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GEOLOGICAL AND ECONOMIC FACTORS AFFECTING
ORE RESERVE ESTIMATION
AND GRADE CONTROL IN PORPHYRY TYPE DEPOSITS

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INTRODUCTION

The mining of porphyry type deposits accounts for about 50% of the world's present copper (Figs.1,2) and molybdenum production and resources. Mining organizations therefore invest substantial amounts of time, money and skills in the location and delineation of these types of deposit. The optimization of this investment effort is based on complex inter-relationships between geological, economic and political factors.

The object of this dissertation is to review the geological and some of the economic aspects involved in the exploration and evaluation of porphyry deposits. These may hopefully provide some practical guidelines for decision making during the exploration and evaluation of such deposits.

For the purpose of this dissertation, the exploration-evaluation of porphyry deposits,has been divided into three main stages:-

Stage 1 : Geological mapping, interpretation of exploration drilling results and other geological factors which may help in understanding the shape and nature of the deposit. A knowledge of existing geological models for porphyry deposits will be essential in understanding the geological factors affecting tonnage and grade of these deposits (see Part 1).

Stage 2 : Determination of grade-tonnage relationships. This is important in order to establish the different tonnage-grade alternatives for the deposit. Based on this, reserve estimations are calculated for different possible scales of mining. Drilling and sampling techniques, as well as statistical and preliminary economic evaluation methods are applied during this stage (see Part 2).

Stage 3 : Mine development and feasibility studies involve factors that influence type and scale of mining, and factors affecting mineral processing and extraction in relation to tonnage-grade alternatives. These factors are reviewed in Part 3.

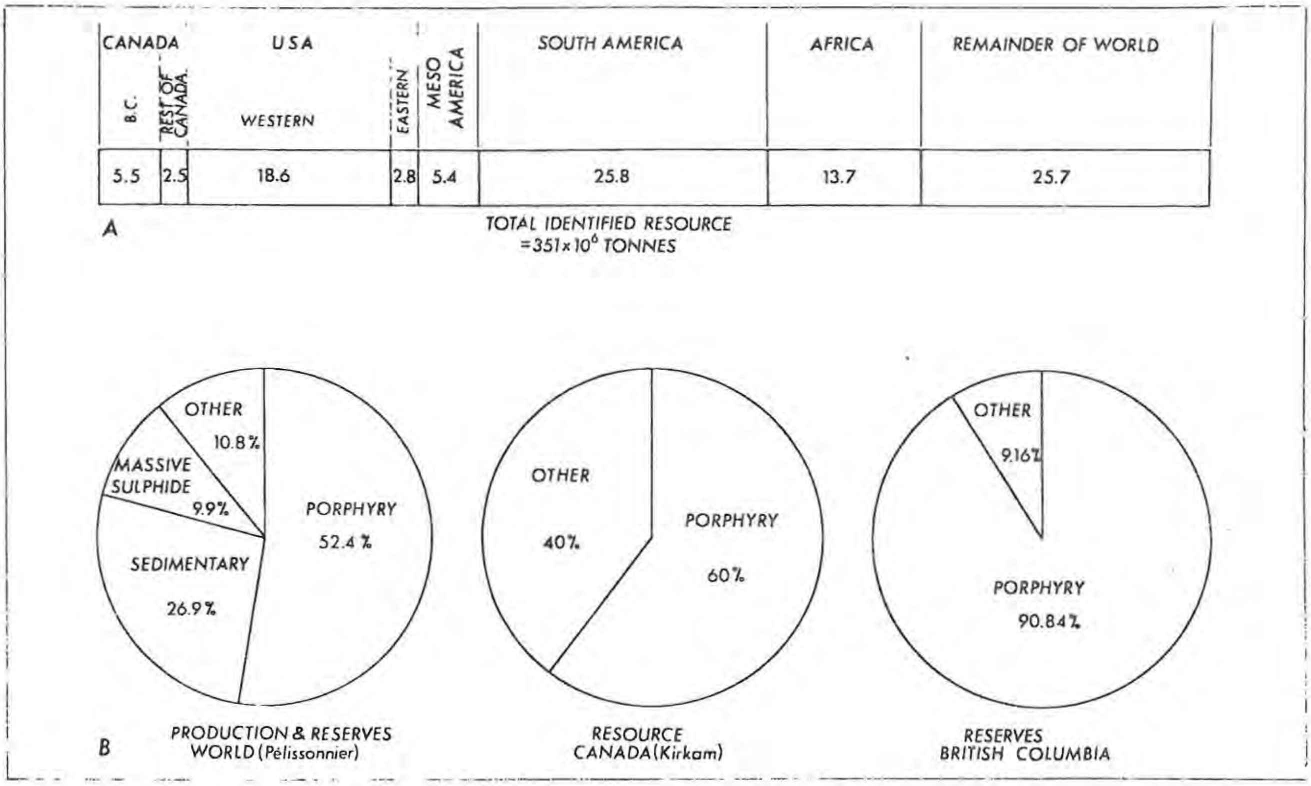

FIGURE 1— Percentage of identified copper resources (a) by region and (b) by type of deposit.

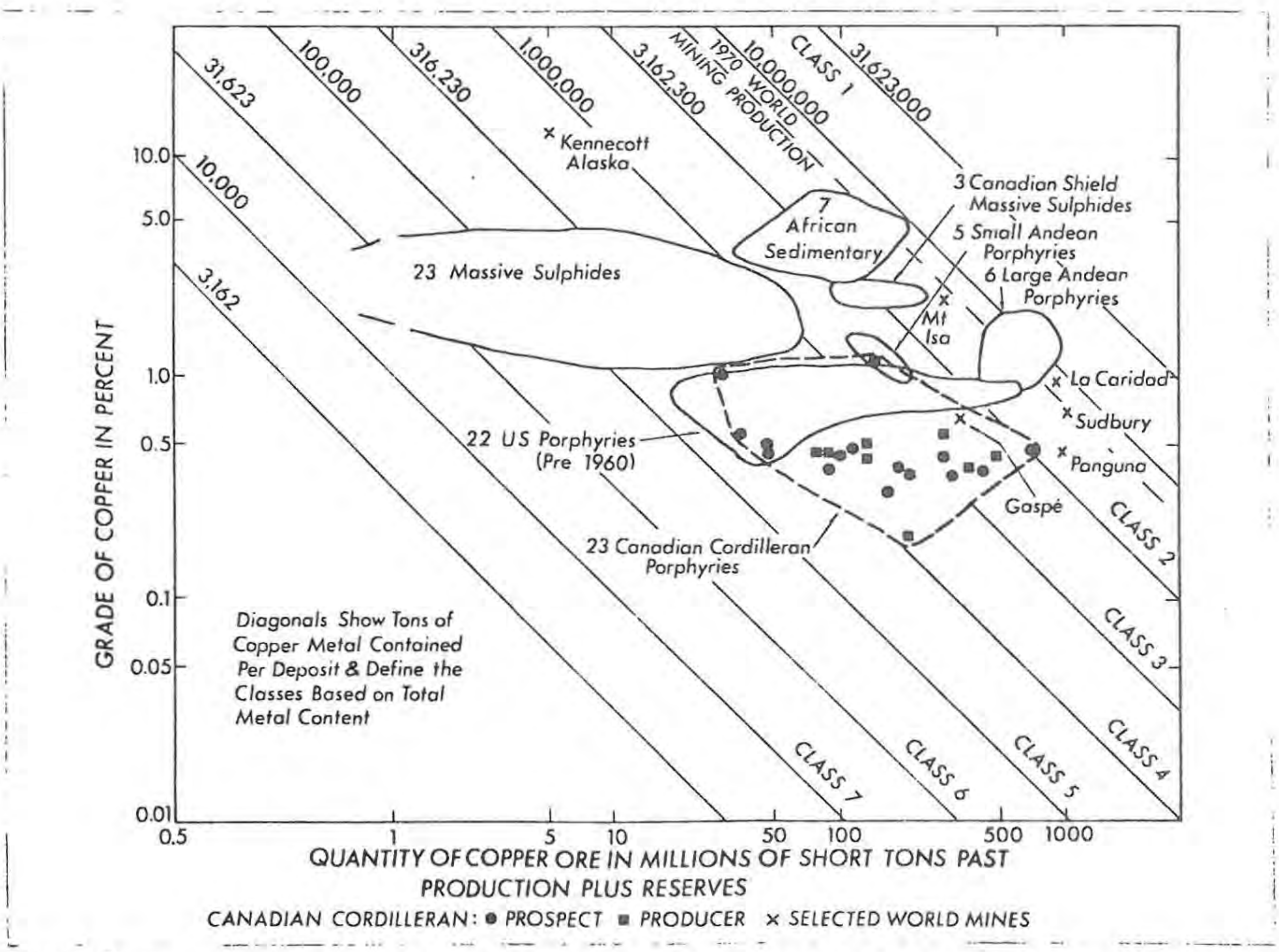

FIGURE 2— Size and grade of selected copper orebodies of differing types and geographic regions after an unpublished compilation by A. A. Kinkel, Jr. (Cox *et al.*, 1973).

SUMMARY AND CONCLUSIONS

Porphyry type deposits present certain geological characteristics that make them unique as deposits from the exploration, evaluation and mining point of view, e.g.

- emplacement near to surface in association with the roof of a granodioritic batholith
- elliptical to circular in plan with an outcrop of 1300 to 2000 m in diameter
- disseminated and stockwork mineralization in the intrusive and wall rocks
- zoned distribution of the sulphide mineralization
- common presence of a supergene enrichment zone.

The generally large size-low grade porphyry deposits have shapes and rock strength characteristics which are favourable in defining large mine and plant production capacities, and open pit or underground block caving mining methods. Because of their large tonnage -low grade relationship these deposits support operations with a low profit margin per ton ore mined. But as lower grade porphyry deposits are being mined, acceptable margins of errors in ore reserve estimation and economic analysis decrease. "Kriging" has proved to be a particularly successful reserve estimation method for reducing these errors. Estimation errors in porphyry deposits usually arise because certain geological features, i.e. stockwork-veinlets, variations in zoning and grade distribution patterns, etc., are not taken into account during the exploration, evaluation and mining.

Several porphyry case histories also show that the distribution pattern of ore and waste in these deposits was neglected during the feasibility stage where an appropriate mining scale had to be chosen. This inevitably resulted in poor ore recoveries and high dilution during mining.

ACKNOWLEDGMENTS

I am thankful to Prof. R. Mason for his guidance and assistance throughout the year and in this dissertation. It was through his suggestion that this dissertation originated and his criticisms are gratefully acknowledged.

I also wish to take this opportunity to thank the staff of Rhodes University Geology Department for the interesting lecturing and assistance during the 1979 M.Sc. course and, specially Dr. R.E. Jacob for his critical review of the dissertation. My thanks are also to Mrs. M. Jackson for her indispensable help and patience in the typing of essays and letters during the year, and to Mrs. J. Gruber for the typing of this dissertation.

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PART 1

1.1 Description of Porphyry Deposits

Porphyry deposits are widely distributed around the world, principally within linear mountain belts of Mesozoic and Cenozoic age, e.g. the Circum-Pacific orogenic belts and the central portion of the Alpide orogenic belt (Fig.3).

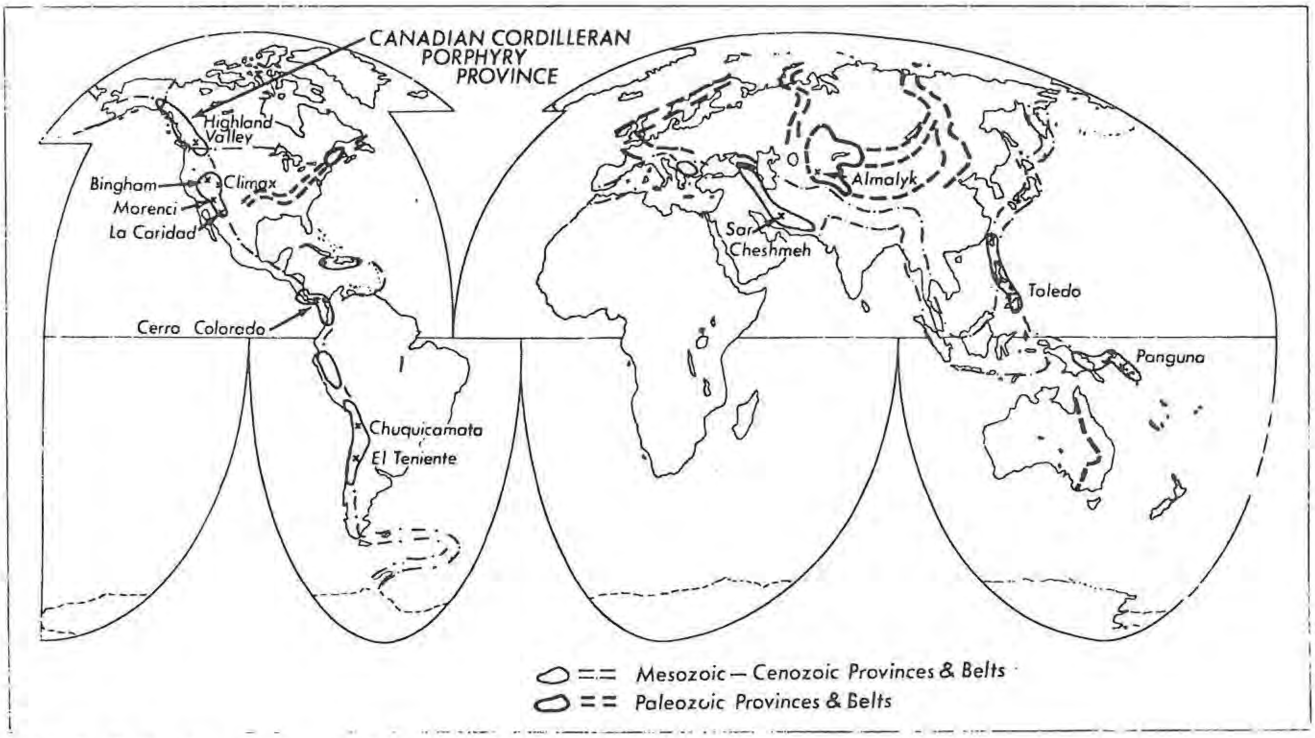


FIGURE 3– Porphyry provinces of the world and related mountain belts.

(Taken from Sutherland Brown and Cathro, 1976.)

Porphyry deposits are also found in pre-Mesozoic mountain belts or their eroded roots, but they appear to be increasingly rare with age and are of lower grade, perhaps because of diminishing chances of survival and changes in the evolution of the earth's crust through time. These linear belts associated with porphyry provinces are characterized by the presence of volcanic rocks of the calc-alkaline suite.

Porphyry deposits form a diverse but genetically related group closely associated with intrusive granitic bodies which were emplaced relatively near the surface. The host-intrusive stock of a porphyry deposit emplaced in the roof of a granodioritic batholith or in the adjacent country rock, is commonly elongate-irregular to circular in shape. It is of the order of 1300 m to 2000 m in outcrop dimensions, and is progressively differentiated from quartz-diorite to quartz monzonite in composition. The host intrusive is more like a stock than a dyke and is controlled by regional-scale faulting. An average of about seventy percent of the ore in these deposits occurs in the igneous host rocks, and about 30% in the pre ore-country rocks. Whatever the case, the distributions of mineralization and hydrothermal alteration normally form symmetrical patterns that reflect the shape of the intrusion. Normally, the different granitoid rocks or stock complex exhibit a porphyritic texture in which some minerals form large irregular crystals within a matrix of smaller grains. The primary mineralization appears as disseminations and as veinlets in fracture stockworks. Some of these stockworks have formed through hydraulic fracturing (Phillips, 1972) during, but relatively late in, the process of emplacement and consolidation of the related intrusives.

The granitic rock suite, veinlet and fracture stockworks, breccia bodies, and hydrothermal and thermal alteration are therefore all closely related in age and part of the porphyry system.

The sulphide mineralization occurs throughout a great volume of rock, e.g. Table 1 illustrates the average geometric and dimensional characteristics of the commercial porphyry deposits of the Canadian Cordillera.

The hydrothermal alteration and mineralization zones are genetically and spatially related to the intrusive stock. They usually show a concentric pattern in relation to the intrusive stock. A typical porphyry copper deposit has a central economic portion characterized by concentric shells of potassium silicate, sericitic, argillic and an outward propylitic alteration zone (Fig.4).

TABLE 1 — Composite Model of Commercial Porphyry Deposits of the Cordillera[1]

Geometry of intrusive complex:	— total area of intrusive: 136 ha (336.4 acres) — length of perimeter of intrusive: 4.5 km (2.8 miles) — area of porphyritic phase: 25 ha (61.6 acres) — area of breccia and diatremes: 11.5 ha (28.3 acres)
Contact alteration zone:	— width: 396 m (1300 feet)
Geometry of orebody[2,3]:	— area of horizontal section: 49.6 ha (122.4 acres) — length: 988 m (3240 feet) — width: 529 m (1740 feet) — thickness: 207 m (678 feet) — tonnage: 249.7 million tonnes (275.3 million short tons)
Portion of tonnage of orebody in wall rock:	— 46.2%
Portion of tonnage of orebody in intrusive:	— 53.8%
Economic characteristics of orebody[3]:	— over-all grade: \$10.01 per tonne (\$9.08 per short ton) — copper grade: 0.67% — molybdenite grade: 0.029% — Pb, Zn, Ag, Au values: \$1.52 per tonne (\$1.38 per ton) — total gross value: 2,450 million dollars — average mining rate: 19,000 tonnes per day (21,000 tpd) — average stripping ratio: 0.8 to 1 — total capital investment required to reach the production stage: \$123 million
Geometry of alteration zone:	— total area: 196 ha (484.5 acres) — area of K-biotite alterations: 57.6 ha (142.3 acres) — area of pyritization: 167 ha (413.2 acres)

(1): modified from De Geoffroy and Wignall (1973; Table 5, p. 39)
(2): based on 0.35% Cu cutoff or MoS_2 equivalent
(3): based on \$0.50 per lb copper and \$1.70 per lb MoS_2

(Taken from Drummond and Godwin, 1976.)

The Fe/Cu ratio of the sulphide mineralization also increases towards the marginal alteration zones. Figure 4 also shows the typical alteration minerals present in these zones. The propylitic alteration zone usually contains only pyorite. The potassic and sericitic alteration zones contain the Cu-Fe and Mo sulphides: chalcopyrite, bornite and/or chalcocite. Minor sulphide associations containing Mo, An, Pb, Zn and Ag are also usually present (Fig.4). Sutulov (1963, 1970) has drawn attention to the high rhemium content of molybdenite in some porphyry copper deposits. On the other hand, rhemium content of molybdenite in stockwork molybdenum deposits is very low.

A characteristic feature of most porphyry copper deposits is a

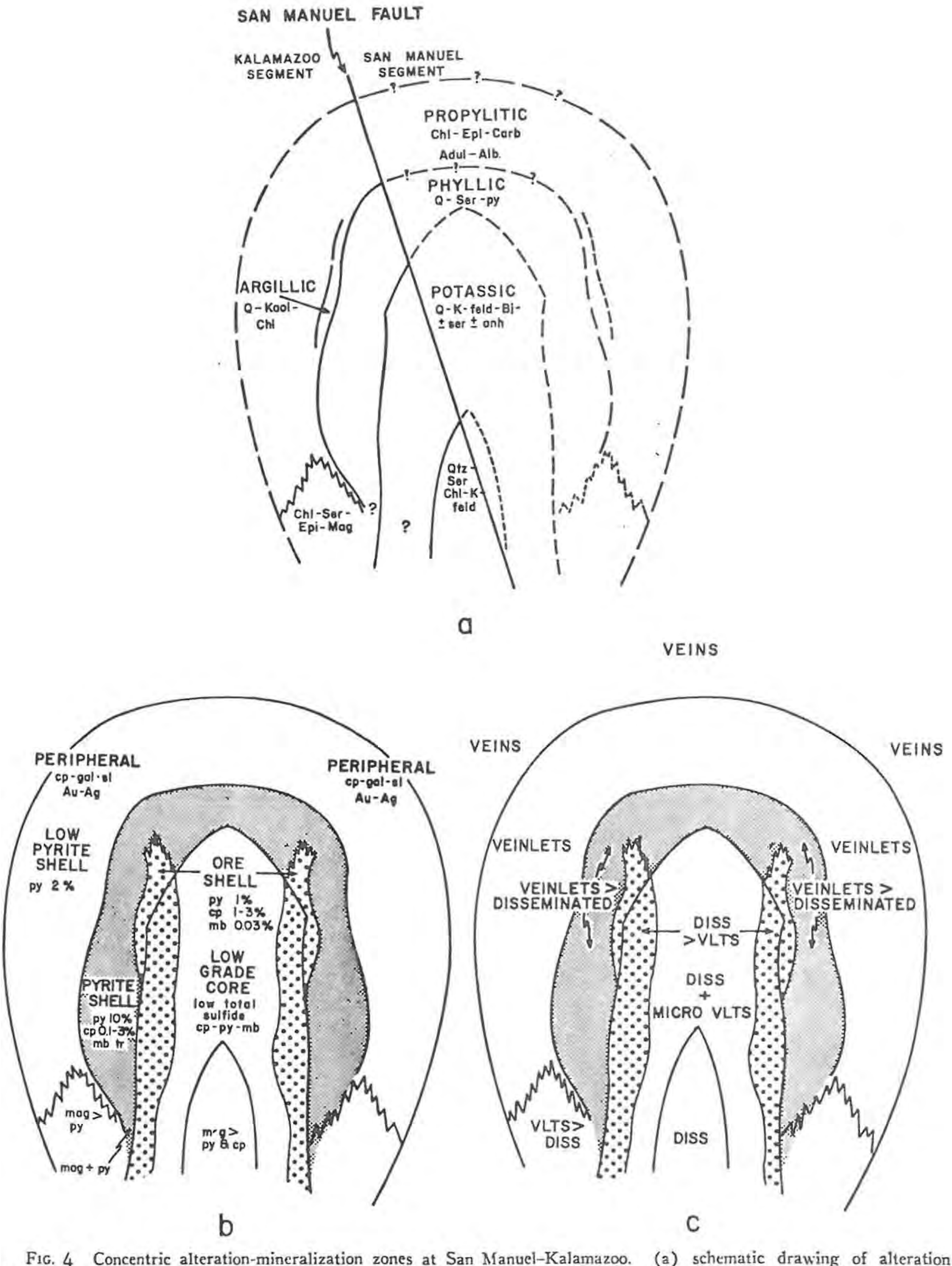


FIG. 4 Concentric alteration-mineralization zones at San Manuel-Kalamazoo. (a) schematic drawing of alteration zones. Broken lines on Kalamazoo side indicate uncertain continuity or location and on San Manuel side extrapolation from Kalamazoo. (b) schematic drawing of mineralization zones. (c) schematic drawing of the occurrence of sulfides.

(Taken from Lowell and Guilbert, 1970.)

more or less developed supergene enrichment. Leaching and supergene enrichment are important processes in many porphyry copper deposits of low grade primary mineralization. These processes provide mechanisms by which a small percentage of metals can be leached from a large volume of low grade mineralized rock and be redeposited to form a higher grade deposit in a smaller volume of rock. The economic value of many porphyry copper deposits being mined at present is a result of these processes (e.g. El Salvador, El Teniente and Chuquicamata, in Chile). There are several factors which can assist the secondary enrichment process:

- en inert wall rock
- a warm climate and a deep fluctuating water table
- a high proportion of pyrite in the leached rocks.

