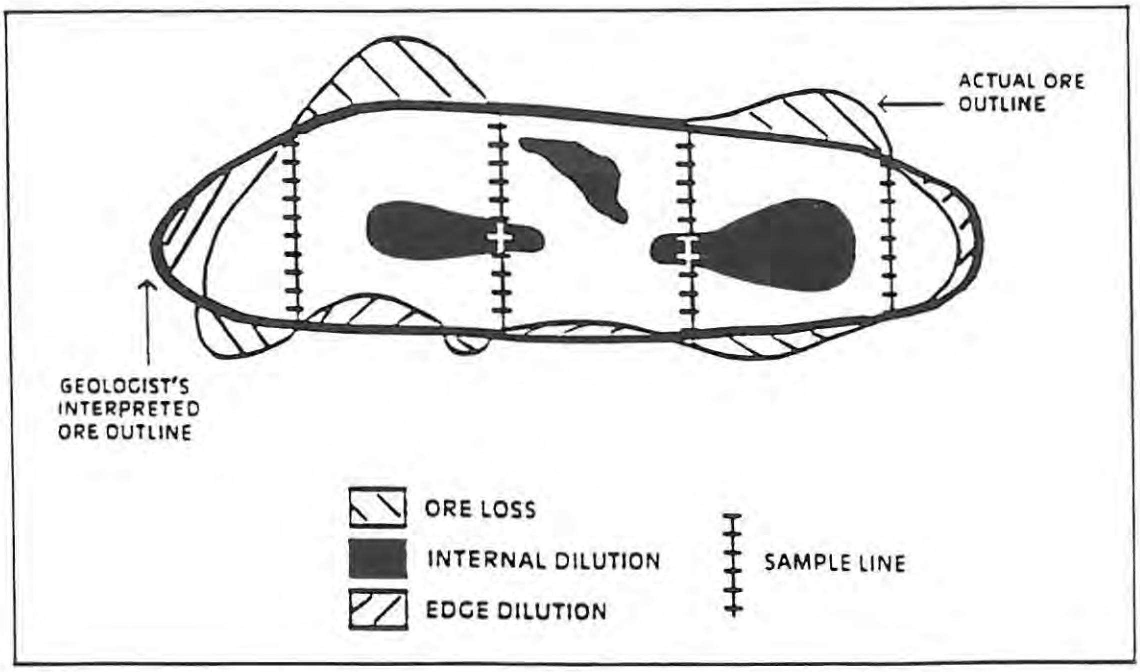


FIGURE 4.15. Dilution and ore concepts (from Carras, 1986).

The progressive causes of ore and gold loss, and dilution gain experienced during the various phases of the mining project are illustrated in a very systematic manner in Fig. 4.16. These range from the estimated in situ resource estimate, the decision to apply a cutoff grade, the design of the mine, to the physical mining and ore treatment. All these items need to be assessed, however, the task of quantifying the sources of ore and gold loss, and dilution gain is more difficult than imagined. There are very few articles that deal specifically with the subject. Examples include

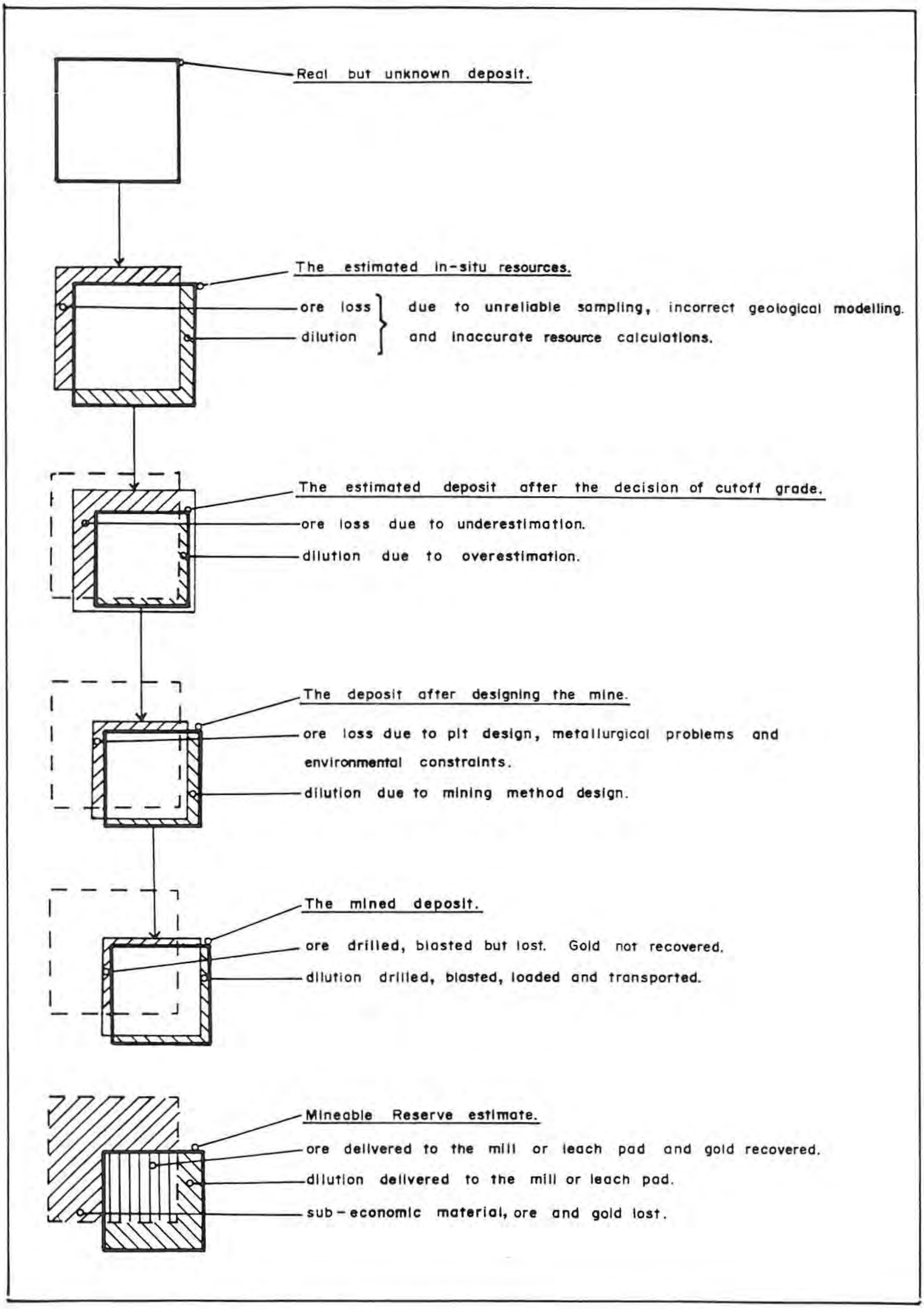


FIGURE 4.16. The sequence of ore losses and dilution gains (modified after Elbrond, 1986).

papers by Bujtor and McMahon (1983), Stone (1986), Carras (1986) and Elbrond (1986). To highlight some of these problems the various causes of ore and gold loss, and dilution gain are discussed below.

The estimated in situ resources. The mineable reserves are ultimately based on the in situ resource estimates, which are themselves a source of ore loss and dilution gain. To reiterate the main sources of error are: 1) unrepresentative sampling; 2) inaccurate sampling and assaying; and 3) incorrect geological modelling. Unrepresentative sampling and inaccurate sampling and assaying results in the wrong allocation of values to mining blocks or the failure to detect ore. Incorrect geological modelling cuts through contacts and excludes some ore from the reserve and includes some barren or low grade material within the estimate.

The estimated "deposit" after the decision of cutoff grade. For global in situ resource estimates, the various methods of calculation often give reasonably close estimates of the mean grade. However, as soon as a cutoff grade is applied to the estimates, some ore which should have been included above cutoff, is in fact excluded and lost. Similarly some material which is really below cutoff grade will be classified as ore. This is due to the error ellipse which arises from the fact that an estimate is not a true value (obvious but frequently forgotten, David, 1972). Fig. 4.17 illustrates this concept and shows the unavoidable ore loss and dilution gain incurred when selecting mineable blocks. The problem is most severe in cases where the size of the ellipse is large

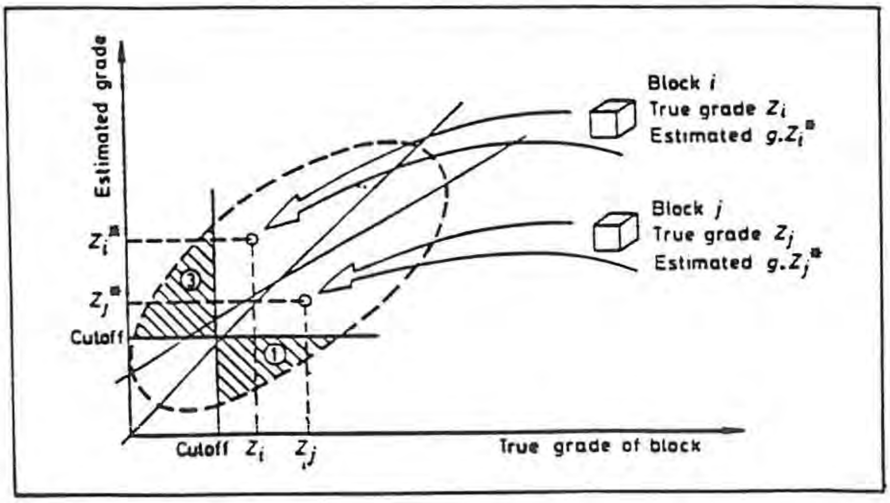


FIGURE 4.17. Correspondence diagram between predicted and actual grades of blocks. Shaded areas show groups of blocks wrongly allocated when a cutoff grade is applied to the estimated reserves. Block (3) represents material estimated as ore which is in fact waste; block (1) represents material estimated as waste which is in fact ore (from David, 1972).

compared to the 45 degree line of perfect correspondence. This situation is often experienced when the reserves are estimated using traditional and classical statistical methods. Better forecasts for planning are provided when geostatistical methods are employed (David, op. cit.; Dagbert, 1987).

An obvious objection to this problem created by inaccurate reserve estimates is that selection of mining blocks will be far better once more sampling information becomes available during mining. Although this is true, it is important to remember that at the final feasibility stage, the mine is designed, optimized and sent into development on the information derived from drill holes and near surface channel samples. No blast hole information is available (David op. cit.).

The "deposit" after designing the mine. During the design of the mine there will be portions of the deposit which may not be mineable because of the size and location of the orebody, metallurgical complications and environmental constraints. Possible sources of ore loss include, amongst others:

- ore lying beneath access roads;
- ore in hazardous areas where unstable ground or strong fissure water is suspected;
- ore located outside the limits of the pit or below the pit bottom;
- refractory material with uneconomic gold recovery; and
- ore that will not be mined due to environmental constraints such as proximity to protected land, danger of groundwater contamination, fear of erosion, increased sedimentation etc.

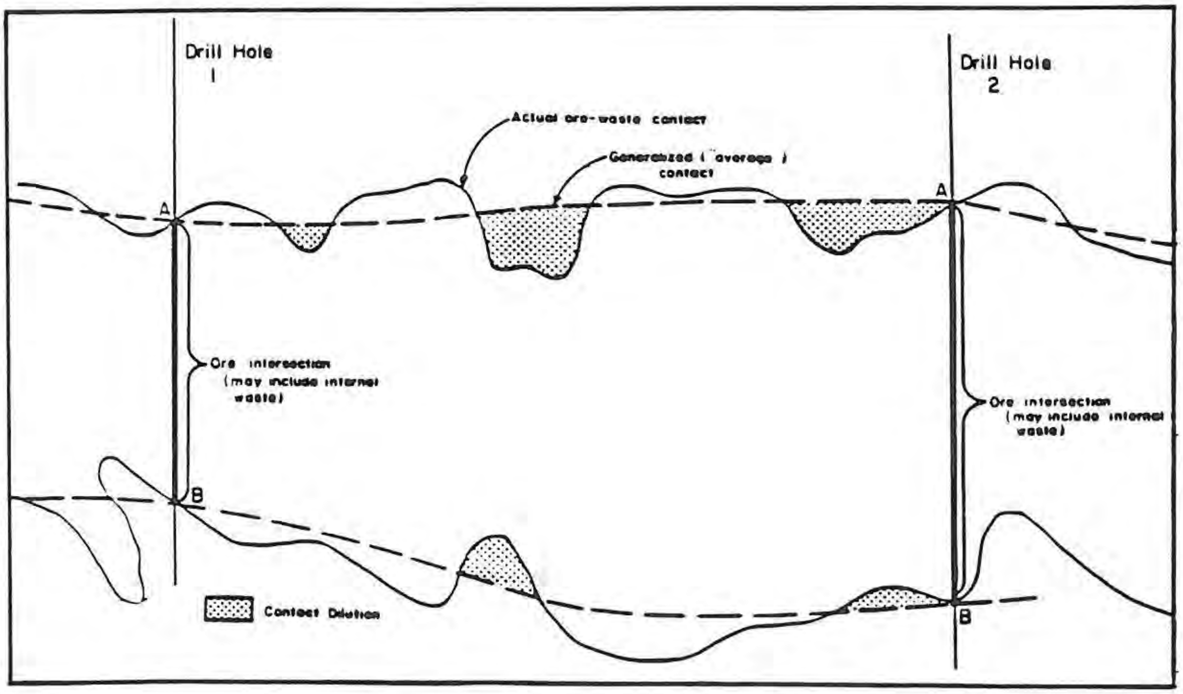
Dilution gains must also be taken into account during the design of the mining method. Normally, the mining method incurs unavoidable dilution because of the size and shape of the orebody. In the case of thin veins which are too narrow to mine selectively, it will be necessary to include external waste material lying outside the boundaries of the vein. On the other hand in a wider ore zone, it may be impossible to avoid mining "fingers" of internal waste from within the orebody envelope. It will be important to ascertain which method was used to estimate the resources, if internal dilution was originally incorporated into the in situ resource estimate and what "geological cutoff" grade was used to define the ore boundary. If traditional methods were used then a mining dilution factor is normally added to the resource estimate. Alternatively methods such as disjunctive kriging and conditional simulation automatically change the support level and estimate the grade of the mining blocks.

<u>The mined "deposit"</u>. During production, ore and gold losses and dilution gains are caused by several interrelated factors, namely:

- irregularities in the outline of the ore zone;
- the sampling method and sample density;
- the attitude of the orebody;
- the bench height;
- human error; and
- gold losses to the tailings or heaps.

These factors, together with ways of quantifying their effect, are discussed in detail by Stone (1986) and Carras (1986). They are mentioned last, as many of the problems can be avoided or at least minimized by trial mining or during production. Trial mining is strongly recommended. Not only does it solve mining, grade control and metallurgical problems prior to mine development, but also many of the other geological and sampling uncertainties associated with estimating the in situ resources.

Irregularities in the outline of the ore zone inevitably lead to problems where the mining method is not selective enough to follow the ore contacts. This effect is illustrated in Fig. 4.18, which shows how the actual ore zone boundary differs from the interpreted (dashed) contact.



<u>FIGURE 4.18</u>. Typical ore/waste contact (modified from Stone, 1986).

The actual section is drawn between two drill holes but the principle would be the same for blast holes or channel samples. If the mining method is only capable of following the dashed contact, a certain amount of waste rock will substitute for ore and a certain amount of ore will be lost. Overbreak of areas will also add to the dilution. In this situation the grade will be less than calculated between the two drill holes but the tonnage will remain unchanged.

The amount of ore loss and dilution gain is a function of the mining method and the irregularity of the contact zone. The severity of the dilution depends on the abruptness of the change in grade across the contact zone. If the grade changes gradually then the dilution will be minor. However, if the grade changes abruptly then barren material is added and the profitability of the mine may be severely affected.

The sampling method and sample density needs to focus on fixing the ore zone contact and the internal waste. Carras (1986) illustrates this problem in Fig. 4.19 which considers a blasting situation where the blast holes are sampled. In this situation the outline of the ore is likely to be incorrectly marked. For example, samples 4 and 9 will be assigned to ore though they contain some dilution. Samples 3 and 10 may go to ore or waste depending on the grade of ore intersected. Samples 2 and 11 will be assigned to waste though they contain some ore.

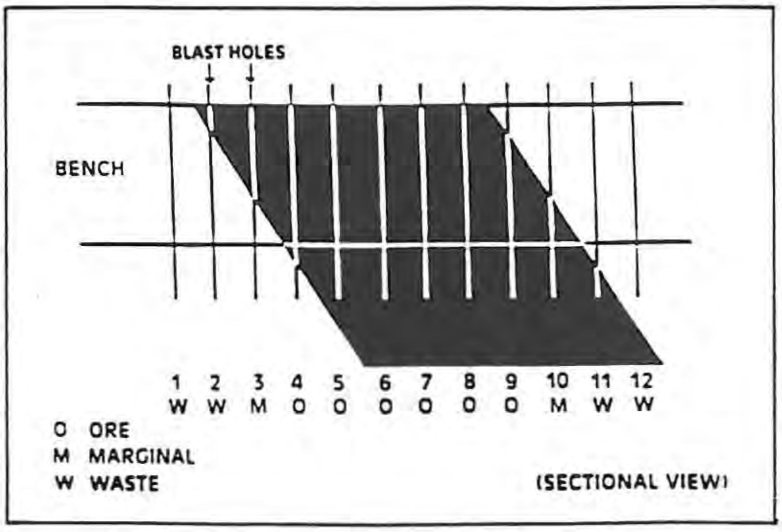


FIGURE 4.19. Effect of sampling design on the delineation of the ore zone (from Carras, 1986).

The relationship between the attitude of the ore zone and the bench height is illustrated in Fig. 4.20. For the smaller bench heights the sampling is able to more closely define the ore zone and reduce the dilution. This implies that selective mining must use a smaller bench height and a close sampling spacing.

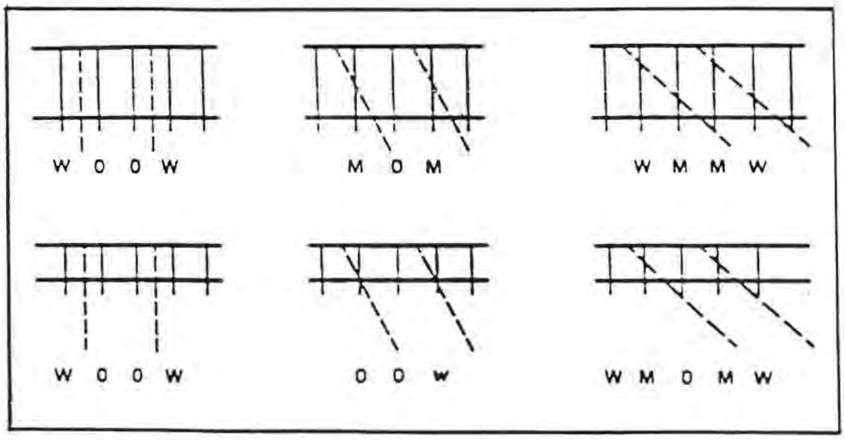


FIGURE 4.20. Relationship between ore zone dip and bench height. Blastholes may be classified as waste (W), ore (O), and either ore or waste (M) (from Carras, 1986).

Human error also needs to be considered. For example ore and waste may be incorrectly routed; the ore zone boundary may be wrongly marked or assay results may be mis-allocated. These factors are almost impossible to quantify but are nevertheless important.

During the ore treatment process some gold will inevitably be lost to the tailings or remain in the heaps. These losses are attributed either to mineralogical complications which prevent gold from being recovered or from inefficiencies in the ore treatment process. To estimate the percentage of recoverable gold, metallurgical recovery factors must be applied to the in situ resource estimates. These factors are derived from metallurgical testwork which is discussed in more detail in chapter 6. However, a point that needs stressing at this stage is that the recovery factors may give an optimistic view of gold recoveries from the actual operating plant. For instance one item often neglected is the variability of the mill feed. In gold processing it is important to try and maintain a constant feed grade so as to prevent gold losses to the tailings. If an excessive amount of waste material suddenly arrives at the mill, the waste material actually removes gold from the system. Since metallurgical testwork is conducted on ore samples these factors may be overlooked. Some caution and the use of probabilistic gold recovery estimates is therefore recommended.

The end result. The end product of all the sampling, assaying, geological modelling, in situ resource calculations, ore and gold losses and dilution gains are the mineable reserves. A common method of presenting mineable reserve estimates are to use grade-tonnage curves (Fig. 4.21). These curves show how many tonnes of ore and marginal material are available at a given cutoff grade and block size. The curves are ideal for performing the economic analysis and determining the scale of the operation.

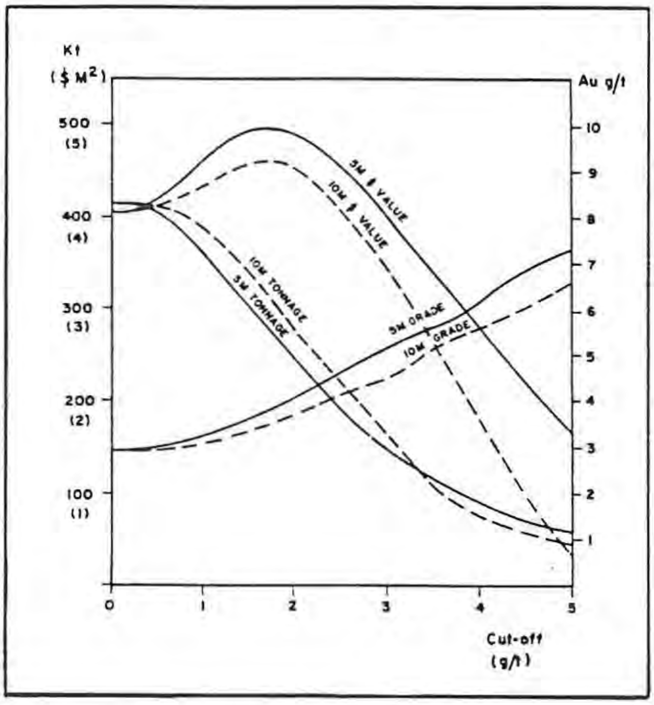


FIGURE 4.21. Typical grade-tonnage curves of a gold mine showing the different grade and tonnage curves, and dollar values produced from 5 and 10m blocks (from Dagbert, 1987).

An important point to be remembered whilst utilizing cutoff grades from grade-tonnage curves, is the size of the mining block being estimated. There is a different distribution curve for each size of block. Consequently the same cutoff applied to different block sizes, produces different financial returns. This is the "volume-variance effect" and is also illustrated in Fig. 4.21, which shows the dollar value curves obtained from using 5 and 10m block sizes. With the larger 10m blocks there is a decrease in the value of the ore for regular cutoff grades. This results from the fact that more tonnes have to be processed to recover about the same quantity of gold. At higher cutoff grades, the drop in the dollar value is more severe and comes from a simultaneous decrease of tonnes and grade. The answer is to select and estimate mineable reserve blocks which correspond to the SMU that will be mined in the open pit. David (1972) provides an excellent summary on the correct use (and misuse) of grade-tonnage curves for economic analysis and mine planning.

4.6. A NOTE ON GRADE CONTROL

Closely related to the estimation of ore reserves is grade control. In fact when the mine is in production the two functions essentially become one. The factors affecting ore reserve estimates - sampling reliability and representativeness, assaying, the accuracy of the extension function, the regression effect, the cutoff grade decision, the bench height and the mining method - are therefore equally applicable to grade control.

Grade control is vital in selective open pit gold mining as excessive ore dilution will erode profits. Although grade control falls within the production period of the mine's life, it is imperative that a suitable programme be designed at the final feasibility stage. Hopefully this will ensure that grade problems are avoided during the critical first few years of production when profits need to be maximized. If a suitable grade control programme is not designed, valuable time and money will almost certainly be lost. The main aims of the grade control programme are as follows:

1) reduce internal and external dilution;
2) prevent mis-classification and incorrect routing of ore and waste material;
3) design a sampling grid which provides adequate ore/waste definition at minimum cost;
4) ensure accurate and precise assays;
5) ensure that assay turnaround and assessment of the sampling data meets the production schedule;
6) accurately predict the mill head, marginal and waste material grades on a day-to-day basis; and
7) assist with short- and long-term mine production planning.

Fig. 4.22 outlines a typical grade control programme. In this programme the two most important functions are the sampling procedure and geological bench mapping. Grade control problems can invariably be traced to either one or both of these functions.