1.2 Porphyry Models

Sillitoe's (1972) proposed model accepts the premise that economic concentrations of copper and molybdenum in a typical porphyry copper system occur in a subvolcanic environment associated with small, high-level stocks. Commonly a porphyry copper-bearing stock grades downward into a pluton of larger dimensions which possesses stockwork mineralization in its upper parts. This hypothetical porphyry copper deposit is overlain by a column of pyritic alteration which transects a calc-alkaline volcanic pile, which culminates upwards in an andesitic strato-volcano with native sulphur deposits (Fig.5). A few small, high-grade silver, gold, lead, zinc epithermal veins are common in the propylitic alteration zone. Sulphide mineralization tends to shift from disseminations in the potassic core of the porphyry through respective zones of microveinlets, veinlets, veins and finally to the individual structures on the periphery which may contain high-grade mineralization (see Fig.4).

The normal location for the economic portions of porphyry ore deposits is in the coeval volcanic piles. Hence, overlying volcanic formations would be expected normally to have been completely eroded from the vicinity of porphyry systems by the time thatu-Mo mineralization is exposed. Stockwork and disseminated mineralization may extend downward from the stock, dying out in depth in the upper parts of the subjacent pluton (Fig.5). Near the upper limit of economic hypogene mineralization, intrusive bodies are likely to be smaller and less regular, and large areas are likely to be occupied by hydrothermal breccias.

As already indicated, porphyry copper deposits may differ in their principal associated metallic element, Mo or Au. Hollister (1975) distinguished Cu-Mo (the Lowell and Guilbert model) and the Cu-Au (the Diorite model) porphyry deposits (Table 2).

The Cu-Mo porphyry deposits developed within a relatively thick continental crust. Their predominant associated intrusives are grano-

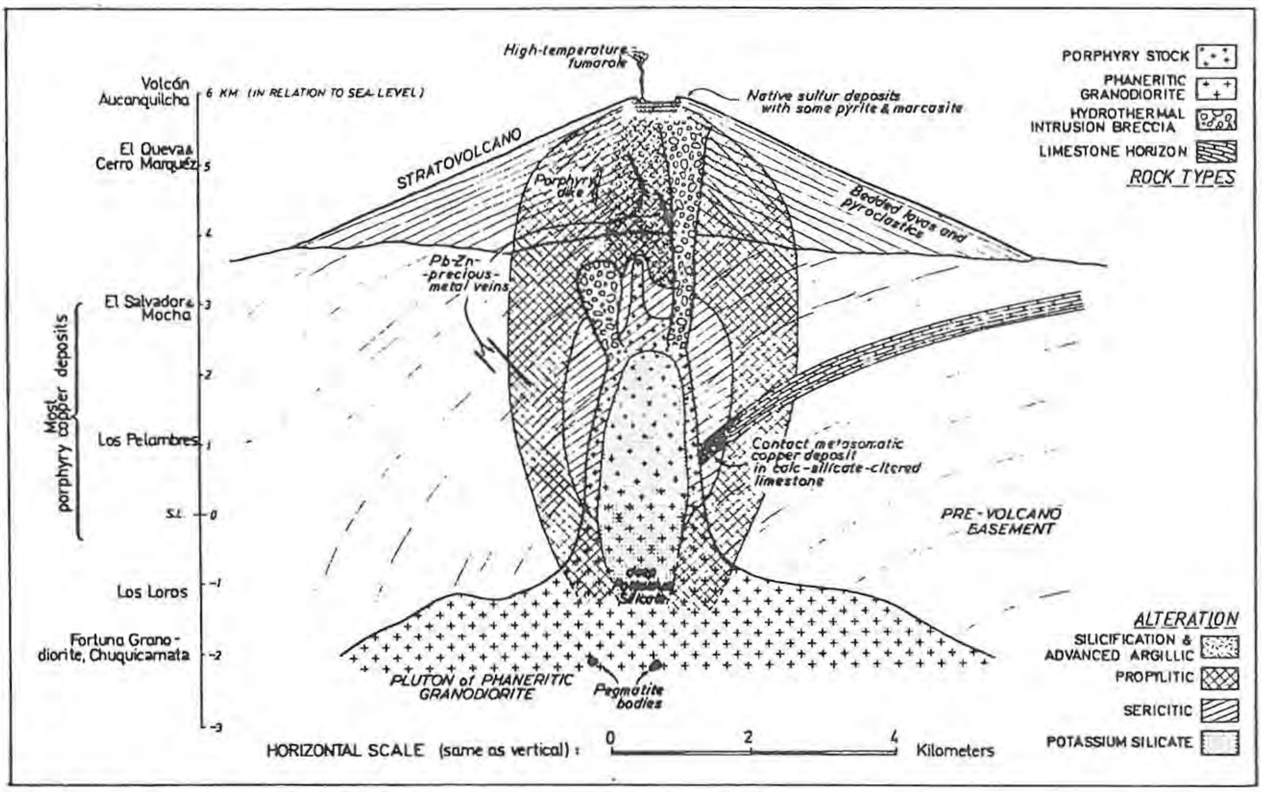


FIG. 45 Idealized cross section of a typical, simple porphyry copper deposit showing its position at the boundary between plutonic and volcanic environments. Vertical and horizontal dimensions are meant to be only approximate.

(Taken from Sillitoe, 1973.)

diorite to quartz monzonite porphyries and contain a well-developed quartz-sericite alteration zone and mineralized veinlet stockwork. These deposits are mainly restricted to the cordilleran environment of the continental margins (e.g. porphyries of the Andean Belt).

The Cu-Au porphyry deposits (Diorite model) developed within a thin continental crust (Hollister, 1975). These deposits are mainly found in island arc environments (e.g. the south-west Pacific, Philippines) and differ from the Cu-Mo deposits in their predominantly syenitic monzonitic and quartz-free associated intrusions. The sulphide mineralization occurs mainly in the form of disseminations and there is generally no development of a quartz-sericite zone or breccia pipes, and there are no well developed veinlet stockwork mineralizations or significant supergene enrichment zones.

TABLE 2
COMPARISON OF LOWELL AND GUILBERT AND DIORITE MODEL FEATURES

Feature	*Lowell and Guilbert model*	*Diorite model*
Intrusive relationships		
Typical intrusion close to ore	Quartz monzonite-granodiorite	Syenite-monzonite
Other intrusions present	Quartz diorite	Diorite
Alteration		
Central Core Area	Potassic (including orthoclase-biotite and/or orthoclase-chlorite)	Potassic (including orthoclase-biotite and/or orthoclase-chlorite)
Peripheral to core	Phyllic (quartz-sericite-pyrite)	Propylitic (chlorite-sericite of chlorite-epidote)
Peripheral to phyllic	Argillic	
Peripheral to argillic	Propylitic (chlorite-epidote)	
Where pervasive pyrite occurs includes	potassic, phyllic zones	Either potassic only or potassic and propylitic
Mineralization		
Quartz in fractures	Common	Erratic
Orthoclase in fractures	Common	Erratic
Albite in fractures	Trace	Common
Magnetite	Minor	Common
Pyrite in fractures	Common	Common
Molybdenite	Common	Rare
Chalcopyrite: bornite ratio	3 or greater	3 or less
Dissemination of chalcopyrite	Present	Important
gold	Rare	Important
Structure		
Breccia	May occur	Rare
Stockwork	Important	Important

(Taken from Hollister, 1975.)

The absence of a well developed supergene enrichment zone is caused by the presence of rather reactive silicates and copper being fixed in place during weathering (Hollister, 1978). The abovementioned features explain the generally lower copper grade in the porphyries of the diorite model.

Porphyry deposits bearing tin (e.g. in Bolivia) and/or tungsten (Yukon Territory) have been described respectively by Sillitoe et al. (1975) and by Cathro (1969). Porphyry tin deposits account for 75% of Bolivia's tin production.

Porphyry deposits form a discrete, genetically related group of mineral deposits. Nevertheless, these deposits are gradational or related to a number of other types of deposits that are associated with igneous processes (see also Fig.5):-

- many porphyry deposits contain a few large veins, e.g. the Stella vein of the Endako Mine (B.C.), the Maria Vein of the El Abra porphyry copper deposit (Chile);

- other porphyry bodies contain breccia pipes or pegmatitic phases that may contain a small but relatively high-grade part of the total reserve, e.g. Copper Mountain (B.C.), Ajo and Santa Rita, Arizona;
- skarn (contact metasomatic) sulphide deposits commonly occur adjacent to the granitic bodies and in some cases may adjoin porphyry deposits . Skarn deposits differ from porphyry deposits in degree of fracturing, shape, size, lack of symmetry and vein stockwork, erratic distribution of metallic minerals and mineralogy. In spite of these differences, skarn deposits may occur as an integral part of porphyry systems such as at Ingerbelle (B.C., Canada) and Bingham (U.S.A.).

Different exploration and evaluation approaches must be considered when other types of associaed ore bodies are present. The level of exposure of the porphyry system is an important variable in this respect, e.g. only at the deepest levels of the San Manuel-Kalamazoo deposit and of the Butte district are the pegmatoid textures and the potassic alteration core present. The level of exposure has also a direct bearing on the mineralogy of the ores present near surface (Figs.4,5).

1.3 Geological Controls influencing Tonnage, Grade and Shape of Porphyry Deposits

Porphyry deposits characteristically show features and mineralization controls of the epizonal type of deposits. Soregaroli (in Clark, 1972) described some of these controlling features at the Boss Mountain porphyry molybdenum deposit (Canada):-

- discordant and chilled sharp margins
- genetically related porphyry dikes
- lack of both foliation and contact metamorphism of adjacent porphyries
- presence of some primary linear structures, breccias and small plutons.

Stringham (1960) established that most productive porphyries, when compared to barren ones, evidence strong crosscutting relationships with the enclosing rock. These differences and other of less importance are shown in the following table.

Table 3.

INTRUSIVE PORPHYRIES
CRITERIA LISTED IN ORDER OF IMPORTANCE

Barren	Productive
1. Classification—forceful. Structure—concordant, sheets, sills, laccoliths, bysmaliths, intrusive breccia, etc.	1. Classification—passive. Structure—cross cutting.
2. Strong flowage features.	2. Weak flowage features (except in smaller bodies, dikes, etc.).
3. Prominant cooling features. Chilled borders. Great variations in grain size, directly related to borders and size of intrusion.	3. Cooling features very weak. Little variation in grain size. Smaller bodies tend to have fewer phenocrysts.
4. May have varied composition. Particularly of basic & alkalic rocks. Where stratification could be prominant.	4. Uniform composition. Rock types restricted to alaskite porphyry through diorite porphyry or rhyolite through andesite.
5. May or may not have glassy groundmass.	5. Glassy groundmass essentially absent.
6. Borders toward related granitoid rock may be gradational.	6. Borders sharp, even toward related granitoid rock.
7. Little contact action on wall rocks.	7. Much contact action where wall rock composition is unlike intrusive (rhyolite—limestone).
8. Usually not altered (propylitization and weathering excepted).	8. May or may not be altered but strong alteration is definitely encouraging.
9. Phenocrysts not likely to be scalloped by groundmass.	9. Phenocrysts may be scalloped by groundmass.

(Taken from Stringham, 1960.)

Porphyry type deposits are host or wall-rock as well as structurally controlled, the host rock providing a regional constraint for the mineralization and structural and alteration features providing more local constraints. Porphyry deposit boundaries represent the outer limits of the effects of mineralizing processes. These processes may produce protore grades of up to 1% Cu. Beyond the deposits boundaries the rocks usually contain 5 to 150 ppm Cu (Singer et al., 1975).

The position of the deposit boundaries and hence the tonnage of the deposit, is a function of the magnitude of the geologic systems involved, whether plutonic or volcanic. The grade depends on a combination of factors relating to the intensity and number of phases of the mineralization process, the concentration of the mineralizing solutions, the rate of change of those concentrations in the depositional process and the availability of sites of deposition. Sites for disseminated mineralization in the host rock include:-

- carbonate wall rocks (skarn-type deposit)
- dispersed iron oxide and iron-magnesian/silicate minerals in both wall rocks and intrusive bodies
- sites around the margins of sericitized feldspars and micas. These sites were generated by an increase in the porosity of the rock during the sericitic-argillic alteration (Barnes, 1967). The overall increase of porosity by this hydrothermal alteration process may reach up to several per cent (op.cit.).
- prior sulphide disseminations that act as nucleii of deposition for later stage sulphides.

veinlets in fractures of the stockwork system are the other important depositional sites for the sulphides. Depositional sites for sulphides may also include the fractures produced by chemical brecciation (Sawkins, 1969) in earlier phase sulphide veinlets, e.g. Chuquicamata.

Grade is also affected by repeated hypogene and supergene mineralization processes superimposed on each other. The latest of the interdependent igneous intrusion and hydrothermal events, tend to mask and even remobilize the early ones (e.g. Chuquicamata being an extreme case of masking and remobilization).

In summary, the grade and the tonnage of a porphyry type deposit seems to be controlled by separate and distinct geological factors (Singer et al., 1975), tonnage being an extensive variable and grade an intensive variable.

The different geological controls affecting the tonnage, grade and shape of porphyry are reviewed in more detail in the following sections.

1.3.1 Lithologic Control

Porphyry deposits are generally spatially and genetically associated with acid to intermediate intrusions with a porphyritic texture and fine-grained phaneritic to aphanitic matrix. The overall configuration of stockwork-type orebodies is to a certain degree controlled by the shape, depth and degree of complexity of the intrusive. The nature of ground preparation and the cooling history of the magmatic and hydrothermal fractions are also equally important.

Porphyry deposits with mineralization restricted to an intrusive body yield a mineralized tonnage that is controlled mainly by the volume of the intrusion. Very few intrusions associated with porphyry copper mineralization are of batholithic proportions; in those of batholithic proportions, only a small part of the intrusion is mineralized or hydrothermally altered (Fig.7b). In other deposits in which favourable wall rocks are mineralized (e.g. El Teniente and Mantos Blancos porphyry copper deposits, Chile; the Questa porphyry molybdenum deposit, U.S.A.) the size of the orebodies may be dependent on the stratigraphic thickness of the favourable beds, on the width of the copper-rich mineral zones surrounding the porphyry deposit. This width is presumably dependent on the temperature, pressure, hydrothermal conditions near the intrusion, and on the physical and mineralogical-chemical character of the adjacent host rocks (Singer et al., 1975).

The igneous activity in porphyry deposits can include the pre- intra-, and post-ore intrusions (e.g. Santa Rita, Cananea, Toquepala) forming a complex stock (Hollister, 1978). Complex ore bodies can be expected as a result of this, e.g. the tops and flanks of each igneous phase at Climax are surrounded by zones of fractured rock into which hydrothermal fluids were introduced. Hemispherical ore bodies cap each igneous phase in this porphyry deposit. These orebodies become smaller and formed closer, although remaining external to the upper contact of their genetically related intrusives (Clark, 1972). Figure 6 (a)shows the localization of the orebodies in the case of a simple intrusion, e.g. the Questa porphyry.Figure 6(b) shows the case of the multiple intrusion at Climax.

According to Hollister (1978), breccia pipes present in deposits with a dominant stockwork structure are generally small and play a subordinate role in the localization of metal sulphide.

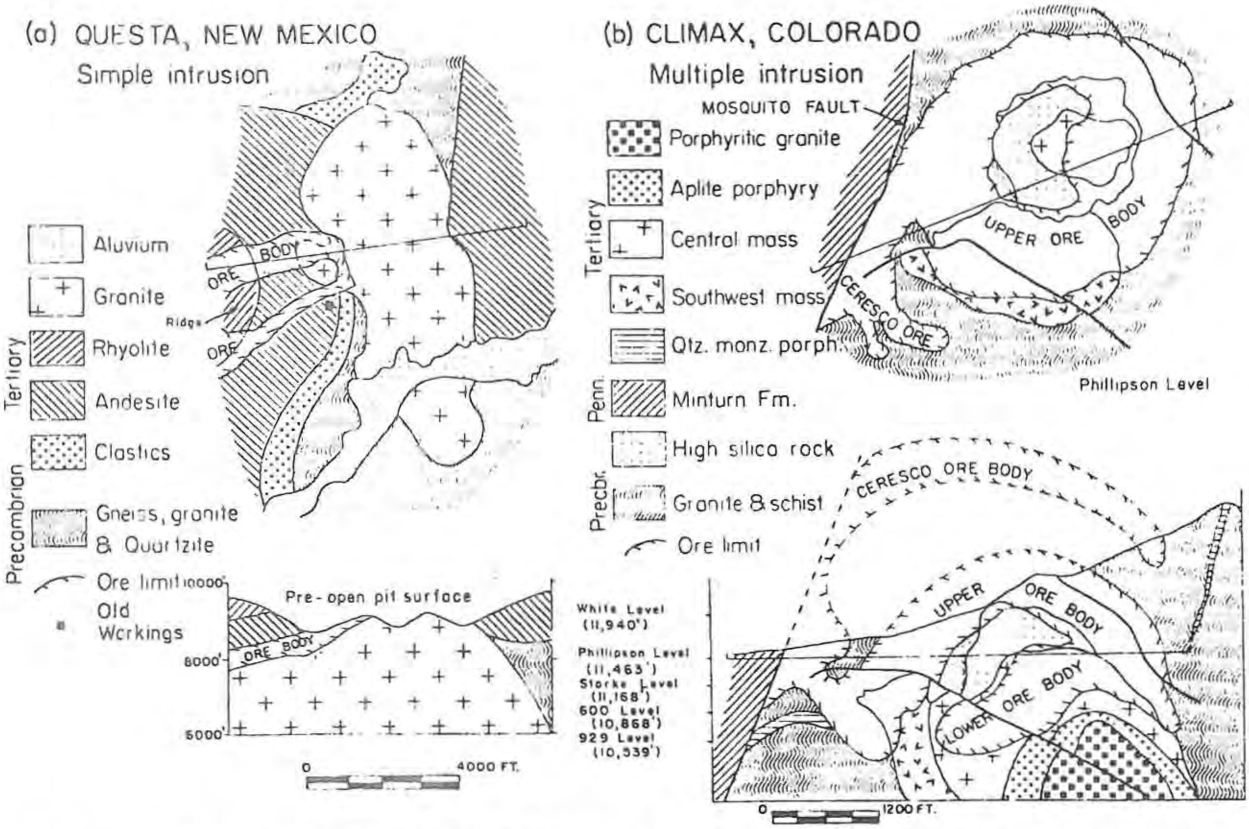


FIG. 6 Plans and sections of two stockwork molybdenum ore bodies and associated host rocks illustrate the classification adopted: (a) Questa-simple intrusion, (b) Climax-multiple intrusion (from Wallace et al., 1968).

(Taken from Clark, 1972.)

Large breccia pipes, on the other hand, may have stockwork structures ringing their periphery, and metal sulphide is primarily controlled by the pipe. The distinction between large and small pipes is economically significant.

Small breccia pipes: small breccia pipes accompany stockwork deposits and may have configurations and other details different from those found in large pipes. Small pipes may have distinct bottoms, blind tops, and grade into breccia dikes along strike.

The bottom configuration of small pipes suggests this breccia may terminate downward in a funnel-shaped neck or grade into a series of tight fractures. Top configuration of small breccia pipes may show that the pipe itself is blind, having developed without relief to the present surface (Perry in Hollister, 1978). Small breccia pipes that do reach the surface may have significantly enlarged cross-sections at depth. Generally the smaller the pipe the greater its tendency to be elongate or grade into a breccia dike (Bryner in Hollister, 1978).

Large breccia pipes: bottom configuration of large breccia pipes remains unknown because no large pipes (e.g. Toquepala, El Teniente, La Caridad) have ever been bottomed. Nor have the roofs of any of these pipes been seen. Refragmentation and multi-stage brecciation with mixing of rock types in fragments appear to be typical of the larger pipes.

Major breccia pipes commonly are ringed with a peripheral annular fracture set, whereas such fractures are absent in the walls of small pipes. Most major breccia pipes also contain a late-stage upward expanding pebble pipe or pebble dike. These pipes and related fracture-vein systems are directly related to the emplacement of the intrusive stock and may contain mineralization, e.g. Boss Mountain porphyry molybdenum deposit (B.C.), (Fig.7a). These breccias appear to terminate in depth against the Boss Mountain stock.

Breccias in the Panguna porphyry copper deposit (Bougainville) occur in nearly all rock types (Baldwin et al., 1978). However the important mineralized ones are related to the Biotite granodiorite

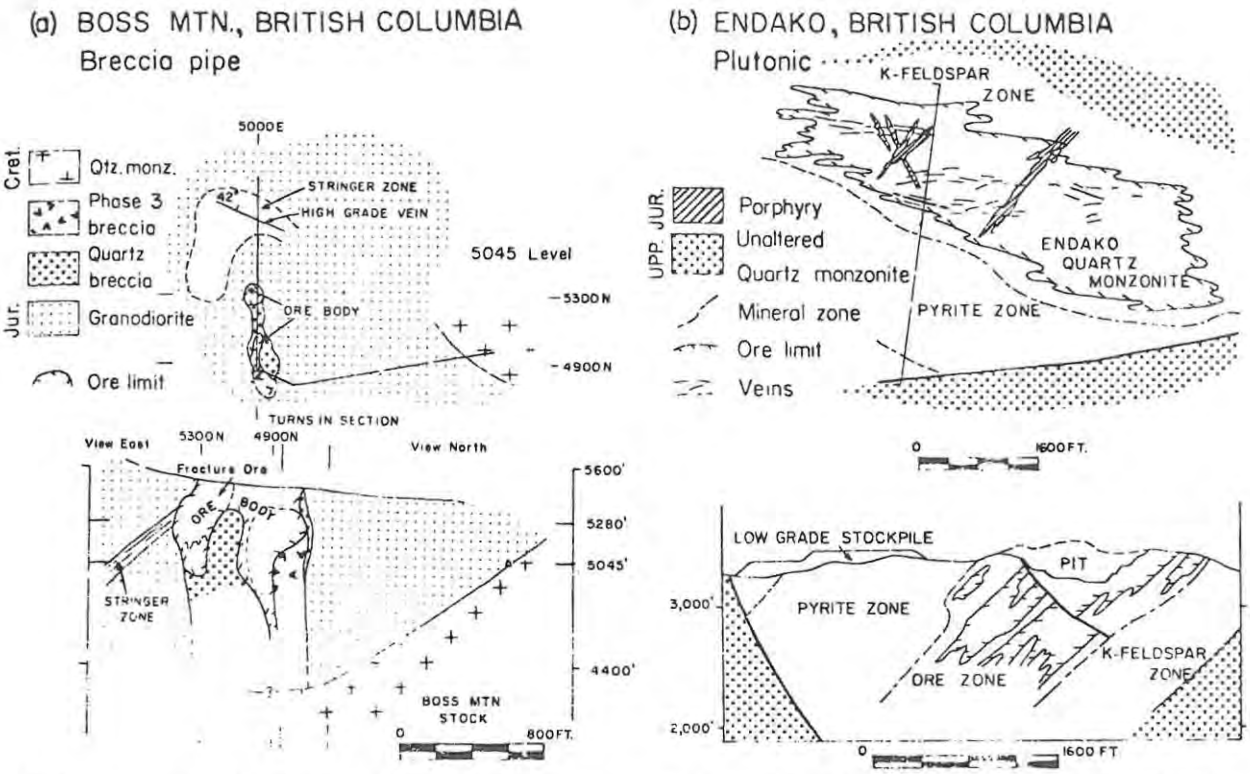


FIG. 7. Plans and sections of two stockwork molybdenum orebodies further illustrates the classification adopted: (a) Boss Mountain-breccia pipe (from Soregaroli, 1968a), (b) Endako-plutonic (from Drummond and Kimura, 1969).