The method of the sampling depends largely on the hardness of the rock. Bulldozer ripping, ditchwitching and vacuum drilling are used in soft formations; RC percussion drilling and blast hole sampling are used in hard formations. Whichever method is employed it is important that an uncontaminated and representative sample is obtained. Judging from reports

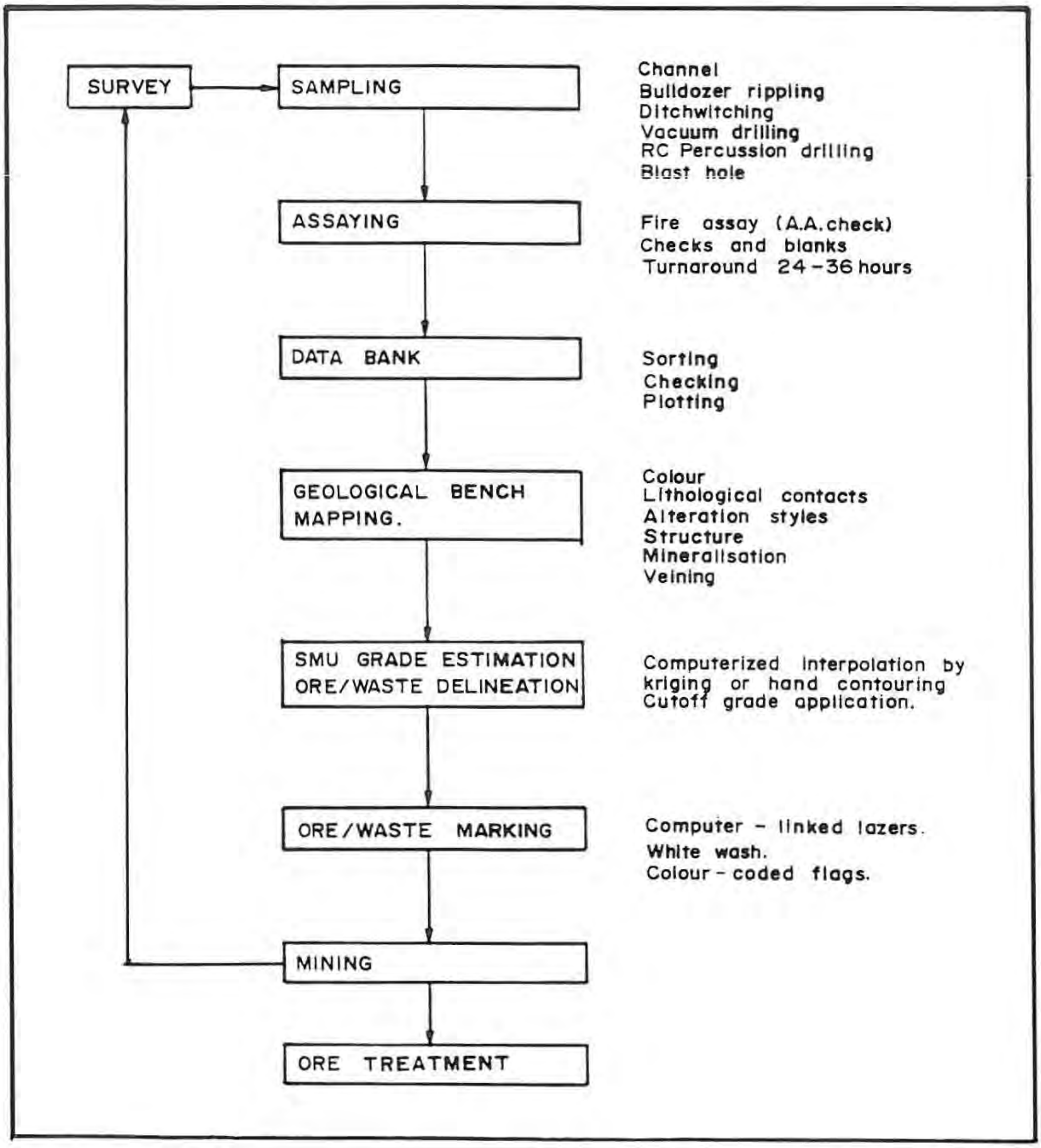


FIGURE 4.22. Outline of typical grade control programme. Note that the programme is usually repeated within 48 hours.

ditchwitch, RC percussion drill and blast hole samples provide the most reliable results (Bell and Dudley, 1986; Sandercock and Amos, 1986). Geological bench mapping is invaluable in grade control. If the gold can be correlated with a mappable geological feature - for instance a specific lithology, structure or alteration style - then an excellent grade control programme can be enforced. Also, fewer samples are required if the ore and waste contacts can be delineated using geological criteria. This will reduce the sampling and assaying workload, and reduce costs. The exploration geologist needs to be aware of this possibility whilst delineating the orebody.

The remainder of the grade control programme involves estimating the grades of the mining blocks, delineating the ore zone boundaries, and marking these boundaries on the benches. Kriging can be used to determine the average grade of the mining blocks. Alternatively hand contouring of the blast hole gold assays may be preferred. Marking of the ore and waste contacts is usually performed using lime or colour coded flags and tape. Sandercock and Amos (1986) provide additional suggestions for implementing a practical grade control programme.

5. GEOTECHNICAL AND HYDROLOGICAL MINE DESIGN

The purpose of this chapter is to examine the applications, data requirements and interpretation of geotechnical and hydrological data. Geotechnical and hydrological data is required for mine design purposes and deals with the realms in which mining engineers work. Far too often these aspects are left until the final feasibility stage. This is a mistake. Although geotechnical and hydrological information is only needed during the preliminary mine feasibility stage it is essential that the exploration geologist sets up procedures to collect the data during the early exploration stage. This is necessary, as final interpretation of the data may require the services of consultant engineers, like Steffen, Robertson and Kirsten (SRK). Obviously consultancy fees can be reduced if the information is already in a useable form. More important, however, is the fact that most of the data is derived from drill core and unless it is logged whilst drilling is in progress, it will be irretrievably lost.

5.1. OBJECTIVES AND SCOPE OF GEOTECHNICAL AND HYDROLOGICAL STUDY

Fig. 5.1 outlines the typical way in which a geotechnical and hydrological study may be conducted. In broad terms the study involves gathering and processing of the data prior to mining; the application of rock mechanic theory to determine the optimum mine design; an assessment of the risks involved; monitoring of the mining parameters during the life of the mine; and lastly execution of remedial actions to maintain the successful operation of the mine. The risk assessment stage is critical, as decisions made prior to mine development may have far reaching consequences on the profitability of the venture. A common practice is to treat the data statistically and determine the risks in probabilistic terms. For example Priest and Brown (1983) used Monte Carlo simulation techniques to test the stability of variable pit slope angles. Additional geotechnical and hydrological data can then be collected if the uncertainties are too great and need to be constrained.

The main objectives of geotechnical and hydrological studies in open pit mines are as follows:

1) determine the optimum pit slope angles (per rock type, interbench and overall);

2) predict the most likely pit slope failure modes (plane, wedge etc.);
3) estimate the pit water pumping and wall drainage requirements;
4) assist in determining the most suitable mining methods (ripping and/or blasting);
5) estimate the ground strength for location of heap leach pads, tailings dams and mine infrastructure;
6) identify suitable material for surfacing of access roads, and lining of leach pads, tailings dams and building foundations; and
7) determine the rainfall and evaporation rates to assist with possible dilution or water loss from the heaps, and potential flooding of the open pit.

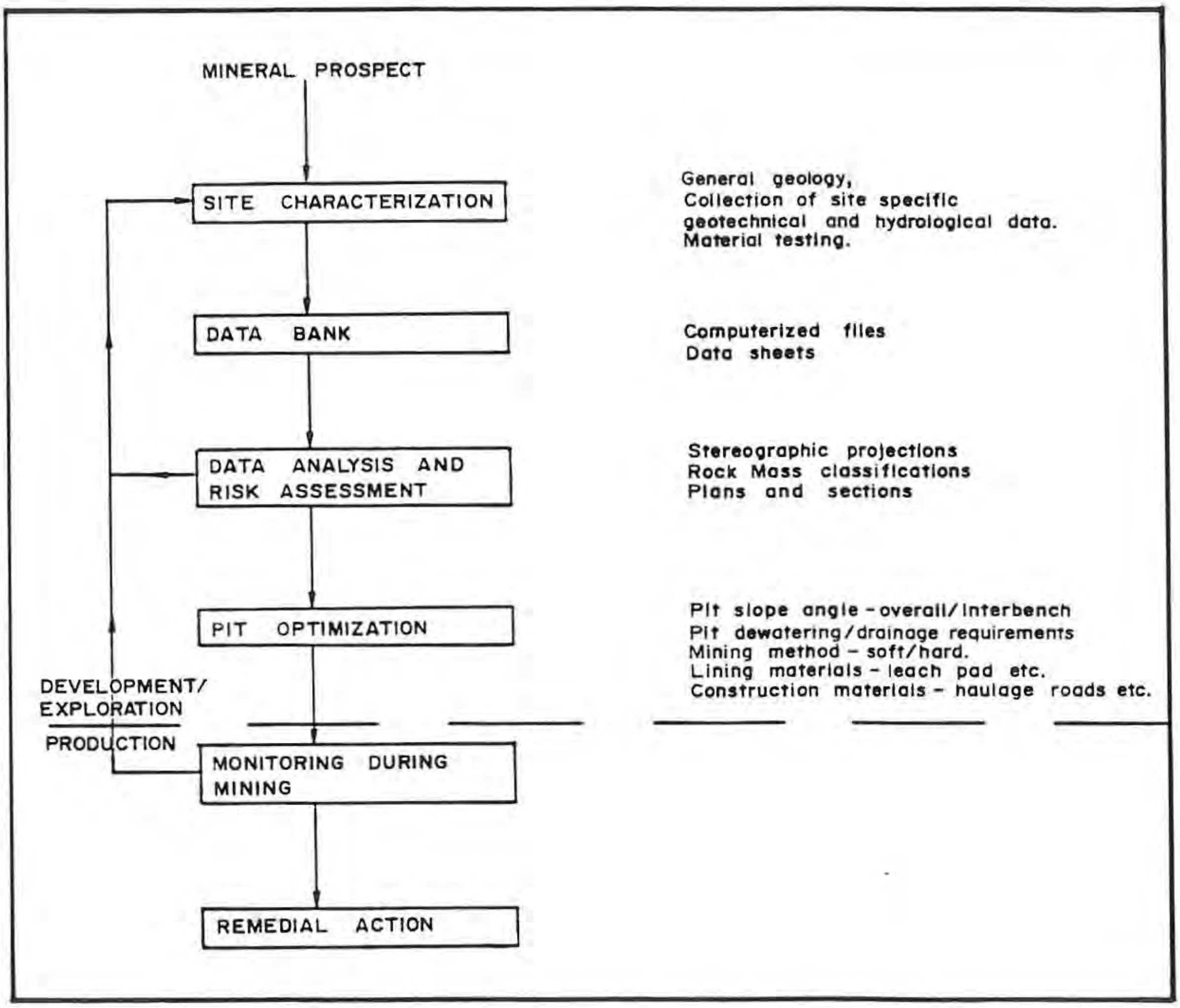


FIGURE 5.1. Major components of a geotechnical and hydrological study conducted during the exploration and mining periods of a mine's life.

Poor evaluation of any of these parameters could seriously impair the profitability of the mining venture. However, of these parameters the pit slope angle is the most critical as it determines the stripping ratio. Fig. 5.2 graphically shows how small changes of only a few degrees in the pit slope angle can significantly change the stripping required in an average sized open pit. At shallow pit depths the changes are not marked but as the pit deepens, the changes in stripping between different slope angles become more pronounced. Obviously the steeper the pit slope angle the better as this reduces the amount of waste mining and increases the profitability of the venture. However, there is a limit to which pit walls can be steepened before instability occurs. Instability can incur serious safety and economic penalties, the most important of which are possible loss of life, loss of ore, extra cleaning up costs to recover the ore that would otherwise be lost, extra costs of rerouting the haul roads, and production delays (Ross-Brown, 1979). This emphasizes the importance of determining the pit slope angle before mining commences. The optimum angle balances the economic aspects of maintaining a favourable stripping ratio with the economic and safety aspects of preventing pit wall failure.

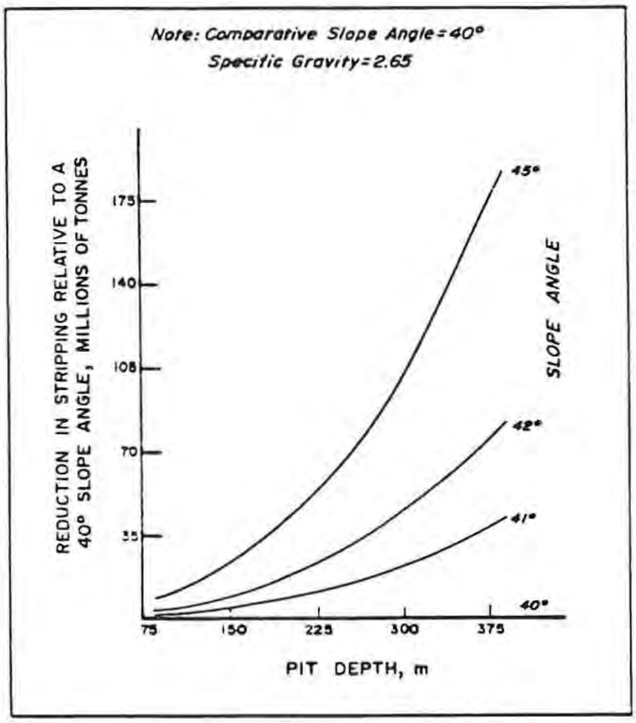


FIGURE 5.2. The effect that changing pit slope angles has on the stripping requirements (from Seegmiller, 1979).

According to Peters (1989) the overall or working pit slope angle may be as low as 20 degrees in strongly fractured rock (Fig. 5.3). In competent rock the overall slope may be as high as 70 degrees. The interbench and final pit slope angles will generally be steeper than the overall slope because it is not necessary to maintain benches after access. Complications can arise due to heterogeneous rock conditions. For instance the rock types and degree of fracturing may be different above and below the orebody; the stability of the overburden material will differ to the bedrock material; groundwater discharge rates may vary in different parts of the pit; and annual fluctuations in the water table may occur. Groundwater promotes slope instability because of the reduction of the effective stress across surfaces on which sliding occurs. Pumping of the pit and induced drainage of the walls help reduce this problem. The pit slope design must take these variable conditions into account and in particular cater for areas where conditions are expected to be hazardous.

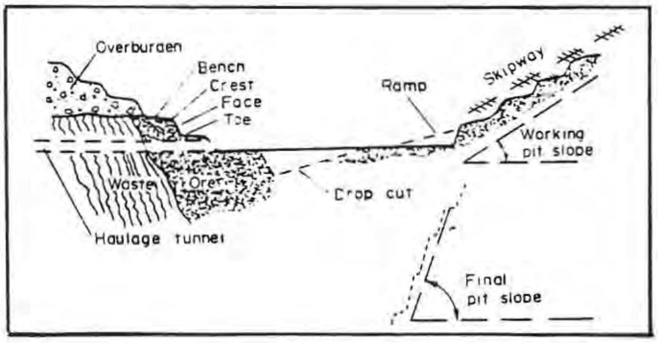


FIGURE 5.3. Open pit mining terminology (from Peters, 1989)

The common modes of failure found in surface pits are illustrated in Fig. 5.4. In general hard rock is liable to fail suddenly and violently by plane and wedge modes. Soft rock commonly fails slowly by circular and non-circular modes.

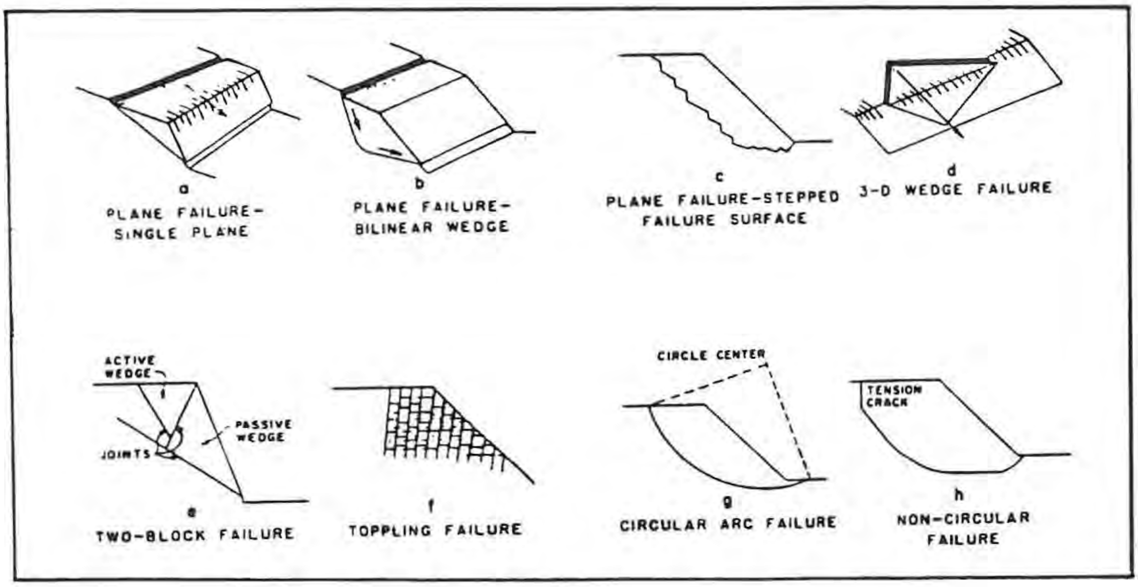


FIGURE 5.4. Failure modes commonly seen in open pits (from Ross-Brown, 1979).

The material strength and properties of the rock are used to determine the most suitable mining method. If the material is relatively soft it can be ripped using bulldozers. Hard rock requires blasting which is considerably more expensive. An assessment of the material properties may also benefit other aspects of the mine. For example clay can be used as an effective impermeable layer in the geotextile lining of heap leach pads.

5.2. DATA REQUIREMENTS

Like the sampling for ore reserve estimates, the collection of accurate and representative data forms the cornerstone of the geotechnical and hydrological study. Even the most sophisticated theoretical analysis becomes meaningless if the geological information on which it is based is inaccurate and inadequate (Hoek, 1987). In order to conduct the study two main categories of information are required:

1) geological discontinuities and material characteristics of the overburden, ore and waste within and adjacent to the pit(s); and

2) groundwater and surface drainage characteristics.

Geological discontinuities include the location, attitude, spacing and filling of faults, shear zones, cleavage, bedding planes and joints. Material characteristics include rock type, alteration style, degree of weathering, material density and strength. Groundwater includes the elevation of the water table and the flow of water through the bedrock. The surface drainage includes the location of all rivers and water runoff channels. Environmental studies also require information on the groundwaters and surface waters so the hydrological study in this instance serves two purposes: mine design and environmental impact assessment.

The bulk of this information can be collected by the exploration geologist during surface and underground mapping and borehole logging. Both mapping and borehole logging complement each other, but where no rock outcrop is present, it may be necessary to expose the bedrock to gain the desired information. The methods of obtaining the data are well known and need not be explained here. More details can be obtained from Seegmiller (1979) and Laubscher (1977). In addition SRK can be contacted for advice on how to record the data for later interpretation. In the following text the items to be recorded during mapping and borehole logging are taken from unpublished geotechnical and hydrological notes provided by SRK.

Geotechnical and hydrological mapping. Surface and underground exposures provide the best opportunity to examine the geological discontinuities and material characteristics. The orientations of structures are readily accessible, and joint spacing and length can be measured. In underground openings a three dimensional picture is afforded whereas surface outcrop generally provides information restricted to two dimensions, unless there are topographical elevation differences. Table 5.1 lists the data that is commonly recorded whilst mapping.

TABLE 5.1. Details of geotechnical and hydrological data recorded from mapping of surface and underground rock exposures, showing commonly used abbreviations (recommended by Steffen, Robertson and Kirsten).

1. Location (XYZ coordinates, peg number).
2. Rock type, degree of weathering, alteration style, colour, grain size, and texture.
3. Rock or soil hardness (a qualitative classification is usually sufficient i.e. R1 very soft rock to R5 very very hard rock).
4. Rock discontinuity - fault F, shear zone SZ, joint J, bedding plane BP, cleavage C etc.
   - thickness and number
   - orientation (dip angle and dip direction)
   - spacing/Rock Quality Designation (RQD)
   - filling (thickness, type and hardness)
   - roughness (slicken sided, smooth, defined ridges, small steps, very rough)
   - waviness (amplitude and wavelength).
5. Density (g/cm^3).
6. Compressive strength (Schmidt Hammer "R" value).
7. Water (Dry D, moist M, seepage S, Flow F, estimate flow l/sec)
8. Comments - blasting effects.

The level of detail required will depend on the time available and the stage to which the prospect advances. SRK recommend four levels of detail: random mapping, detailed line or cell mapping, major feature mapping, and critical structural mapping. Random mapping of scattered outcrop may be sufficient during the geological feasibility stage whilst major feature and critical structural mapping will be necessary during the preliminary mine feasibility stage.

Geotechnical and hydrological borehole logging. Generally, boreholes provide the only means of evaluating the character of the rock and groundwater conditions at depth. Owing to the fact that drilling is needed to evaluate the orebody, minimal additional costs will be incurred if geotechnical and hydrological data is also gathered. Geotechnical borehole logging is best performed on core or orientated core samples and should be conducted in the field immediately after the core has been retrieved from the core barrel. This ensures that the rock is tested in its near natural state, i.e. before the core has dried out or been further broken up by handling and splitting. Chips from percussion holes are not suitable for determining geological discontinuities but can be used to provide information on the rock type, alteration style, degree of weathering and water table elevations. The holes should always be left open in order to monitor the groundwater conditions using piezometers.

Table 5.2 lists the geotechnical and hydrological data that is commonly recorded during borehole logging.

TABLE 5.2. Details of geotechnical and hydrological data recorded from borehole logging, showing commonly used abbreviations (recommended by Steffen, Robertson and Kirsten).

1. General borehole statistics (location, collar elevation, azimuth, core size).
2. Lithology (depth, rock type, degree of weathering, alteration style, colour, grain size, texture, zones of leaching and cavities).
3. Rock or soil hardness (a qualitative classification is usually sufficient i.e. R1 very soft rock to R5 very very hard rock).
4. Rock discontinuity - fault F, shear zone SZ, joint J, bedding plane BP, cleavage C etc.
   - orientation (relative to core axis and reference point)
   - spacing/Rock Quality Designation (RQD)
   - filling (thickness, type and hardness)
   - roughness (slicken sided, smooth, defined ridges, small steps, very rough)
   - waviness (amplitude and wavelength).
5. Density (g/cm^3).
6. Compressive strength (point load test).
7. Water - down hole estimate of water table elevation, multiple aquifers if any and flow estimate l/sec.
8. Core recovery parameters (total core recovery TCR, and solid core recovery SCR).
9. Comments.

In addition to borehole logging, the drillers report can provide useful information and SRK recommend that operators should be encouraged to complete records of drilling conditions and progress, paying particular attention to water strikes and losses, and drilling conditions.

Under certain circumstances it may be impossible to collect all the necessary data. In this case priority should be given to collecting the most important data. In order of decreasing priority SRK recommend the following list of geotechnical information:

1) location, orientation and extent of faults, shear zones and slips.
2) surface features, widths and fillings of these features;
3) location and extent of lithologies with significantly different strengths or structural characteristics, e.g. dykes, leach zones and degraded shale horizons;
4) orientation of major joints and slips;
5) spacing of major joints and slips;
6) filling and roughness of major joints and slips;
7) orientations of other joint sets;
8) spacing and continuity of joints in each set;
9) filling and surface features of joints in each set; and
10) strength of intact rock material.

5.3. DATA INTERPRETATION

Data interpretation involves the application of geotechnical theory to the data in order to meet the objectives listed earlier. Laubscher (1977) recommends that the interpretation techniques be straightforward, so that they can also form part of the ongoing mining operation. Highly sophisticated techniques are time consuming and unlikely to be of practical benefit. The emphasis here will be to highlight the practical applications of the various techniques available. It is not intended to describe in detail the theoretical foundations to each technique as these have already been covered in the references cited below. Four main categories of techniques can be distinguished. They are commonly used together and are listed below.

1) Stereographic projections for analysis of geological discontinuities and prediction of modes of failure (Ross-Brown, 1979).

2) Rock mass classifications for mine design. These include:
   (i) Rock Mass Quality "Q" index to determine the rock mass strength for construction purposes (Barton et al., 1974);
   (ii) Mining Rock Mass Rating (MRMR) to help determine the overall pit slope angle (Laubscher 1990); and

(iii) Excavation "N" index to help determine the most suitable excavating equipment (Kirsten, 1982).

3) Limiting equilibrium for pit slope design. This includes the factor of safety rating to help determine the overall and interbench pit slope angles and to quantify the instability of the slopes (Ross-Brown, 1979).

4) Plans and sections. These include amongst others:
   (i) cross sections of the pit outline;
   (ii) geotechnical hazard plans for highlighting problem areas; and
   (iii) groundwater level contour plans for determining aquifer and pit drainage requirements.

In stereographic projections the equal area (Schmidt) net provides the ideal tool for classifying the discontinuity sets. The nets allow hundreds of strike and dip pairs to be plotted and contoured. By using the orientations of the discontinuities and the estimated pit slope face, it is possible to determine the modes of failure (Fig. 5.5). The particular mode of failure is required for the factor of safety rating (item 3).

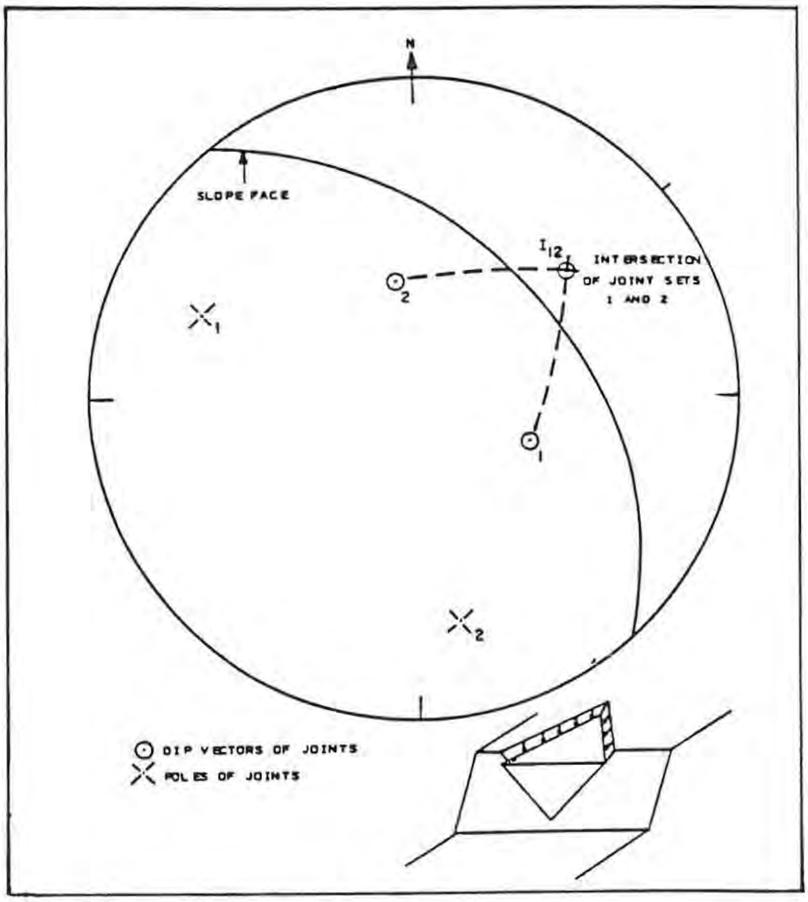


FIGURE 5.5. Typical example of equal-area stereographic net showing 3D wedge failure (from Ross-Brown, 1979).

The rock mass classifications are very useful engineering tools. By assigning a value to the rock and not a vague descriptive term, the classifications allow the mining engineer to make a sound, professional decision. SRK recommend that more than one classification be used initially until it becomes apparent that one method is more suitable than the others. This will depend to a large extent on local ground conditions. It is also likely that some modifications of the chosen system will be necessary once mining starts. The reason for this is that although the classifications are theoretically correct, mining is not an exact science. Empirical observations and common sense still play an important role.

The Rock Mass Quality, "Q," index uses RQD, joint characteristics, and water and stress factors. The technique was developed primarily for civil engineering purposes and may therefore be conservative in a mining environment. On the other hand, the MRMR is derived from theoretical and empirical observations. It was developed specifically for mining applications and is probably the best method to use. Brook and Dharmartne (1985), examined several classifications for tunnel support before and after mining (including the "Q" index), and found that the MRMR provided the closest prediction of the actual conditions. In the MRMR five parameters - RQD, intact rock strength, joint spacing, condition of joint and groundwater - are used to obtain an in situ rock mass rating. The in situ rock mass rating is then adjusted according to weathering, field and induced stresses, joint orientations and blasting effects to give the MRMR. The in situ rating is relatively easy to obtain. The MRMR, however, is more difficult to judge and requires a person with geotechnical experience. The greatest advantage of the MRMR is that it is the only method that can be applied to boreholes and rock exposures. This advantage, together with its proven track record and practical application make it superior to any other rock mass classification for determining the overall pit slope angle.

The excavation "N" index uses the uniaxial strength, dry density, RQD, and condition of the joint sets - their orientation, alteration, roughness and filling - to calculate a value for N from which to select the most suitable excavating equipment. To facilitate the choice of equipment, the index has been classified into eight categories which correspond with the excavating equipment used in industry. These range from D3 to D10 Caterpillar bulldozers, and blasting. A major advantage of this classification, besides being practical, is that it can be applied over the full range of natural materials ranging from the weakest soils to the hardest rocks. It can also be applied to drill core and rock exposures. A further advantage is that the classification is based entirely on the in situ properties of the natural materials. This allows the engineer to select the most appropriate equipment for the job, i.e. the size or method of excavation does not dictate the class of excavation (Kirsten, 1982).

The factor of safety rating is based entirely on theoretical assumptions which describe slope stability in terms of opposing forces: gravity on the one hand and frictional shear stress and the forces of cohesion on the other. The ratio of the opposing forces defines the factor of safety, F.

$$F = \frac{\text{forces tending to restore equilibrium}}{\text{forces tending to disturb equilibrium}}$$

where, F=1, equals the unity at the point of limiting equilibrium.

A factor of safety greater than 1 is necessary to ensure an acceptably low chance of failure. In this method the main parameters measured are the slope geometry, material properties and groundwater conditions. This information is then incorporated in equations to derive the factor of safety for different slope angles and block failure modes. The advantage of this method is that interbench and overall slope angles can be expressed in terms of probability of failure curves (Ross-Brown, 1979, p. 182). These curves effectively quantify the risks and allow the engineer to select a "safe" slope. A major drawback is that the orientations and characteristics of the discontinuities must be known. Therefore the method cannot be applied to drill core which means it may have limited application during the delineation programme. Due to poor rock exposure, the discontinuity orientations and modes of failure are unlikely to be known in sufficient detail until mining commences. Also, even if the discontinuity orientations are known, they may not be representative of ground conditions at depth. It is also uncertain if the method has a proven track record.

Finally plans and sections provide an ideal means of pictorially representing the geotechnical and hydrological information. For instance Laubscher (1977) recommends that the rock mass classes be plotted and contoured to highlight hazardous areas where extra precautions will be required once mining commences.

6. GOLD RECOVERY AND ORE TREATMENT OPTIONS

The purpose of this chapter is to: 1) examine the mineralogy of gold ores and discuss the significance of mineralogy in gold recovery; 2) present a suitable procedure to test gold ores; and 3) outline the common ore treatment processes.

It is essential that the metallurgical characteristics of the ore be determined soon after significant gold mineralisation is discovered. This will establish if the gold can be extracted economically from the rock. The emphasis here is on economic. Any gold can be extracted from a rock, but the very definition of ore requires that the metal be extracted for a profit. Fortunately gold metallurgy is technically advanced and there are several ore treatment options to select from. Amongst the most common are carbon-in-pulp (CIP), carbon-in-leach (CIL), resin-in-pulp (RIP), heap leaching, roasting, pressure oxidation and bacterial oxidation (BIOX). The precise option or combination of options depends primarily on the ore's mineralogy.

6.1. OBJECTIVES AND SCOPE OF METALLURGICAL STUDY.

The main objectives of the metallurgical study are to:

1) classify the ore types and determine the gold recovery for each type of ore; and

2) design a suitable ore treatment process which is technically, economically and environmentally justified in terms of the project's overall costs, risks and returns.

Fig. 6.1 illustrates how these objectives are incorporated into a typical metallurgical study. The most important steps are sampling, mineralogical investigations and metallurgical testwork. Like the ore reserve and geotechnical studies, the outcome of the metallurgical study is only as good as the data on which it is founded. Accurate and representative sampling, expert mineralogical investigations and definitive metallurgical testwork will be required to successfully classify the ore types, determine the gold recoveries and optimize the design of the metallurgical plant. An optimum plant must take into account the mine infrastructure, mine and mill capacities, equipment requirements, unit operations, capital cost, capital scheduling, operating cost, profit expectations and production efficiencies. Details of the ore types, their respective gold recoveries, and the capital and operating cost of the plant are incorporated into the economic analysis of the mining venture. Sufficient data is required so that the results can be treated statistically during the feasibility studies. If the ore types and recovery rates are uncertain then further mineralogical investigations and metallurgical testwork are necessary. If the plant is technically and economically non-viable, then further optimization or an alternative treatment process may be necessary.

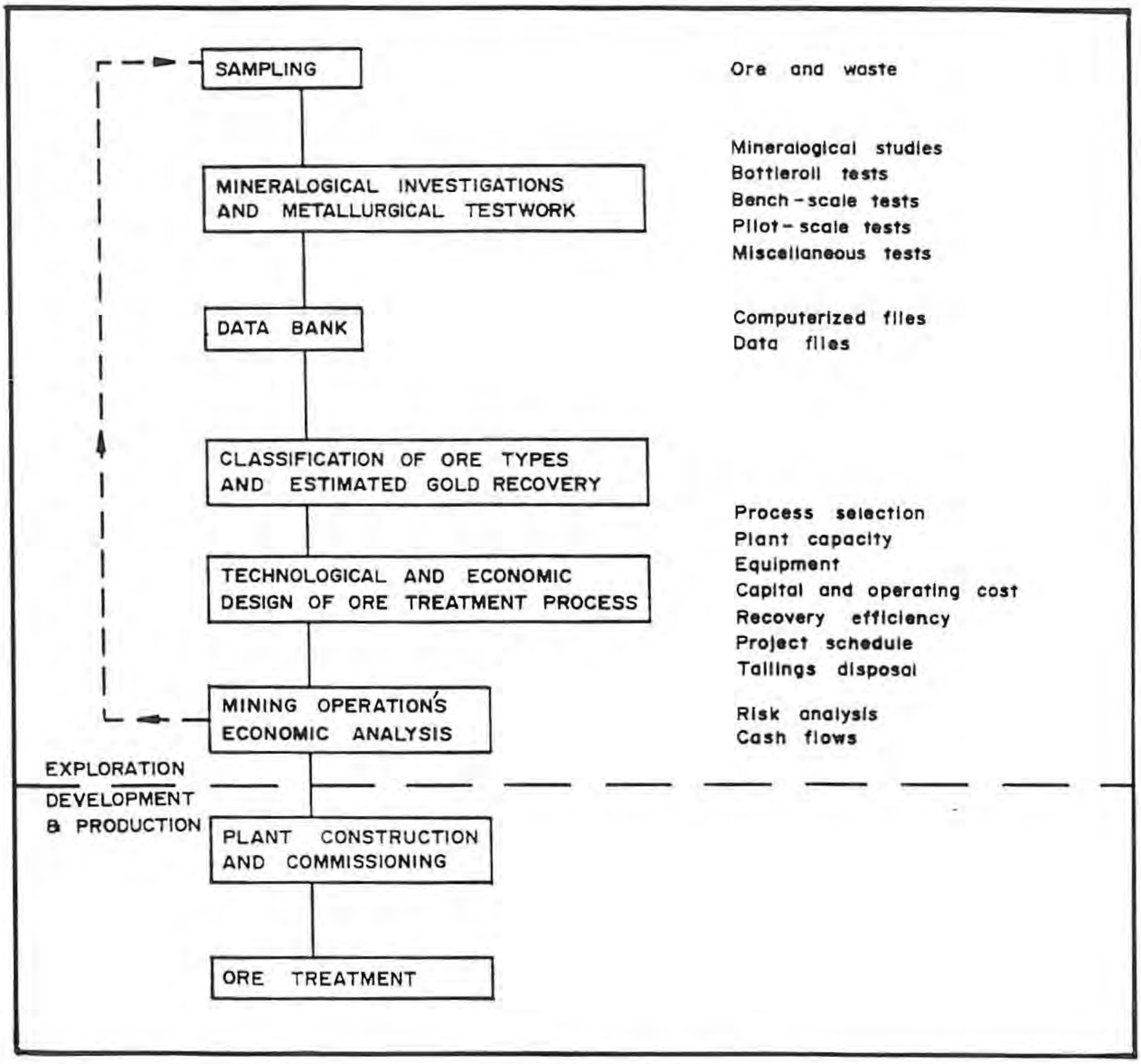


FIGURE 6.1. Major components of a metallurgical study during the life of the mine.

Since almost all the ore treatment processes use cyanide solvent to recover gold from the ore or concentrate (cyanidation), it is convenient to classify the ores metallurgically as free milling or refractory. Free milling ores or concentrates are amenable to gold extraction by direct cyanidation and can be treated by relatively inexpensive processes like carbon-in-pulp, carbon-in-leach, resin-in-pulp and heap leaching. The ore or concentrate is refractory if less than 80 per cent of the gold can be extracted by direct cyanidation after grinding (Petruk, 1989). Refractory ores require pre-treatment by roasting, chlorination, pressure oxidation or bacterial oxidation in order to expose the gold particles for dissolution by cyanide. Free milling and refractory gold ores can be further classified as oxidized and sulphidic types. Oxidized ores contain oxides of iron and sulphur and are developed in the weathering environment, as in laterite gold deposits. Sulphidic ores contain pyrite and/or arsenopyrite as the main sulphide minerals and are typically present below the limit of weathering. In general, most, but not all oxidized ores are free milling, but not all sulphidic ores are necessarily refractory.

6.2. GOLD ORE MINERALOGY AND ITS SIGNIFICANCE TO GOLD RECOVERY

The mineralogy of gold ores and the significance of mineralogy in gold recovery has been reviewed in a number of publications (Henley, 1975; Gasparrini, 1981 and 1983; Taylor et al., 1989; Petruk, 1989; Harris, 1990). In addition Boyle (1979) provides a comprehensive account on the geochemistry of gold and its deposits. The effect of mineralogy in gold extraction can be conveniently discussed under the following seven headings:

1) the gold-bearing minerals;
2) the grain size of the gold-bearing minerals;
3) textural association of the gold-bearing mineral and host mineral;
4) chemical lock-up or "invisible" gold;
5) the nature of the gangue minerals;
6) cyanicides; and
7) coatings on gold.

The gold-bearing minerals. The gold-bearing minerals include native gold, several gold alloys, the tellurides and some metallic gold compounds such as aurostibnite and maldonite (Table 6.1). The most common of these are native gold and electrum; the other alloys and tellurides are scarce; the metallic gold compounds rare. The important point to note is that both native gold and electrum are amenable to direct cyanidation. The other gold-bearing minerals are refractory.

The purity of native gold is expressed as fineness which is the proportion of pure gold in the mineral in parts per thousand. Thus native gold 900 fine contains 10 per cent impurity. Silver is the most common impurity and electrum is a variety of gold containing more than 20 per cent silver (Gasparrini, 1983). Harris (1990) reports that electron microbe analyses of gold grains throughout the world indicate that Precambrian lode gold deposits are commonly characterized by restricted gold fineness in the range 890-960, whereas epithermal deposits exhibit a variable gold fineness, in the range 450-900. Of the other alloys (Cu, Pt, Pd, Rh, Ir, Bi), there is some speculation if the impurities are in fact substituting in the gold lattice. Boyle (1979) suspects that most of the gold alloys with the platinum group elements are probably fine intergrowths.

TABLE 6.1. Gold-bearing minerals (from Boyle, 1979).

Native elements, alloys and metallic compounds	
Gold	Au
Argentian gold (electrum)	(Au.Ag)
Cuprian gold (cuproauride)	(Au.Cu)
Palladian gold (porpezite)	(Au.Pd)
Rhodian gold (rhodite)	(Au.Rh)
Iridic gold	(Au.Ir)
Platinum gold	(Au.Pt)
Bismuthian gold	(Au.Bi)
Gold amalgam	Au_2Hg_3 (?)
Maldonite	Au_2Bi
Auricupride	$AuCu_3$
Palladium cuproauride	$(Cu.Pd)_3Au_2$
Sulphide	
Uytenbogaardtite	Ag_3AuS_2
Tellurides	
Calaverite	$AuTe_2$
Krennerite	$(Au.Ag)Te_2$
Montbrayite	$(Au, Sb)_2Te_3$
Petzite (antamokite)	Ag_3AuTe_2
Muthmannite	(Ag.Au)Te
Sylvanite	$(Au.Ag)Te_4$
Kostovite	$AuCuTe_4$
Nagyagite	$Pb_5Au(Te.Sb)_4S_{5-8}$
Antimonide	
Aurostibite	$AuSb_2$
Selenide	
Fischesserite	Ag_3AuSe_2
Tellurate	
Gold tellurate (?)	

The gold tellurides are reported to occur mainly in deposits in volcanic assemblages. Particularly notable are epithermal veins and fissures in Tertiary rocks - for example the Emperor deposit, Fiji; and Precambrian greenstone hosted lodes - for example the Kalgoorlie deposit (Boyle, op. cit.). Gold tellurides present metallurgical complications during cyanidation. They dissolve slowly in cyanide solvent and require fine grinding, intense aeration, and long agitation (Harris, op. cit.)

The gold compounds, aurostibnite and maldonite, do not appear to be related to any specific geological environment. They are found where stibnite and other antimony or bismuth minerals occur. Aurostibnite and maldonite are not attacked by cyanide solvents and the necessary treatment to extract the gold is poorly understood. Pre-treatment of the concentrate or ore by roasting, pressure oxidation, bacterial oxidation or chlorination may be considered (Taylor et al., 1989).

<u>The grain size of the gold-bearing minerals</u>. The grain size of the gold-bearing minerals varies from submicroscopic gold bound within sulphide minerals to gold nuggets many centimetres in diameter. Mesothermal lode gold deposits are generally marked by a fairly uniform particle size, whereas epithermal vein and laterite gold deposits are often characterized by variable gold particle sizes. In epithermal environments this variability probably reflects the sites of gold precipitation which are more open (Boyle, 1979). In laterite environments secondary mobilization and precipitation of gold produces new fine-grained gold, commonly 5-20 microns diameter, and nuggets (Mann, 1984 a; Wilson, 1984).

The gold grain size affects the milling and cyanidation requirements of the ore. Over grinding is costly and fine grinding impedes gold recovery. Material ground to less than 10 microns blocks macropores in the carbon, thereby reducing carbon activity in carbon-in-pulp circuits. In filtration circuits, fine material less than 75 microns reduces the filtration efficiency and causes increased soluble gold losses to the tailings. On the other hand coarse gold grains cause difficulties in the cyanidation process. Due to its low surface to weight ratio, coarse gold dissolves slowly in cyanide solvent and can be lost to the tailings if the retention time in the solvent is not long enough. Gravity concentration is required to recover coarse gold from the milled ore before cyanidation.

<u>Textural association of the gold-bearing mineral and host mineral</u>. For metallurgical purposes it is important to establish if the gold is distributed along fractures, at the border between grains of the same host mineral, two different host minerals, totally enclosed in the host mineral or a combination of these (Gasparrini, 1981). Gold occurring along grain boundaries can often be liberated easily for cyanidation during milling, but where gold is enclosed in another mineral, no route exists for the cyanide to dissolve the gold. In this situation the cause of refractoriness is described as physical lock-up or encapsulation. Host minerals that commonly encapsulate gold are pyrite, arsenopyrite and quartz. To treat these ores the host mineral would have to be chemically dissolved or oxidized. Alternatively, both host mineral and gold could be dissolved. Gold that is strongly encapsulated in sulphide minerals is usually

liberated by roasting, pressure oxidation or bacterial oxidation (Swash, 1988). Less strongly encapsulated gold in sulphide minerals may be liberated by hot aqueous oxidation or chlorination (Hausen, 1981). If time is not important it may be possible to oxidize the sulphide minerals and liberate the gold by exposing the ore to the atmosphere. Gasparrini (1983) reports that natural oxidation may be achieved after six months to one year depending on the fractures and porosity within the sulphide minerals. Where quartz encapsulates gold it may be possible to crack the quartz and liberate the gold by repeatedly heating and quenching the ore. Alternatively the quartz could be dissolved by strong acids. However, both these methods are expensive and likely to make the operation uneconomic.

Chemical lock-up or "invisible" gold. Gold can also occur as a minor or trace constituent in silicates, sulphides, carbonates and oxides (Boyle, 1979). Metallurgically, the most important are the sulphide minerals pyrite and arsenopyrite. Several reports (Hausen 1981; Swash 1988) suggest that submicroscopic (invisible) gold is present in pyrite and arsenopyrite. The gold may occur as extremely fine (less than 0.02 microns) grains within the sulphide mineral lattice or be in solid solution. Ores containing invisible gold are refractory. No amount of fine grinding will ever liberate the gold for cyanidation. Pre-treatment of the concentrate or ore involving complete oxidation of the host mineral by roasting, pressure oxidation or bacterial oxidation is necessary.

The nature of the gangue minerals. The nature of the gangue minerals can have a detrimental effect on the handling and treatment of the ore and the effectiveness of cyanidation. For example very "hard" ores containing quartz are abrasive and expensive equipment like mill linings will need to be replaced more often. Ores hosted in chert break into sharp fragments which can cut through conveyor belting and further add to costs. Talc minerals make grinding difficult and add to settling and filtration problems (Harris, 1990).

Ores containing carbon can cause problems with the cyanidation process. Where "active" carbon is present, the gold is preferentially absorbed by the carbon from the cyanide solutions and lost to the tailings - a term known as preg-robbing. Likewise, if any "active" carbon is present in heap leaching ore, the gold will be retained within the heaps. Charcoal from wood fires can also cause gold loss. If forested areas need to be cleared in order to expose the orebody for mining, burning the trees should be avoided. A further problem caused by carbonaceous material in the ore is that a reducing environment is created in the leaching vessels. This inhibits gold dissolution as the cyanidation reaction requires oxygen. To overcome the negative effects of carbonaceous material several options are available: 1) roasting can be employed to oxidize the carbon to carbon dioxide; 2) aqueous oxidation and chlorination can be used to oxidize or decrease the preg-robbing capabilities of the carbon (Hausen, 1981); 3) more "active" carbon can be added to the leach as soon as the gold is in solution as in carbon-in-leach; or 4) various surfactants (shales or graphitic carbon) which inhibit the absorption of gold onto preg-robbers can be added to the leach.

Clay or any fine material can also cause gold recovery complications. Where heap leaching is being considered a high clay content causes channelling on the surface of the heap, thereby preventing the cyanide solutions from reaching the gold within the heap. Agglomeration of the ore prior to stacking on the heap is necessary to overcome channelling. Where carbon-in-pulp is being considered a high clay content may "bind" or clog the macropores in the carbon, thereby inhibiting gold absorption.

Cyanicides. Cyanicides are deleterious constituents that inhibit cyanidation by altering the chemistry of the leach. Minerals containing nickel, iron, manganese, cobalt and copper, such as pyrrhotite (FeS), pyrolusite (MnO_2), covellite (CuS), chalcocite (Cu_2) and native copper (Cu) form cyanide complexes thereby consuming cyanide which would otherwise be used to dissolve gold. Antimony and arsenic do not form cyanide complexes, but they react with lime in the cyanide leach to form compounds which then consume cyanide (Worstell, 1986). In addition sulphide and sulphate ions, thiosulphate, arsenites and ferrous ions can consume oxygen and thereby inhibit the gold cyanide dissolution process.

Excessive cyanide consumption adds to operating costs and is one of the main causes for concern in heap leaching. At the Drylands open pit, heap leach gold mine in the Transvaal, South Africa, cyanide consumption accounted for just over a quarter of the total working cost, or R10.4 per tonne of agglomerated ore (Pelly, 1990). At Drylands the gold is hosted in weathered manganese-rich material and it is probable that large amounts of cyanide consuming pyrolusite are present in the ore. The adverse effects caused by cyanicides can only be eliminated by removing the deleterious constituents from the ore prior to cyanidation. Treatment of the ore by gravity concentration (to separate the gold) or chemical treatment of sulphide float concentrate can be considered.

Coatings on gold. Coatings can have adverse effects on the gold recovery. For example coatings of gold by iron oxides and hydroxides in laterite deposits may prevent or at least inhibit gold dissolution by cyanide solutions. In this instance identification of the gold grain size and its textural association with the iron oxides and hydroxides will normally give an indication of the most appropriate treatment method.

In addition coatings can develop on the gold particles during ore treatment. For example during roasting, precipitates of iron oxides, silicates, and/or lead, antimony and arsenic compounds can coat the gold particles and prevent dissolution. These negative effects can be minimized by maintaining the correct roasting temperature, however, problems do arise where pyrite and arsenopyrite occur together. Gasparrini (1983) reports that the roasting temperature for arsenopyrite must be kept lower than that for pyrite, otherwise arsenates coat the gold particles. In these situations it is common practice to pre-treat the pyrite and arsenopyrite separately by operating a two-stage roaster in series.

Coatings can also be a problem in the cyanide leach system. Gasparrini (op. cit.) reports that electrum can develop a coating of silver sulphide which appears to mix with other particles and insulate the electrum from dissolution by cyanide solutions.

6.3. MINERALOGICAL INVESTIGATIONS AND METALLURGICAL TESTWORK

Mineralogical investigations and metallurgical testwork ideally need to be conducted whilst the orebody is being drilled. During the early stages of exploration, it is the responsibility of the exploration geologist to ensure that the work is carried out. Later, during the preliminary mine and final feasibility stages an experienced metallurgist and mining engineer are required to design an ore treatment process and plant.

Table 6.2 lists the scope and purpose of the mineralogical investigations and metallurgical testwork that are conducted throughout the exploration period. Overall, the studies involve increasing definition of the ore and refinement of the ore treatment process.

For mineralogical investigations an optical microscope is usually adequate. The gold-bearing mineral, host mineral, grain sizes and textural associations can be determined from polished mounts or thin sections. One problem that may be encountered is actually being able to find the gold in low concentrations. Petruk (1989) discusses several techniques to improve the success of finding the gold grains. These include: 1) concentrating heavy minerals; 2) preparing special polished sections; 3) searching with an image analyser; and 4) dissolving associate silicate and sulphide minerals. In cases where the gold is fine (less than 10 microns) or where compositions of the gold-bearing minerals need to be determined, then a scanning electron microscope (SEM) coupled with an electron microbe will be required (Hausen, 1981).

TABLE 6.2. Scope and purpose of the mineralogical investigation and metallurgical testwork (modified after Joung, 1983).

FEASIBILITY STAGE	TEST METHOD	TEST MATERIAL	PURPOSE
GEOLOGICAL	MINERALOGICAL INVESTIGATION BOTTLEROLL	EXPLORATION DRILL CORES PERCUSSION CHIPS. CHANNEL AND GRAB SAMPLES	CLASSIFY ORE TYPES ESTIMATE GOLD RECOVERIES 10 % Accuracy DETERMINE ORE MINERALOGY CONCEPTUAL ORE TREATMENT PROCESS PRELIMINARY CAPITAL AND OPERATING COST ESTIMATE
PRELIMINARY MINE	BENCH-SCALE TESTS MINERALOGICAL INVESTIGATIONS	INFILL DRILL CORES AND PERCUSSION CHIPS BULK REPRESENTATIVE SAMPLES	GOOD GOLD RECOVERY ESTIMATE 5 % Accuracy FIRM ORE TREATMENT PROCESS PRELIMINARY FLOWSHEET UNIT OPERATIONS DESIGN GOOD CAPITAL AND OPERATING COST ESTIMATE CAPACITY ESTIMATE
FINAL	PILOT-SCALE TESTS (optional)	BULK REPRESENTATIVE SAMPLES FROM MINING	FIRM GOLD RECOVERIES <5 % Accuracy PROCESS CRITERIA FIRM FLOWSHEET EQUIPMENT SELECTION PRELIMINARY ENGINEERING DESIGN FIRM CAPITAL AND OPERATING COST ESTIMATE OPERATOR FAMILIARISATION TROUBLE SHOOTING

Metallurgical testwork includes numerous techniques designed to assess the chemical and physical characteristics of the ore. These may be categorized as bottleroll, bench-scale, pilot-scale and miscellaneous tests. They are briefly discussed below.

Bottleroll tests. These tests provide a quick and inexpensive indication of the likely gold recovery by direct cyanidation and cyanide consumption at different particle sizes. The tests are a useful initial means to determine if the ore is free milling or refractory. In general bottleroll tests reflect the gold recovery that can be expected from carbon-in-pulp, carbon-in-leach and resin-in-pulp circuits. If heap leaching is being considered, then lower rates of extraction will almost certainly be achieved from the heaps as real conditions will not match the laboratory tests. The cyanide consumption will indicate if cyanicides are present and provide an estimate of the cyanide consumption costs. Bottleroll samples typically weigh 4-6 kilograms.