(Taken from Clark, 1972.)

in time and space and are clustered around the margins of this intrusion. The breccias are mainly pipe-like but may coalesce to form linear belts along the contact. Outlines of these mineralized breccias from bench to bench are irregular, but they are generally persistent with depth.

Three different types of breccias are present at Panguna:

- collapse breccias : these contain copper sulphides
- intrusive breccias : contain copper sulphides and minor amounts of Au in bornite
- contact breccias : mineralization is highly erratic; parts of the breccias are barren.

Intramineral and post-ore porphyry intrusions may partly destroy (and remobilize?) the original ore bodies, e.g. the Urad-Henderson (Colorado) complex of intrusives and associated subvolcanic porphyry

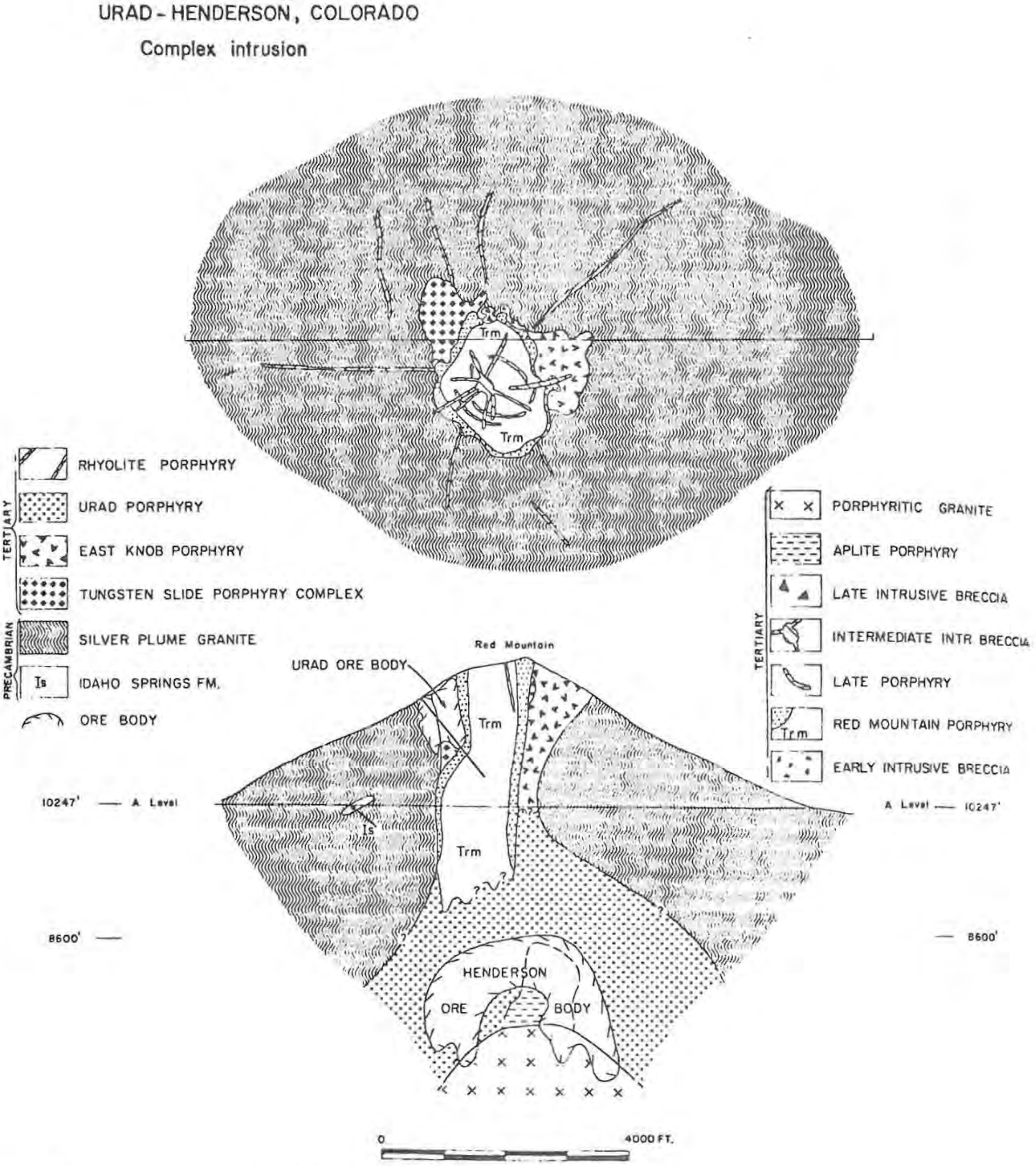


FIG. 8 Surface plan and section of the Urad and Henderson Ore Bodies illustrates a sequence of igneous and mineralization events of the complex sub-class (from Wallace, MacKenzie, and Blair, 1967; MacKenzie, 1970).

(Taken from Clark, 1972.)

orebodies are shown in Fig.8. The original ore bodies in this case were destroyed by the post-ore Red Mountain porphyry and the remaining Urad ores appear at present as being peripheral to this post-ore porphyry. Similar post-ore intrusions destroying the original ore bodies exist in several other porphyry deposits, e.g. El Salvador (Gustafson and Hunt, 1975), Panguna (Baldwin et al., 1978).

1.3.2 Structural Control

The initial function of structural control in hydrothermal porphyry type mineralization is to increase the permeability of the rocks that will host the mineralization. This form of ground preparation in porphyry deposits is represented by tension openings either as breccias columns and/or as a stockwock fracture system. The location of these tension openings - or conduits for the hydrothermal mineralizing fluids - will affect the distribution of the hydrothermal alteration and mineralization, and the corresponding alteration-mineralization zoning symmetry. The size and shape of these deposits will be related to this zoning. Usually, the more intensive and extensive the mineralized stockwork fracture system, the better the chances of encountering a higher grade protore and large tonnage porphyry deposit.

The stockwork system of porphyries may be of a tectonic and/or hydraulic origin. The hydraulic type of stockwork is generally developed at the periphery of the stock complex and in the adjacent intruded country rock. This type of stockwork is better developed in porphyry deposits of the Lowell and Guildbert model, in which breccia pipes are also common characteristic features (see Table 2). Hydraulic stockworks are produced in response to the build-up of hydraulic pressure of the residual gaseous phase during the cooling of a near surface magma (Gibson et al., 1975). Repetitive movements, or emplacement of later plutonic phases accompanied by pulses of hydraulic pressure built up, result in complex structures, e.g. Climax, Urad-Henderson porphyries (Clark, 1972). Stockworks of hydraulic origin present an irregular pattern and their fractures are rather discontinuous, resembling streaks.

Stockworks of tectonic origin in porphyries are developed in regionally localized areas in response to pressures of tectonic nature, e.g. the Haib porphyry copper prospect, South West Africa.

This type of stockwork is usually associated with some of the following tectonic features:-

- intersecting faults not demonstrably part of a conjugate set as exemplified at Chaucha, Michiquillay porphyries, Peru.
- conjugate fractures in either wall of one major fault as shown at the Chuquicamata and Copaquire porphyry deposits, Chile.
- shear zone fracturing as demonstrated at Butte.
- annu lar, radial and concentric fracturing caused by an intrusion (diapirism) as exemplified by Campana Mahuida (Argentina) and Hudson Bay porphyries (Fig.9).

Figure 10 illustrates the different structures and sequence of tectonic events in the development of the Endako "tectonic" stockwork. Stockworks of tectonic origin tend to present rather regular and continuous fracture patterns, (e.g. a system of joints, for example Haib). In most of the economic porphyry deposits emplaced in the Laramide stocks of Arizona, two sets of the regional system are mineralized. In the unproductive porphyries only one set tends to be mineralized (Rehrig and Heidrick, 1972). The presence of this type of stockwork may indicate lower hydraulic pressures and probably a weaker hydrothermal activity in the "tectonic" type stockwork. The level of exposure of the porphyries must certainly also be considered in the evaluation of these features.

An obvious combination of the "tectonic" and "hydraulic" type of stockworks exists in some porphyry deposits, e.g. Chuquicamata. At Chuquicamata the tectonic stockwork, bearing mineralized veinlets of the hydrothermal phase, was superimposed on the hydraulic stockwork system containing mainly magmatic and early-stage hydrothermal mineralization.

Major linear tectonic structures also play an important role in many porphyry deposits. Faulted and displaced porphyry deposits are more difficult and expensive to explore, to evaluate and to mine.

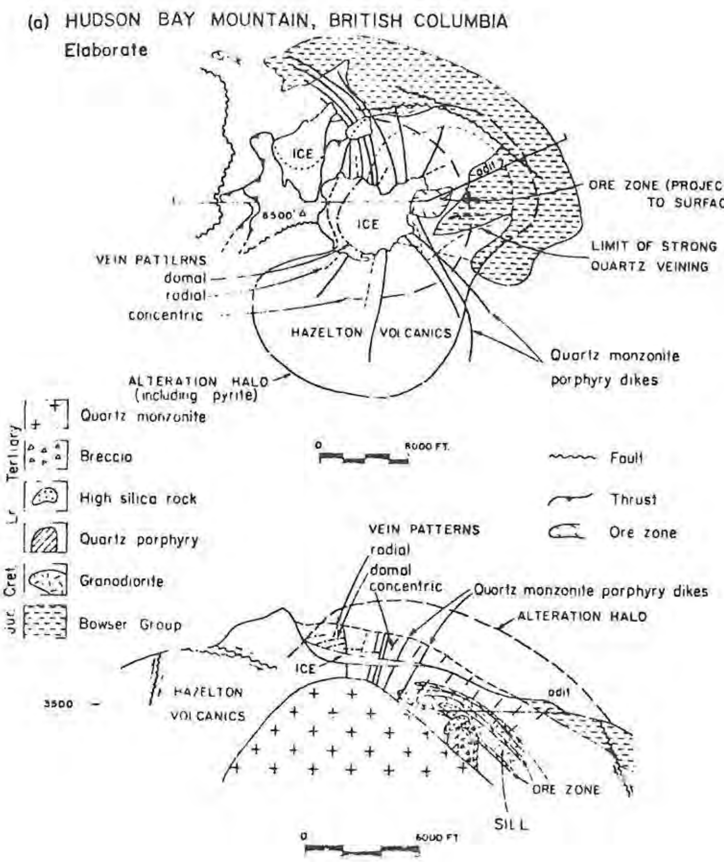
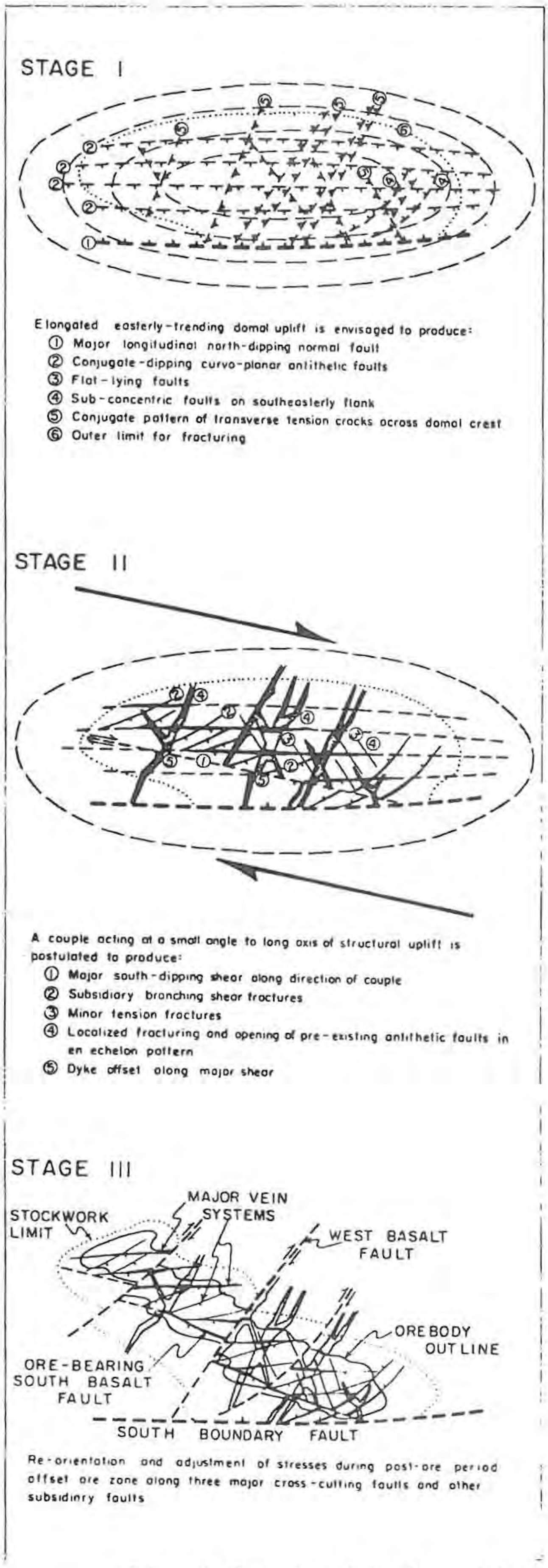


Fig.9. Plan and section of the Hudson Bay Mountain (B.C.) stockwork molybdenum deposit, showing type of fracturing (vein patterns) related to the intrusion.

(Taken from Clark, 1972.)



– Schematic illustrations of the elongated domal structure to show the sequential development of Endako stockwork.

Fig.10.
(Taken from Kimura and Bysouth, 1976.)

Although large ore bodies tend to be circular to oval in shape, the elongate nature of some deposits lacking breccia pipes, almost certainly represents the effects of pre-ore structural control. Butte, Silver Bell and Chuquicamata show fault-controlled leakage of the volatiles. The pressure of mineralizers in these deposits may not have built up to the explosive breccia-forming stage. In some deposits, post-mineral faulting followed by erosion has changed the tonnage and shape of the ore deposits. Ore bodies may be cut or displaced by major post-mineral faults, e.g. at Chuquicamata the uplifted western half of the original deposit is assumed to have been eroded away. The hypogene ore zone at the Morenci and Metcalf porphyry deposit was cut and severely displaced by a series of graben-like faults (Fig.11).

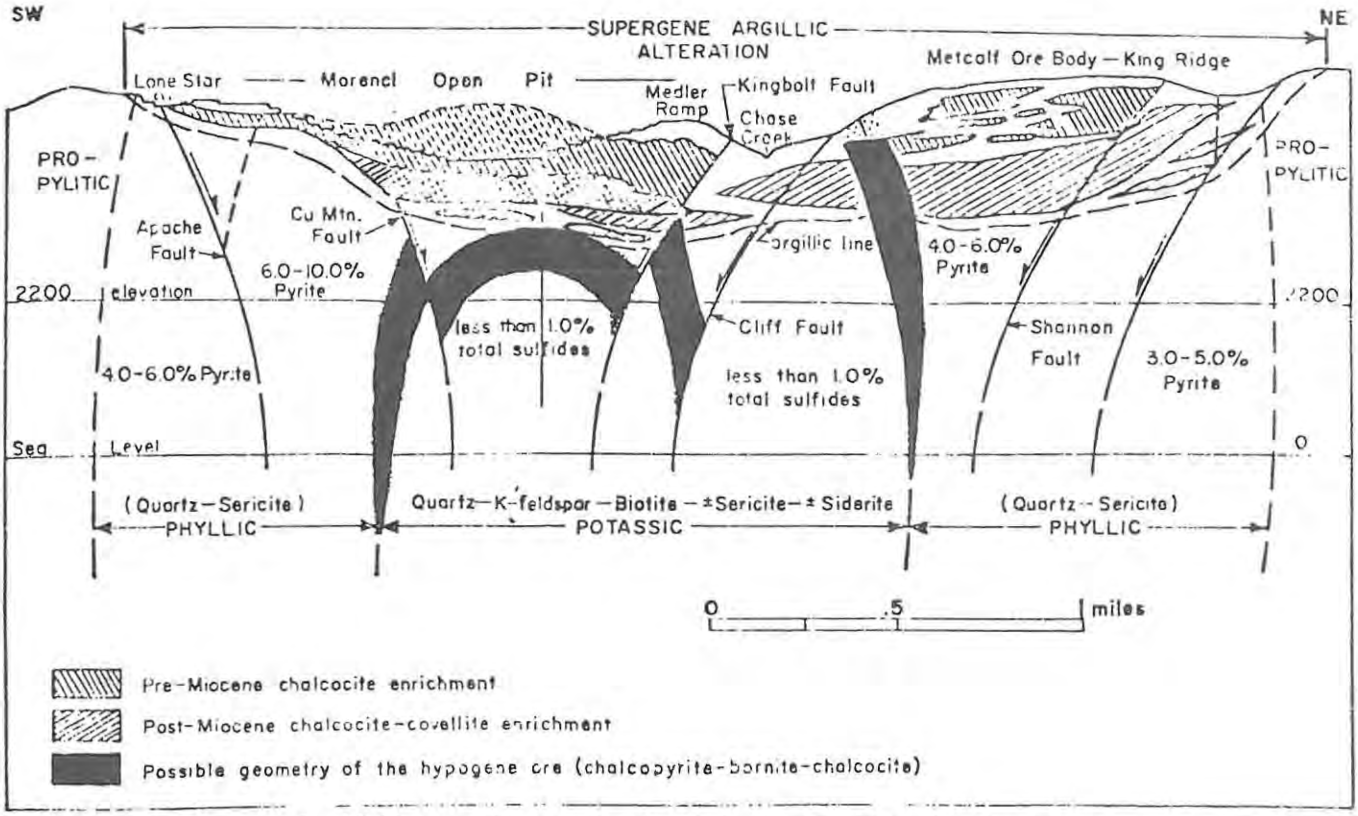


Fig. 11—Diagrammatic cross-section through the Morenci and Metcalf ore deposits.

(Taken from Langton, 1973.)

The peculiar dumbbell shape of the Morrison porphyry copper deposit (Canada) is the result of the original annular shaped deposit having been faulted and displaced horizontally about 300 m (Carson and Jambor, 1976). The lower-grade copper and higher zinc core of this deposit coincides with this fault zone (Fig. 12).

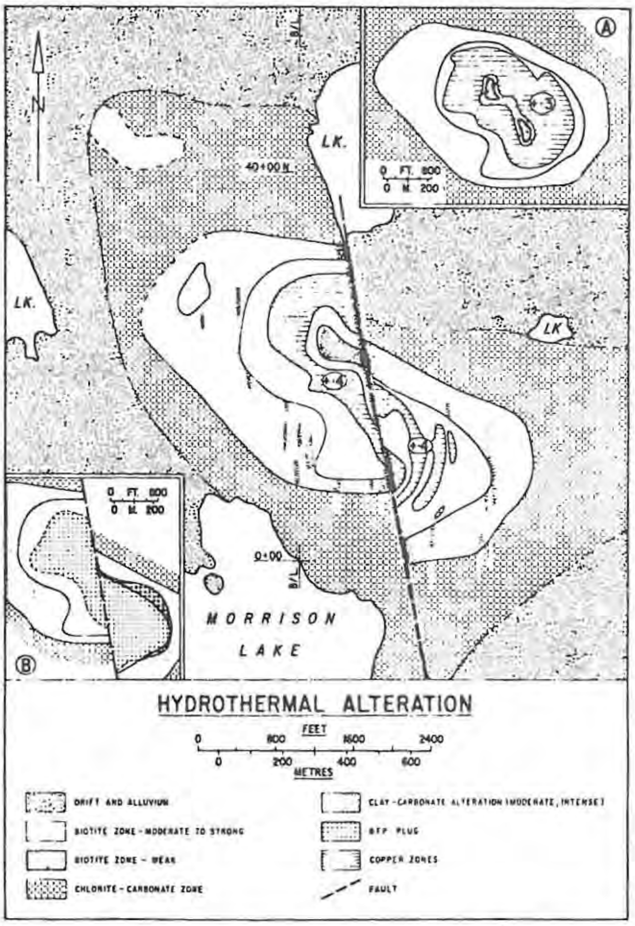


FIGURE 12- Hydrothermal Alteration Zones, Morrison Deposit. The main diagram illustrates the spatial relationship between the + 0.4 per cent copper zone and the moderate to strong biotitization. Intense clay-carbonate alteration occurs along the Morrison fault, which has offset the copper zone and alteration haloes approximately 300 metres horizontally. The northerly elongation of many smaller lenses and patches of clay-carbonate alteration is partly interpretive.
Inset A shows the biotite zone and + 0.3 per cent copper zone restored to their original, pre-faulting position, neglecting the vertical component of the offset. Inset B shows the BFP plug and biotite zone in their present, offset positions. Pyritization (> 1 per cent pyrite) occurs throughout the property.

(Taken from Carson and Jambor, 1976.)

1.3.3 Control by Alteration

As already mentioned before, each successive intrusive phase of a porphyry stock is accompanied by its suite of hydrothermal products, e.g. Climax (Wallace et al., 1968), El Salvador (Gustafson and Hunt, 1975). The pattern of hydrothermal mineralization in porphyries, although spatially related to the porphyry stocks is centered and is concentric with the respective alteration zones (Fig.4). The close relationship existing in these stockwork deposits between the hypogene alteration and mineralization zones, their mineralogy, grade and Fe/Cu ratios, is a function of distance from the heat source represented by the intrusive porphyries.

Complications of alteration-mineralization patterns can also be expected in the shallow volcanic-subvolcanic environment of porphyries. Contrasting adjacent rock types, structural anisotropism, and steep physical changes and chemical environmental gradients may produce telescoping of assemblages and short-range variation in alteration intensity and extensiveness (Guilbert and Lowell, 1973). In addition to this, the axis of each of the alteration-mineralization phases is not necessarily conincident, and the intensity of each of these events diminishes with the distance from the axis. Nevertheless, the several stages of mineralization, e.g. magmatic, early and late-stage hydrothermal mineralization phases added almost onto the same space can produce different mineral zones, e.g. copper zones, molybdenum zones, e.g. Butte porphyry (Fig.13), and/or an overall increase of grade within the same space (e.g. Chuquicamata).

Superimposed brecciation events on a typical porphyry alteration-mineralization pattern can also add complexity to the system, e.g. at the La Caridad porphyry copper deposit (Mexico), brecciation events control the economic mineralization.

a) Variations in the Size-Grade and Alteration-Mineralization Zoning Relationships

According to Lowell and Guilbert (1974), the phyllic quartz-sericite-pyrite zone is the principal ore bearer in most porphyry Cu-Mo deposits, with the overlap of phyllic and potassic zones assuming greatest importance in hypogene zoning (Fig.4). Rose (1970) indicated that the highest copper content normally occurs within the biotite-or the orthoclase zone or at its outer border with quartz-sericite alteration. Hypogene copper only tends to occur in the argillic zone in the larger porphyries (e.g. Chuquicamata) and in this case it tends to be in association with other sulphides. Generally, the smaller the porphyry deposit, the less likely is the copper to occur outside the phyllic zone.