Bench-scale tests. These tests include any controlled experiment designed to solve specific unit operations and select a suitable flowsheet to process the ore and recover the gold. For example in column leach tests a typical heap is simulated to test optimum crush size, agglomeration requirements, leach time, cyanide concentration and solvent application rates (Showell and McCrabb, 1988). Column leach tests are more expensive to perform than bottleroll tests but the added information gained compensates for the greater expense.

Pilot-scale tests. These tests are designed to simulate the ore treatment processes that will operate at the time of mining. The objective of the pilot-scale test should not be to select the flowsheet, but to evaluate the ore treatment process and demonstrate that the process is risk free. Pilot-scale tests have the advantage of being able to operate continuously and in a closed circuit, rather than batch-wise as in most bench-scale testwork (Stanley, 1987). Young (1983) describes the benefits of the pilot-scale tests as follows:

1) examination of the various ore types at operational conditions, on a sustained basis;
2) determination of process requirements, energy and water;
3) observations of hazardous materials, recycles and their effect;
4) study of waste disposal/environmental controls;
5) development of definitive engineering design criteria;
6) determination of instrumentation/process controls;
7) development of operating procedures;
8) preparation of quality products for market assessment;
9) satisfy investors that the concept is viable; and
10) crew training.

Pilot-scale testwork is the ultimate metallurgical test and can be very expensive (upward of R1 million). Consequently it should only be considered where the ore treatment process is new, or under circumstances when the company is inexperienced. The Council for Mineralogical Technology

(MINTEK) may be contracted to assist with the design and operation of the pilot plant. Ideally the pilot plant should be built at the mine site so that the effect of local water quality on gold recovery and environmental problems can be observed. Also the pilot-plant can then eventually serve as part of the full-scale equipment. To keep capital costs low, the equipment need not be new as it is highly unlikely that the gold recovered from the pilot plant will cover the cost of conducting the test.

Miscellaneous. Included in this category are various tests to determine the physical and chemical properties of the ore. The most common tests performed are: 1) micro track analysis; 2) size fraction analysis; and 3) bond work index. The micro track analysis determines the percentage of fines in the ore which may be required for agglomeration testwork. The gold size fraction analysis determines the percentage of gold in each size fraction. This information is required if gravity concentration is being considered. The bond work index estimates the energy required to grind one tonne of ore to a specific particle size. This information is critical in the mill design. Milling is expensive and it is important to select the mill with the correct lining and capacity before mining starts.

During the early stages of exploration the mineralogical investigation and metallurgical testwork will primarily be concerned with establishing the causes of refractoriness. The difficult (or refractory) gold ores require detailed mineralogical investigation to determine the optimum recovery process, whereas the free milling ores need not be subjected to the same scrutiny. Based on the foregoing pages the most common causes of refractoriness are summarized as follows:

1) physical lock-up or encapsulation;
2) chemical lock-up or "invisible gold";
3) insoluble gold alloys and compounds;
4) cyanicides that consume cyanide or oxygen;
5) carbonaceous material; and
6) insoluble coatings on the gold bearing minerals.

Since there are several reasons for an ore to be refractory, a systematic method to establish the cause(s) of refractoriness is required. Fig. 6.2 outlines a suitable procedure that can be followed to establish these causes. The first step (stage 1) is to determine gold recovery by bottleroll tests. If the gold recovery is greater than 80 per cent there is no need for detailed mineralogical investigations and conventional

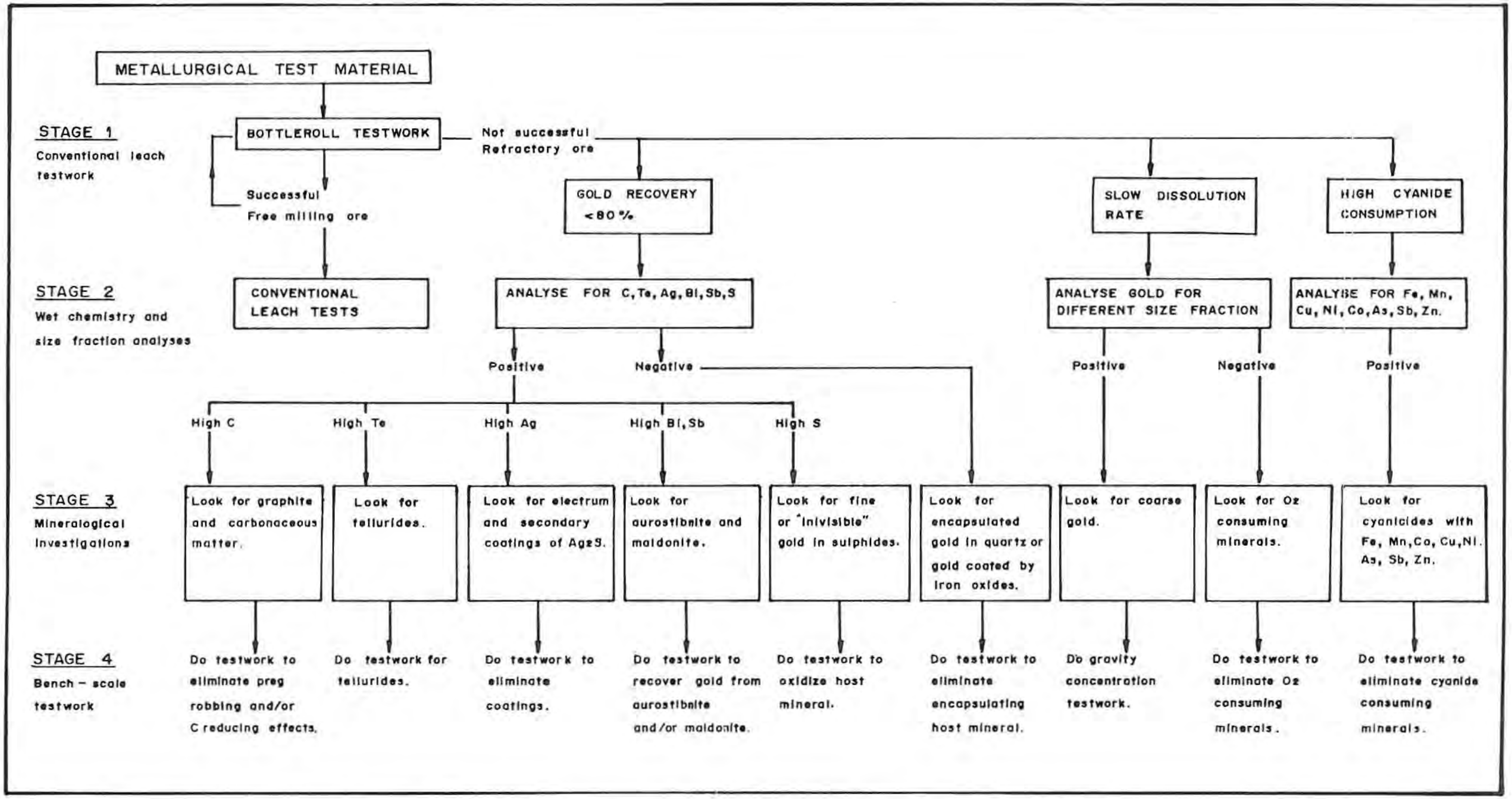


FIGURE 6.2. Effective procedure for studying gold ores and establishing the main causes of poor gold recoveries, slow rates of gold dissolution and high cyanide consumption (modified after Gasparrini, 1983).

cyanidation tests can be performed. However, if the gold recovery is less than 80 per cent, or if the dissolution rate is slow, or if cyanide consumption is high, the next step is to isolate the reason for the unsatisfactory results. This is determined by a combination of wet chemistry (stage 2) and mineralogical investigations (stage 3). Analysing the test material for C, Te, Ag, Bi, Sb and S will help establish whether the refractoriness is caused by "active" carbon, insoluble tellurides, coatings on electrum, insoluble aurostibnite and maldonite, submicroscopic gold in sulphide minerals, encapsulated gold in quartz, iron oxide coatings, or a combination of the above. Slow gold dissolution rates are invariably caused by coarse native gold. A size fraction analysis and gold assay of each size fraction will establish if coarse gold is present. If no coarse gold is found, then it is likely that oxygen consuming minerals are causing the slow dissolution rates. Where cyanide consumption is high, then cyanicides may be present in the test material. Analysing for Cu, Ni, Fe, Mn, Co, As and Sb should help establish which mineral is responsible for the high cyanide consumption. The final step (stage 4) will involve bench-scale tests to solve the problems causing the low gold recovery, slow dissolution rates and high cyanide consumption.

6.4. ORE TREATMENT OPTIONS

The recent boom in gold mining has also seen many new innovations in treating gold ores. In particular these innovations have been directed not only towards the economic aspects of increased efficiencies and reduced costs, but also the environmental aspects related to poisonous gaseous emissions and liquid effluents (Wall et al.,1987). The most significant historical advances have been in carbon technology, the pre-treatment of refractory gold concentrates, heap leaching technology, liquid effluent technology and alternative lixiviants (Wall et al., op. cit.).

Carbon technology. The gold industry was revolutionized in 1973 when it was found that gold could be absorbed by, and later stripped (eluted) from "active" carbon[1]. The technique is efficient and replaced the traditional recovery processes which used filtration or counter-current decantation in

1. In this instance the gold absorbing qualities of "active" carbon in the reduced form of coconut shells or peach pips are used to the benefit of the treatment process. It does not refer to the naturally occurring carbonaceous material which causes refractoriness.

conjunction with Merrill-Crowe zinc precipitation (Dahya and King, 1983). The economic benefits of carbon technology are evident from the number of carbon-in-pulp and carbon-in-leach plants now operating worldwide. In South Africa no conventional filtration plant has been built since the carbon-in-pulp system was successfully commissioned in about 1977 (MINTEK Report, 1987).

Pre-treatment of refractory gold concentrates. Since many ores are refractory, considerable effort has been focussed on obtaining higher gold recoveries at reduced cost and less pollution from these ores. In particular bacterial oxidation and pressure oxidation offer non-polluting alternatives to the traditional method of roasting gold concentrates. In South Africa, Genmin has pursued bacterial oxidation at Fairview Mine in the eastern Transvaal where the company treats refractory sulphidic concentrates in a semi pilot-scale plant. The results have so far been encouraging and Fairview Mine has recently decided to dismantle its Edwards roaster and treat all refractory concentrates in a full-scale bacterial oxidation plant.

Heap leaching technology. Heap leaching of gold ores was largely advanced in North America to treat low-grade oxidized gold ores in Nevada. Subsequently the technology has found application in other regions of the world, as for example at many open pit gold mines of western Australia.

Liquid effluent treatment. Due to increasing environmental constraints, the traditional methods of treating cyanide effluents by alkaline chlorination or hydrogen peroxide, are being replaced by alternative means using an enzyme (cyanide hydratase) or a SO_2-air process (Wall, et al., 1987).

Alternative lixiviants. Due to the toxicity of cyanide, attempts have been made to find an alternative non-toxic lixiviant. Thiourea is known to dissolve gold more quickly than cyanide but commercial applications are not yet widely practiced and more research is required (Deschenes, 1986).

Fig. 6.3 illustrates the present-day options for processing and extracting gold from ores and concentrates. As discussed earlier the precise route (flowsheet) depends on the mineralogy of the ore. The flowsheet will involve a combination of two or more of the following unit processes: crushing, milling, gravity concentration, flotation, pre-treatment, cyanide leaching, carbon absorption, tailings disposal, electrowinning, zinc dust precipitation, amalgamation and smelting. These unit processes open up a vast subject of extractive metallurgy which cannot be addressed here. Wills (1979) and Stanley (1987) comprehensively cover the subject and the reader is referred to their texts for more detailed information. In the evaluation of selective open pit gold deposits there are four key areas which will probably play an important role. These are gravity concentration, pre-treatment of refractory sulphide concentrates, carbon and resin based technology, and heap leaching. The relevant details of each are discussed below.

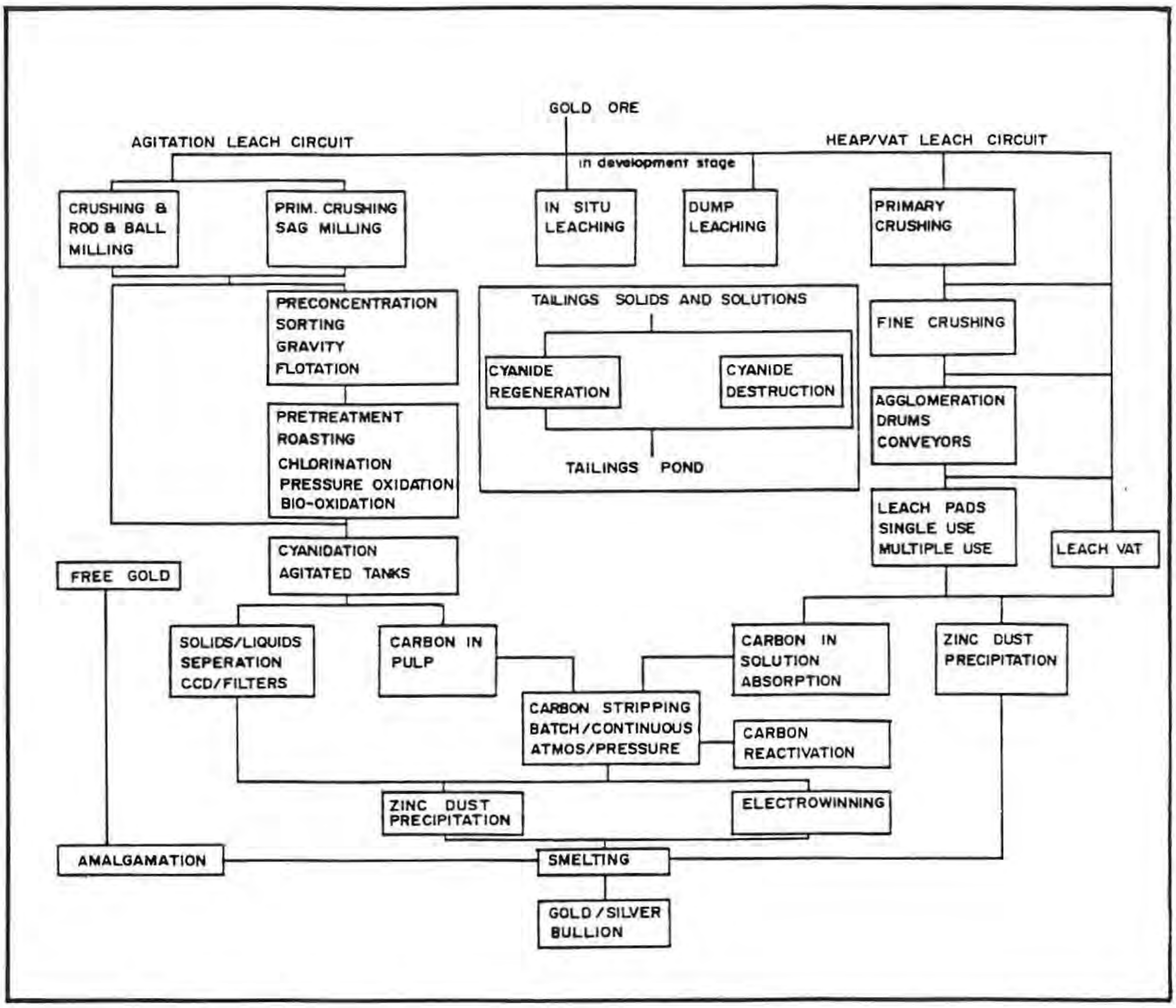


FIGURE 6.3. Gold ore treatment selection (modified from Young, 1983)

Gravity concentration. Gravity concentration involves the separation of one mineral from another based on the relative density (r.d.) differences between the two minerals. The high density of gold (r.d. 19.3) and pyrite (r.d. 5.0) compared to the gangue minerals (e.g. quartz r.d. 2.7) makes gravity concentration ideal for recovering gold from milled ores. Despite this, gravity concentration has tended to be neglected from hardrock flow-sheets, probably because more attention has been focussed on the flotation and cyanide leach aspects of gold recovery. Yet gravity concentration offers several economic and environmental advantages, the most important of which are listed below, after Burt and Mills (1985) and Penman (1987).

1) Early recovery of gold.
2) Improved recovery resulting from the exclusion of coarse gold from the cyanide leach circuit where insufficient residence time can result in gold loss to the tailings.
3) Improved recovery resulting from the specialized treatment of the concentrate. For example coatings of iron oxides on the gold particle can be removed.
4) Relatively low capital outlay.
5) Energy efficient (lower installed cost per tonne of throughput than flotation).
6) No expensive reagents are used, i.e. less prone to spiralling reagent costs.
7) Non-polluting.
8) Simple operation.

Opposed to these benefits are several disadvantages which need to be considered carefully before installing gravity concentration equipment. First, there is increased security risk caused by gold thefts. Second, there is a lower limit of gold particle size that can be separated. Even with sophisticated equipment this limit is still about 10 microns (Burt and Mills, op. cit.). This means that gravity concentration alone cannot recover all the gold from the ore. A flotation unit may still have to be installed and the addition of gravity concentrators could complicate the circuit. Fig. 6.4 shows the operational particle size ranges of the more common gravity concentration units available. Traditionally Johnson drums, plane tables, riffled tables, corduroy shaking tables, strakes and sluices

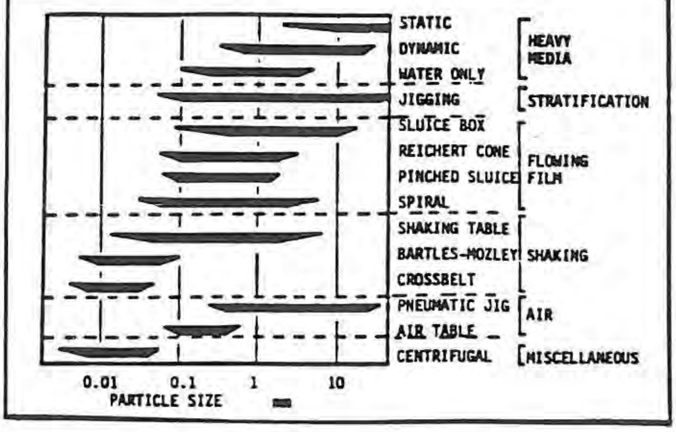


FIGURE 6.4. Particle size operating efficiency of common gravity concentration equipment (from Burt and Mills, 1985)

have been employed to recover gold particles greater than about 50 microns. Below this particle size gold recovery becomes prohibitively more difficult. However, several interesting new innovations in the fine particle recovery go some way towards resolving this problem. The Reichert cone and spiral separator, and the Knelson concentrator may now offer practical solutions to recovering fine gold particles (Richards and Bangerter, 1984; Harris, 1984 respectively).

Judging from present mining operations the principle application of gravity concentration is to complement, rather than replace the flotation and cyanide leach processes. Richards and Bangerter (op. cit.) report that at Kambalda in western Australia, Reichert cones and strake tables recover 40 to 50 per cent of the gold. This has the advantage of reducing the capacity requirements and reagent costs of the carbon-in-pulp circuit. In addition fluctuations in grade of the ore reporting to the carbon-in-pulp circuit are avoided thereby assisting metallurgical treatment and reducing cost. The grade of ore is also important. Penman (1987) reports that gravity concentration is only effective where high-grade ores are being treated. Presumably where low-grade ores are treated, the large tonnage throughput cannot be efficiently handled by gravity concentrators.

Pre-treatment of refractory sulphide gold concentrates. As mentioned previously the main commercial methods of pre-treating refractory sulphide flotation concentrates include roasting, pressure oxidation and bacterial oxidation. In open pit mines it is likely that weathering near surface may have rendered the ore free milling, but as mining extends below the limit of weathering, refractory ores are generally encountered. The economic and environmental aspects of each method must therefore be considered. Useful articles which describe and compare the three processes are presented by Taylor et al. (1989), and Haines and van Aswegen (1990).

In roasting the concentrate is heated to temperatures around 650 - 750 ^{o}C for periods of 20 - 30 minutes. The objective is to oxidize the refractory sulphide minerals into porous iron oxides (calcines) so that the gold is liberated for subsequent cyanidation. The roasting process also produces sulphur dioxide which can be used to manufacture sulphuric acid. In addition, if arsenopyrite is present in the concentrate, gaseous arsenic trioxide is also produced. The arsenic trioxide forms a by-product, but it must first be further refined to very high purities before the product is marketable.

The main advantages of roasting are high gold recoveries of about 90 per cent, short roasting periods, and removal of arsenic or antimony cyanicides. These benefits, however, are severely curtailed by environmental problems related to sulphur dioxide and arsenic trioxide emissions. Methods are available to remove the harmful emissions but they can never be 100 per cent efficient. There are also operational and economic disadvantages. Incorrect roasting temperatures can drop gold recoveries; high carbonate content in the concentrate can evolve enough CO_2 to blanket the roaster bed and stop oxidation of the sulphides; calcine ash containing gold can be lost to the atmosphere via the stack; secondary minerals such as iron arsenates produced during roasting can coat the gold particles and inhibit cyanidation; and cyanicides such as ferrous sulphate may be formed during the roasting process. The capital cost for a roaster is less than a pressure oxidation plant, but higher than a bacterial oxidation plant.

Pressure oxidation can be used to treat ores and concentrates. In this method oxidation of the refractory sulphide minerals and liberation of the gold for cyanidation is achieved by heating the ore or concentrate in autoclaves at temperatures of 170 - 190 ^{o}C and pressures of about 2200 kPa. The oxidative process produces sulphuric acid (of very poor quality) and ferric iron, both of which are corrosive. The process also requires pure oxygen so an oxygen plant is necessary. Once the sulphide minerals have been oxidized, lime is usually added to the pulp to bring the pH up to 10.5 for cyanidation.

The main advantages of pressure oxidation are economic and environmental. On the economic side, high gold recoveries in excess of 90 per cent and generally better than roasting can be achieved after a few hours of autoclaving. The adverse effect of cyanicides are likewise reduced. A related process (high pressure/low alkalinity cyanidation) has also proved effective in treating antimony-rich concentrates (Wall, et al., 1987). On the environmental side, pollution of the atmosphere is eliminated. Harmful arsenic and sulphur are precipitated out as stable ferric arsenate and calcium sulphate. According to Haines and van Aswegen (1990) neutralization of the toxic arsenic is highly efficient if the ferric:arsenic molar ratio exceeds 4:1.

The main disadvantages of pressure oxidation are complex engineering design, operational breakdowns and relatively high operating costs. Due to the corrosive nature of the products and high pressure required to oxidize the sulphide minerals, sophisticated materials and equipment need to be designed. Operational problems are encountered with agitation, sulphur clogging of the autoclaves, instrument failure, overheating, off-gas control, corrosion, control of the process, and understanding of the engineering problems. Operational breakdowns are common, consequently plant availability is poor. Pressure oxidation also requires a high level of training. Skilled supervisors are therefore needed to monitor the process. The relatively high operating costs are caused by the oxygen and lime consumption. In addition ores rich in silver are also at a disadvantage as refractory argentojarosite is formed in the autoclave.

In bacterial oxidation bacteria Thiobacillus ferrooxidan and Thiobacillus thiooxidan are commonly used to breakdown the sulphide minerals and liberate the gold for subsequent cyanidation. The process can be performed on rock dumps and tailings, but improved gold recoveries are obtained when concentrates are treated in tanks under controlled conditions. The optimum conditions require that air and small amounts of nutrients (nitrogen, phosphate) be supplied to the bacteria, and that a pH of 1.8 be maintained. Temperature is also important. The bacteria perform best between 35 - 40 ^{o}C, but since the reaction is exothermic it is usually necessary to cool the oxidation vessels. Once the refractory sulphide minerals have been "eaten", lime is added to the pulp to bring the pH up to 10.5 for cyanidation.

Bacterial oxidation has several economic and environmental advantages. On the economic side, gold recoveries in excess of 90 per cent can be obtained. At Fairview Mine, recoveries of 93 per cent have been consistently obtained from the pilot-scale plant. Compared to pressure oxidation the main advantages are ease of operation. Bacterial oxidation plants experience fewer stoppages than pressure oxidation plants and are easier to control. Unskilled labour can monitor the process so the practice is ideal for remote areas. Also capital costs are lower for small plants, operating costs are about equal and silver-rich ores can be treated (Taylor et al., 1989; Haines and van Aswegen, 1990). Compared to roasting the main advantages are environmental and improved gold recovery. Like pressure oxidation, no pollution problems are encountered. The resultant ferric sulphate, arsenous acid and sulphuric acid are neutralized to products suitable for safe disposal on the tailings dam.

Despite the strong economic and environmental advantages, commercial use of bacterial oxidation is limited. The main reason is that the technology is still being refined. Mining companies are reluctant to use a new process until it has proved itself economically viable in a full-scale operation. Also there is the aspect of marketing. Up until recently Genmin has kept its bacterial oxidation technology confidential. Only now are details of the process being released, as Genmin now wishes to recoup its investment by selling the technology to industry.

The main disadvantage of bacterial oxidation are the relatively long residence time of 3-4 days. Some operational problems are also apparent. The high acidity causes corrosion of the equipment whilst a breakdown in the cooling system may destroy the bacteria thereby delaying production until the culture regenerates itself. Lime and cyanide consumption is also high and therefore costly. Also sulphur contents are important. Concentrates with a very high sulphur content require longer residence times for the bacteria to "eat" the sulphide minerals.

<u>Carbon and resin technology</u>. Fig. 6.5 schematically illustrates a typical carbon-in-pulp circuit. The process consists principally of four operations: 1) absorption of the gold from cyanide solutions by activated carbon in counter-current contactor containers; 2) elution of the gold from the loaded carbon; 3) electrowinning of the gold from the pregnant stripping solutions; and 4) reactivation of the carbon for recycling in the absorption system.

In carbon-in-leach, stage (1) is eliminated and activated carbon is added to the pulp simultaneously with the cyanide lixiviant. Carbon-in-leach is primarily used to prevent the adverse preg-robbing effects of natural carbonaceous material in the ore.

In resin-in-pulp, ion exchange resin fibers or beads are used as an alternative to activated carbon to absorb gold from the cyanide solutions. The advantages of using resin are faster loading of gold, elimination of complex elution methods and ease of regenerating the resins. At present, the technology is still at the development stage, therefore commercial application is subject to the same uncertainties as bacterial oxidation. However, MINTEK have reported encouraging results from a semi-pilot-scale plant being used at Golden Jubilee open pit mine in the eastern Transvaal, South Africa (Mining Journal, 1990 September 28, p. 236). Carbon-in-pulp was initially tried at Golden Jubilee but proved unsuccessful, principally I believe because humic acids and clay were binding the carbon and inhibiting efficient gold absorption.

The advantages of carbon-in-pulp, carbon-in-leach and resin-in-pulp technology are their relatively short residence periods, high gold recoveries (commonly in excess of 95 per cent), relatively low capital and operating costs, and ease of operation. The disadvantages are mainly operational with problems related to activated carbon pores becoming clogged by fines, abrasion of the carbon and associated gold loss, screening, and elution. However, many of these problems continue to be eliminated with improved research and development (MINTEK report, 1987).

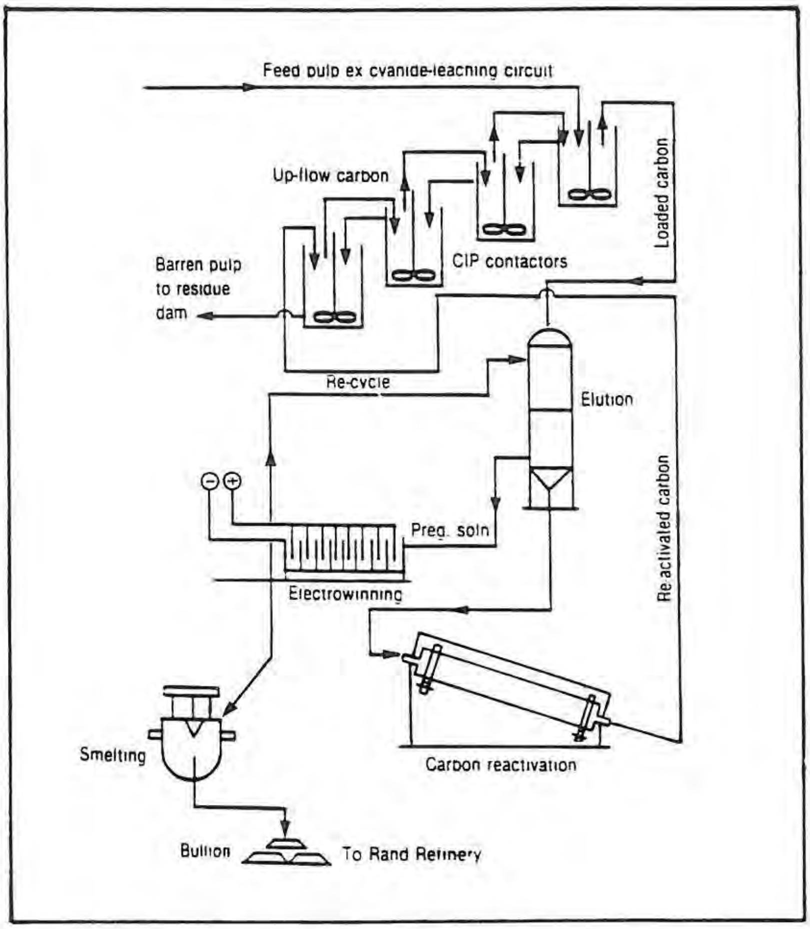


FIGURE 6.5. Typical carbon-in-pulp process for the recovery of gold (modified from MINTEK report, 1987)

Heap leaching. Fig. 6.6 schematically illustrates a typical heap leach circuit. The process consists principally of four operations: 1) preparing the ore by crushing and/or agglomeration with sodium cyanide and lime; 2) sprinkling cyanide solution onto the heap; 3) collecting the resultant pregnant solution in a pond; and 4) removing the gold from solution either by carbon absorption and electrowinning, or precipitation on zinc dust (Merrill-Crowe process). Heap leaching can be used to treat ore, rock dumps and tailings. A detailed account of heap leaching practices are presented by Worstell (1986), and van Zyl and Hutchison (1988).

The main advantages of heap leaching are simplicity of design and operation; lower capital and operating cost than carbon-in-pulp, carbon-in-leach and resin-in-pulp; short start up time of 4 to 12 months; no milling requirements; water efficiency; and no need to establish tailings dams. The disadvantages of heap leaching are low gold recovery (commonly only about 70 per cent); high cyanide consumption where cyanicides are present; and long residence times of about 1 to 3 months. Also heap leaching can be adversely affected by the climate. In cold climates problems arise from freezing of the cyanide solutions whilst in wet climates dilution of the cyanide solutions requires make-up of cyanide.

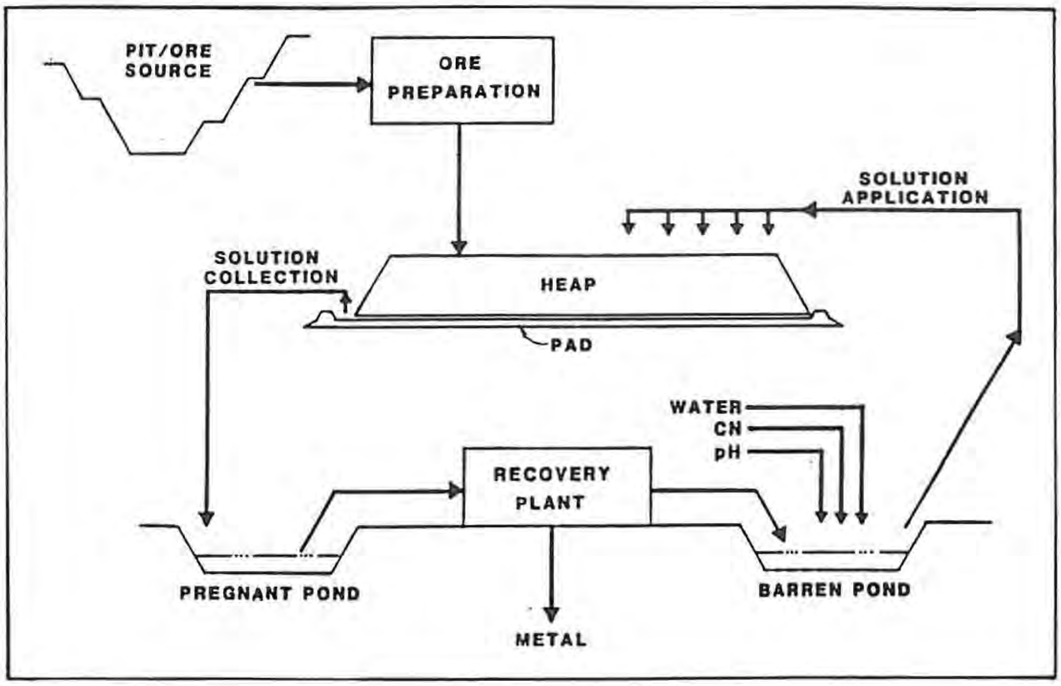


FIGURE 6.6. Schematic heap leach process (from Dorey and van Zyl, 1988).

Since gold heap leaching has been introduced mining companies have had difficulties in deciding whether to opt for carbon-in-pulp or heap leaching. The low capital costs and short start up times of heap leaching are very attractive incentives to achieving a quick return on the initial capital investment. However, the comparatively low gold recoveries and longer residence times adversely affect the longer term profitability of heap leach operations. Many mines have decided to employ both techniques. Sometimes heap leaching is employed initially in order to quickly generate enough capital to construct a carbon-in-pulp plant. In other situations heap leaching is used to treat low-grade (less than 2 g/t) ores whilst the high-grade ores are treated in a carbon-in-pulp circuit. Although the latter approach may be profitable a careful assessment of the two options should be made before the mine is developed. Battaglene et al., (1984) used standard discounted cash flow methods with variable gold prices, gold head grades, gold recoveries, and capital and operating costs, to show that carbon-in-pulp offered a better return on investment than heap leaching, even for low-grade ores. Their results excluded other variables such as throughput rates and contract mining alternatives, but the analyses demonstrate the importance of performing an economic analysis in order to select the most efficient and profitable treatment process. In my opinion heap leaching is best used in situations where the reserves are too limited to sustain production for more than a few years or where the company cannot afford the cost of constructing a mill and carbon-in-pulp plant. Mining companies that opt for heap leaching simply because it is "in vogue" could be disappointed. Well tested and proven methods like carbon-in-pulp usually prove better in the end.

7. PROJECT ENVIRONMENTAL APPROVAL

The purpose of this chapter is to: 1) examine the environmental problems related to gold mining; 2) outline the principles of Integrated Environmental Management - a new environmental policy introduced by the Council for the Environment, South Africa; and 3) offer some guidelines to obtaining environmental approval of the mining venture.

The environment forms an integral and increasingly important part of mineral prospect evaluation. In the past, mining companies have generally shown little concern for the effect that their exploration and mining activities have had on the environment. Ample evidence of this is presented by the abandoned workings, derelict structures, scarred landscape, unvegetated tailings dams, and polluted groundwaters commonly seen in mining districts. This situation is now rapidly changing. Since the late 1960's there has been growing concern that the environment is being destroyed. These concerns were first voiced in North America and Europe, but have subsequently started to spread worldwide. Quite rightly, the environment should concern everyone - developer and conservationist alike. Newspapers and television programmes talk almost daily of pollution and destruction of the earth's atmosphere, oceans, rivers and land. In South Africa leading industrialists now publicly plead for environmental management not only to sustain economic development but also to ensure survival (Egan, 1990).

7.1. WHY ENVIRONMENTAL APPROVAL

The term environment can be confusing and requires defining. Originally environment referred to physical, biological and chemical components of a specific site. More recently it has been expanded to include socio-economic components. Boardman et al. (1978) define environment as follows:

- air, land and water;
- plant and animal life including man;
- the social, economic and cultural conditions that influence the life of man or a community;
- any building, structure, machine or other device or thing made by man;
- any solid, liquid, gas, odour, heat, sound, vibration or radiation

resulting directly from the activities of man; or
- any part or combination of the foregoing and the interrelationships between any two or more of them.

Thus where mining is concerned the term environment relates both to the development of the mineral resource for the benefit of mankind and conservation of the mineral and other (land, wildlife, scenic, recreational etc.) resources for the benefit of future generations.

Several good reasons can be found as to why the mining company must obtain environmental approval during the exploration stage of the mine's life. These are listed below.

1) Since environmental legislation is becoming more stringent it is important that mining companies anticipate these changes well in advance of them being enforced. This will ensure that adequate provisions to preserve the environment are incorporated into the mine plan before development starts.

2) No mining can start until all the necessary mining and environmental permits have been obtained.

3) It is a moral obligation and responsibility of the mining company to conserve the environment.

4) Mining companies must not be perceived by the public as purposely neglecting the environment. Neglect of the environment leads to bad publicity, lawsuits and unwanted financial settlements.

5) The government and public must be satisfied of the company's good intentions, as well as their technical and financial ability to conserve the environment.

6) Environmental approval is costly and time consuming. The economic outcome of the project will be adversely affected if there are delays in obtaining the mining permits.

7) Environmental planning and management are in the interests of the company. An effective policy has economic benefits as environmental disasters are more costly to rectify than prevent.

7.2. ENVIRONMENTAL PROBLEMS RELATED TO OPEN PIT GOLD MINING

In open pit gold mining the main environmental problems are related to several causes, the most common of which are as follows:

1) erosion of tailings;
2) pollution of the atmosphere by emissions;
3) dust and noise pollution;
4) changes to the ecology;
5) visual impact;
6) water pollution;
7) disruption of the surface or groundwater supply; and
8) disruption of current and future land use.

Erosion of tailings. Tailings erosion has become a problem due to the finer grinding now commonly used to separate the low-grade gold from the ore (Cannon, 1989). Wind erosion can pollute the air with fine silica and cause silicosis - a lung disease. To prevent wind erosion the tailings must be compacted and the surface stabilized chemically or with vegetation. Tailings dams should not be located near any community. Water erosion of the tailings also needs to be avoided as this may silt-up rivers.

Pollution of the atmosphere by emissions. Harmful emissions (mainly from roasting stacks) commonly include sulphur dioxide, carbon monoxide, arsenic trioxide and ash. Although there are devices (gas scrubbers, baghouse filters and electrostatic precipitators) to collect these emissions before they are discharged directly into the atmosphere, the devices are never completely effective (Cannon, op. cit.). Such emissions inevitably destroy nearby vegetation and pollute the atmosphere. Also arsenic trioxide eventually finds its way into the groundwaters. The Edwards roasters are notorious for their inability to prevent gaseous emissions. Far better emission control is provided by the flousolids roasters, or better still, opt for non-polluting pressure oxidation or bacterial oxidation (Haines and van Aswegen, 1990),

Dust and noise pollution. Blasting and haulage of materials into and out of the open pit produces dust and creates noise. Spraying the benches and haulage roads with water to prevent dust and keeping noise levels below the legal limits is required.

Changes to the ecology. The mining operation is almost certainly going to disturb the delicate ecosystem of the local fauna and flora. Some undisturbed land may have to be set aside to preserve the natural habitat for later colonization once mining is complete. Alternatively endangered animals and plants may need to be relocated to safe regions.

Visual impact. Open pit mining has far greater visual impact on the landscape than underground mining. The pits, tailings dams, waste dumps, heaps, settling ponds, pregnant solution ponds, processing plants, infrastructure, haulage roads, and parking lots are unsightly to most people! With the obvious exception of the open pit, the other features should be located in areas where there is minimum visual impact, i.e. concealed behind trees or a hill. Also disturbance of hill tops and ridge lines, or any picturesque landscape should be avoided.

Water pollution. Water pollution of surface and groundwaters can be a major environmental problem in open pit gold mining. The main sources of possible pollution include cyanide, arsenic and sulphuric acid.

During the cyanidation process highly poisonous hydrogen cyanide gas is produced if the leach is not maintained at a high pH. Also neutralization of the cyanide into non-toxic cyanide complexes or compounds is required before the residues from the metallurgical plant are disposed to the tailings dam; otherwise water seepages from the tailings will eventually pollute rivers and groundwaters. Some methods to accomplish neutralization were mentioned briefly in section 6.4. Where heap leaching is concerned, cyanide in the old heaps must be neutralized before the dumps are rehabilitated (Mining Journal Supplement, 1991). Also during the operation of the mine, any pregnant solution ponds containing cyanide solvent must be covered with nets to prevent birds from drinking the water.

Arsenic poisoning is a problem where arsenopyrite-rich ores require pre-treatment by roasting. Arsenic trioxide discharged from the stack or remaining in the calcines can find its way into the surface and groundwaters. Arsenic is highly poisonous and only small amounts entering the blood stream cause death. Treatment of the gold concentrates by bacterial oxidation or pressure oxidation eliminates this problem.

Pollution of surface and groundwaters by acids commonly occurs where tailings dams and waste dumps containing pyrite are exposed to weathering. Pyrite decomposes when it comes into contact with air and water to form ferrous hydroxide and sulphuric acid. Ferrous hydroxide is precipitated but the sulphuric acid increases the acidity of the water and destroys organisms in streams, kills trees, corrodes any metal products and renders the water unacceptable for recreational purposes. Methods to treat acid waters are expensive and should be prevented before the problem becomes acute. Treatment may include ponding and controlled release of the waters, permanent sealing of drainage points, disposing of tailings and wastes into water-tight clay-lined pits, and neutralization of the waters with alkalis (Cannon, 1989).

Disruption of the surface or groundwater supply. Open pit mining will probably change the aquifer regime which in turn can disrupt the supply of water to the local community and ecosystem. An assessment of the effect that mining will have on the quantity and quality of the waters will be required in order to expedite any remedial action. Such matters are dealt with in more detail by Brassington (1982).

Disruption of current and future land use. Plans to rehabilitate the mine to its original land use are usually required once all mining activities are complete. Alternatively other arrangements could be acceptable. For example it may be possible to flood the open pit and use the area for recreational purposes or as a natural wildlife sanctuary. Wells (1986) offers some useful guidelines to the long-term rehabilitation of open pit workings.

7.3. INTEGRATED ENVIRONMENTAL MANAGEMENT

In most countries government legislation aimed at preventing pollution and preserving the environment is mandatory. The legislation may seem strict and preventative of economic development. Certainly in the USA, Environmental Impact Assessments originally focussed only on the detrimental, rather than the socio-economic benefits of mining. In South Africa the anti-development aspects of Environmental Impact Assessments were appreciated and the Council for the Environment has recently proposed the term Integrated Environmental Management (IEM) to replace Environmental Impact Assessments (Council for the Environment, 1989). IEM pertains to be a holistic environmental policy which tries to balance conservation with development. This is the crux of an environmental policy, namely: to prevent pollution and save some of the mineral resource and all other resources for the future, whilst at the same time exploiting the mineral resource for the benefit of the investor, local community and country. IEM tries to ensure that a systematic and structured approach to environmental concerns is adopted at all stages of the mine's life. This is appropriate and no matter where in the world or what legislature is enforced, a balanced approach to environmental concerns seems the most reasonable.

A complete appraisal of IEM is beyond the scope of this report and only an outline is presented here. Egan (1990) summarizes the policy whilst the Council for the Environment (1989) provides a detailed explanation of its policy. In essence there are four stages in IEM: the proposal generation, assessment, decision and implementation stages (Fig. 7.1).

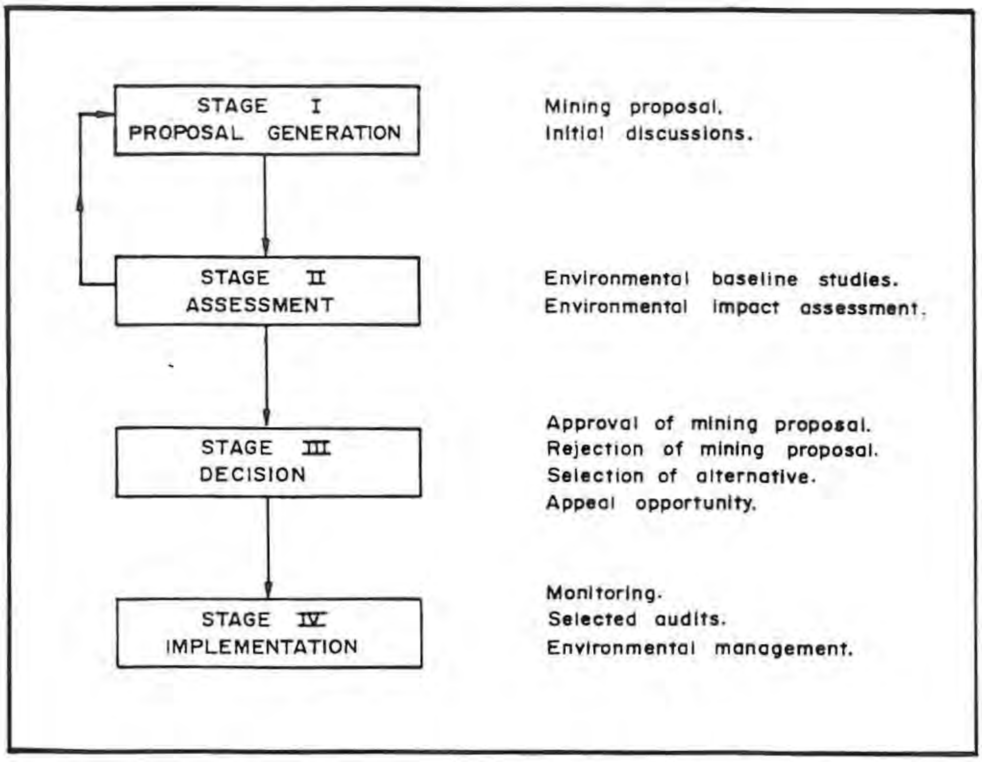


FIGURE 7.1. Main stages in the Integrated Environmental Management procedure as proposed by the Council for the Environment, South Africa.

The proposal generation stage is concerned with formulating a mining proposal, as well as viable alternatives to the proposed mining plan. The assessment stage is concerned with investigating and evaluating the impact that mining will have on the environment during the life of the mine and after mining operations have ceased. This was the traditional focus of the Environmental Impact Assessment. An environmental assessment will most likely alter the original mining proposal in which case the procedure starts at stage (1) again. The decision stage is concerned with identifying and formally approving the mining proposal which is in the best overall interests of all parties concerned, i.e. both conservation and development demands are met. Finally the implementation stage is concerned with ensuring that the approved mining proposal is implemented. Stages (1), (2) and (3) fall into the exploration period of the mine's life and therefore warrant further discussion.

Stage (1) is initiated when the mining company submits a mining proposal to the relevant regulatory government body. In South Africa this involves informing the government mining engineer. However, there are likely to be several government agencies that need to be contacted depending on the country and actual location of the mineral prospect within the country. The mining proposal furnishes basic information about the mine, details of the mine planning and description of the operation. At the geological feasibility stage such information is only likely to be rudimentary, nevertheless, it would be expedient to approach the authorities in order to open discussions. This allows the mining company to explain to the authority the purpose and need for the proposed mine, and gives the authority an opportunity to recommend a suitable course of action to make the mining proposal environmentally acceptable. The discussions should ensure that the mining company does not become committed to one course of action before considering possible objections to the mining venture.

During stage (2) a number of environmental baseline studies are conducted to determine the impact that mining will have on the environment. Baseline studies involve collecting accurate and representative on-site environmental data from the mining lease area and adjoining district. Like the ore reserve, mine design and metallurgical studies, baseline studies are the foundation to the whole environmental evaluation. The most important items that must be collected for environmental baseline studies are listed below, after Wells (1986).

1) Climatic conditions. Items such as rainfall, temperature, evapotranspiration, winds and extreme climatic conditions all have a bearing on the selection of plants for rehabilitation, the time of planting, and the length of the growing season.

2) Land use and capability. This information is required in order to restore the land to its original use or plan for alternative uses.

3) Flora and fauna. A study of the ecosystem is required to determine the overall impact that mining will have on the wild plants and animals. In particular rare species must be protected.

4) Soils. A detailed description of the soils including the types, chemistry, thickness, location, stability, and slope on which they occur is very important. Soils will need to be removed and later replaced after mining operations have ceased. In laterite gold deposits the soil may even have to be processed in order to recover the gold. These items are very relevant to mine rehabilitation.

5) Surface and groundwaters. The quality and quantity of surface and groundwaters require careful monitoring during and after the mine's life. Adequate sampling and analyses for dissolved constituents, toxic materials, acidity, and the level of the water table need to be recorded. Surface waters and groundwaters affect the pit drainage requirements, the supply of water to the local community and the local ecosystem.

6) Mineral resources and geology. The depths and types of the mineralisation, and overburden thickness need to be noted for the mining proposal.

7) Geographical factors. Detailed information on the location of towns, roads and dwellings together with information on the topography, drainage pattern and land use need to be recorded for visual impact, pollution and rehabilitation.

8) Noise and dust. The ambient noise levels and dust concentrations at various locations and times need to be noted.

9) Traffic. Increased levels of traffic need to be determined in order to assess the ability of current roads to cope with the extra traffic.

10) Archaeological and cultural factors. Sites of historical or special interest (public footpaths, picnic sites, cemeteries etc.) need to be noted as they could be disturbed by the mining operation.

Baseline studies can be conducted throughout the exploration period and preferably need to be started at the geological feasibility stage. Much of the information can be collected by the company's exploration team. Other information, especially that related to the ecosystem requires environmental scientists. If adequate environmental input has been included in the formal mining proposal, the assessment stage will be simplified. In fact the environmental assessment report required at the end of stage (2) may contain baseline information already collected during stage (1). This prevents delays, which at the final feasibility stage need to be kept to a minimum. Also it ensures that the information relates to the longest possible period of time. Subtle changes to the environment sometimes become apparent only after several years of study.

The final environmental assessment report will generally include the following contents:

1) basic introductory information on the proposed mine;
2) detailed description of the mining operation;
3) a detailed description of the environment;
4) a description of the impact that mining (and alternatives) will have on the environment;
5) a discussion of how negative environmental impacts will be prevented, reduced, controlled, and/or neutralized during the life of the mine;
6) socio-economic improvements to the environment;
7) rehabilitation plan during and after the life of the mine; and
8) management and funding details.

Appendix 4 provides specific details on these eight items and can be used to help the evaluating team design an effective environmental policy.

During stage (3) the authority will decide whether to approve the mining proposal, grant approval conditional upon specific mitigation measures, reject the proposal, or select some other alternative (including the alternative of maintaining the status quo). The authorities objective at this stage is to make the procedures as open as possible. This means that the public must be informed of the proposed mine and invited to attend the hearing. Any public concerns about the proposed mining activities can then be taken into account and acted upon. Appropriate channels to contest the decision must also be left open should it be disagreeable to any party.

7.4. PERMITTING STRATEGY

A review of the literature on environmental matters indicates that there are several steps that the mining company can take to make the process of acquiring the mining permits a lot easier. Selected articles on this subject include Thatcher and Struhsacker, (1987); Fletcher, (1987); and Alberts (1990). Some of the most important aspects of permitting strategy to emerge are: 1) maintaining good communications; 2) unravelling the regulation maze; and 3) formulating an environmental policy.

Maintaining good communications. Environmental planning should be one of the first steps in evaluating a mineral prospect. Too many companies make the mistake of approaching the authorities when senior management have virtually decided to develop the mine. One reason why the government authorities and public may appear anti-development is that they are only informed about the proposed mine at the final feasibility stage. Such delayed tactics should be avoided. It is imperative that the relevant agencies be contacted during the early exploration stage, even if the mineral prospect may never be developed. This will open the channels of communication and the authorities will appreciate the forward and honest approach set by the mining company. Informing the public and inviting their opinions is also important as negative public opinion can destroy the chances of developing a mine. Actually involving the public in the environmental planning may be one way to avoid this possibility. For example at Homesteak Mining's open pit gold mine in South Dakota, U.S.A., the pit is being extended through the middle of the town! Such an operation would not have been possible without community participation brought about by regular meetings, briefings, hearings and prompt responses to any concerns (Mining Journal Supplement, 1991).