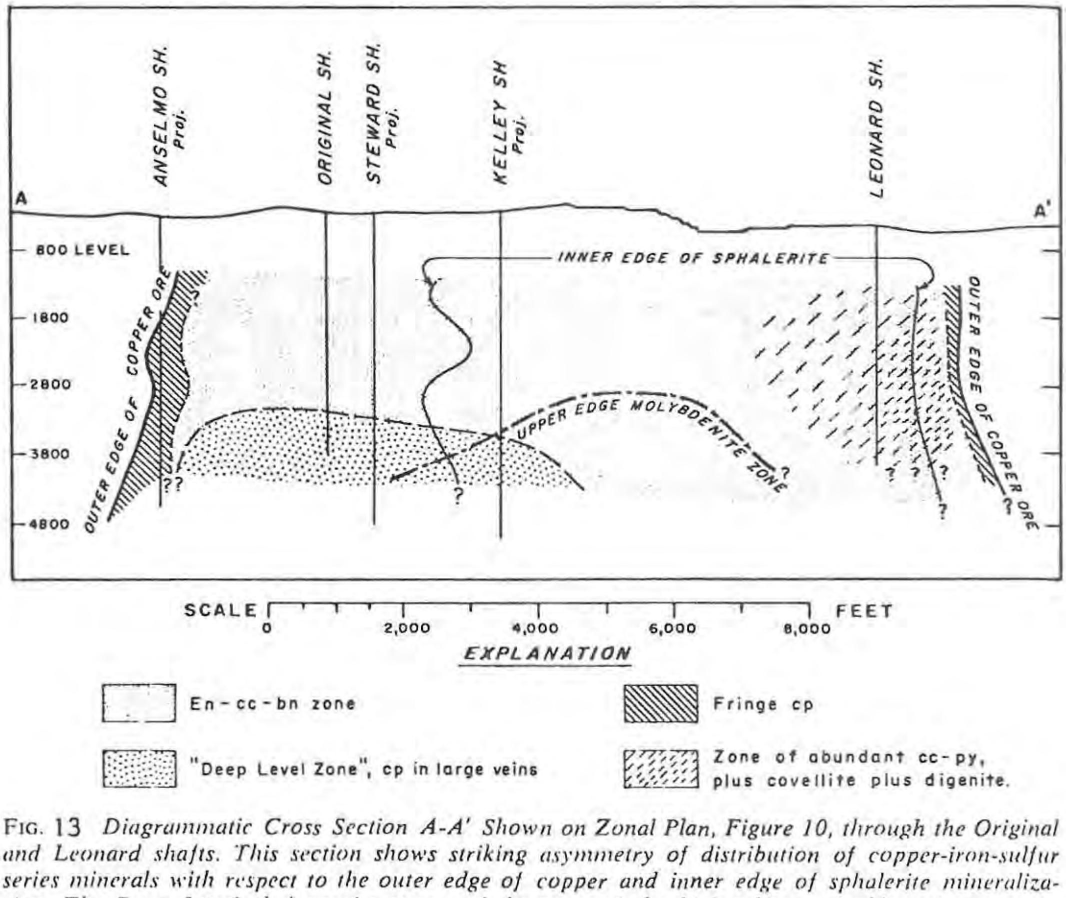


FIG. 13 *Diagrammatic Cross Section A-A' Shown on Zonal Plan, Figure 10, through the Original and Leonard shafts. This section shows striking asymmetry of distribution of copper-iron-sulfur series minerals with respect to the outer edge of copper and inner edge of sphalerite mineralization. The Deep Level chalcopyrite zone and the zone of the high-sulfur assemblage (py, cc, cv, dg) are both within the zone of predominant quartz-pyrite-chalcocite-bornite-enargite ore which is characteristic of most of the copper zone, but they are in different structural positions and both overlap in part with sphalerite. The molybdenite zone is cut by both the sphalerite and deep level chalcopyrite lines.*

(Taken from Meyer et al., 1970.)

In some porphyry deposits the phyllic zones are not well developed and the ore is closely associated with secondary biotitization, e.g. porphyry deposits of the Babine Lake area, Western Canada (Carson and Jambor, 1974). In these deposits the copper-bearing zones and highest copper grades are contained within somewhat larger zones of hydrotherm biotite that are in turn surrounded by wide areas of chloritized rocks (Fig.14).

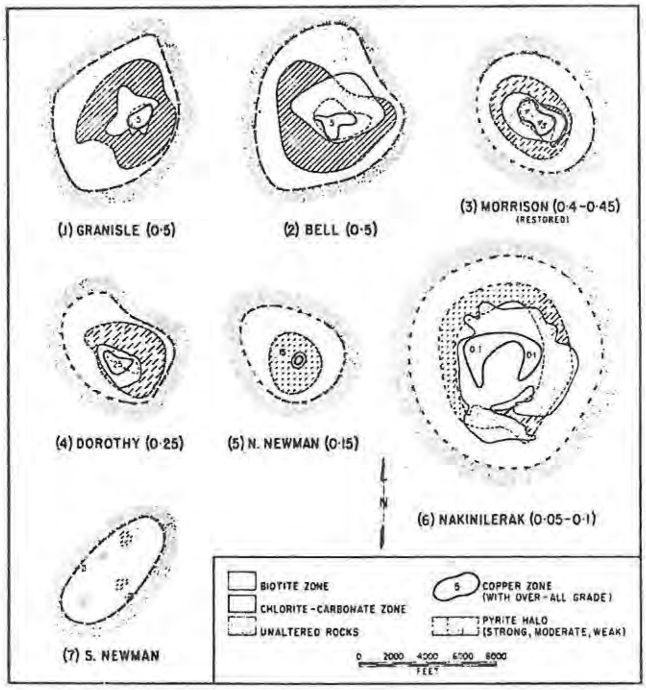


FIGURE 14 — Composite diagram of the Babine deposits showing the relative sizes and grades of the copper zones and associated alteration haloes. The deposits are numbered in the order of decreasing copper grades; note the corresponding decrease in the intensity of the pyrite haloes. The biotite zones contain both weak and intensely developed hydrothermal biotites, and therefore do not include a distinguishing "quality" factor. The North Newman biotite zone is schematic. The outer limits of the Nakinilerak zone are approximate.

(Taken from Carson and Jambor, 1974.)

Carsons and Jambor (1974) also noted that the economic potential (e.g. size and grade) of porphyry deposits in the Babine area corresponds with the areal extent and quality of hydrothermal biotite. Thus, the interior portions of biotite haloes that enclose copper zones exceeding 0.4% Cu, have the features characteristic of good-quality biotitization as shown in Table 3. Biotite in the outer, less copper-rich portions of these haloes is of a poorer quality and is accompanied by increased chloritization (see Table 3). The biotite zone and copper mineralization are very weak and erratic at the lower-grade deposits (e.g. 0.04-0.1% Cu at Nakinilerak).

TABLE 3— Characterization of Hydrothermal Biotites in Terms of Quality

	Good Quality	Moderate Quality	Poor Quality
Colour	deep brown	brown	pale brown to yellowish, pale red, *greenish brown* to *green*
Relative grain sizes (Figs. 17—20)	coarse	medium	fine
Pseudomorphism of mafics	coarse grained, sugary-textured	medium to coarse, sugary-textured	fine grained, commonly with intimately associated fine-grained chlorite
Presence in matrix	well-dispersed in matrix	generally absent	absent
Distribution in thin section	abundant: all amphibole replaced; dispersed in matrix	abundant; all amphibole replaced, but matrix biotite low or absent	variable; if abundant most of the above features are present
Effect on biotite phenocrysts	no effect to replacement of phenocryst edges	no effect	no effect
Synonyms	intense biotitization; strong biotitization	moderately intense biotitization	weak biotitization
Areal abundance and general relationship to Cu grades	present throughout the > 0.3% Cu zones; abundant except at Bell	common within 0.2-0.4% Cu zones	rare in zones containing > 0.3% Cu; common where Cu = 0.1%-0.2%; where Cu < 0.15% nearly all biotite is of this type

(Taken from Carsons and Jambor, 1975.)

Porphyry deposits in the Babine area lacking hydrothermal biotite do not contain significant copper mineralization (e.g. South Newman porphyry). However, the highest grade copper deposits in the Babine area contain sericite alteration in significant amounts in a position roughly similar to that of the phyllic zone of Lowell and Guilbert (1970). Higher grade deposits in this area also have the most distinct annular shapes of the pyritic haloes around the copper-rich core. This is because of their greater pyrite content and their stronger segregation of sulphides; e.g. the Granisle and Bell pyrite haloes are the most continuous and strongest, averaging about 10% pyrite (Fig.14).

At the Lime Creek porphyry molybdenum deposit the molybdenite mineralization forms a ring pattern slightly elliptical in outline. This ring occurs in the transition zone between the quartz-orthoclase and the phyllic alteration zones and also conforms roughly to the contacts of the stocks (Hollister, 1978). The pyritic halo at Lime Creek is external and partially overlapping the molybdenite ring. At Climax, the tungsten/molybdenum ratio increased with each successive hydrothermal event. In addition, tin is associated with the tungsten stage and zone in the Upper Ore Body (Fig.6(b)). At the Panguna (Baldwin et al., 1978), as well as at Chuquicamata, zones of intense quartz-sericite-kaolin-pyrite alteration obliterate locally the earlier K-feldspar alteration and associated lower copper grade. At Panguna (diorite model porphyry) the only late-stage quartz-sericite-pyrite alteration present occurs in structurally controlled narrow seams which do not constitute a well defined phyllic alteration zone.

Some porphyry deposits may also contain economic mineralization outside the main core area, e.g. at the Rio Vivi porphyry copper deposit, Philippines, which contains no recoverable molybdenite, a small alteration zone related to a small stock contains molybdenum rich protore (Cox, in Kesler, 1973). In some porphyry deposits of the diorite model of the Philippines, molbydenum protore is also far outweighed by peripheral gold vein deposits (Kinkel et al., 1956).

Most higher-grade (+ 0.6% Cu) primary porphyry deposits, such as San Manuel (Lowell, 1968), Bingham (Bray, 1969) and Los Pelambres (Sillitoe, 1973), have strong potassic alteration zones. At lower-grade deposits (e.g. Granisle, Panguna, etc.), the potassic alteration, though widespread, is spotty and not as pervasive. The potassic zone in this case is intermingled with either phyllic, propylitic or argillic alteration. Potassic zones at the poorest porphyry deposits (± 0.1% Cu, e.g. Nakinilerak, Santa Rita (Nielson, 1968),) may also contain large patches of fresh-unaltered rock. Most viable porphyry deposits have a stronger and more extensive (300 m or more) prophylitic alteration zone developed at their margins.

In summary, the size, grade and regularity of the mineralization-alteration zones decreases as the intensity of the hydrothermal processes also decrease. In this case, the alteration zones become more diffuse and the grade of the copper becomes more dependent on the composition and textures of the host rocks. Consequently the separation of a "higher grade" from "lower grade" ore zones, such as potassic and prophylitic, becomes less distinct, and the application of selective mining methods and programmes (high-grade ore zones first), become more difficult.

Obviously, the level of erosion of each porphyry system must be considered in the evaluation of all these grade-size-alteration relationships described. Some sub-economic porphyry deposits may be at erosional levels that are either too shallow or too deep in the alteration-mineralization column (Figs.4,5, and 15). Tilting of a porphyry system may also produce an alteration pattern on surface similar to the one indicated in Figure 15 (e.g. the Haib porphyry copper, South West Africa).

Variations in the size and shape of the alteration-mineralization zones can also be produced by different compositions of the pre-ore wall rocks. Porphyry deposits that are essentially wall rock porphyries may impose alteration and mineralization on a host rock with a chemical composition and physical response markedly different from that of a granodiorite or quartz-monzonite intrusion. Different wall rock compositions are therefore obvious obstacles to the development of alteration-mineralization symmetric zones (Lowell and Guilbert, 1974). Chemical and probably thermal gradients are so telescoped at contacts with carbonate sections that sudden precipitation of metal values commonly produce high-grade but narrow skarn zones (e.g. Christmas and Morenci, Arizona, and Santa Rita, New Mexico). At the Christmas porphyry the potassic, phyllic and prophylitic alteration zones are well developed in an elongate quartz-monzonite porphyry stock, but the zones are sharply cut off on either side by substandially richer skarn zones in carbonate sediments into which the stock has intruded. Only rarely is widespread skarn mineralogy produced hydrothermally in carbonate rocks, as at Mission-Pima and Twin Buttes, Arizona.

Mafic volcanic rocks such as basalts, diabases and andesites (Fig. 16) are relatively rich in iron and magnesium and bias the system toward the formation of more mafic alteration minerals, favouring the development of prophylitic alteration zones (e.g. Mantos Blancos porphyry copper deposit, Chile).

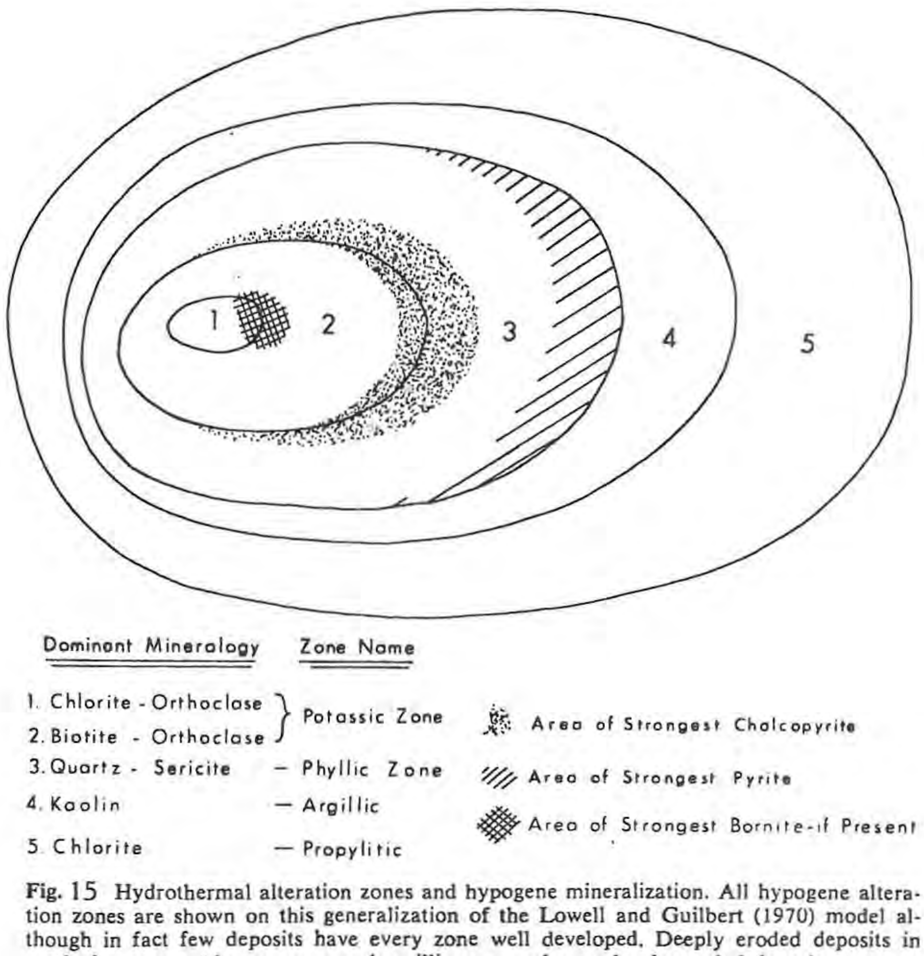


Fig. 15 Hydrothermal alteration zones and hypogene mineralization. All hypogene alteration zones are shown on this generalization of the Lowell and Guilbert (1970) model although in fact few deposits have every zone well developed. Deeply eroded deposits in particular may not have an exposed argillic zone and very deeply eroded deposits may not contain a phyllic zone. Phyllic zones may also be absent from dry deposits. Areas of pyrite and chalcopyrite concentrations are shown where they exist for many, but not all, deposits in this model.

(Taken from Hollister, 1978.)

Figure 17 illustrates the shapes of some Canadian Cordilleran porphyries as well as the characteristic relationship existing between the sulphide zoning, the Cu/Fe ratio, and the copper grade.

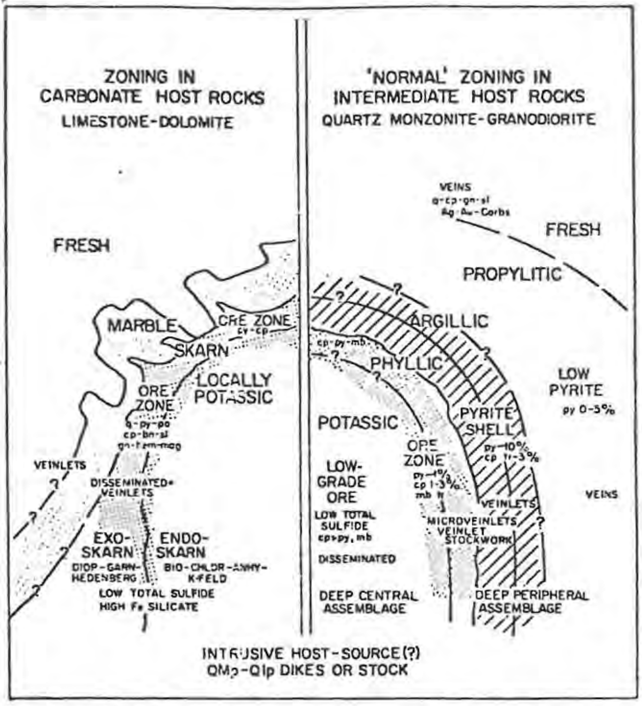
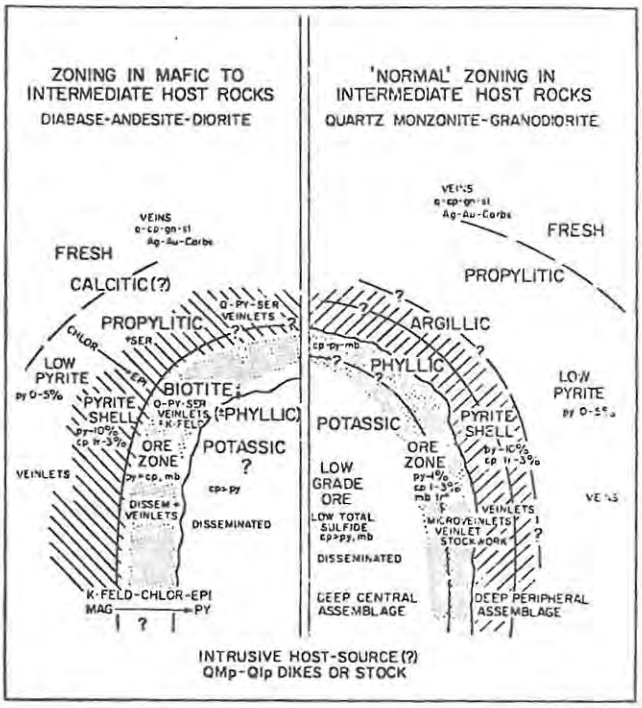


FIGURE 16— Schematic comparison of alteration and mineralization in carbonates and intermediate rocks.



— Schematic comparison of alteration and mineralization in mafic and intermediate rocks.

(Taken from Guilbert and Lowell, 1974.)

b) Relationships of Stockwork Veinlets to Alteration and Mineralization Phases

The presence of abundant hydrothermal veinlets filling the fractures of the stockwork is typical of economic porphyry deposits. The occurrence of sulphides tends to shift from disseminations in the potassic core of the porphyry through to the surrounding alteration zones where microveinlets, veinlets and veins predominate.

The presence of a dense stockwork system and of abundant early and late-stage hydrothermal veinlets and veins is favourable and in direct relation to the protore grade of these deposits, e.g. at the Panguna porphyry, the copper grade rises rapidly where the frequency of quartz-veins increases (Baldwin et al., 1978), at the Lime Creek

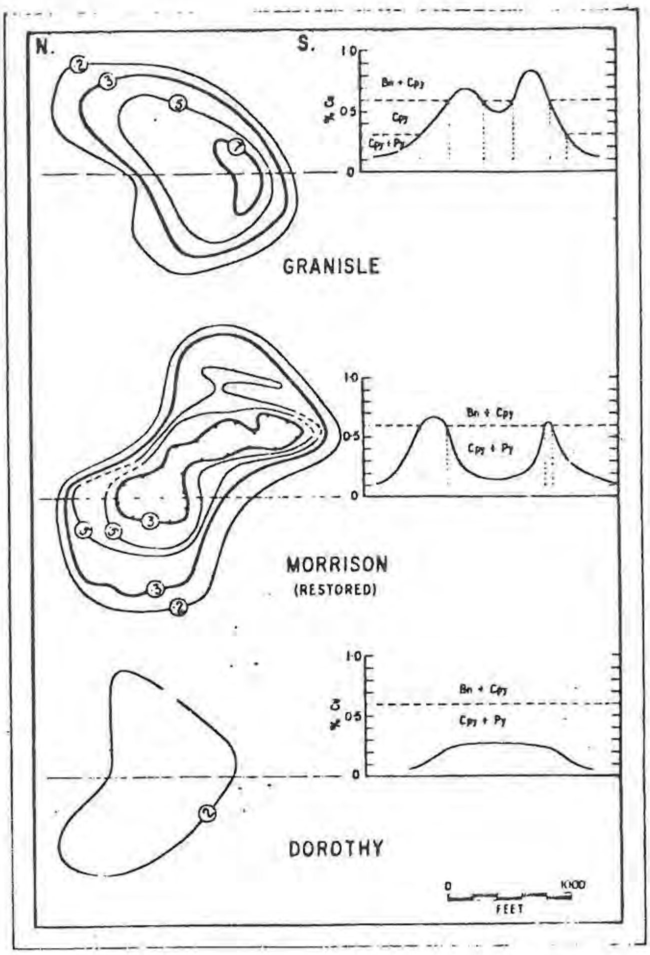


FIGURE 17— Grade zones and grade profiles for copper at Granisle, Morrison and Dorothy. The diagram illustrates the shapes, sulphide zoning and copper abundances of these deposits. The horizontal dashed lines in the grade profiles show the relationship between % Cu and the sulphide mineralogy; the vertically dashed lines indicate that the sulphide zones are steeply dipping. The locations of the 0.5 and 0.7 grade lines at Granisle are based largely on field observations and are therefore approximate.

(Taken from Carsons and Jambor, 1978.)

molybdenum porphyry ore deposit (B.C.) four separate but superimposed substages of molybdenite mineralization have contributed to increase the grade of the protore (Hollister, 1978).

The presence of veinlets of the hydrothermal phase is especially important because they contain a higher proportion of sulphides, either included in the centre line, along their boundaries, or within their associated sericitic or argillic alteration haloes. In the presence of a dense fracture system this will result in a more pervasive hydrothermal alteration of the host rock. A more pervasive and intense sericitic or argillic alteration means an increased porosity of the rock and of precipitation sites for sulphides. Magmatic veinlets on the other hand are more massive like gangue veinlets, usually with no centre line and only with an occasional K-feldspar alteration halo. This halo has no associated increase of porosity in the host rock. The magmatic type of veinlets are poorly mineralized.

Table 4 and Figure 18 illustrate the veinlet sequence and the respective characteristics of the different veinlets at the Endako porphyry molybdenum deposit.

TABLE 4 — Relative Ages of Veins and Envelopes

Stage	Vein	Envelope
1 (oldest)	qz qz-mo qz-mt (± py)	a) Ks b) Ks-bi c) Ks-qz
2	qz-mo minor Ks qz-mt qz-mo qz-mt-mo (all ± py, cp, bn) qz-py (± mo, mt)	(qz-ser-py) (?) qz-ser-py qz-ser-py qz-ser-py qz-ser-py (Pyrite Zone)
3	qz qz-mo and coarse mo qz-mt qz-mt-mo (all ± py, cp)	(No envelopes)
4	qz qz-py	(No envelopes) (Occasional "bleached halo")
5	sp, minor qz calcite chalcedoney	(No envelopes)
6 (youngest)	Late unfilled fractures	

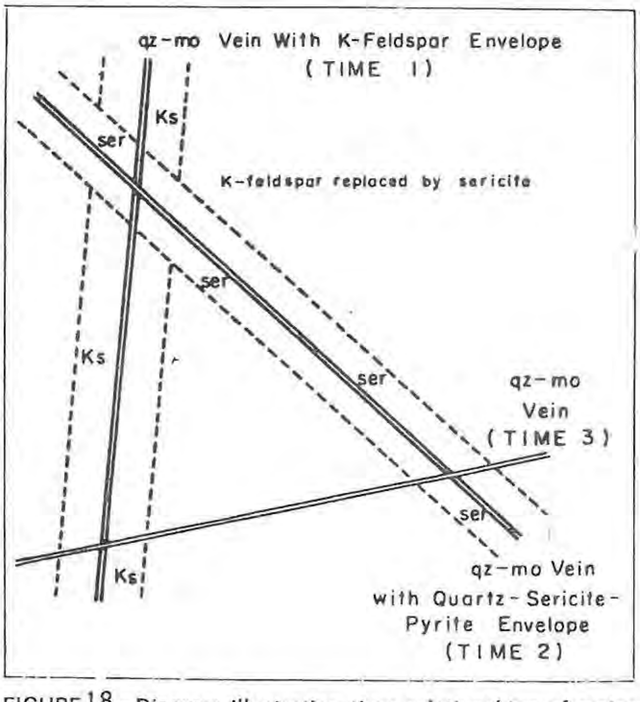


FIGURE 18 — Diagram illustrating time relationships of major veining events during mineralization. (Host rock is pervasively altered outward from veins with or without envelopes.)

(Taken from Kimura and Drummond, 1976.)

In several porphyry deposits criteria for distinguishing the several types of veinlets are difficult to apply because of intergradational relationships (e.g. Butte, Meyer et al., 1970).

1.3.4 Supergene Processes

Supergene processes may have enriched a low-grade uneconomic porphyry copper deposit into an ore body of economic grade, e.g. the enriched ore zone at La Caridad has a copper content of about 3.5 times that of the underlying hypogene mineralization.

Supergene enriched porphyry coppers are particularly associated with regions that have had a semi-arid climate during relatively recent geologic time and have escaped fluvial or glacial erosion, e.g. South-Western U.S.A., Mexico, Peru, northern Chile. Supergene enrichment in most of these deposits occurred mainly during periods of rapid differential uplift of fault blocks, e.g. Morenci (Langton, in Lowell, 1974), La Caridad (Saegart et al. in Lowell, 1974), Chuquicamata, etc. In some porphyry coppers lateral migration of the enriched ore has also taken place into areas not marked by surface evidence (e.g. indigenous limonite). There are also many instances of oxidized copper mineralization occurring some distance from their assumed primary source, e.g. the Exotica oxidized copper ore body in alluvial gravel is located 3 km from the Chuquicamata porphyry. The Sagasca oxidized copper deposit (Chile) is another example.

From the economic and mining point of view, these supergene enriched zones represent ore bodies of higher grade that can easily be mined selectively during the early stages of the operation, thus allowing for a quicker return on the investment.

The geological-geomorphological factors involved in the secondary enrichment processes of porphyry coppers (e.g. amount of pyrite, behaviour of the water table, topography, etc.) are described extensively in Titley and Hicks (1966).

PART 2

2.1 Tonnage-grade Relationships among Porphyry Deposits

Knowledge of the tonnage-grade relationship is useful when specific sets of economic conditions are imposed on geological assessments of tonnage and grade to derive estimates of ore reserves.

Lasky (1950) found that within porphyries the cumulative tonnage increases at a constant geometric rate as the grade decreases anthmetically. The method of using cumulative tonnage could be applied across porphyry deposits, and the results would be consistent with Lasky's results (J. Whitney, in Singer at al., 1975). Lasky's cumulative contained copper curves become flat at the zero cut-off grade. Zero cut-off grade represents the outer limit of the effects of the special geologic conditions that gave rise to the ore deposit. Copper content at this limit tends to approach average abundance of the country rocks.Several recent studies however indicate that Lasky's rule only applies to the medium-grade porphyry copper deposits, for which it was initially formulated. Figure 19 shows the exponential (Lasky's) distribution of ore reserves and metal content as function of cut-off grade and average grade of ore reserves. These curves illustrate the following exponential relationships:-

$$R(G) = Ae^{-G/K}$$

$$M(G) = KAe^{-G/K}(1+G/K)$$

$$Gavg\ (G) = M(G)/R(G) = K + G$$

where

G = cut off grade

R(G) = tonnage of ore reserves above cut off grade G

M(G) = metal content in ore reserve above cut off grade G

Gavg(G) = average grade of ore reserves above cut off grade G

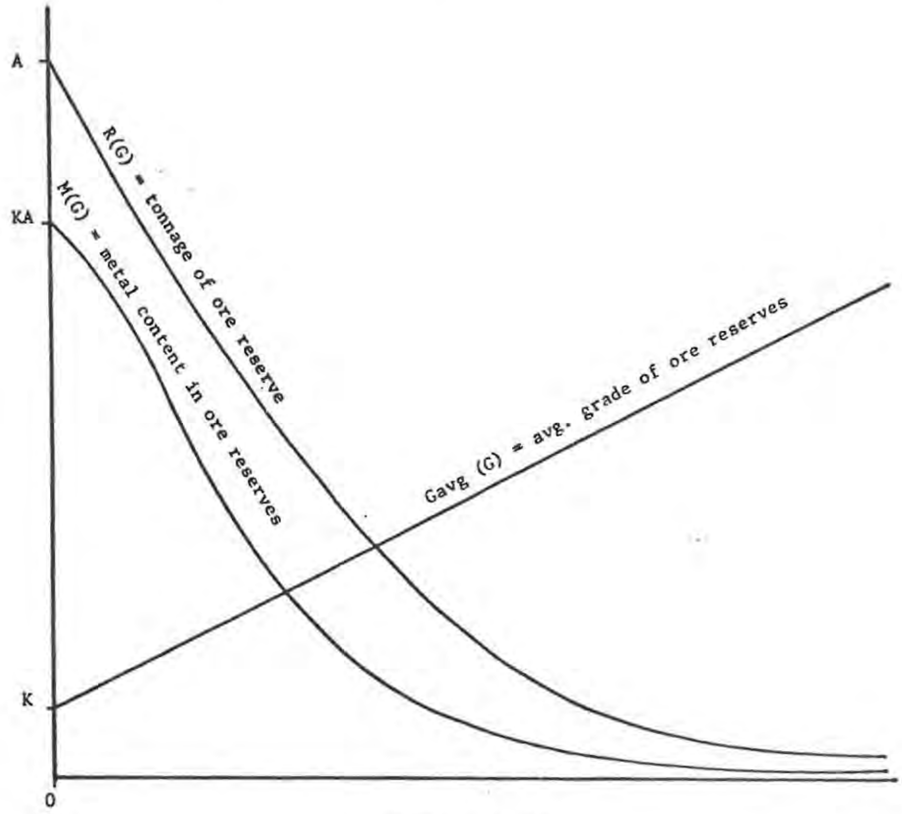
A = total deposit tonnage; R(O)=A

K = average grade of total deposit

e = 2.718 (base of natural logarithm)

Figure 19

EXPONENTIAL TONNAGE-GRADE RELATIONSHIP



(Taken from Mackenzie, 1979.)

The following table taken from Mackenzie (1979) illustrates the application of hte exponential tonnage-grade relationship for a hypothetical hypogene porphyry copper deposit containing 200 million tons (A) with an average grade of 0.5% Cu (K).

Cut-off Grade (% Cu)	Ore Reserves (tons)	Metal Content (tons)	Average Grade (% Cu)
0.00	200,000,000	1,000,000	0.50
0.50	73,580,000	735,800	1.00
1.00	27,070,000	406,000	1.50
1.50	9,960,000	199,100	2.00
2.00	3,660,000	91,600	2.50
2.50	1,350,000	40,400	3.00
3.00	496,000	17,350	3.50
3.50	182,000	7,290	4.00
4.00	67,000	3,020	4.50

The tonnage-grade relationship as conceived above or as otherwise estimated from assay frequency distribution is a useful construct for certain types of policy analysis. However, it is important to realize that such a relationship does not show where the high-grade reserves are nor does it indicate the boundaries between ore and waste. Only the overall content of the deposit is known. Therefore, this type of relationship is of little use for mine planning. For this purpose, more practical estimates of tonnage and grade must be made for each possible mining block.

The porphyry population was also analysed by the U.S.G.S. According to this study it would appear that both tonnage and grade have a geologic upper limit and an economic lower limit changing with time. A wide scatter of tonnage at each level can be expected for lower-grade levels. This means, however, that total metal content of lower-grade deposits will also be lower than for deposits known to date.

The frequency distributions of the different grade ranges within porphyry deposits are also distinct:

(i) low-grade assays within porphyry deposits are generally positively skewed

(ii) medium-grade assays tend to be normally distributed

(iii) higher-grade assays in porphyries are negatively skewed (Mackenzie, 1979).

The majority of assay distributions of porphyry deposits are low-grade and positively skewed, the lognormal distribution providing in this case an approximation (Fig.20). If these assay values are expressed in terms of logarithms, the frequency distribution of the logarithm values of assays within the logarithmic grade intervals becomes a normal distribution, and a plot on log-probability paper of the cumulative logarithm values of these assays would give a distribution like the one shown in Fig.21.

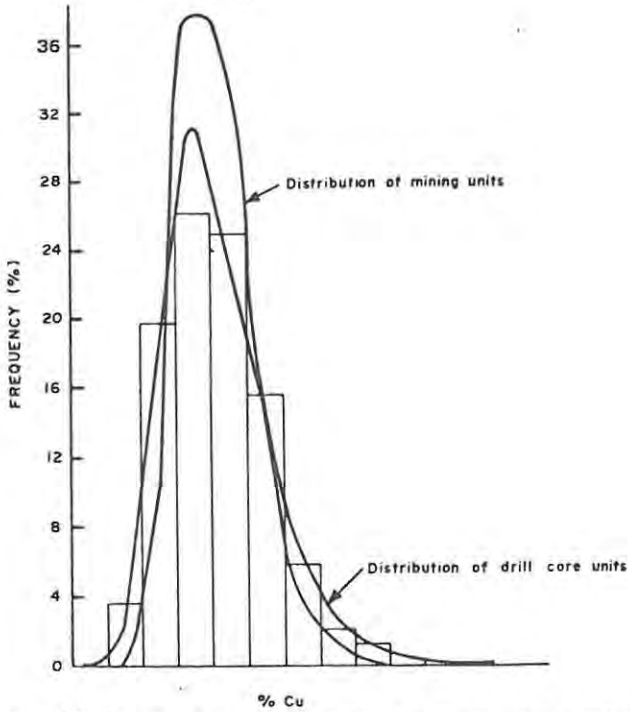


Fig. 20. Distribution of the grade of drill core units and of mining units in the Bougainville porphyry copper deposit. After Blackwell (1972).

(Taken from David, 1977.)

Such transformations are useful for the statistical analysis (confidence limits, error of estimation etc.) of these data.

The estimated copper content in individual deposits of the Cu-Mo and Cu-Au porphyry models (see chapter 1.2) is shown in Figure 22. Because the classes in Figure 22 are divided exponentially, improbably high errors in size estimates would be necessary to change the forms of the distributions. Comparisons of the two populations in Figure 22 indicate that the upper contents as well as the tonnages of the two models differ at the 95% confidence level.

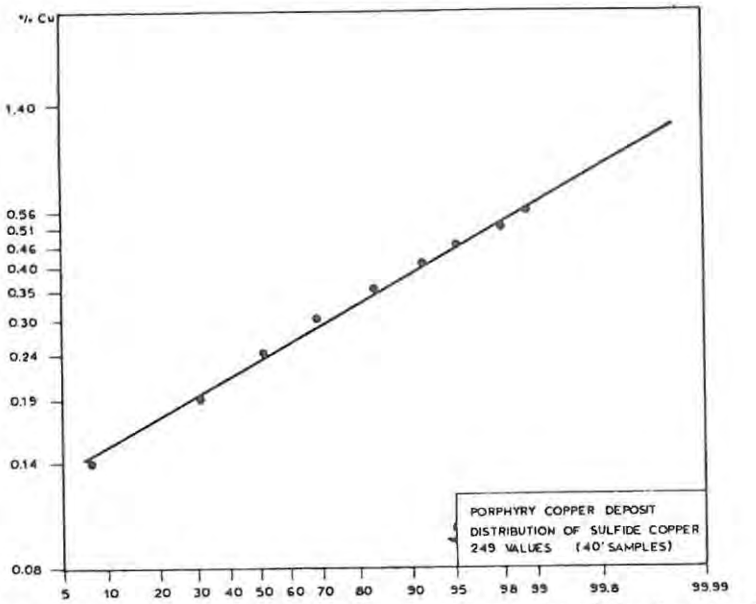


Fig.21 . Plot on log-probability paper of the cumulative distribution of the sulfide copper grade of 249 40' samples in a porphyry copper deposit.

(Taken from David, 1977.)

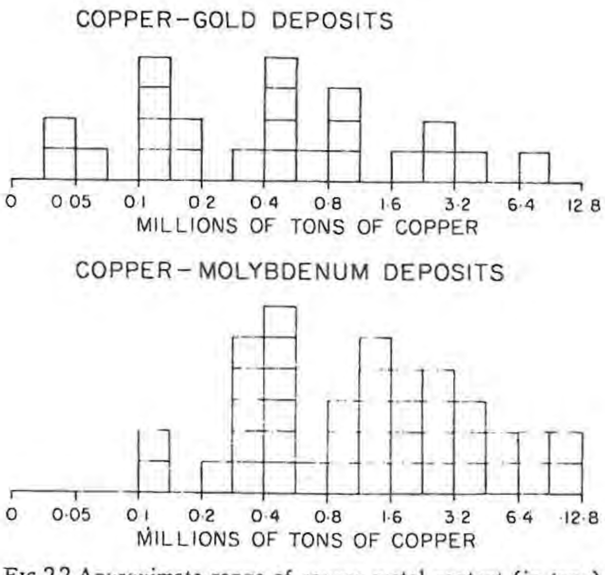


FIG.22 Approximate range of copper metal content (in tons) for copper-molybdenum and copper-gold deposits.

(Taken from Kesler, 1973.)

It can be concluded that the average Cu-Au (Diorite Model) porphyry is smaller than the Cu-Mo (Lowell and Guilbert Model) porphyry, although to what extent this difference can be attributed to the longer time period over which most Cu-Mo deposits have been explored and operated is not known (Kesler, 1973).

A log-log graph of copper content vs. molybdenum content for porphyry copper deposits and porphyry molybdenum deposits of North America also showed distinct separation into two populations with very few deposits having transitional grades (Hollister, 1978). Outstanding differences also exist for the ratio between other metals and molybdenum. The tungsten/molybdenite content for stockwork molybdenite deposits is much greater than that for most porphyry copper deposits. The rhenium/molybdenum ratio for stockwork molybdenite deposits is less than one-fifth that for porphyry copper deposits.

Singer et al. (1975) used univariate statistical analysis to determine which theoretical distribution models best describe the distributions of the average grade and tonnage of 103 porphyry copper deposits. He recognized 4 subdivisions based upon the metallogenic province within which these deposits occurred (see Tables 5 and 6).

Within the porphyry deposits, the average grade ranges from 0.49% Cu for the Canadian deposits to 0.99% Cu for the South American deposits. The average grades for the United States-Mexico deposits (0.59% Cu) and the south-west Pacific deposits (0.52% Cu) are close to the average grade of the Canadian deposits.

The porphyry deposits contain an average of 548 million tons of ore. Within the four sub-classes of porphyry deposits, the median metal content ranges from 608,000 tons for the south-west Pacific to 3,214,000 tons for the South American deposits. Singer et al. (1975) indicates that with the exception of the south-west Pacific porphyry deposits, the lognormal distribution model is superior to the normal distribution in describing the observed distribution of mean grade

TABLE 5—*Summary statistics for the average grades of copper deposits*

Type of deposit	Number of deposits	Arithmetic data: Mean grade	Arithmetic data: Standard deviation	Arithmetic data: Median grade	Logarithmic data: Mean, $\log_{10}$ of grades	Logarithmic data: Standard deviation, $\log_{10}$ of grades
Porphyry:						
Canada	21	0.0049	0.0017	0.0047	−2.331	0.147
United States and Mexico	38	.0059	.0014	.0057	−2.243	.101
South America	20	.0099	.0036	.0093	−2.033	.160
Southwest Pacific	24	.0052	.0012	.0051	−2.296	.114
World	103	.0063	.0027	.0059	−2.233	.162
Massive sulfide	146	.0292	.0235	.0229	−1.640	.303
Strata-bound	18	.0378	.0118	.0362	−1.442	.132
All	267	.0210	.0213	.0140	−1.855	.392

TABLE 6—*Summary statistics for the tonnage of copper deposits*

Type of deposit	Number of deposits	Arithmetic data: Mean (millions of metric tons of ore)	Arithmetic data: Median (millions of metric tons of ore)	Arithmetic data: Mean (thousands of metric tons of copper)	Arithmetic data: Median (thousand of metric tons of copper)	Logarithmic data: Mean, $\log_{10}$ of metric tons	Logarithmic data: Standard deviation, $\log_{10}$ of metric tons
Porphyry:							
Canada	21	245	177	1,210	824	8.247	0.354
United States and Mexico	38	815	338	4,781	1,932	8.529	.629
South America	20	773	347	7,622	3,214	8.540	.610
Southwest Pacific	24	203	120	1,058	608	8.080	.436
World	103	548	234	3,452	1,368	8.369	.565
Massive sulfide	146	10.3	2.26	301	52	6.354	.828
Strata-bound	18	91	41.4	3,453	1,496	7.617	.690
All	267	223	16.5	4,679	230	7.217	1.208

(Taken from Singer et al., 1975.)

and tonnage. The degree of association or interdependence of the tonnage-grade variables in these porphyry deposits were also examined by means of the correlation coefficient by Singer et al. (1975). The sample correlation coefficients were first tested against the "null" hypothesis in order to determine if the population of grades and tonnages has a zero correlation (complete linear independence) (see Table 7).

TABLE 7 –*Correlation coefficients between average copper grade and total tonnage*

[NS, not significant; **, significant at the 1-percent level]

Type of deposit	Number of deposits	Correlation coefficient: Arithmetic data	Correlation coefficient: Logarithmic data
Porphyry:			
Canada	21	−0.16NS	−0.22NS
United States and Mexico	38	.00NS	−.09NS
South America	20	−.07NS	−.17NS
Southwest Pacific	24	−.05NS	−.07NS
World	103	.09NS	.05NS
Massive sulfide	146	−.13NS	−.42**
Strata-bound	18	−.10NS	−.19NS
All	267	−.22**	−.67**

(Taken from Singer et al., 1975.)

Sample correlation coefficients calculated for both the arithmetic data and the logarithmic data for each group of porphyry coppers and for all the porphyries together provide no basis for rejecting the null hypothesis of zero correlation between average grade and tonnage (Fig.23). In fact, less than 5% of the variation of tonnage is explained by the average grade for these deposits. This independence of grade and tonnage for porphyries is therefore convenient for modelling of grades and tonnages of yet unmined deposits. As the grades and tonnages are essentially lognormally distributed for the porphyries, the calculation of the probability of the occurrence of any particular range of grade or tonnages, given that a porphyry exists, is an easy task.

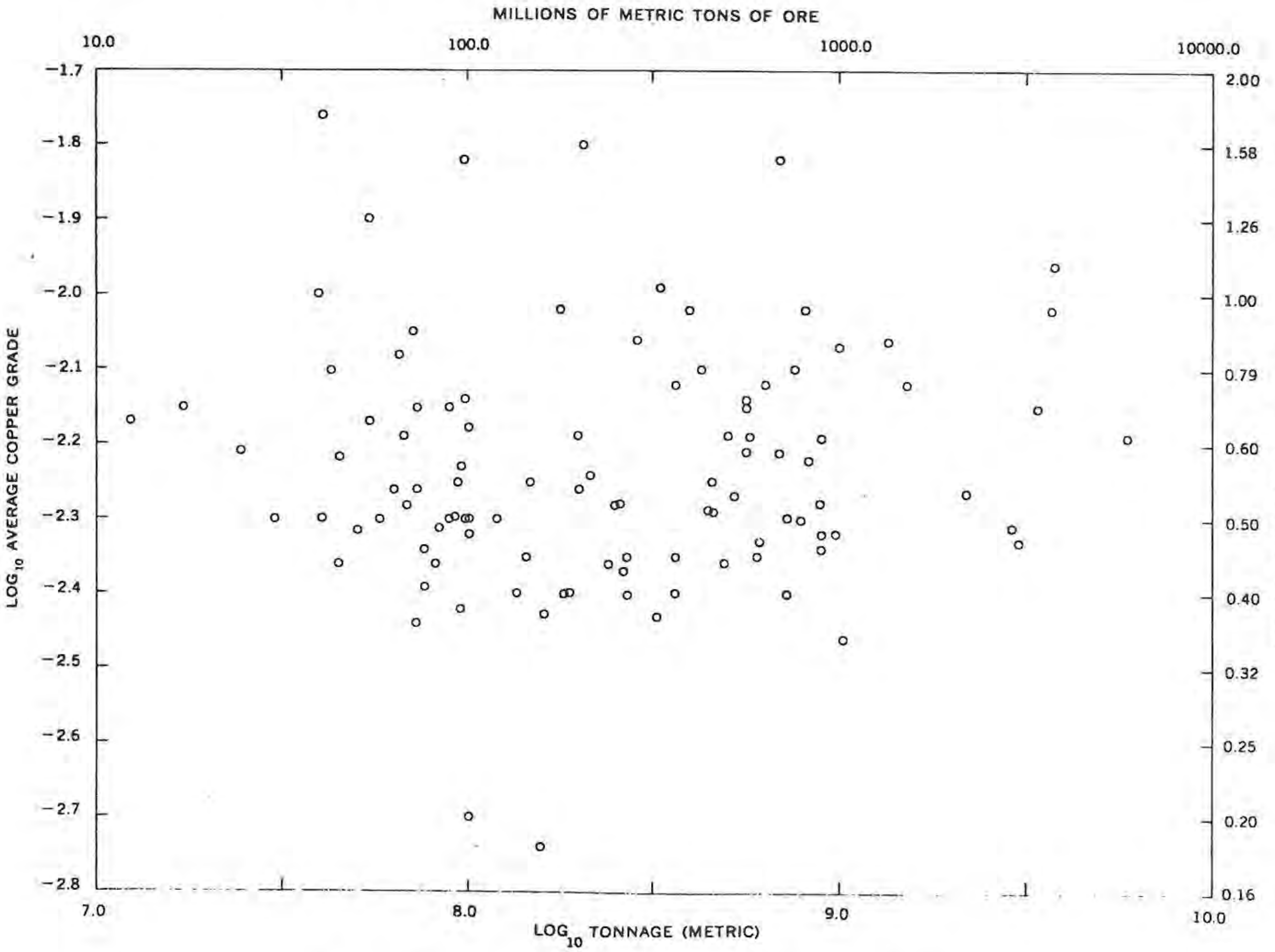


FIGURE 23-Grades and tonnages for porphyry copper deposits.