Unravelling the regulation maze. The number of agencies, environmental protection laws and regulations can be complex and will vary depending on the country of location. Changes to pre-existing laws and new regulations also appear from time to time. They will all need to be unravelled early on during the exploration period in order to assist with implementing the on-site environmental baseline studies and environmental planning.

Formulating an environmental policy procedure. If the company wants to expedite an effective environmental policy it must first design its own guidelines. Those outlined by the Council for the Environment (1989) seem appropriate. However, it would be worth mentioning some points made by Fletcher (op. cit.).

1) One person in the company should be appointed as a spokesman to interface with the regulatory agencies.

2) The individual responsible for interfacing with the agencies must have a complete understanding of the laws, regulations and guidelines which dictate the process of approval.

3) A personal and sincere approach to understanding the regulatory process, work load of the agencies, and realistic time schedule is essential.

4) All critical information should be hand-carried to its destination and explained.

5) All requests or questions must be responded to promptly and clearly.

In conclusion environmental problems should not be considered an obstacle to successfully developing a new mine and a sensible and honest approach by the mining company will in most cases be rewarded by a positive development decision. One thing is certain, environmental legislation and public reactions will become increasingly more demanding. Mining companies that anticipate and cater for these changes will emerge successful. For instance governments will probably insist that a trust be established to fund rehabilitation once mining is complete. Very often it is the small, so called "fly by night" companies that get into difficulties and find themselves unable to finance the cost of rehabilitation.

8. ECONOMIC ANALYSIS OF THE MINERAL PROSPECT

The purpose of this chapter is to: 1) outline a suitable method to help determine the economic potential of the mineral prospect; and 2) discuss the investment criteria that must be considered in order to make a sound investment decision. Selected references covering these topics include Mackenzie (1981), Gentry (1988), and Mallinson (1989).

In the previous chapters we examined the geology, ore reserves, mine design, metallurgy and environmental aspects of selective open pit gold deposits. This chapter will show how these items are combined with marketing, financial and government related aspects into an economic analysis of the mineral prospect. Determining the economic potential of the mineral prospect is the ultimate reason for evaluating the property. The investors want to know what profit can be expected if the property is sent into production. However, as illustrated in Fig. 2.5, the economic (financial) analysis is only as good as the assessments which underlie it. This emphasizes the danger of being mislead by an impressive rate of return whilst forgetting that it is the reliability and accuracy of the data on which the economic analysis is based that is far more important.

8.1. OBJECTIVES AND SCOPE OF ECONOMIC ANALYSIS

The economic analysis of a mineral prospect is not too dissimilar to an economic analysis of any business venture. There are, however, some important differences which set mineral investments apart from other business investments. The most notable is that the orebody has a finite life. The return on the investment must therefore be accomplished within the life of the mine. Another is that the gold price - and consequently the project's profitability - is often subject to unexpected fluctuations. Also the risks associated with mining ventures are generally greater; mining is usually more capital intensive; and the mining investment may have a longer start up time to production (Gentry, op. cit.). These features must constantly be kept in mind during the economic analysis.

In order to conduct a comprehensive economic analysis, the key elements that need to be estimated over the life of the mine are the likely costs, risks and returns of the investment alternative. Thus the main objective of the economic analysis is to integrate and quantify these elements accurately, as it is only with their appreciation that a sound investment decision can be made. The costs include all capital and operating costs, and royalty, lease, tax, and debt payments that must be incurred to exploit the orebody. The risks relate to the uncertainties associated with the mining investment and include any elements which may affect the present and future economic outcome of the property. Table 8.1 lists the common risk elements which are of economic significance. These are broadly divided into geological, geographical and time related risks. There is considerable overlap between the three categories and not every risk item is always significant. For instance the threat of a military coup may be eliminated in times of political stability. However, items like the ore reserve estimates are nearly always risk elements. The returns are the net profits realized after all costs and revenues have been accounted for. In this instance the revenues are derived mainly from the sale of gold which is a function of the metal recovered and gold price.

The economic analysis should also allow for comparisons between competing investments. By comparing all the investment alternatives, senior management can select those projects which have the greatest economic potential and most likely chances of success. This equates to choosing an investment which provides shareholders with a satisfactory capital growth rate and dividend payment at minimum risk. To distinguish between satisfactory and unsatisfactory investments, a hurdle rate, which is commonly based on the cost of capital, is set by the company. Investment opportunities with a rate of return above the hurdle rate are accepted whilst those below are rejected. In South Africa the hurdle rate is typically about 8 per cent (in constant money units), but may vary from 5 to 15 per cent depending on the company's profit expectations and view of the associated risks. High risk investments usually require a higher hurdle rate whereas the reverse is true for low risk investments.

Fig. 8.1 outlines the main components that may be incorporated into a comprehensive economic analysis of the mineral prospect. At the start of the economic analysis are the input variables. These include all the geographical, geological, mining, metallurgical, environmental, marketing, government and financial factors needed to determine the economic potential of the mineral prospect. In order to arrive at a sound investment decision two paths are recommended. These are described as the cash flow analysis and subjective analysis paths.

In the cash flow analysis path, various input variables are required to estimate the property's cash flows. Financial yardsticks calculated from

Table 8.1. Risk elements affecting the economic potential of gold mineral prospects.

Geological Risk Elements

- mineable reserves (grade, tonnage, distribution and types)
- gold recovery and loss from plant
- size and shape of the orebody (attitude, width, strike)
- depth of orebody below surface
- structural complexity (faulting, folding, dykes)
- regularity and continuity of the orebody
- ground conditions (hardness, strength, abrasiveness, competency)
- instability of pit slope angle (jointing, fissures, faults)
- refractoriness (textural complexity and metal content of the ore minerals; grain size and distribution of the ore minerals)
- type and nature of the gangue (cyanicides, clay, talc, carbon)
- amount and quality of groundwaters
- natural catastrophe (earthquakes, volcanic eruptions)

Geographical Risk Elements

- remoteness (location of prospect with respect to transportation, access routes, water, power source, towns, supplies, manufacturers, housing and recreational facilities
- availability of labour, danger of indigenous diseases
- topography, terrain, elevation above sea level drainage
- the climate (danger of flooding, drought, high winds, temperatures)

Time Risk Elements

- gold price (short, medium and long term fluctuations)
- inflation/exchange rates (short, medium and long term fluctuations)
- changes in mineral right leasing policy
- changes in mining tax and royalty payment
- tax laws (personal income, property, dividends, capital)
- environmental sensitivity (public reaction, disasters, pollution)
- manpower requirements, union conditions of employment, foreign labour restrictions, lack of skilled personnel and strikes
- politics (threat of military coup, unrest, repatriation of profits, nationalization and sanctions)
- poor funding arrangements (market sentiment, equity, loans and forward gold sales)
- management (confidence in their skills, incompetence)
- unforeseen financial commitments
- gold, ore and concentrate thefts
- engineering (inadequate design, over design, equipment failure)

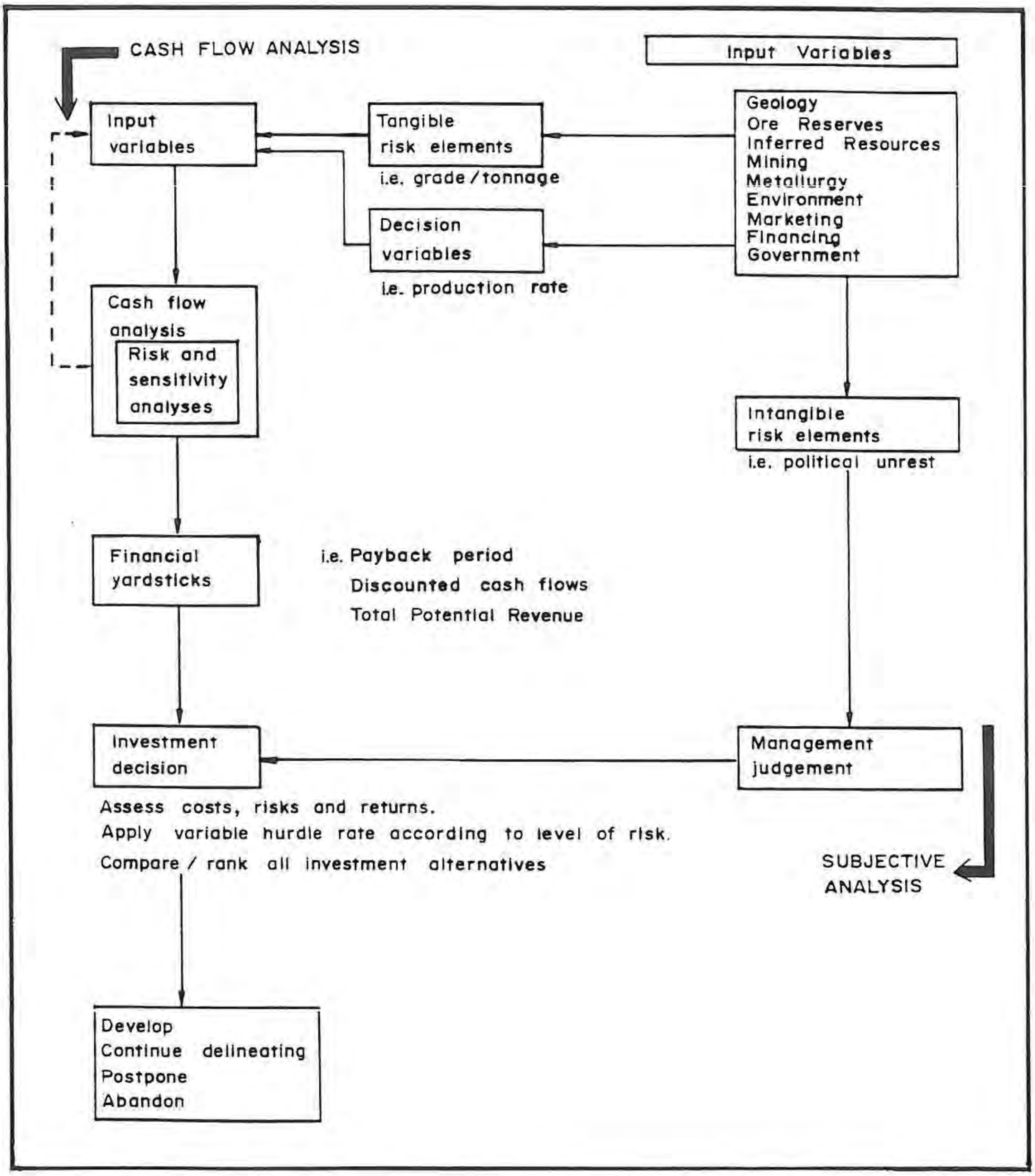


Figure 8.1. Main components incorporated into an economic analysis of the mineral prospect.

the cash flows are then used to provide a quantitative assessment of the likely costs, risks and returns associated with the investment alternative. As will be discussed later, the risk analysis forms a powerful means of effectively assessing these three key elements. The input variables required to determine the property's cash flows are divided into decision variables and tangible risk elements. An example of a decision variable would be the milling rate. At some stage the evaluator decides from a number of production alternatives that a milling rate of say 10,000 tonnes per month is optimum for the type and size of deposit being mined. In this case the milling rate is eventually fixed and no longer constitutes a variable (Mallinson, 1989). Examples of tangible risk elements are grades, tonnages, metal price forecasts, stripping ratios, and capital and production cost estimates. The uncertainty in these risk variables cannot be removed by decision but can (and must) be quantified for inclusion into the cash flow calculations.

In the subjective analysis path, input variables not incorporated in the cash flow analysis are judged. This path is more difficult to define and relies mainly on experience and sound judgement. The input variables that require subjective judgement are essentially intangible risk elements or subtle items that cannot be adequately quantified for inclusion into the calculation of cash flows. Examples of intangible risk elements would include amongst others: political threat, the likelihood of a labour strike, a natural disaster, the chances of finding additional reserves, ground conditions or the amount and amplitude of faulting. In general the most significant intangible risks elements are geological. Table 8.2, from Mallinson (1989) lists some intangible geological risk elements that may be expected to influence the various costs during the delineation, mining, treating and refining processes. To demonstrate the subtle nature of these risk elements an attempt is made to indicate the order of magnitude of the effects. I1..I3 represents increasing magnitude of increased costs and D1..D3 represents decreasing magnitude of decreasing costs.

The investment decision that culminates from the economic analysis will be based on financial yardsticks generated from the cash flow analysis and subjective judgement resulting from an assessment of the intangible risk elements. In effect the cash flow analysis and subjective analysis paths complement one another and neither should be conducted without the other. The reason for adopting this approach is to minimize the chances of making an unsound decision. It is important to integrate and evaluate all aspects affecting the economic outcome of the investment alternative. To rely solely on the calculation of cash flows is dangerous and likely to court disaster. On the other hand a decision based solely on subjective judgement of the intangible risk elements is equally dangerous. The eventual decision will be whether to develop, continue delineating, postpone or abandon the mineral prospect. If the decision is to continue delineating the prospect, the whole process of gaining more information on the mineral prospect's geology, ore reserves, mining, metallurgical, environmental, marketing, government and financial factors will be repeated. This demonstrates the iterative process of mineral prospect evaluation.

Table 8.2. The effect of different geological factors on the different cost areas (from Mallinson, 1989).

GEOLOGICAL FACTORS	DELINEATION COSTS	PRE-PRODUCTION COSTS	MINE CAPITAL COSTS	MINE DEVELOPMENT COSTS	MINE OPERATING COSTS	MILLCAPITAL COSTS	MILL OPERATING COSTS	OFF MINE REALIZATION COSTS	
increasing SIZE	I_3	I_3	I_3	D_2	D_3	I_3	D_3		
increasing STRUCTURAL COMPLEXITY	I_3	V_1	V_2	V_3	V_2				Either primary or secondary, could decrease costs if favourable eg. fold duplication.
more complex SHAPE	I_3	I_1	I_1	I_3	I_3		I_2	I_2	
less CONTINUITY	I_3	I_1	I_1	I_3	I_3		I_2	I_2	These factors could affect the milling costs if more waste dilution occurred, especially if the waste contained constituents delitreous to the milling - floatation process.
less REGULARITY	I_3	I_1	I_1	I_3	I_3		I_2	I_2	
narrower WIDTH	I_3	I_1	I_2	I_3	I_3				
changing ATTITUDE	V_3	V_3	V_3	V_3	V_3				Major effect on all of these, either increasing or decreasing costs depending on what the attitude is!
less well defined BOUNDARY	I_1			I_1	I_1				Could increase development and operating costs as more sampling / drilling would be necessary to define the orebody.
poorer GROUND CONDITION	I_1	I_1	I_2	I_3	I_3		I_2	I_2	
less favourable TOPOGRAPHY	I_1	I_1	I_1		I_1	I_2	I_2		Affects roadways, access, tailings disposal etc..
less favourable GROUND WATER CONDITIONS		I_1	I_2	I_3	I_1		I_2		More complex flotation cicuits would be required if more material was oxidized.
increasing VARIABILITY OF ORE DISTRIBUTION	I_1		I_1	I_2	I_1	I_1	I_1	V_3	May well increase milling costs but could be benificial overall in allowing for sequencing and specific timing of cash flows.
HARDER material			I_1	I_1	I_1	I_3	I_3		
more ABRASIVE material			I_1	I_1	I_2	I_1	I_3		
less favouralbe CONDITION OF ORE MINERALS						I_1	I_3	I_1	
less favourable NATURE OF GANGUE						I_1	I_3	I_3	
increasing TENOR of ore						I_1	I_2	I_3	
increasing METAL CONTENT of ore minerals						D_1	D_2	D_3	
increasing ORE IMPURITIES						I_1	I_3	I_3	
increasing ORE TEXTURAL COMPLEXITY						I_2	I_3	I_3	
smaller GRAIN SIZE						I_3	I_3	I_1	
POLYMETALLIC						I_3	I_3	I_3	

I = Increased costs, 1 ... 3 = increased magnitude of effect.
D = Decreased costs, 1 ... 3 = increased magnitude of effect.
V = Variable costs (I or D), 1 ... 3 = increased magnitude of effect.

- 119 -

Naturally there is considerable overlap between the cash flow analysis and subjective analysis paths. For instance Mallinson (1989) incorporates a favourability factor into his cash flow analysis in order to try and quantify the subjectivity related to the intangible geological risk elements shown in Table 8.2. This facilitates decision making, however, in order to incorporate this information into the cash flow calculations, the exploration geologist must have a good working knowledge of geology and the influence that geology has on the mining operation. An alternative approach would be to raise or lower the hurdle rate according to the seriousness of the intangible risk elements. For instance a prospect with unfavourable geological conditions would require a higher hurdle rate.

The economic analysis should be conducted continuously throughout the exploration period. This ensures that delineation funds are channelled towards defining key factors that have the most significant effect on the economic potential of the project. It also ensures that funds do not continue to be spent on mineral prospects that show little potential of ever becoming a mine. However, in order to obtain meaningful results, it is important that the exploration geologist and mining engineer be thoroughly acquainted with all the aspects related to mine economics. This will ensure that economically beneficial and detrimental factors are identified and evaluated at an early stage.

8.2. CASH FLOW ANALYSIS

Mackenzie (1981) defines cash flow as the difference between benefits and costs for a specified time period (usually annually).

Thus: Cash Flow = Annual Benefits - Annual Costs

where Benefit Elements= Sales Revenue
Salvage Value
Return on working Capital
Tax Credits

Cost Elements = Capital Expenditure
Operating Cost
Taxation Payment
Royalty Payment.

If the annual benefits exceed the annual costs, the cash flow is positive; if the annual costs exceed the annual benefits, the cash flow is negative.

The potential profitability of the mineral investment is described by its time distribution of cash flows. It is usual practice to calculate a time distribution of cash flows over a period of about 10 years, or if the mine's life is less than 10 years to the end of the mine's life. Salvage value is normally excluded from the cash flow analysis, unless it is known that the mine has a particularly short life and that the equipment can be sold after mining operations cease. The basic input variables required to determine cash flows are listed in Table 8.3. Guidelines to the construction of cash flows are presented by Mallinson (1989).

Table 8.3. Basis input variables required for the cash flow analysis of a typical selective, open pit gold prospect.

1. Ore Reserves
    a. mineable tonnage
    b. mill head grade (average g/t gold, silver)
    c. scheduling (yearly change in grade or tonnage)
2. Production rate (tonnes per year)
    a. ore
    b. waste
3. Stripping ratio
4. Plant capacity.
5. Plant efficiency (per cent loss of gold from plant if any).
6. Gold price (in local currency units).
7. Exploration Expenditure
8. Capital Expenditure
    a. pre-production (mine, mine access, plant, infrastructure)
    b. working
9. Royalty and lease payments.
10. Operating Costs
    a. mine
    b. plant
    c. administration
    d. environmental control
    e. safety
    f. refining
    g. other (insurance, overheads)
11. Taxation rate/credits.
12. Exchange rate.
13. Inflation rate (optional, only included in current money units).

As mentioned earlier many of the input variables will not be known with certainty. Items such as the gold price are notoriously difficult to forecast over long periods and are unlikely to be predicted correctly. However, other items such as the ore reserves, capital and operating costs, gold recovery rates and stripping ratios will be estimated more accurately as the delineation programme advances towards the final feasibility stage. Other items such as royalty and taxation formulae will be known exactly. It is important that the evaluator concentrate on estimating the site specific input variables. Although the gold price has a major influence on the cash flows, the market forces influencing the price are beyond the control of the evaluator. Therefore there seems little reason to spend a lot of time trying to forecast the future gold price. A gold price forecast generated by the company's "in-house" market analysts or an outside "expert's" opinion could be used.

At the early stage of exploration when there is little site specific information available, it will be difficult for the geologist to make a realistic estimate of the capital and operating costs, and mill capacity. One way of estimating these factors is to make an educated guess based on other mines exploiting similar deposits. However, a danger to this approach is that the estimates will be unable to take into account the changing economies of scale, i.e. an increase in the production rate reduces the operating cost but increases the capital expenditure as a larger mill will be required to treat the additional ore. It is highly unlikely that another mine will be exploiting a similar deposit under the same circumstances. A solution to this problem, which takes into account the changing economies of scale, is to use empirically derived formula as exemplified by Mackenzie for Canadian mines and used by Mallinson (1989) in his cash flow computations.

Cash flows can be conducted in either <u>constant</u> or <u>current</u> money units. The difference between the two is the rate of inflation and it is important that the units be clearly stated. This is vital where projects are competing for funds. Radically different and potentially misleading results are produced if constant money units used with one project are compared with current money units used with another project, or worse still if the two money units are mixed in the economic analysis of one project. The influence of inflation and the allied problem of interest and escalation are addressed in more detail by Mackenzie (1979) and Smith (1987). Since the issues are important some of their conclusions are detailed below. To clarify the issues Fig. 8.2 compares the cumulative cash flow of a hypothetical project expressed in constant, current (fully escalated), and real (de-escalated current) money units.

In constant money units there is no inflation. The value of money is measured by what it can purchase today. This assumes that in the long-term cost inflation will be matched by escalation in the gold prices. Constant money units have the advantage of being more readily grasped as all prices are quoted in today's money. The cash flow analysis is also easier to perform. However, by failing to anticipate the rising costs during the pre-production period, the actual capital expenditure and debt repayments may be understated. Also taxes tend to be understated because the declining value of capital cost deductions, relative to inflating costs and revenues, is not recognized (Smith, 1987). Overall, constant money units may have the net effect of slightly overvaluing the property.

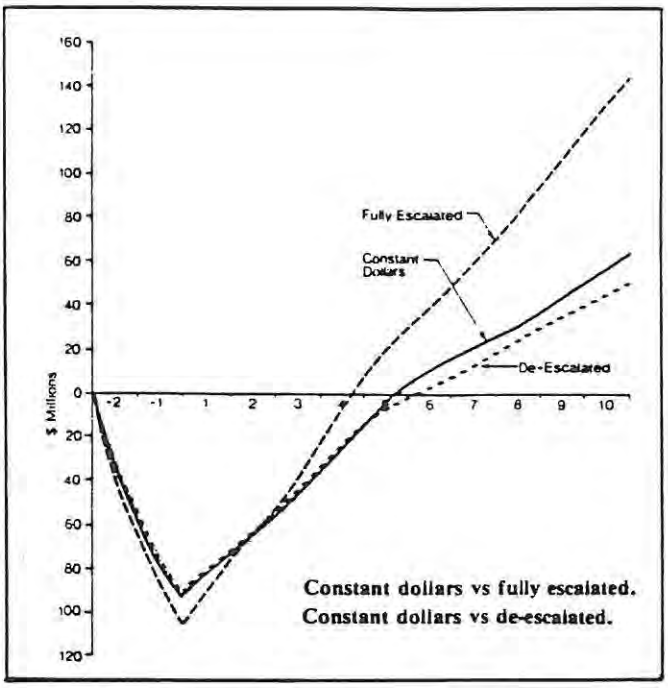


Figure 8.2. Cumulative cash flows for the same investment alternative expressed in constant, current (fully escalated) and real (de-escalated) money units (from Smith, 1987)

In current money units, inflation is applied throughout the mine's life and all values reflect the purchasing power of money during the year it is used. Current money units have the advantage of more accurately estimating the capital expenditure and tax deductions when the mine is being developed and in production. However, a serious drawback is that by forecasting the inflation rate over long periods, another uncertainty is introduced into the cash flow calculations. There are also other problems: 1) due to the compound effect, small cost errors can be over exaggerated in later years; 2) it is more difficult to comprehend the purchasing power of inflated money; 3) the cash flows are more complicated to perform; and 4) critical periods, such as a dip in the grade or a planned production increase can become masked in an inflated cash flow. Some of these problems can be removed by de-escalating the current cash flows. However, this raises the additional uncertainty and complication of predicting by how much to de-escalate the inflated cash flows (Smith, 1987).

There appears to be no right way with constant versus current money units. Constant money units offer fewer complications, however, it would be wrong to ignore inflation. Mackenzie (1979) and Smith (op. cit.) offer several recommendations, five of which are given below.

1. If investment alternatives are being compared, similar money units and assumptions should be used.

2. The assumptions made must be clearly stated. The evaluator must understand the impact of the assumptions and make certain that management also understands the assumptions and their impact.

3. All values should be expressed in <u>local</u> currency terms so as to avoid the added inflationary problems of trying to predict exchange rates among currencies in which investments are made and product sold.

4. During the geological and preliminary mine feasibility stages, the cash flows should be conducted in constant money units. Introducing inflation at these stages can confuse and cloud other more fundamentally important issues.

5. During the final feasibility stage, inflation should be included in order to determine the actual cash flows that will be experienced when the mine is being developed and in production. This will facilitate funding, debt and tax scheduling.

8.3. DEFINITION OF FINANCIAL YARDSTICKS

The common financial yardsticks used to measure the profitability of the mineral prospect are the payback period, Net Present Value (NPV) and Internal Rate of Return (IRR). The discounted cash flow criteria (NPV, IRR) are the most useful as they take into account the time value of money. However, it is normal practice to quote all the yardsticks as investment alternatives should be compared using multiple criteria. As most financially orientated publications show how the criteria are derived only the relevant details of each are discussed here.

The payback period is the time required to recover the initial capital expenditure from operating profits on an after tax basis, measured from the time investment is complete, i.e. the start of production. The method is simple to use and serves to screen projects quickly but the results are basically inadequate as selection criteria. The main weaknesses are, after Mackenzie (1981):

1) it fails to take into account the differences in timing of cash flows within the payback period;
2) it fails to give any consideration of cash flows beyond the payback period; and
3) it neglects the time value of money.

The payback period can play a useful role in evaluating financial and business risks where time is a critical factor; for example in a politically unstable country, where recouping the investment within a short period is the main consideration.

The Net Present Value provides a quick way of screening investment alternatives. NPV is defined as the cumulative value of all cash flows, both positive and negative, discounted at some specified rate to the present. NPV is calculated for each time distribution of cash flows and the value represents the anticipated return on the investment over and above the specified discount rate. Usually a discount (interest) rate is selected which is equivalent to the company's minimum rate of return or hurdle rate. A positive NPV indicates that the returns on the investment will exceed the hurdle rate. When NPV equals zero the investment is said to be at breakeven. A negative NPV indicates that the investment will not meet the company's minimum rate of return. In this way investment alternatives below breakeven are rejected. When several investments are to be selected from a pool of projects, then a derivative of the NPV, the Present Value Ratio (PVR), is required for project selection. Details of the PVR are provided by Mackenzie (1981, p. 202).