(Taken from Singer et al., 1975.)

Because of the independence of grade and tonnage, the probability of a deposit falling in particular ranges of grades and tonnages is simply the product of the probabilities of the range of grades and of the range of tonnages. The probability of a very low-grade deposit, say less than two standard deviations below the mean (less than 0.28% Cu) is equal to 0.023 (2.3%) from tables of the areas of the normal curve. The probability of a very large tonnage, say more than 3 s.d. above the mean (greater than 11×10^9 tons), is 0.0013. Thus, the probability of a porphyry having a grade less than 0.28% Cu and a tonnage greater than 11×10^9 tons is $0.023 \times 0.0013 = 0.00003$; that is, if 33,000 deposits were discovered, only one of them would be likely to have these characteristics.

If improvements in technology allow mining of lower and lower-grade porphyry deposits, the most probably tonnage of these new deposits will be roughly the same as those presently being exploited; thus, each deposit will contain less copper.

The lack of correlation of tonnage and grade for the porphyry deposits has implications not only for quantitative modelling, but also for qualitative modelling. Qualitative modelling is often used for future supply studies of natural resources.

2.2 Delineation of Porphyry Deposits

The discovery of positive indications of mineralization and mineable widths on a surface outcrop or by exploratory borehole provides justification to initiate a preliminary delineation program. The objective of such a program, at this early stage of discovery, is to estimate the horizontal and vertical extent of mineralization. Diamond drilling (and also trenching, when the mineralization outcrops) is generally used at this stage because of the relative ease, speed and low cost involved for obtaining the required information. As soon as the continuity and extent of the mineralized mass is verified, preliminary computations determine whether or not the deposit can be profitably mined. Very crude estimates of grade, tonnage, commodity price(s) and capital, mining and processing costs are combined to evaluate the profitability of the exploration project. Since the amount of information concerning the deposit is too sketchy to assess the reliability of geological variables used in the evaluation, experience and intuition guide the decision to proceed to the next step, an initial delineation program.

The purpose of the delineation stage is to provide reliable estimates of the mineral deposit's physical characteristics (shape, size, mineral content, grades, mineralization trends, etc.) for use in a feasibility study. For this purpose, a drilling grid must be established. Previously drilled exploratory boreholes may be used as a framework and the grid tailored to the preceived shape and attitude of the deposit. Porphyry deposits are normally delineated by vertical or subvertical boreholes, drilled on a regular square pattern, but anisotropy in the distribution of the mineralization (e.g. Chuquicamata) or certain structural controls of the mineralization (e.g. if mineralization appears predominantly in a certain system of fractures) and topopographic features may warrant some changes in the grid pattern and attitude of the boreholes.

Under favourable geologic and drilling conditions, the diamond drill cores should give fairly good and precise information about the abovementioned characteristics of the deposit. The core data of the different holes drilled can then be statistically and/or geostatitically analysed to determine the expected value of the various physical characteristics of the mineral deposit as well as the reliability of these estimates. In practice, most delineation programs are terminated when the deposit is believed to be sufficiently proven.

2.2.1 Geological Factors affecting the Selection of a Drilling Grid

a) Size and shape:

The large circular or elliptical mass of porphyry deposits can be delineated by vertical boreholes set on a square grid pattern (e.g. 100 m grid at the Sar Cheshmeh porphyry, Iran; 75 m at the Haib porphyry, South West Africa; 400 ft. at the Panguna porphyry, Bougainville). As the mineral content gradually decreases towards the outer limits of mineralization (e.g. towards the prophylitic alteration zone), it is not necessary to locate the boundaries of the geological reserves. The horizontal extent of the drilling grid is therefore limited by some form of minimum possible cut off grade (e.g. 0.4% at Sar Cheshmeh, 0.35%% Cu at Haib). If mineralization does not decrease and eventually terminate with depth, the vertical extent of the boreholes will be limited by open-pit mining constraints as reflected by operating cut-off grade (e.g. approximately 350 m of depth at Haib, 500 ft. at Sar Cheshmeh). Operating cut-off grade raises as ore is mined down to the deeper levels in the pit; this is because mining costs and the stripping ratio increase with depth. Thus the bottom of the pit is set at the level where the mineral content of the deposit (if already known, or if it is broadly estimated) equals the operating cut-off grade.

At Sar Cheshmeh the holes were drilled vertically because the supergene zone is essentially subhorizontal, so giving the maximum number of intercepts for a given total footage. However, vertical holes are not the best for delineating steep sided dykes, of which numerous exist at Sar Cheshmeh. Nevertheless, sufficient ore tonnage was not a problem at at this deposit, whereas accuracy of grade estimation was much more important to the project (Selection Trust Ltd., 1970). During mining however these dikes may strongly dilute the ore and the mill head grades may be much lower than expected and/or the actual ore tonnage may be lower than the expected because of misplacements of the geological contacts (e.g. Panguna; Baldwin et al., 1978).

b) Stockwork-veinlet System

Porphyry deposits are frequently explored by drilling a regular grid of vertical holes, into what is a deposit with normally a sub-vertical fracture-veinlet system (Dixon, 1979). The assumption made is that since there are many mineralized fractures, there will be a good statistical chance that the "misses" will be balanced by the "hits". This assumption may be valid over the whole drilling program, e.g. the expected grade of the resource based on all the exploration sampling will be an unbiased estimate of its natural grade. However, if one examines borehole sample data from porphyry coppers in relation to production records it is often found that the drill-hole sample grades tend to over-value high-grade and under-value low-grade ore. This is what would be expected because low-grade porphyries tend to have less numerous mineralized fractures and there is a lower probability that the drill holes will score "hits" in the right proportion if the drill hole and sample spacing remains the same. Under-valuation can more easily occur in the "tectonic" type of stockworks, because the veinlets tend to follow a preferred joint orientation; e.g. the Haib porphyry, South West Africa. There can also be two preferred fracture and veinlet orientations but only one of which is more obvious in surface outcrops. Normally, in such a case a pattern of inclined drill holes is then set out to give a good intersection angle; the samples taken from the holes will give a good report of the grade of mineralization in the known more obvious fracture system, but it may be that a different proportion of the mineralization (e.g. different fracture density and/or different type of hydrothermal veinlets-richer or poorer in metal content) is contained in the other fractures that are orientated very unfavourably in relation to the boreholes.

Differential core loss, due to geological features, can also produce under-valuation of stockwork-veinlet deposits, e.g. in these deposits, core tends to break along fractures, and if these are mineralized even with high core recoveries, there will be a higher loss of the softer, sulphide minerals than of the silicates. In porphyry deposits the softer, late-stage - more intensely mineralized hydrothermal veinlets with a centre line - will be the ones contributing most to the loss of sulphides. To some extent the under-valuation of the stockwork deposits produced due to this differential core loss can be remedied by sludge sampling but intense hydrothermal alteration can leave the host-rock too porous and so sludge samples can become unreliable.

From the above analysis one might be tempted to think that it would be better to explore hypogene porphyry deposits - with no supergene enrichment zone - by inclined or even horizontal drill holes. Unfortunately this usually turns out to be more expensive or even impossible because of the topography.

c) Effect of Scale

Another geological effect to be considered in the selection of the drilling and sampling grid is the effect of scale.

Many porphyry deposits show grade variation patterns that are anisotropic (e.g. Chuquicamata); and so are many commonly used sampling schemes applied at these deposits.

At Chuquicamata most diamond drilling from the pit is inclined (approximately 43-47%) and perpendicular to the main N-S trend of mineralization. If there is a preferred orientation in the feacture system of a porphyry, then the apparent reserve/grade relationship and the block uncertainties will be different depending on which direction the samples are taken. A common manifestation of this effect is in circular open pits where it is often found that it is much harder to hold grade on one part of a bench than on another part that has a different strike (Dixon, 1979). In some porphyry coppers with a supergene enrichment a subhorizontal fabric is also imposed on the subvertical fabric of the hypogene mineralized zone, e.g. Sar Cheshmeh, Chuquicamata, La Carídad. In this case these deposits must be sampled, in such a way that all the directional effects can be allowed for in planning a drilling program. The oxidized zone in the porphyry copper deposits presents the greatest anisotropy in the vertical direction. In the supergene zone there is a combined effect of the vertical distribution of the hypogene mineralization and the subparallel disposition of the reduction surface. The hypogene zone presents the greatest anisotropy in the direction perpendicular to the main structures and the vertical range in maximum. Drill holes in the oxidized zone are therefore generally drilled vertically; holes in the supergene enriched zone are more conveniently drilled inclined (e.g. Chuquicamata) and holes in the primary zone are recommended to be inclined and parallel to the maximum anisotropy.

The different types of variations in the geological fabric of a porphyry deposit can be shown where a tunnel through such a deposit is sampled. Suppose a few, widely spaced samples are taken, starting in the feebly mineralized propylitic alteration zone and then in the other alteration zones. A jump in grade probably would occur somewhere at the outer limits of the sericitic zone, and the grade would generally rise and then fall away again as the opposite propylitic alteration zone is reached (see Fig.17). However, if this tunnel is sampled on very close intervals (e.g. 1 m or less) a random mixture of very high and low-grade samples will be obtained, depending on whether the samples comprised mineralized vein or inter-vein material. Geostatistics is of great use for analyzing the scale of sampling, the grade distribution and ore trends. Geostatistics can indicate from the geological point of view, the optimal spacing of the drill hole grid. This optimization provides the exact number of samples needed to represent the grade and dimensions of an orebody. Optimization is easy if the zone of influence intended for each sample in ore-reserve calculations could be demonstrated to be the actual zone of influence in nature. Zones of influence are often related to adjacent samples on a geometric basis; yet if two adjacent samples cannot be correlated at some acceptable confidence level neither one can really be expected to have an actual and measurable influence in the zone between them. Where adjacent samples show a strong correlation, sampling is adequate, a zone of influence can be assigned, and further sampling would be a waste of effort and money. Some approaches to finding actual zones of influence around sample locations use statistical methods such as correlation coefficients, the mean square successive difference test or the variogram function, which is a geostatistical method.

(i) Statistical procedures: these are used in the early period of delineation to estimate deposit characteristics and assess their reliability. In this early period of delineation, the existing grid dimensions are greater than the distance of influence of each sample and only overall deposit characteristics can be considered for estimation purposes. Since no part of the deposit can be estimated, cut-off grade as well as mine sequencing cannot be analyzed. Thus, geological estimates are limited to overall average grade and total deposit tonnage (Mackenzie, 1979).

The use of correlation coefficients to control the number of sample locations in a large disseminated porphyry orebody is shown in Figure 24.

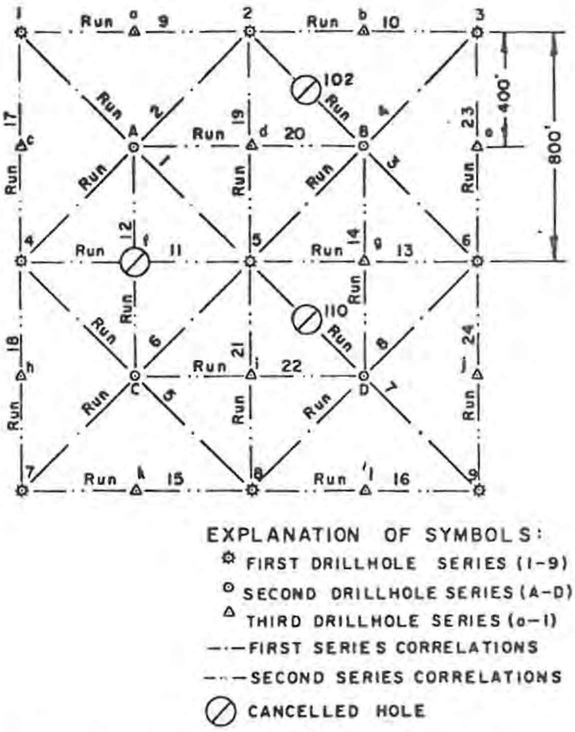


Figure 24 Drill-hole pattern for large disseminated orebodies, based on a method developed at the Christmas mine, Arizona. (From J. G. Kuhn and J. D. Graham, "Application of correlation analysis to drilling programs: a case study—technical notes," *Transactions*, v. 252, no. 2, Fig. 1, p. 142. Copyright © 1972, American Institute of Mining Metallurgical and Petroleum Engineers, Inc. Used by permission of the publisher.)

(Taken from Peters, 1978.)

In this example geologic information and experience were used to design a drilling program on a 122 m square grid. Grade estimates were needed at a precision of 0.08% Cu. An initial series of holes (1-9) was drilled on 244 in centres, twice the designed grid size. The assays were composited to standard intervals in each hole, and a linear correlation between the data was sought along diagonals in statistical runs 1 to 8. No significant relationship was found, so a second series of holes A-D, was drilled. In run 11, a suitable correlation was found between holes 4 and 5 in run 11; an acceptable average confidence interval of $\pm$ 0.08% was also calculated for the two holes. On this basis, hole f was cancelled; after drilling the rest of the holes in the series new statistical runs were used to test the need for the holes in the centre of each 122 m square.

Diagonal runs indicated significant correlations between holes B and 2 and between d and b; hole 102 was cancelled. If further evaluation had been considered on the basis of information from the completed pattern, the statistical process could have been carried down to a 61 m grid scheduling only those holes actually needed.

Although classical statistic procedures can determine and evaluate the directional trends and area of influence of the samples, the spatial distribution of these variables is not considered.

(ii) Geostatistical procedures; the variogram function: this function deals with regionalized variables that have specific distance and directional characteristics. This function can be used once the geological data base has become large enough to require the estimation of spatial correlation in the deposit, the average grade and estimation variances of mining blocks, etc. This technique is therefore used in the late stages of the delineation process, and to obtain accurate ore-reserve calculations.

The incorporation of cut-off grade, enabling the derivation of a grade-tonnage relationship, is valid at this stage. This optimization technique, continued with sequencing and selective mining policies will improve the profitability of the different mining investment alternatives in a deposit.

The theoretical formulation of the variogram function is described by many authors (e.g. David, 1977; Royle et al., 1974; Davis, 1973; Matheron, 1972) and will not be presented here. Figure 25 illustrates one of the basic ideas in geostatistics. Case I and Case II have the same range in values, but in Case I the values are highly erratic and in Case II they are symmetrically distributed around a high-grade centre. In terms of classical statistics, the two populations would be identical, they have the same average grade and the same standard error of the mean and would give the same frequency histogram. But the two cases would certainly present different problems during the evaluation and mining. In Case I there is no specific change with distance, on the other hand in Case II there is a

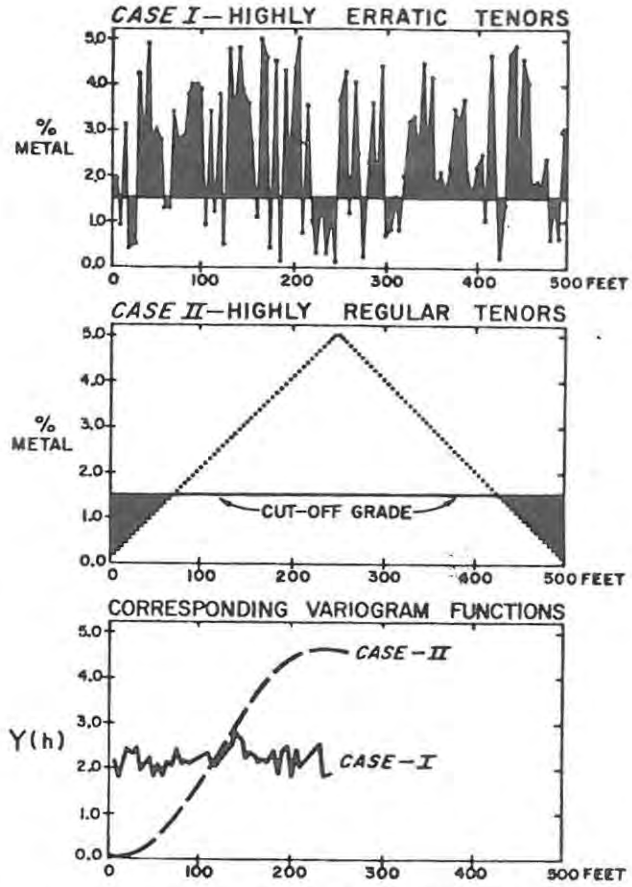


Figure 25 Silhouette diagrams of two vastly different populations of identical assay values, with corresponding variogram functions. (From R. A. Blais and P. A. Carlier, "Applications of geostatistics in ore evaluation," CIM special volume No. 9, *Ore Reserve Estimation and Grade Control*, Fig. 2, p. 44, 1968. Used by permission of the Canadian Institute of Mining and Metallurgy.)

(Taken from Peters, 1978.)

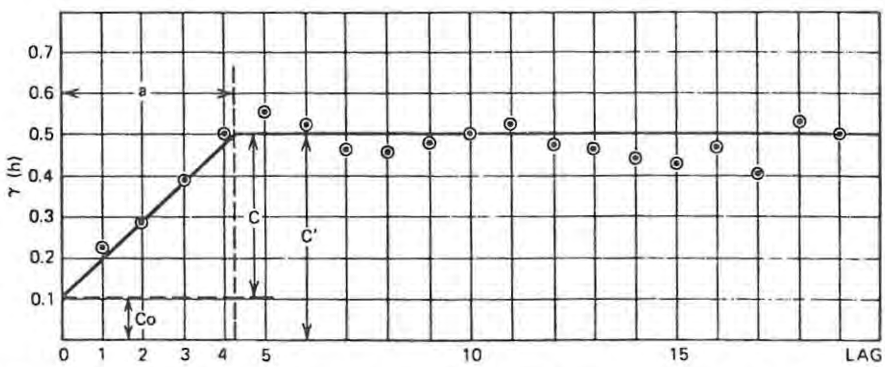


Figure 26. Variogram from a large stratabound Canadian uranium deposit. The regionalized variable is the tenor in uranium along a line of drill holes at 15-m intervals. The range (a) of the variogram is at a lag of 4.2 (63 m). The significant geologic dimension is thought to be the width of a buried channel. (After R. A. Blais and P. A. Carlier, "Applications of geostatistics in ore evaluation," CIM Special Volume No. 9, *Ore Reserve Estimation and Grade Control*, Fig. 15, p. 55, 1968. Used by permission of the Canadian Institute of Mining and Metallurgy.)

(Taken from Peters, 1978.)

regular rate of change. Case I would need a large number of drill holes for adequate sampling, regardless of the statitical treatment; Case II, which uses the geostatistical approach, would need only three. The variogram function of Figure 26 changes in a specific direction as more sample values are added until at a certain distance, called the "range" of the variogram, the influence of the closer samples is lost and the plot approaches an independent (random) behaviour. The optimum spacing between drill holes or sample locations in this particular direction is indicated by the range of the variogram. Samples taken at a greater distance would miss the significant correlation; samples taken at a closer distance would show the correlation but would be unnecessarily close.

In practice, however, the pattern of drilling is a function of the drilling method, the accessibility of the site, the geologic situation, the mechanical characteristics of the rocks, the objectives of the project, the requirements for statistical analysis and the costs of drilling. It is generally a compromise between what is preferable and what is convenient or economical (Peters, 1978); e.g. at the Sar Cheshmeh porphyry copper deposit the diamond drilling programme was planned to provide a grid of vertical holes over the area of economic mineralization. This was undertaken in three stages, commencing with a primary grid of holes at 200 m centres - the "200 m grid". In the second stage, holes were drilled at the centres of these 200 m squares - designated the "141 m grid". For the third stage the outstanding holes to complete a 100 m grid network were drilled, but the second and third stages were not completed in the central portion of the deposit which consists mainly of low-grade porphyry dyke. On completion of the grid drilling, an infill series of irregularly spaced vertical and inclined diamond drill holes was drilled to provide specific local information on grade and dyke position and thickness (Selection Trust Ltd., 1970). In general, a 50% change in grade between adjacent holes on the 100 m grid was the limit accepted; more than that called for infill investigation.

2.2.2 Factors Affecting the Selection of a Drilling Technique

Drilling is an important factor in the evaluation of large size deposits like porphyry deposits. It is the principal tool used in obtaining reprsentative samples for ore reserve estimations. Detailed, three dimensional sampling of the target nearly always consists of the same combination of methods; but the techniques used are adap ted to the degree of consolidation of the material to be sampled, to the topographic and climatic conditions, to the chemical and physical properties of the valuable mineral (Cu, Mo, Au, W, Sn, etc.) sought, the sample and information requirements, the geometry of the orebody, and depth to be drilled (Bailly, 1976).

Among the available methods in exploration drilling, three are by far the most popular: diamond core drilling, rotary drilling, and percussion drilling. The principal features of these and other methods are given in Table 8.

Table 8 Exploration Drilling Methods and Normal Characteristics

<table>
<tr><th></th><th>Diamond Core</th><th>Rotary</th><th>Continuous Coring</th><th>Downhole Rotary</th><th>Downhole Hammer</th><th>Percussion</th><th>Churn</th></tr>
<tr><td>Geologic information</td><td>good</td><td>poor</td><td>fair</td><td colspan="4">(—— poor ——)</td></tr>
<tr><td>Sample volume</td><td>small</td><td>large</td><td>small</td><td colspan="2">(—— large ——)</td><td>small</td><td>large</td></tr>
<tr><td>Minimum hole diameter</td><td>30 mm</td><td>50 mm</td><td>120 mm</td><td>50 mm</td><td>100 mm</td><td>40 mm</td><td>130 mm</td></tr>
<tr><td>Depth limit</td><td>3000 m</td><td>3000 m</td><td>1000 m</td><td>3000 m</td><td>300 m</td><td>100 m</td><td>1500 m</td></tr>
<tr><td>Speed</td><td>low</td><td colspan="5">(—— high ——)</td><td>low</td></tr>
<tr><td>Wall contamination</td><td colspan="2">(—— variable ——)</td><td>low</td><td colspan="4">(—— variable ——)</td></tr>
<tr><td>Penetration—broken or irregular ground</td><td>Poor</td><td colspan="3">(—— fair ——)</td><td colspan="3">(—— good ——)</td></tr>
<tr><td>Site, surface and underground</td><td>S + U</td><td>S</td><td>S</td><td>S + U</td><td>S + U</td><td>S + U</td><td>S</td></tr>
<tr><td>Collar inclination, range from vertical and down</td><td>180°</td><td>30°</td><td>0°</td><td>30°</td><td>180°</td><td>180°</td><td>0°</td></tr>
<tr><td>Deflection capability</td><td colspan="2">(—— moderate ——)</td><td>none</td><td>high</td><td colspan="3">(—— none ——)</td></tr>
<tr><td>Deviation from course</td><td colspan="2">(—— high ——)</td><td colspan="3">(—— little ——)</td><td>high</td><td>little</td></tr>
<tr><td>Drilling medium, air or liquid</td><td>L</td><td>A + L</td><td>L</td><td>A + L</td><td>A</td><td>A + L</td><td>L</td></tr>
<tr><td>Cost per unit depth</td><td>high</td><td>low</td><td>moderate</td><td colspan="3">(—— low ——)</td><td>high</td></tr>
<tr><td>Mobilization cost</td><td>low</td><td colspan="4">(—— variable ——)</td><td>low</td><td>variable</td></tr>
<tr><td>Site preparation cost</td><td>low</td><td colspan="4">(—— variable ——)</td><td>low</td><td>high</td></tr>
</table>

(Taken from Peters, 1978.)