The Internal Rate of Return is that rate of discount that causes the negative and positive cash flows to be exactly equal. In other words IRR is the compound interest rate that makes the NPV exactly equal to zero. The discount rate for the IRR is not set and is normally found by trial and error. In economic terms, the IRR represents the compound interest rate of return that will be realized by investing in the project. The minimum acceptable condition for selection is a rate of return greater or equal to the hurdle rate.

The NPV and IRR are usually quoted together as both complement each other. The NPV provides a measure of the monetary gain at the specified discount rate, whilst the IRR gives an indication of the rate of return that will be achieved from the investment. There is one weakness though. Neither NPV or IRR provide an indication of the mine's size. This can be obtained by the Total Potential Revenue (TPR), as introduced by Mallinson (1989). The TPR is obtained by the calculation:

> expected tonnage processed * expected recovered metal(s) grade * expected metal(s) price. (*: multiply).

The TPR is useful for differentiating between large and small mineral prospects which may be competing for funds. Even though a small prospect may have a higher NPV and IRR than a large prospect, the larger prospect will be more attractive to the investor. This is because larger deposits tend to have longer lives and are therefore more likely to generate greater long-term profits when market conditions become favourable, i.e. if the gold price were to suddenly rise. Also there is a lower limit where the deposit just becomes too small to warrant its development.

8.4. RISK ANALYSIS

It is very important to assess what effect the risks will have on the profitability of the mining venture. As mentioned earlier the risks can be broadly divided into tangible risk elements that are quantified for inclusion into the cash flow analysis and intangible risk elements that require subjective judgement. In this section we refer to the former.

There are essentially three different approaches to handling risk using the cash flow computations. They are conservatism, sensitivity analysis and risk analysis. The risk analysis is by far the most powerful method. It not only quantifies the risks but also provides a measure of the quality of the evaluation and the probability of achieving the specified return on investment. Before expanding on the risk analysis it is first necessary to discuss the merits and demerits of the other methods.

The conservatism approach is to apply conservative point estimates to the input risk elements. The method suffers from being extremely subjective and is not recommended as it could lead to rejection of a potentially economic investment. There is no knowing how conservative one should be and it is almost impossible for another evaluator to repeat the exercise. Furthermore, conservatism is unable to assess the overall risks, nor indicate the upside and down side profit potential of the investment. Also, due to its subjectivity it cannot be used to adequately compare investment alternatives. The reason for its continued use is that some people like to give conservative estimates so that the project has a better chance of succeeding, or rather they do not want to be blamed if the project fails. The correct way is to give the most likely estimates (with variances) and know the probability of possible failure. The most opportune moment to apply conservatism is while making an investment decision. Here, the decision maker can apply a higher hurdle rate should he wish to be conservative.

The sensitivity analysis measures how the discounted cash flow criteria will change as the input variables are changed independently of each other. The analysis is easily performed by systematically altering the value of the input variable being examined whilst fixing the remaining input variables. If all the input variables are then assessed in turn, their effect on the profitability of the investment can be compared. The sensitivity analysis does not quantify the overall risks. However, it does identify the input variables that have the most significant impact on the profitability of the project. Fig 8.3 illustrates a typical sensitivity analysis. In this example increases in the gold recovery, gold price and ore grade have greater benefit on the operation's profitability than the tonnage throughput and milling rates. In the opposite sense, a decline in profitability is most strongly influenced by increases in production costs, then capital cost and lastly the stripping ratio.

The main application of the sensitivity analysis is during the delineation programme. By identifying the most significant risks, the sensitivity analysis can be used to direct funds into resolving key issues that are crucial to making a go or no-go decision. Provided the variables are alterable then the evaluator can focus on these items and avoid wasting time and money on items that have little impact on the project's profitability. For example if the economic potential of the mineral prospect is most sensitive to securing a low-cost supply of water, there seems little reason in spending additional funds on infill drilling to more closely define the orebody. In this situation the funds would be better allocated to locating an alternative water supply. If all attempts failed to locate an alternative low-cost water supply, then the factor becomes unalterable and funds can then be allocated to resolving the next most significant risk element. This iterative process of identifying and resolving key risk elements ensures that delineation funds are used judiciously, which itself is important to maintaining a high rate of return.

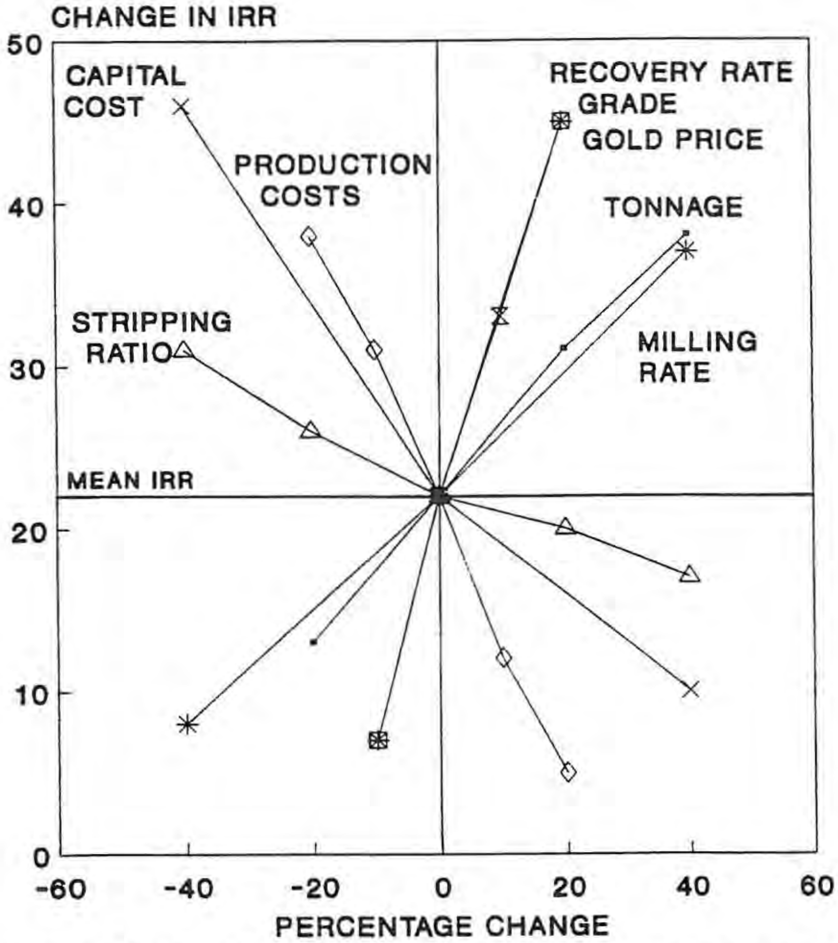


Figure 8.3. Sensitivity of IRR to changes in input variables in a typical open pit gold prospect (from Pelly, 1990).

The risk analysis is the only method that is able to effectively quantify the overall costs, risks and returns. This is achieved by assessing the effect that all the input variables have on the profitability of the project. Since there are so many risk elements to consider and because their interaction requires complex calculations, the only practical method available is to perform the cash flow calculations on a computer using a Monte Carlo simulation (Mackenzie, 1979; Barnes, 1980). A good example of how a robust, practical risk analysis programme can be applied is presented by Mallinson (1989). In his particular risk analysis, Mallinson (op. cit.) elected to construct the programme using Lotus 1-2-3 spreadsheet software. Once set up on the computer, the programme is quick to use and very versatile in making delineation and investment decisions.

Fig. 8.4 illustrates the typical Monte Carlo risk analysis simulation. In this technique the approach is to quantify the variability of the input variables, sample the distributions in a random manner, calculate the cash flows for each set of input data, repeat the procedure many times, and display the results as a probability distribution of expected IRR, NPV and TPR etc.

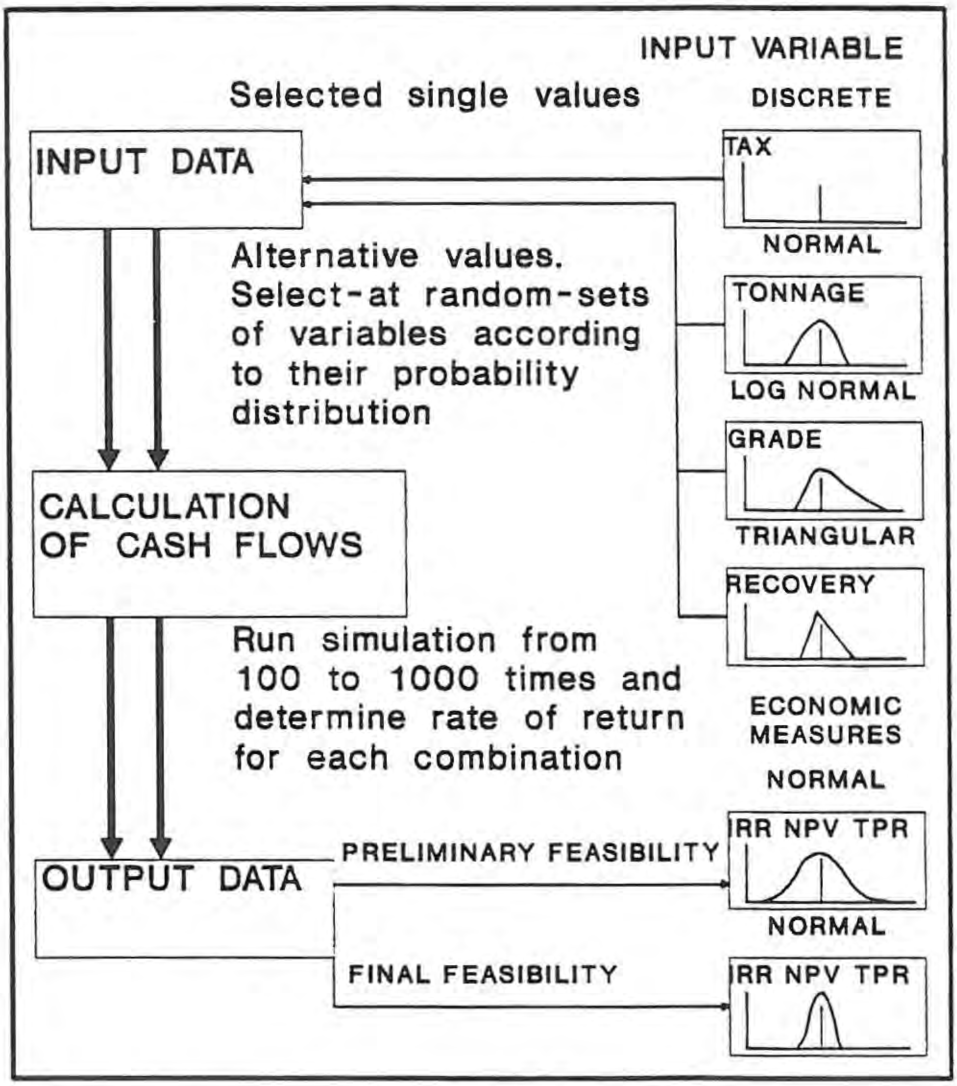


Figure 8.4. The risk analysis procedure (from Pelly, 1990).

The advantage of the Monte Carlo simulation is that for every iteration, a different set of input data are randomly selected. Since each input variable is sampled according to its estimated range of values, every set of cash flows will represent a possible realistic mining situation. The difference between the predicted distribution of cash flows and the actual cash flow will of course depend on how accurately the evaluator has been able to estimate the form (mean, range, standard deviation) of the input variables. Herein lies the strength of the risk analysis. It is easier and more correct to predict a distribution of possible values rather than a point value.

Since the input variables are described as a probability distribution, the rate of return is also described as a probability distribution. Typically a normal, bell-shaped distribution of IRR values are produced. One way of presenting such information is to construct a cumulative frequency curve of the IRR values (Fig. 8.5). This curve is ideal for quickly assessing the potential profitability of the project. For instance, in the example shown one can quickly read from the curve that there is only a 10 per cent chance of achieving an IRR greater than 22 per cent, or that there is likewise a 10 per cent chance that the IRR will be less than 7. The most likely IRR is 14.5 per cent.

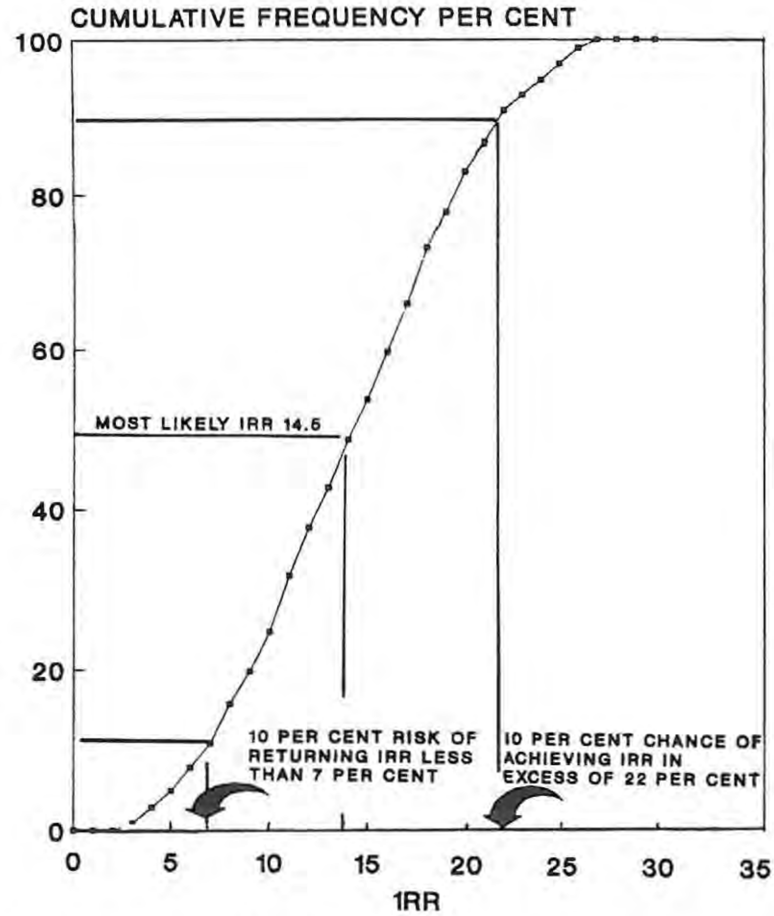


Figure 8.5. Illustrative example of cumulative frequency curve of IRR values generated from a risk analysis (modified after Mallinson, 1989).

The "steepness" of the curve is a measure of the risks and uncertainties. As the delineation programme advances from the geological to final feasibility stage and as the risk elements become better defined, so the curve will steepen. Towards the final feasibility stage the rate of change becomes sluggish and eventually the curve will reach a maximum limit where additional data collection has little influence on further steepening the curve. At this point the population spread represents the non-removable risk elements (Mallinson, 1989). It is important to note that during the delineation programme - as more time and money is spent defining the mineral prospect - the curve moves progressively towards the left. This has the adverse effect of reducing the IRR on the investment and unless the exploration expenditure is "written off", what was once a potentially economic project may slowly become uneconomic. Thus another application of the risk analysis is to make the evaluator aware of the moment in time when additional expenditure is unlikely to improve the project's profitability (Mallinson, op. cit.).

Another useful method of pictorially presenting the results from the risk analysis is to plot the TPR against IRR (Fig 8.6). In this plot investment alternatives of different sizes can be compared. As mentioned earlier larger deposits are preferable, therefore the line separating economic projects from uneconomic projects is curved as shown in the figure.

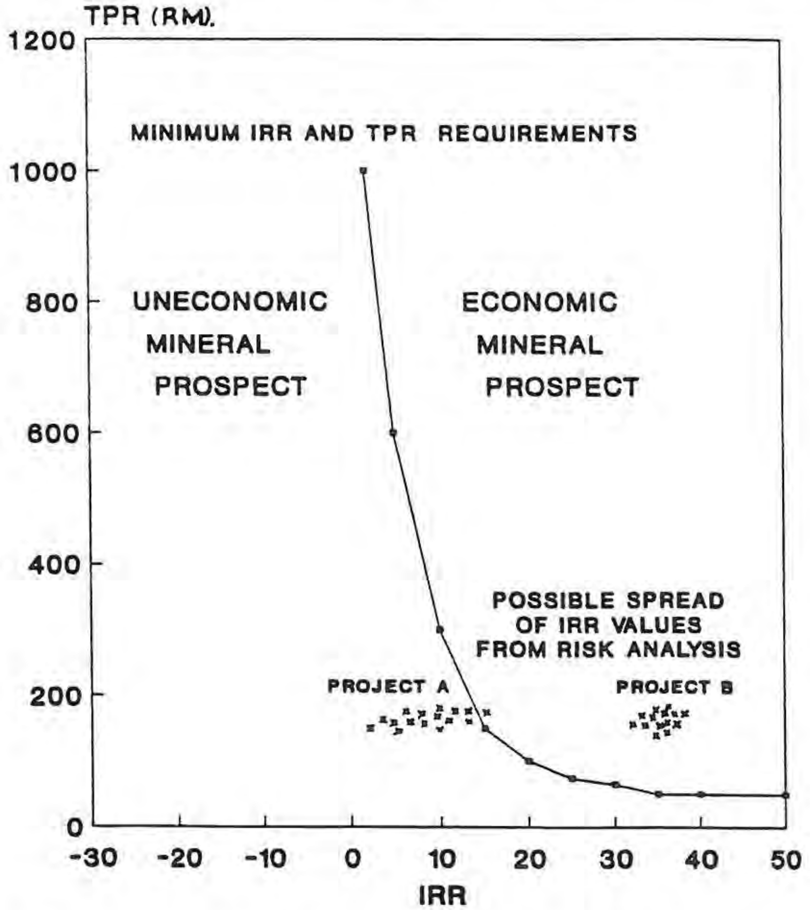


Figure 8.6. Plot of TPR versus IRR. The plot forms an ideal means of comparing investment alternatives.

Many other types of graphs and forms of presenting the data are available. For example there are utility graphs for comparing investment alternatives (Mackenzie, 1979) and scatter plots (Mallinson, 1989). To describe all these forms of presentation will serve little purpose here. The main point is that the graphs are presentable and convenient to use. Sundry scraps of paper are avoided and provided the input variables are realistic of the actual situation, then management can make sound investment decisions based on quantified criteria. This builds confidence into the no, no-go decisions. In addition there are also other reasons why risk analysis is preferable. These are listed below.

1) It provides a measure of the quality of the investment. The spread of expected discounted cash flow yardsticks show how successful the delineation programme has been in defining the tangible risk elements.

2) It forces the evaluator to quantify and define all the input variables. This avoids the proverbial "back of a cigarette packet" or mental type of calculation which is prone to forgetting important factors.

3) It is reproducible. Regardless of the evaluator, risk analysis offers a convenient standard within the company that everyone can relate to.

4) It allows for comparisons to be made between competing investments. Ranking of projects is therefore facilitated.

5) It is practical and efficient. Tedious and complex calculations are immediately computed once new information is incorporated in the cash flow calculations. The economic potential of the project can therefore be constantly monitored.

6) It can also be used as a sensitivity analysis. Critical items can therefore be identified and targeted for further delineation.

Overall, risk analysis is a very powerful and useful tool. The main complication is the actual construction of the programme. Serious errors, manipulations and potentially misleading results can arise through the mathematical formulae themselves. The evaluator needs to be aware of how the calculations are derived and how the input variables interrelate.

Another problem is the one caused by correlations. Correlation is the effect that one input variable may have on another. For example an obvious one is the gold and silver contents of the ore. An increase in gold grade is usually associated with an increase in the silver grade, in which case the two are said to be positively correlated. Other correlations are less obvious, but it is important in the risk analysis to incorporate their effect. This is achieved by using correlation coefficients in the calculation. However, practical problems can arise where the correlation is not obvious or if there is insufficient information to determine the correlation coefficient. This situation is aggravated by an increase in the number of input variables used in the cash flow calculation. A practical solution is therefore to limit the number of input variables which are correlated (Mackenzie, 1979 p. 243).

8.5. INVESTMENT DECISION CRITERIA

For mineral investment decisions, Mackenzie (1981, p. 463) and Gentry (1988) recommend three investment criteria, namely: 1) expected profitability criterion; 2) risk criterion; and 3) competitive cost criterion. They are briefly discussed in turn below.

Expected profitability criterion. Mineral properties with higher expected profitability are preferred. For example the average or 90 per cent upper limit NPV, IRR and TPR can be used to select properties with the highest profit potential. Properties with returns below the company's hurdle rate are rejected. Marginal properties can be postponed until the economic conditions become more favourable.

Risk criterion. Mineral properties with less risk are preferred. As seen in Fig 8.5, risk can be judged by the variance of the probability distribution of possible rates of return. The investor is concerned mainly with the lower limits of the distribution and requires assurance that some minimum level of profitability (usually measured as the cost of capital) can be obtained with a degree of confidence. For example the 90 per cent lower limit of NPV or IRR could be used to measure the risk criterion.

Competitive cost criterion. Mineral properties with operating costs in the lower quartile of those of all major gold producers are preferred. The competitive cost criterion is essentially a safeguard against over estimates made in the cash flow projections. For one reason or another (but almost certainly because the input variables were inaccurately evaluated to begin with) discounted cash flow projections for many mining investments have not been met. Realizing this, senior executives have adopted a more conservative approach by selecting projects with competitive operating costs. The assumption is that if cash flows are not met - perhaps as a result of low gold prices - then all gold producers will be adversely affected. Under these circumstances the first mines to close will be those with the highest operating costs or lowest profit margins. If the company's investment choice is a low-cost gold producer, then the mine has a greater chance of surviving the period of depressed gold prices and will be well placed to take advantage of any subsequent rise in the gold price. In South Africa the operating gold mines are commonly ranked according to their operating cost per kilogram of gold produced (Rands per kilogram gold). This cost factor is more useful than simply stating the cost per tonne as the grade of the ore milled is also taken into account. A similar costing could be used as the competitive cost criteria for selection of new mineral investments.

As can be seen these three investment criteria address the principle factors that need to be assessed in order to evaluate the mineral prospect, namely the costs, risks and returns associated with the investment alternative.

9. CONCLUSIONS

The evaluation of selective open pit gold deposits is an iterative process. From the moment significant mineralisation is discovered, the evaluator is repeatedly involved in collecting additional information on the prospect's geology, ore reserves, mining, metallurgical and environmental aspects. Throughout the delineation programme the economic potential of the mineral prospect must be continually assessed. This will ensure that the costs, risks and returns of the investment alternative remain acceptable to the company's growth strategies and profit expectations.

The best means of directing and co-ordinating the delineation programme is through the use of feasibility studies. These studies purposely force the evaluator to adopt a holistic approach to mining investments by taking into account all the scientific, technological and economic aspects affecting the investment alternative. Likewise feasibility studies address the investors' concerns by concisely and accurately evaluating the mineral prospect. Although the studies are comprehensive documents, their subject matter shifts progressively with time. During the geological feasibility stage ore reserve verification is paramount. The ore reserves are the prime asset of the property, yet, they are the most difficult item to evaluate. During the preliminary mine feasibility stage, mining, metallurgical and environmental aspects become increasingly more important. Towards the final feasibility stage attention shifts towards the competitive assessment, marketing and financial aspects of the mining venture.

Investment decisions are ultimately based on the economic potential of the mineral prospect. This requires that the costs, risks and returns of the investment alternative be quantified using discounted cash flows derived from a risk analysis. The risk analysis is an integral part of the feasibility study and is an extremely powerful tool. However, it should not be used as the sole investment decision maker. Subjective decisions based on experience and judgement of the intangible risk elements are also needed to make sound investment decisions. In today's uncertain gold investment climate, the investor needs to choose a selective open pit gold prospect which not only has high expected returns but is also cost competitive and relatively risk free.

Throughout this report it has been stressed that evaluation of the geological, ore reserve, mining, metallurgical and environmental aspects requires accurate and representative data. This is singularly the most important factor of mineral prospect evaluation, for no matter how good the economic potential of the property may appear, the results of the economic analysis will only be as good as the data on which it is founded. Mining companies would do well to focus on this basic aspect of mineral prospect evaluation, rather than place false hopes on complex manipulations of the data. In this regard the exploration geologist must play a vital role. Through his field mapping and borehole logging, the exploration geologist is in the best position to prejudge the main risk elements before mining commences. However, in order to make relevant observations such a person requires broad mining experience and a good understanding of mining economics.