(a) Rotary Drilling : depth of drilling with air cirulation may be limited by water inflow (e.g. from the water table) and by pressure and capacity of the air compressor. Rotary drill rigs are usually heavier than most diamond drill rigs, less flexible in where they can be placed, and limited in their ability to drill inclined holes. Rotary drilling, up to 180 m deep holes, was used at Sar Cheshmeh for the infill program between the diamond drill grid. It also proved to be particularly useful in delineatingthe margin of this deposit where diamond drills had made slow progress and achieved poor core recoveries. The disadvantage of rotary drilling is that identification of the rock types and other geological characteristics (type of veinlets, structural orientations, etc.) are difficult or impossible to obtain from the cuttings that are recovered.

(b) Percussion Drilling : this is a popular method for outlining shallow ore (down to 50 m) and where dry ground conditions permit drilling with air (if the down-hole method is used). Conventional percussion methods, e.g. Churn drill, can drill in almost any type of ground - for example, overburden, hard abrasive rock, dry or wet ground, and so on. Percussion drilling is quick, cheap, and it provides finely broken rock chips in a stream of air (down-hole method) or water (Churn drill). Churn drill holes have been used extensively 20-50 or more years ago to drill several porphyry coppers, e.g. Chuquicamata, Mantos Blancos, Lomas Bayas,(Chile), Santa Rita (New Mexico), etc.

At Lomas Bayas, Churn drill holes reached depths of 700 m.

Percussion drilling is frequently used in scout drilling programs to define the outline of mineralized zones and grade distributions in order to justify and plan more expensive diamond drilling; an example is Rössing Uranium deposit, South West Africa. Percussion drilling proceeds rapidly and operations can be conducted in rough terrains.

(c) Diamond Drilling : has the advantage that it can be done at any angle, positive or negative, with a large range in core diameter and borehole depth. Diamond core drilling supplies actual samples of the rock and not cuttings as with percussion and rotary drilling. Coring is therefore preferred for detailed geological and geotechnical studies.

Core drilling is expensive (5 to 10 times more per metre than percussion or rotary drilling) and detailed planning is therefore required to provide optimum information.

Since a major objective in sampling porphyries is to obtain enough material for accurate ore-grade calculations and metallurgical testing as well as delineation of the orebody, A to N size core (30 to 55 mm diameter) is taken in preference to the less expensive E size core (21 mm). The larger core sizes are especially preferred in irregularly mineralized and low-grade deposits (e.g. porphyries) (Peters, 1978); for example, at Sar Cheshmeh most of the core size was NX size, and the core diameter was kept as large as possible to improve recovery and reduce differential core loss in the generally bad ground conditions (supergene clays, brittle rock, etc.) of this deposit. To achieve better core recovery in this case triple tube core barrels and wireline equipment was also used (Selection Trust Ltd., 1970).

Where core recovery is poor, sludge is sampled and the sludge assays are combined with the corresponding core assays, but with an air of suspicion if the combined weights add up to other (more or less) than 100% of the weight of material that should have come out from the hole. An ideal geological sample from a diamond hole would consist of 100% of the rock displaced recovered in core and sludge, the chemical analyses of which could then be averages by weighting them according to the cross-sectional areas of the core and of the annular opening between the core and the wall of the hole (Koch and Link, 1970). However, core recovery is never 100% and sludge recovery is usually worse because of the following factors:-

- insufficient water pressure to wash the material and the heavier sulphides out of the hole;
- and/or because the material is washed into cracks in the rock;
- and/or it does not settle in the tanks at the collar of the hole. Differential settling of the heavy sulphide minerals in the sludge may warrant the use of special sludge cutters, as at Sar Cheshmeh, where Vaughan Thompson sludge cutters were used.

For these reasons percentages of recovery must be estimated and the weighting factors adjusted. The weighting factors must be determined empirically because the observations for core and sludge constitute two statistical samples, each with its own mean and variance. At the Chuquicamata porphyry copper mine, Waterman (1955) determined the following core-sludge relationship (Table 9):

TABLE 9 SLUDGE AND CORE ASSAYS FROM DIAMOND-DRILL HOLES IN THE CHUQUICAMATA COPPER MINE, CHILE*

Item	Core recovered (%)	Core assay (% copper)		Sludge recovered (%)	Sludge assay (% copper)	
		Soluble	Insoluble		Soluble	Insoluble
Observation	44	0.46	0.39	81	0.72	0.13
	50	1.12	0.35	67	0.72	0.10
	58	2.42	0.21	95	1.35	0.10
	57	0.95	0.22	102	1.31	0.10
	61	0.30	0.23	73	0.95	0.11
	80	1.51	0.23	48	0.98	0.12
	64	1.18	0.13	115	1.05	0.12
	83	1.24	0.28	91	1.07	0.10
Mean	62.13	1.14	0.26	84.0	1.02	0.11
Standard deviation	13.52	0.65	0.08	21.3	0.23	0.01
Coefficient of variation	0.22	0.57	0.32	0.25	0.23	0.11

* After Waterman, 1955, p. 60.

(Taken from Koch and Link, 1970.)

As part of the diamond drilling program at Sar Cheshmeh, sludge samples were taken to check that the core recovered was representative of the rock drilled. Both core and sludge values for the different holes showed a close agreement (see Table 10, e.g. DDH-3), but direct use of sludge assays was not possible at Sar Cheshmeh because of dilution by drilling mud.

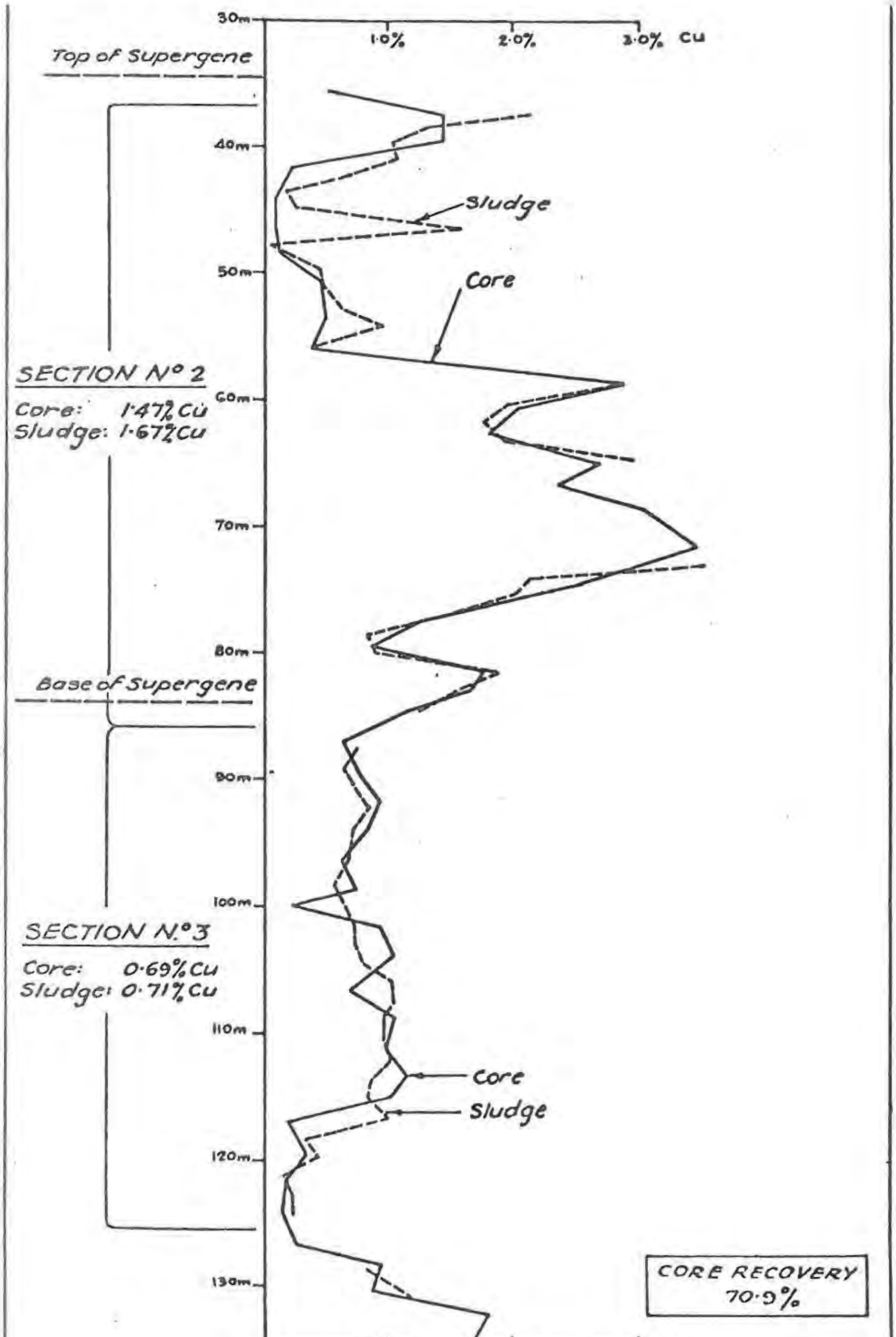


Table 10 : Sludge and core Profile DDH-3.
(Taken from Selection Trust Ltd., 1970.)

Errors and bias in sampling can have several causes and depend on the drilling technique used, the way it is applied and/or be produced by the sampling apparatus used. Cuttings from non-core drilling and the sludge sometimes recovered in diamond core drilling purport to represent specific increments in depth. The cuttings that arrive in a sample at the drill-hole collar are composed of minerals with a wide range in density and therefore with a range in travel time from the bottom of the hole. Some of the material may have come from the walls of the drill hole; some may have been lost in fractures; some may have accumulated in open fissures and vugs until it was regurgitated into a later sample. Where drilling fluid is recycled, as in permafrost and desert areas, some of the finer mineral particles are recycled as well. In the most unreliable situation of all, a sample from an unsurveyed part of a deep drill hole may have been taken far from its assumed location.

2.2.3 Drilling Costs influencing the Selection of Drilling Grids

Hewlett (1965) studied in approximately 50 porphyry copper deposits the relationship between the desired precision of preliminary grade estimates and drilling costs in order to determine the most economic drill hole spacing in these deposits. Based on such a relationship, decisions can be made concerning investments on following sequential drilling for obtaining acceptable precision for the final grade estimate.

The standard error of the mean ($S_{\bar{x}}$) is used for computing the precision of the estimate of the average grade. The standard error of the mean is the standard deviation (S) of the assays divided by the square root of the number of assays ($\sqrt{n}$);

$$S_{\bar{x}} = S / \sqrt{n}$$

The standard deviation can be obtained from the formula:

$$S = \sqrt{\frac{\Sigma X^2 - \bar{x}\,\Sigma X}{n-1}}\ ;\ \text{where}$$

X = represents individual assays

$\bar{x}$ = is the mean or average grade of the assays.

The confidence interval (CI) about the estimated mean grade is computed by

$$CI = \bar{x} \pm (S_{\bar{x}} \; t_{\alpha})$$

t_{α} = table value of "t" statistic at the α confidence level.

The following example of the Liberty porphyry copper deposit (U.S.A.) illustrates the interrelationship between drilling cost, precision of the estimate of grade of ore, and drill hole spacing:

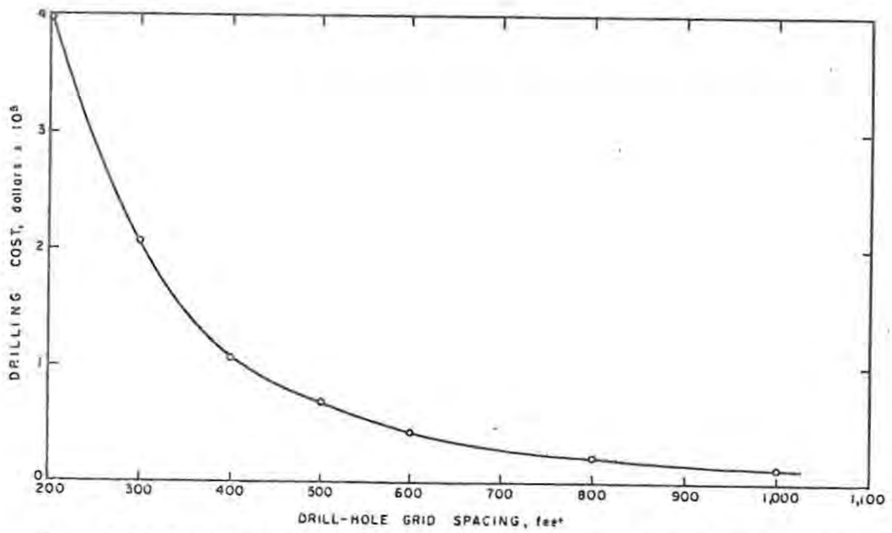


FIGURE 27- Drilling Cost for Drill-Hole Grid Spacings for Liberty-Type Copper Deposit.

(Taken from Hewlett, 1965.)

Figure 27 illustrates the drilling cost for various drill-hole grid spacings. It can be seen from Figure 27 that the drilling cost increases rapidly for grid spacings that are less than 400 feet because of the increased number of drill holes.

Figure 28 illustrates the values of the standard error of the mean against the grid spacing for the same deposit. It can be seen that the greatest rate of change of the standard error of the mean is within the grid spacing of 800 to 1000 feet, and the rate of change is quite consistent, nearly linear, below a spacing of 800 feet. Thus, characteristics of the mineralization in this deposit, as measured by the standard deviation "s" ($s = s_{\bar{x}} \sqrt{n}$), indicate that drill-hole spacing must be less than 800 feet before the standard deviation is sufficiently stabilized to produce a standard error of the linear that is less than 0.02.

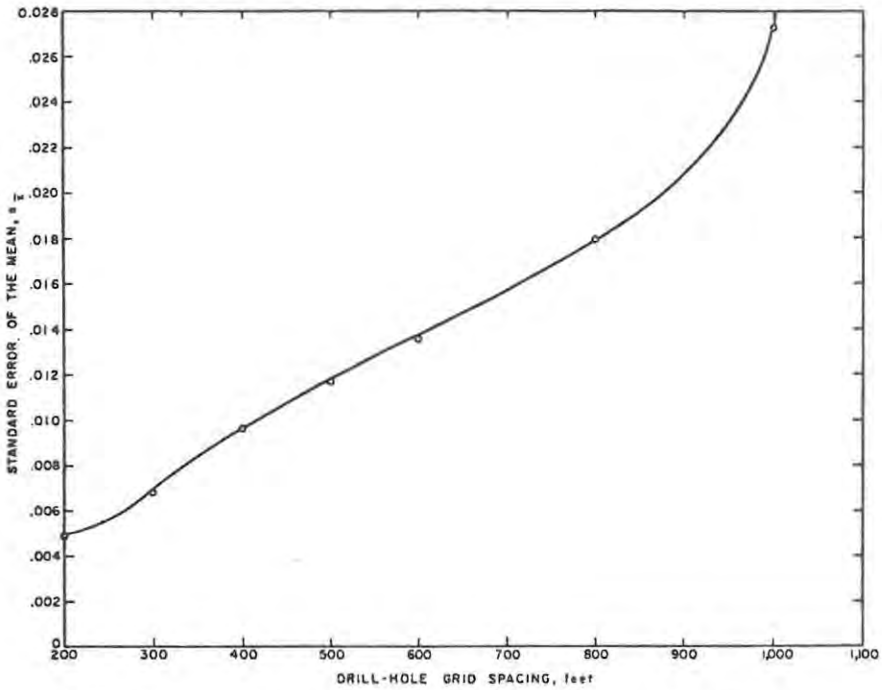


FIGURE 28 Standard Error of the Mean for Drill-Hole Grid Spacing for Liberty-Type Copper Deposit.

(Taken from Hewlett, 1965.)

Any grid spacing less than 800 feet will produce a nearly linear reduction in the standard error of the mean and, therefore, a reduction in the precision of the estimate of grade of ore. Any desired increase in the precision of the grade estimate will be increasingly expensive for drill-hole grid spacing less than about 600 feet. Figures 27 and 28 therefore illustrate and quantify the interaction between characteristics of mineralization of a porphyry deposit and the drilling cost to explore it.

The increase in drilling cost for an increase in precision of 0.001 of a standard error of the mean for each grid spacing is shown in Figure 29. This figure shows that the greatest increase in drilling cost occurs with a spacing of less than 600 to 700 feet. If the precision were acceptable by using a grid of 800 feet, this would be the most economical spacing.

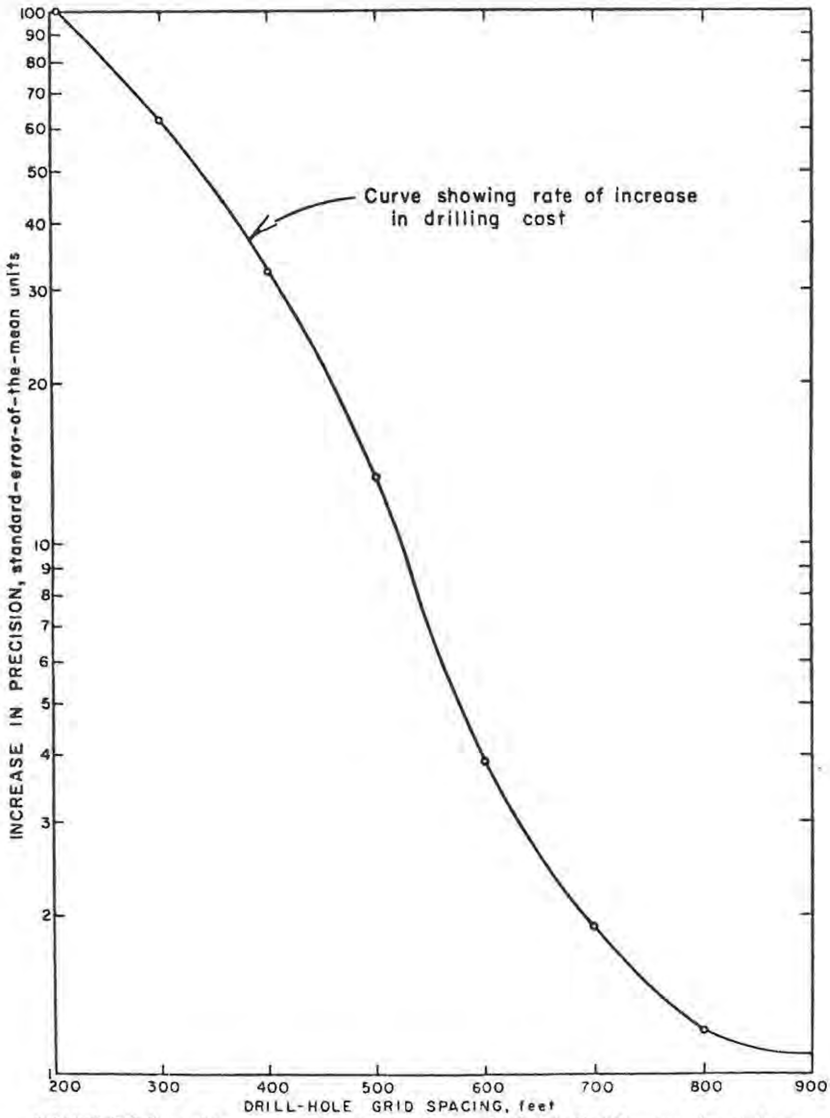


FIGURE 29 Cost of Increasing Precision for Various Drill-Hole Grid Spacings for a Liberty-Type Copper Deposit.

(Taken from Hewlett, 1965.)

It is expected that most porphyry copper deposits will fall either on the no-trend model line or right of the line, as shown in Figure 30. This is because geological control involved in the processes of mineralization (see chapter 1.3) and secondary enrichment commonly causes changes in the mean that result in multimodal distributions (Fig:31). Supergene

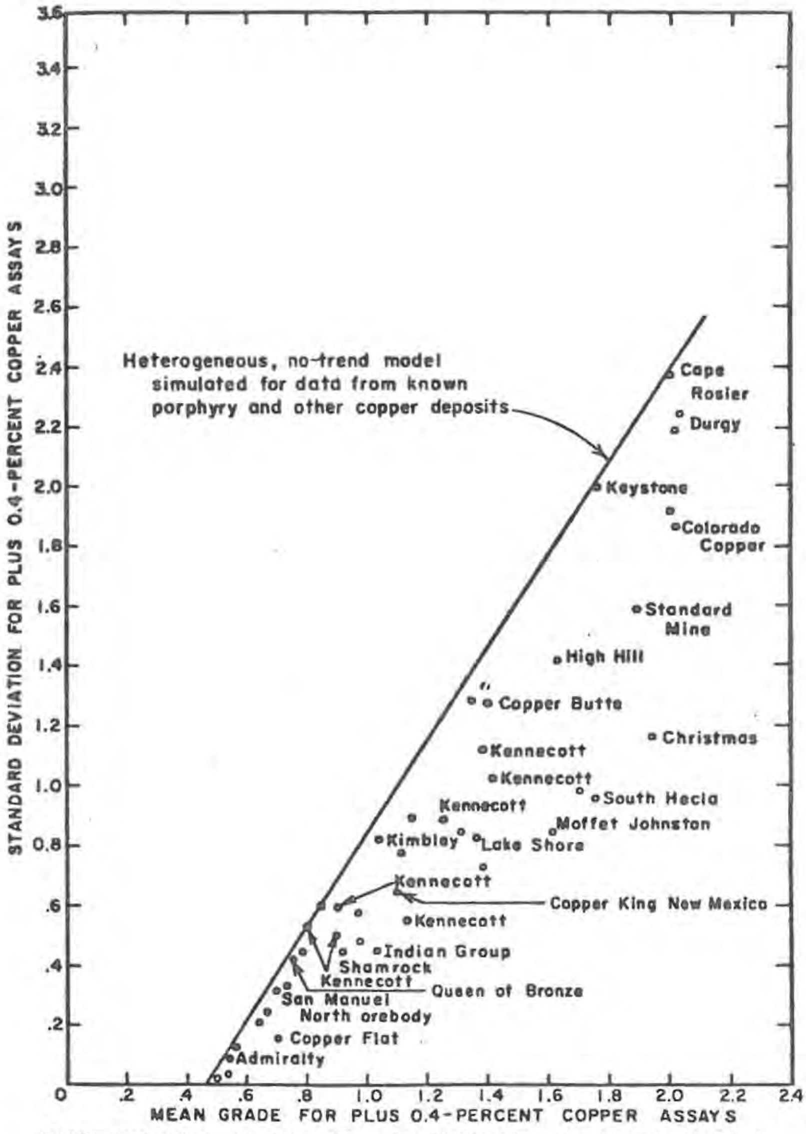


FIGURE 30- Relationship of Actual Deposits With Three-Dimensional Trend to No-Trend Simulated Model.