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APPENDIX 1

BASIC FORMAT FOR FEASIBILITY STUDIES

1. BASIC FORMAT

    1. Define the resource - area of interest
       - ore reserve

    2. Describe the mining operation
       - mining
       - metallurgy
       - marketing
       - personnel
       - infrastructure
       - environment

    3. Financial/economic assessment and justification

2. SUGGESTED CONTENTS (modify as required)

    Summary

    1. Introduction

    2. Conclusions

    3. Recommendations

    4. Location and Infrastructure

    5. Holdings and Titles
       5.1 Mineral rights
       5.2 Water and surface rights

    6. Historical, Political and Sociological
       6.1 Past exploration and production
       6.2 Political system
       6.3 Local population

    7. Work done

8. Geology
    - 8.1 Regional and local geological setting
    - 8.2 Stratigraphy
    - 8.3 Structure
    - 8.4 Alteration and weathering
    - 8.5 Mineralisation

9. Mineral Resources and Reserves
    - 9.1 Selected mining area
    - 9.2 In situ resources
    - 9.3 Mineable reserves
    - 9.4 Summary

10. Mine Design
    - 10.1 Options
    - 10.2 Geotechnics
    - 10.3 Pit design
    - 10.4 Production schedule
    - 10.5 Mining methods
    - 10.6 Grade control

11. Recovery Plant and Process
    - 11.1 Mineralogical study
    - 11.2 Metallurgical testwork
    - 11.3 Plant design
    - 11.4 Tailings disposal

12. Personnel
    - 12.1 Policy
    - 12.2 Requirements
    - 12.3 Safety and security

13. Infrastructure Requirements
    - 13.1 Power
    - 13.2 Water
    - 13.3 Access
    - 13.4 Buildings and installations
    - 13.5 Vehicles and other

14. Environmental Assessment and Management
    14.1 Description of environment
    14.2 Environmental impact assessment
    14.3 Rehabilitation plan
    14.4 Management and funding

15. Competitive Assessment
    15.1 SWOT analysis

16. Market Analysis
    16.1 Gold price
    16.2 Rand/dollar exchange rate
    16.3 Inflation rate

17. Financial and Economic Analysis
    17.1 Capital costs
    17.2 Operating costs
    17.3 Cash flow analyses
    17.4 Investment criteria summary

18. References

APPENDICES

A Holdings and titles

B Exploration/delineation techniques

C Geology plan and stratigraphic section

D Structural plan and section(s)

E Mineralisation characteristics

F Mineral resources and reserves

G Geotechnical and hydrological information

H Pit design, layout and general

I Mineralogical studies, metallurgical testwork and plant design

J Personnel requirements and organizational structure

K Surface infrastructure layout

L Environmental assessment report

M Capital and Operating cost breakdown

3. CONTENTS DETAILS

Main Report		Appendices
Text	Figs. and Tables	
SUMMARY		
- Location; lease/mining area	Table of details	
- Work done		
- Deposit geology		
- In situ resources (inferred, indicated and measured; tonnes + grade)		
- Mineable reserves (proven and probable; tonnes + grade)		
- Mine + plant design capacities + schedules		
- Grade control		
- Personnel requirements		
- Infrastructure requirements		
- SWOT analysis		
- Market assumptions (gold price, exchange + inflation rates)		
- Financial analysis		
- capital cost		
- operating costs		
- risk and sensitivity analyses		
- main investment criteria (costs, risks and returns)		
- Conclusions		
- Recommendations		

Main Report		Appendices
Text	Figs. and Tables	
1. INTRODUCTION - Aims/objectives - Background information - Layout of report		
2. CONCLUSIONS - Summary of results and inferences - Use firm unambiguous statements		
3. RECOMMENDATIONS - Best future options (abandon, postpone, develop and mine, continue delineation)		
4. LOCATION AND INFRASTRUCTURE - Location, terrain, climate (temperature, rainfall, freak weather, seasonal variations,net evaporation vs. precipitation) - Power, water, access, towns	Plan of region showing area of interest and infrastructure.	

Main Report		Appendices
Text	Figs. and Tables	
5. HOLDINGS AND TITLES		
5.1 Mineral Rights - summary - Statement (outright, options, joint ventures, shares; areas)		A Table of details and Plan (Mineral, Water and Surface rights)
5.2 Water and Surface Rights - - Summary statement (owners, areas)		
6. HISTORICAL, POLITICAL AND SOCIOLOGICAL		
6.1 Past exploration and production - Summary details (boreholes, production, yield)		
6.2 Political system - Summary statement (past, present & future; violence)		
6.3 Local population - Summary details (skills, unemployment, availability, work attitude, culture, religion, holidays)		

Main Report		Appendices
Text	Figs. and Tables	
7. WORK DONE		
- Drilling (number + holes) - Sampling and analysis - Trial mining - Metallurgical investigation - Metallurgical testwork - Other (specify)		B Details of techniques used. (Drilling - percussion / diamond; sampling - size width; analysis - fire assay ICP, AA, name of laboratory; testwork - bottleroll, bench-scale, pilot-scale)
8. GEOLOGY		
8.1 Regional and local geological setting		
- Major lithologies/ contacts - Metamorphic history - Deformation history - Igneous history - Metallogenesis	Simplified geology plan	C Detailed geology plan with detailed stratigraphy and descriptions of major lithologies
8.2 Stratigraphy		
- Local stratigraphy adjacent to orebody, thickness, implications to mining	Simplified stratigraphic column	Refer to C.
8.3 Structure		
- Major structural features (faults, shear zones fissures, folds - size, frequency and orientations	Simplified structural plan and section	D Detailed structural plan + sections(s)

Main Report		Appendices
Text	Figs. and Tables	
- Minor structural features (joints, folds, faults etc.) - Implications to mining		
8.4 <u>Alteration and weathering</u>		
- Alteration styles (nature and area of influence) - Weathering (depth, degree, variability, causes - palaeo or recent) - Implications to mining/ ore mineralogy	Simplified contour plan and section showing alteration halos & depth of weathering	Incorporate with E below if necessary.
8.5 <u>Mineralisation</u>		
- Ore description (variations, types, controls) - Orebody geometry (thickness, attitude, size, and shape) - Value distribution (compared to ore type and variability within orebody) - Mineralogy (ore & gangue, oxides and sulphides, secondary and primary)	Simplified figures showing mineralisation features.	E Detailed plans and sections showing drillholes, orebody details - thickness attitude, displacement extensions, variations, alteration halos.

Main Report		Appendices
Text	Figs. and Tables	

9. MINERAL RESOURCES AND RESERVES (For classification see Appendix 3)

9.1 Selected Mining Area
- Basis for area selection (geological, geostatistical, mining rights)

9.2 In situ Resources
- Assessment criteria used (specific gravity, assay grade geological cutoff, support, drill hole intersections, reliability of assays, geological interpretation, internal waste etc)
- Allowance for faulting, dykes
- Methods used (traditional, classical, geostatistical)
- Measured, indicated, inferred (tonnes and grade)
- Ore types (refractory, free milling, sulphidic, oxidized)
- Distribution

9.3 Mineable Reserves
- Assessment criteria used (selective mining unit, dilution/ ore loss factors, metallurgy)
- Methods used (traditional, conditional simulation, geostatistical)
- Proven and probable reserves (tonnes and grade at cutoff grade); and
- Ore types: refractory, free milling, sulphidic, oxidized)
- Distribution

Main Report		Appendices
Text	Figs. and Tables	
9.4 Summary		
- Average grade 90% upper and lower confidence all categories, SMU size	Tabulate relevant details	F Resources and reserve
- Average tonnes (90% upper and lower confidence), all categories		- resource/reserve assessment criteria
- Ore types		- borehole locality plan
- Distribution		- borehole value & width tabulation
		- point value histograms
		- log prob. frequency plot + additive constant
		- Sichel's mean + 90% confidence
		- experimental variograms
		- kriged mean + confidence
		- tonnage-grade curves for various cutoff grades & block sizes
10. MINE DESIGN		
10.1 Options		
- Options (open pit, underground)		
- Proposed alternative (scale financing, risk, flexibility)		
10.2 Geotechnics		
- Geotechnical summary (rock stability, pit slope angles)	Summary tables	G Details of geotechnical and hydrological characteristics)
- Hydrological summary (water fissures)		Data analysis, classification used, results, hazard plans & sections

Main Report		Appendices
Text	Figs. and Tables	
10.3 Pit design - Parameters used (bench height and width, slope angles, depth, area, stripping ratio - Pit layout (bench positions, haulage ways) - Waste dumps - Drainage requirements	Simplified plan showing pit outline and orebody.	H Detailed plans and sections showing pit design.
10.4 Production schedule - Strategy, rates, tonnage generated, ore exposure rate, ore and waste scheduling	Tables on rates and scheduling for life of mine with predicted mean ore grades.	

10.5 Mining method
- Options, preferred alternative
- Mining method
  - soft or hard (ripping/blasting)
  - contractor or mining company
  - equipment (excavators, haulage, auxiliary)
  - maintenance		

10.6 Grade control - Geological and/or assay ore/waste definition - Sampling methods (bulldozer ripping, ditchwitching, RC percussion, blasthole) - Schedule/responsibilities - Assay analyses and data manipulation	Simplified sketch showing sampling positions relative to orebody and procedure.	

Main Report		Appendices
Text	Figs. and Tables	
11. RECOVERY PLANT AND PROCESS		
11.1 Mineralogical study - Summary of mineralogy - gold-bearing minerals, grain size, host minerals, textural association - Ore type (refractory/free milling; sulphide/oxide; causes of refractoriness)		I Details of mineralogical study and metallurgical testwork and plant design (techniques, laboratories used, results, engineering drawings, planning, costs)
11.2 Metallurgical testwork - Method (bottleroll, bench-scale pilot-scale) - Gold recovery vs time, pH, reagent consumption, descalants, run of mine/crush size, agglomeration, materials handling - Comment on optimum gold recovery/extraction		
11.3 Plant design - Options; preferred alternative process (heap leach, CIP, CIL, RIP, roasting, BIOX, pressure oxidation) - Design (flowsheet, description capacities, scheduling) - Construction (pad, liner, materials, equipment) - Pollution control, neutralization of toxic waste and gas emissions	Gold treatment flowsheet	Refer to I

Main Report: Text	Main Report: Figs. and Tables	Appendices
11.4 Tailings disposal		
- Options, preferred alternative		Refer to I
- Design (construction, capacity)		
- Construction materials		
- Cyanide and acid neutralization		
12. PERSONNEL		
12.1 Policy		
- General policy, selection criteria remuneration, pension, condition of employment, training, housing.		J - Organizational structure diagram - productivity calculations.
12.2 Requirements		
- Organizational structure - Departmental manning levels, phasing. - Productivity requirements.	Table on dept. manning levels.	
12.3 Safety and Security		
- Safety - policy, systems. - Security - policy systems.		
13. INFRASTRUCTURE REQUIREMENTS		
13.1 Power		
- Requirements, installation phasing.	Table detailing requirements	K Surface layout of: plans showing road, rail, buildings, parking lots, tailings dams, plant.

Main Report		Appendices
Text	Figs. and Tables	
13.2 Water		
- Requirements, source (dams, boreholes) pumping facilities, reticulation, clarification, service or fissure water.	Table of requirements	K water reticulation flow sheet
13.3 Access		
- Road/rail requirements parking, design.		
13.4 Buildings and installations		
- On mine - office, stores, mill, plant, security, change house, workshops, magazine, canteen, training, accommodation, sewage, heating plant etc.		
- Off mine - accommodation, recreation.		
13.5 Vehicles and other		
- Requirements/details.		
14. ENVIRONMENTAL ASSESSMENT AND MANAGEMENT (Also see Appendix 4, this report)		
14.1 Description of environment		L Environmental assessment report
- Physical factors		
- Biological factors		
- Human factors		
14.2 Environmental impact assessment		
- Positive and negative aspects, conclusions		

Main Report		Appendices
Text	Figs. and Tables	
14.3 Rehabilitation Plan		
- Policy, plans, controls		Refer to L
14.4 Management and Funding		
- Financing, responsibilities, scheduling		Refer to L
15. COMPETITIVE ASSESSMENT		
15.1 SWOT Analysis		
- Strengths	Tabulate details	
- Weaknesses		
- Opportunities		
- Threats		
16. MARKET ANALYSIS		
16.1 Gold Price		
- Past, present and future forecasts	Tables and graphs	
16.2 Rand/Dollar Exchange Rates		
- Past, present and future forecasts	Tables and graphs	
16.3 Inflation Rates		
- Past, present and future forecasts	Tables and graphs	

Main Report		Appendices
Text	Figs. and Tables	

17. FINANCIAL AND ECONOMIC ANALYSIS

17.1 Capital costs

- Base date, cost centres, capital and working, scheduling	Tables of cost centres, expenditure scheduling	M Detailed capital and operating cost breakdown into exploration, mining, metallurgy, infrastructure, administration, miscellaneous.

17.2 Operating costs

- Base date, cost centres, total expenditure and per unit, fixed and variable costs	Table of cost centres (total and unit)	Refer to M

17.3 Cash Flow Analysis

- Input variables; cash flows, constant or current money units; financial yardsticks (PP, NPV, IRR, TPR); sensitivity and risk analyses; life of mine	Table giving key input variables and graphs for sensitivity and risk analyses	

17.4 Investment Criteria Summary

- Profitability, risk and cost competitiveness	Graphs and tables	

APPENDIX 2

THE ROLE OF GEOLOGY IN MINERAL PROSPECT EVALUATION

Asterisk denotes essential items.

Design of exploration programme (drilling, sampling, assaying, budgeting).

- General genetic model of the gold deposit.
- Expected location and geometry (size, shape, and orientation) of gold deposit. *
- Expected nature of mineralization, alteration styles and weathering.
- Ground conditions, logistics, exploration techniques and expenditure.

Ore reserve estimation.

- Oxidation, and leaching and dispersion of gold.
- The different ore types.*
- The size, shape, limits and attitude of the orebody. *
- The structure of the orebody and its surrounds. *
- Continuity and controls of mineralisation. *
- Resource/Reserve classes and definitions (see appendix 3 this report). *
- Sampling: representativeness, technique, type, size and splitting. *
- Assaying: method, sample preparation, laboratory, checks, precision. *
- Verification of accuracy and reliability of sampling and assaying. *
- Value distributions, means, grades, variance, coefficient of variation, thickness, specific gravities.
- Sample support, compositing and declustering. *
- Geological modelling of the experimental variogram(s). *
- Resource calculation: traditional, classical and/or geostatistical.
- Reliability/confidence of resource estimate. *
- Mineable reserve calculation: dilution, recovery, cutoff and economic parameters used. *
- Grade-tonnage curves and block size with 90% confidence limits.

Mine design.

- Depth and character of overburden and weathering.
- Drillability and blasting characteristics of the rocks.
- Geotechnical and hydrological data for pit slope angles, hazard plans. *
- Hydrological data for groundwater conditions and locality of water supply.
- Size, shape and attitude of the orebody. *

- Distribution of rock types.
- Condemnation drilling of sites for buildings, roads, leach pads, waste dumps and tailings dams.
- Location of construction materials - clay, sand and gravel, for leach pad linings, access roads and foundations.
- Mineable reserves. *
- Grade control design: alteration, rock type, mineralization, sampling. *

Ore treatment.
- The different ore types. *
- Host rock.
- Mineralogy of ore and gangue: gold-bearing minerals, gold grains size, host minerals and textural association. *
- Causes of refractoriness: physical or chemical lock-up, insoluble gold alloys, cyanicides, carbon rich material, coatings. *
- Rock hardness, abrasiveness, bond work index and cohesion.
- Potential co-products.
- Location of appropriate sites for representative metallurgical samples.*

Environmental Baseline studies.
- Soils: types, thickness, location, susceptibility to erosion and slope.
- Surface and groundwaters: quantity and quality of water.
- Mineral resource and geology (mineral and chemical makeup of ore/waste for possible pollutants; acids, mercury, arsenic etc.).
- Geological and surface features map.

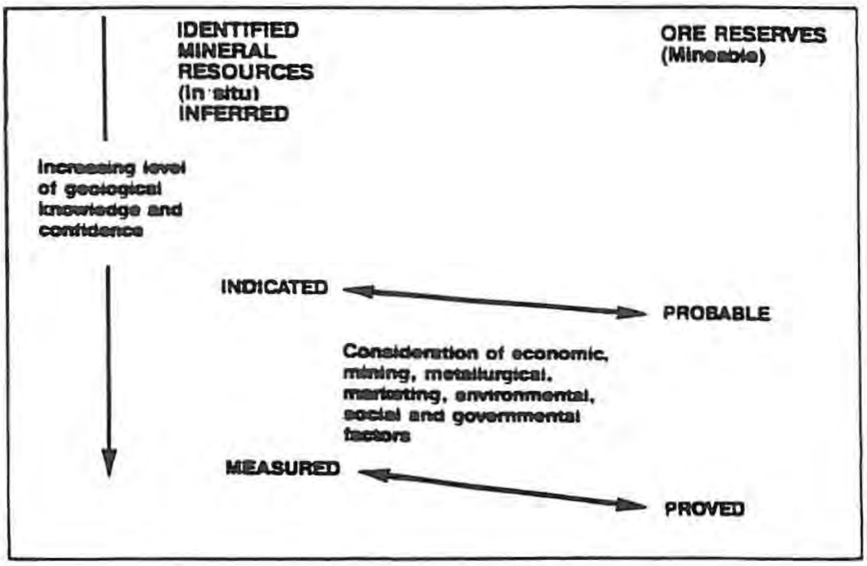
Optimizing scale of operation (mainly for pit and plant optimization).
- Ore deposit model (capabilities of orebody to sustain designed production rates). *
- Local reserve estimation: ore type, grade, block variance and distribution. *
- Potential reserves for future exploitation.

APPENDIX 3

CODE FOR REPORTING IDENTIFIED MINERAL RESOURCES AND ORE RESERVES

(Extracts taken directly from the report of the joint committee of the Australasian Institute of Mining and Metallurgy and the Australian mining industry council, June 1988.)

1. The terminology used for classifying mineral resources and ore reserves is outlined below:



2. The definitions of mineral resources and ore reserves are as follows:

RESOURCE is defined as an identified in situ mineral occurrence quantified on the basis of geological data and a geological cut off grade only. A resource may be reported as:

- an Inferred Resource (least reliable)
- an Indicated Resource
- a Measured Resource (most reliable)

Note: the term in situ reserves is no longer valid.

In defining a resource, the competent person will only take in consideration geoscientific data. It must be appreciated, however, that in reporting on a resource, it is implied that there are reasonable prospects of economic exploitation.

Inferred Resource is an estimate, inferred from geoscientific evidence, drill holes, underground openings, or other sampling procedures and before testing and sampling information is sufficient to allow a more reliable and systematic estimation.

Indicated Resource means a Resource sampled by drill holes, underground openings, or other sampling procedures at locations too widely spaced to ensure continuity but close enough to give reasonable indication of continuity and where geoscientific data is known with a reasonable level of reliability.

Measured Resource means a Resource intersected and tested by drill holes, underground openings and other sampling and procedures at locations which are spaced closely enough to confirm continuity and where geoscientific data is reliably known.

ORE RESERVE means that part of the Measured or Indicated Resource, which could be mined together with dilution and from which valuable or useful minerals could be recovered economically under conditions realistically assumed at the time of reporting. Ore reserves should be reported as:

- Probable Ore Reserves (least reliable)
- Proven Ore Reserves (most reliable)

Note: Possible Ore Reserves, i.e. that part of the Inferred Resource is no longer considered a justified category.

Probable Ore Reserves means ore reserves stated in terms of mineable tonnes or volumes and grades where the conditions are such that ore will probably be confirmed but where the in situ identified Resource has been categorized as indicated.

Proven Ore Reserves means Ore Reserves stated in terms of mineable tonnes or volumes and grades in which the in situ resource has been categorized as a Measured Resource.

3. In Report Writing it is recommended that the following be adapted:

(i) Company reports should standardize mineral resource and ore reserve terminology.

(ii) Mineral resource and ore reserve estimates prepared for public release should be undertaken only by appropriately qualified persons.

(iii) Any report should state the pertinent data on which the mineral resource and ore reserves are based, e.g. drill holes. Full information on the nature of the sampling, sampling intervals, assay data and positions must also be given. See Table below:

Resource assessment criteria	Explanation
Data density	Whether sample density is sufficient to ensure continuity as well as provide an adequate data base for the estimating proceeding used.
Accuracy of location of sampling points	This variable refers to how well the location of a sample position is known and its effect on the resource estimate.
Drilling technique	Whether core, rotary, percussive or percussion and if non-core, whether open hole or reverse circulation.
Sampling technique	If core, whether cut or chisel broken and whether quarter, half or all core taken. If non-core, whether riffled, section cut, tube sampled, or whatever, and whether sampled dry or wet. If wet, what precautions taken to maximise recovery and minimise fines loss.
Proportion of core recovery in mineralised zone	
Tonnage factor (SG)	Whether assumed or determined and, if determined, by what method and how frequently. If assumed, are assumptions valid and the basis for those assumptions.
Quality of assay data	Whether reproducible and whether representative. Substantial quality control and umpire assaying is necessary to identify any deficiencies in assay quality.
Quality of data description	Whether core logged in detail; whether all significant lithologic, structural, mineralogic, alteration or other geological or geotechnical characteristics and properties recorded competently. If underground chip samples, whether channel cut or chipped linearly or whether randomly taken from a face. If, linear, whether horizontal or vertical.
Geological interpretation	Whether based on sufficient data or postulated assumptions, whether constrained by one model or whether consideration given to alternative possible interpretations.
Estimation techniques	A clear description of estimation techniques and key assumptions.
Cut-off grades	What cut-off grades have been assumed.

(iv) Mineral Resources and Ore Reserves and their respective categories must be clearly distinguished in the report. The words "ore and "reserve" should not be used in stating Resource estimates as these terms imply technical feasibility and economic viability and are only appropriate when technical and economic factors have been considered.

(v) All tonnage or volume and grade figures for both Resources and Ore Reserve estimates should be rounded off to the second or third significant figures depending on the reliability of the estimate.

APPENDIX 4

ENVIRONMENTAL ASSESSMENT REPORTING FOR OPEN PIT GOLD DEPOSITS

(Adapted from Thatcher and Strohsacker, 1987; Wells, 1986; Alberts 1990)

BASIC FORMAT

1. Define the resource and mining plan
2. Describe the environment
3. Describe the effect of mining on the environment
4. Describe mitigation measures and management

SUGGESTED CONTENTS (modify as required)

Summary

1. MINING PROPOSAL

1.1 Background information

1.2 Detailed information of mine planning and description of operation

2. DESCRIPTION OF ENVIRONMENT

2.1 Physical factors

2.2 Biological factors

2.3 Human factors

3. NEGATIVE IMPACT OF MINING ON THE ENVIRONMENT

4. POSITIVE IMPACT OF MINING ON THE ENVIRONMENT

5. REHABILITATION PLAN

6. ENVIRONMENTAL MANAGEMENT

6.1 Funding arrangements

6.2 Responsibilities and scheduling

Note: Item 1 forms the mining proposal issued to the regulatory agency in order to initiate discussions; item 2 forms the environmental baseline studies resulting from the discussions; and items 3 to 6 complete the environmental assessment study.

1. MINING PROPOSAL

1.1 BACKGROUND INFORMATION

a Owner
b. Manager
c. Owner of mineral rights
d. Owner of surface rights
e. Mine product and by-products
f. Production rate
g. Planned life

1.2 DETAILED INFORMATION OF MINE PLANNING AND DESCRIPTION OF OPERATION

a. Location and access from major and nearby towns
b. Total mining area
c. Topography of surroundings, drainage network, other surface features
d. Location of orebody
e. Layout of pit and haulage network
f. Position of box cuts and final voids
g. Location of infrastructure, metallurgical plant, heap leach pads, waste tips, tailings dams, storm water reticulation, sewerage, offices, housing and recreational facilities.

2. DESCRIPTION OF THE ENVIRONMENT

2.1 PHYSICAL FACTORS

2.1.1 LOCATION

2.1.2 GEOMORPHIC/PHYSIOGRAPHIC

a. Surface contours
b. Geological hazards
c. Unique land forms

2.1.3 CLIMATE

a. Rainfall: average rainfall, rainfall distribution through the year, seasonal variations.
b. Temperature: mean annual temperature, maximum and minimum, frost expectancy, daily variations.
c. Evapotranspiration: mean annual rates, and summer and winter rates.
d. Winds: predominant wind directions and speeds.
e. Extreme climatic events: frequency of droughts, floods, hail, very high winds.

2.1.4 SOILS

a. Soil types and depths of A and B horizons
b. Productivity
c. Hazards
   (1) Erosion characteristics
   (2) Mass failure
   (3) Soils unsuitable for rehabilitation (i.e. laterite, gravel, sodic-saline soil)

2.1.5 MINERALS AND ENERGY RESOURCES: Other than mining proposal

2.1.6 VISUAL RESOURCES

2.1.7 CULTURAL RESOURCES

a. Archaeological
b. Historical
c. Architectural

2.1.8 WILDERNESS RESOURCES

2.1.9 WILD AND SCENIC RIVERS

2.1.10. WATER RESOURCES

a. Surface water: location, drainage density, dams pans, wetlands.
b. Groundwater: location, water table level, aquifer size.
c. Quality and quantity of surface water and groundwater.
d. Uses and requirements of water resources.

2.1.11. AIR QUALITY: Including quantity, location and distribution of dust and toxic emissions.

2.1.12. NOISE: Including blasting and other mine related causes.

2.1.13. TRAFFIC

2.1.14. FIRE

a. Potential wildlife hazard
b. Role of fire in the ecosystem

2.1.15. LAND USE

a. Present use: farming, forestry, grazing.
b. Capability as based on natural criteria and not actual present use.

2.2 BIOLOGICAL FACTORS

2.2.1 FLORA

a. Forest, including diversity of tree species
b. Open grassland, including conditions and trends
c. Other major vegetation types
d. Threatened and endangered plants
e. Ecosystem
f. Diversity of plant communities
g. Noxious plants

2.2.2 FAUNA

a. Habitat/migrations
b. Populations
c. Threatened and endangered species
d. Diversity of animal communities
e. Ecosystem
f. Animal damage control

2.2.3 RECREATIONAL RESOURCES

2.2.4 INSECTS AND DISEASES

2.3 HUMAN FACTORS

2.3.1 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

2.3.2 SPECIAL CONCERNS

a. Minority: civil rights

b. Historical, archaeological and cultural

2.3.3 WAYS OF LIFE, AS DEFINED BY:

a. Subculture 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

2.3.4 LAND TENURE AND LAND USE

2.3.5 LEGAL CONSIDERATIONS

3. NEGATIVE IMPACT OF MINING ON THE ENVIRONMENT

a. Tailings erosion

b. Emissions

c. Dust and noise pollution

d. Changes to ecosystem

e. Visual impact

f. Water pollution

g. Disruption of surface and groundwaters

h. Disruption to current and future land use during life of mine

4. POSITIVE IMPACT OF MINING ON THE ENVIRONMENT

a. Roads

b. Trails

c. Water collection, storage and distribution

d. Communications system

e. Solid waste collection

f. Sanitary waste collection

g. Hospitals, schools, recreation

h. Fauna and flora sanctuary

i. Wealth creation

j. Job creation

5. REHABILITATION PLAN DURING AND AFTER LIFE OF MINE

a. Prevention, reduction, control neutralization of negative influences

b. Topsoil stripping, storage and replacement

c. Revegetation of waste tips, tailings dams and heap pads

d. Pollution control of air, land and waters

e. Pit reclamation

f. After mine land capabilities

g. Final contour plans of rehabilitated area

6. ENVIRONMENTAL MANAGEMENT

6.1 Funding arrangements

6.2 Responsibilities and scheduling