(Taken from Hewlett, 1965.)

enrichment introduces a higher grade for a population of assays-increasing the mean- enrichment also causes a higher standard deviation by introducing a wider range of assay values. Because

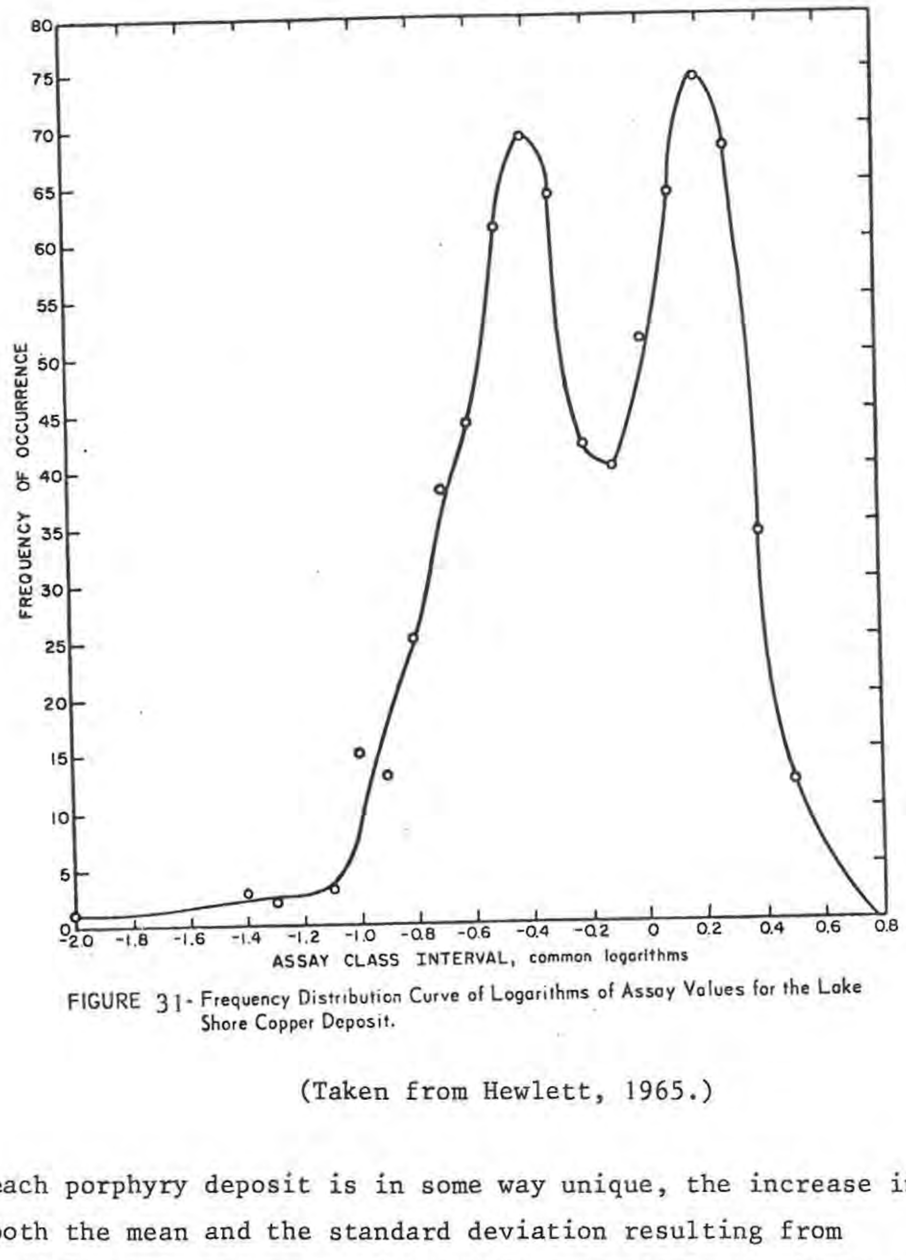


FIGURE 31 - Frequency Distribution Curve of Logarithms of Assay Values for the Lake Shore Copper Deposit.

(Taken from Hewlett, 1965.)

each porphyry deposit is in some way unique, the increase in both the mean and the standard deviation resulting from enrichment is not expected to conform precisely to the linear relationship shown in Figure 30.

Based on Figure 30, Hewlett (1965) proceeded to compute for these 50 porphyry deposits the drilling requirements of the number of ore assays required for a specified precision, e.g. a standard error of the mean of 0.02 equivalent to a confidence interval of $\pm$ 0.04% Cu. The number of ore assays at a cut-off grade of

0.40% Cu was computed for the standard error of the mean of 0.02 from:

$$n = S^2/(S_{\bar{x}})^2 = S^2/(0.02)^2$$

The results are plotted in Figure 32, which is a modified form of Figure 30, where (S), the ordinate in Figure 30, is coded by the proportion $S^2/(0.02)^2$ to result directly in the number of ore assays (or drill grids) required for a standard error of the mean of 0.02, that is, n for $S_{\bar{x}} = 0.02$.

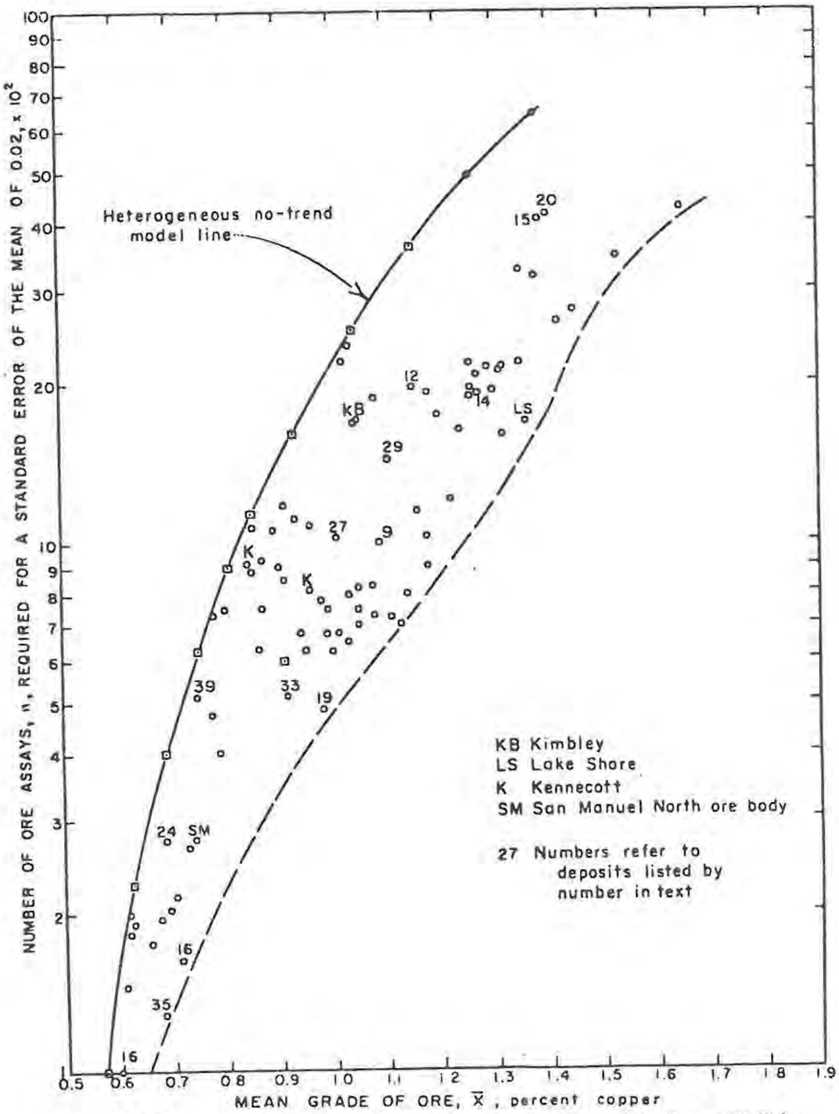


FIGURE 32. Number of Ore Assays Required for a Standard Error of the Mean of 0.02 for Various Mean Ore Grades for Numerous Copper Deposits.

(Taken from Hewlett, 1965.)

An attempt to illustrate the effect of the sequential drilling of a mineral deposit upon the improved estimate of the average grade is shown in Figure 33 for assay data from numerous porphyry

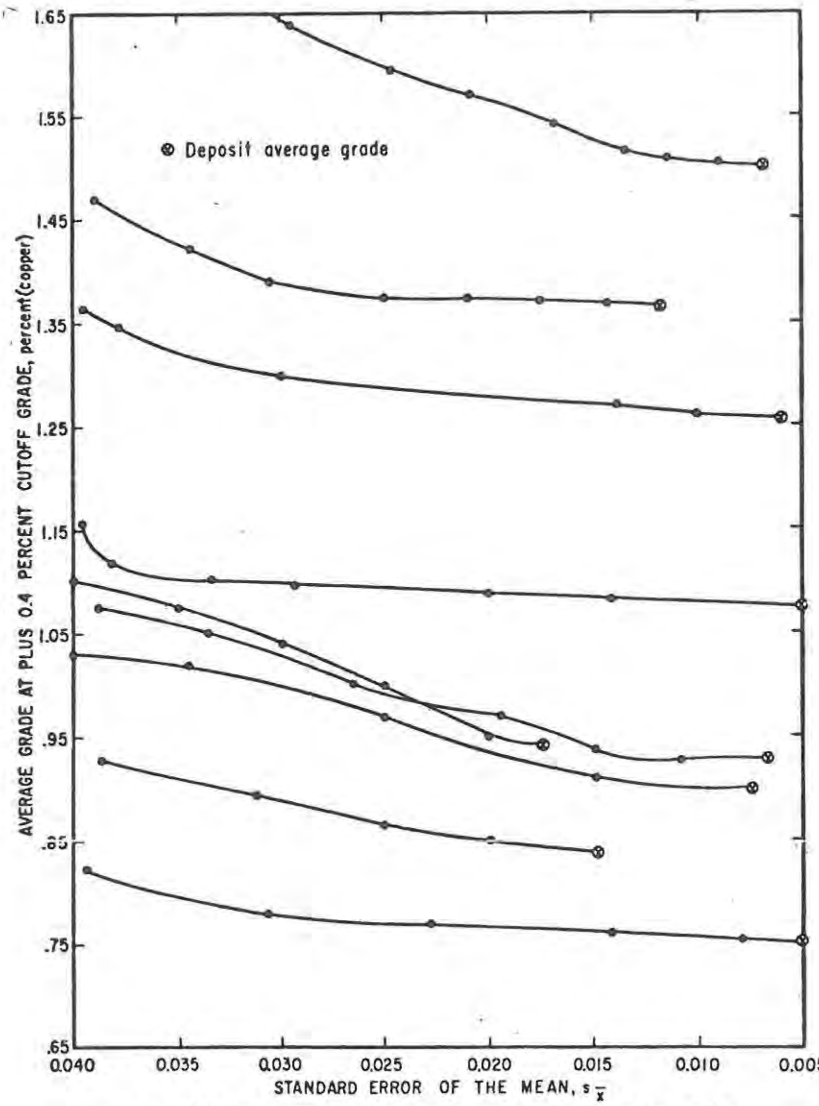


FIGURE 33- Comparison of Accuracy and Precision for Estimating Grade of Ore for Numerous Porphyry Copper Deposits.

(Taken from Hewlett, 1965.)

copper deposits. It shows that historically the ore grade of the deposits was higher during the early drilling of each of the porphyries. This could be a result of:-

- early drilling to test the high-grade zone or zones
- subsequent drilling to delimit the orebody.

Figure 33 also shows that the average grade estimated during the drilling sequence is related to the error of the mean, which is $S_{\bar{x}} = S/\sqrt{n}$; or a measure of not only the precision but also the number of samples. Therefore, the last point to the right on the figure represents the best estimate of the grade of the ore in the entire deposit. In general, the estimate of average grade does not change appreciably for a precision greater than a standard error of the mean of 0.02.

The relationship between the stimated tonnage of ore and the standard error of the mean for various stages of sequential drilling is shown in Figure 34. It is seen in this figure that at a standard error of the mean of 0.02, the estimated tonnage is very close to that actually mined from each deposit.

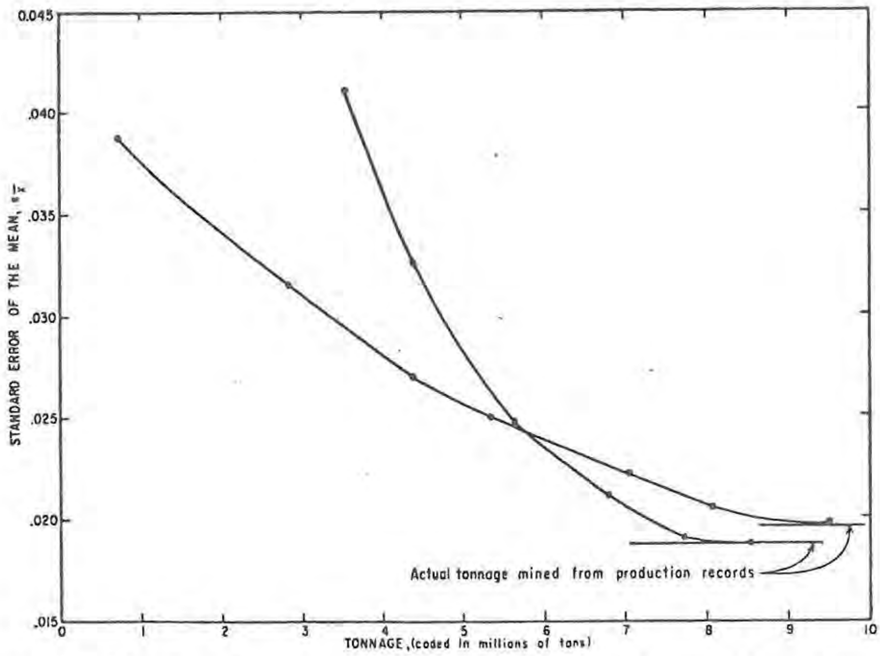


FIGURE 34 - Comparison of Accuracy and Precision for Estimating Tonnage of Ore for Two Typical Low-Grade Porphyry Copper Deposits.

(Taken from Hewlett, 1965.)

This points out that the standard error of the mean used as the criteria for the precision of the estimate of the average grade is also an indication of the accuracy of the tonnage estimate.

The following procedure has been used by Hewlett (1965) to determine the future drilling requirements for a deposit that has been sampled by only a few drill holes:

1. From available drill hole data the following ratios are calculated:-

    a) $\dfrac{\text{number of ore assays}}{\text{number of total assays}}$

    b) $\dfrac{\text{number of total assays}}{\text{number of drill holes}}$

2. Compute the mean grade and standard deviation of available ore assays - in the present case assumed to be a cut-off grade of 0.40% Cu.

3. Compute the number of ore assays for each combination of grid spacing.

4. Using "n" and "s", calculate the standard error of the mean for each grid spacing.

This procedure is used sequentially whenever additional drill-hole data becomes available so that the drilling requirements are known as accurately as possible at all times during the drilling period. Figure 35 represents the plot of standard error of the mean at any time during the drilling of different porphyry deposits. The progressively larger number of ore assays accumulated during drilling is used to express time during drilling to provide a common basis to relate the standard error of the mean to time. Application of the data in Figure 35 to

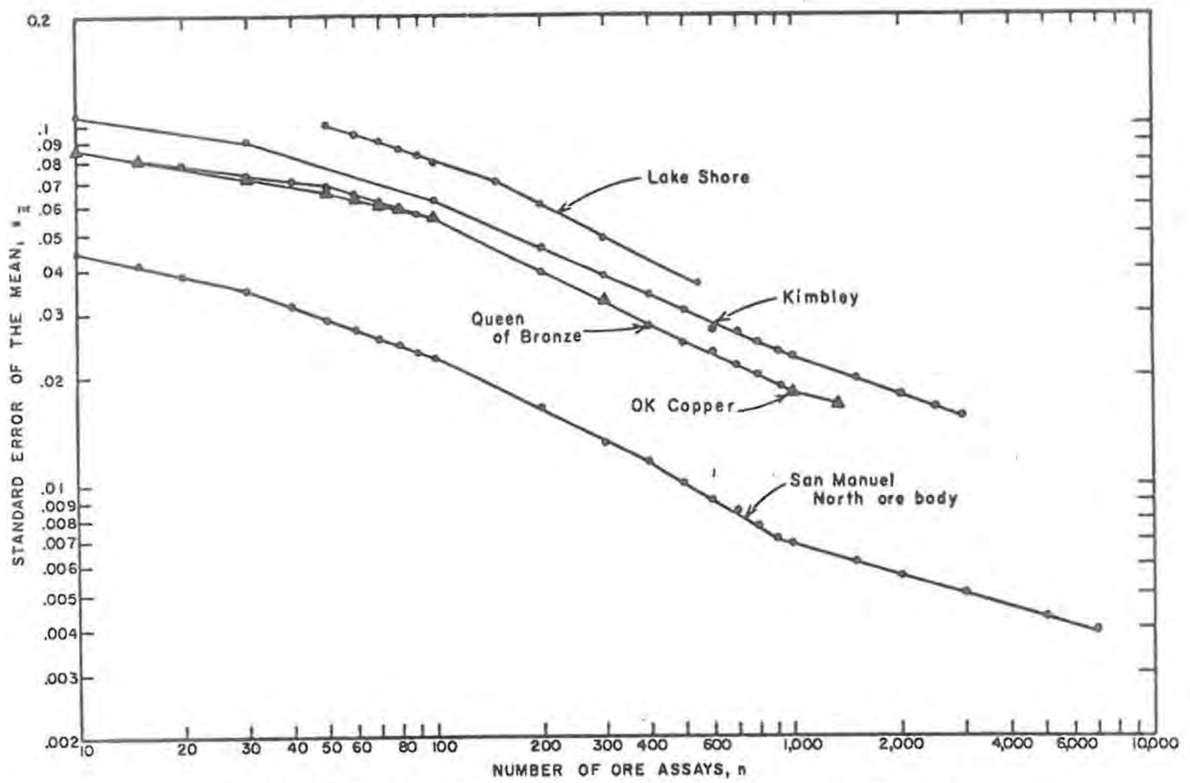


FIGURE 35 - Plot of Standard Error of the Mean and Number of Ore Assays Computed for Natural Grade Trend in Various Deposits.

(Taken from Hewlett, 1965.)

a new prospect can be made in order to obtain an estimate of the drilling requirements by using available geological data concerning the prospect. These drilling requirements, in terms of number of ore assays, were converted into standard error of the mean by use of Figure 35 and plotted against grid spacings (Figs.36, 37 and 38).

The data from Figures 36, 37 and 38 were replotted to summarize the relatship between standard error of the mean, grid spacings (cost), estimated size of the deposit and possible mineralization type, based on the Kimberley and Queen Bronze deposits (see Fig.39).

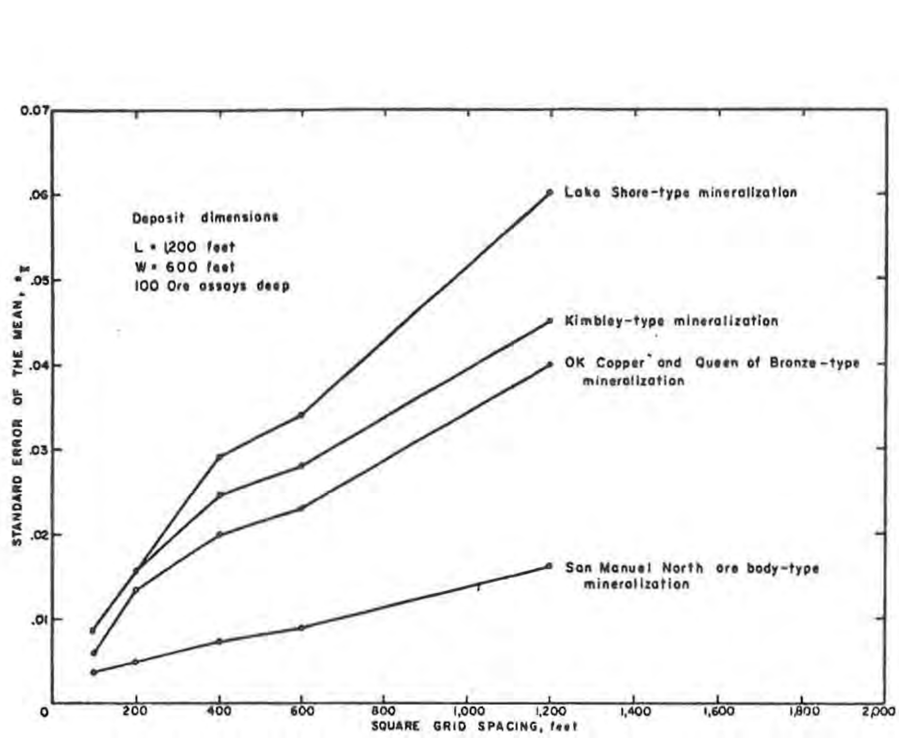


FIGURE 36. Plot of Standard Error of the Mean and Grid Spacing for Various Types of Low-Grade Copper Mineralization (Deposit Length of 1,200 Feet).

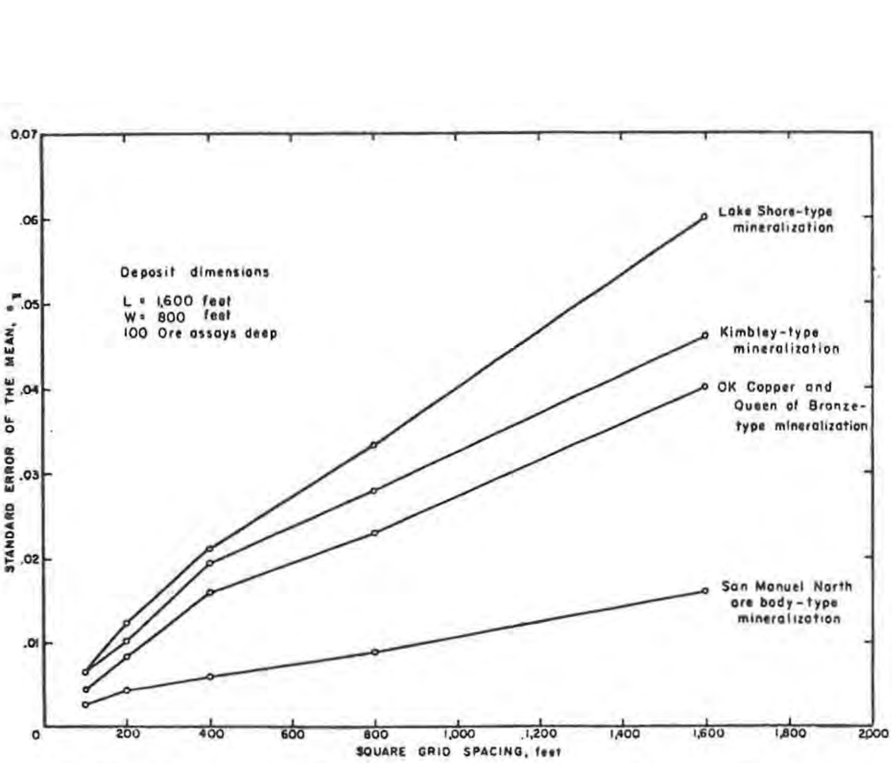


FIGURE 37. Plot of Standard Error of the Mean and Grid Spacing for Various Types of Low-Grade Copper Mineralization (Deposit Length of 1,600 Feet).

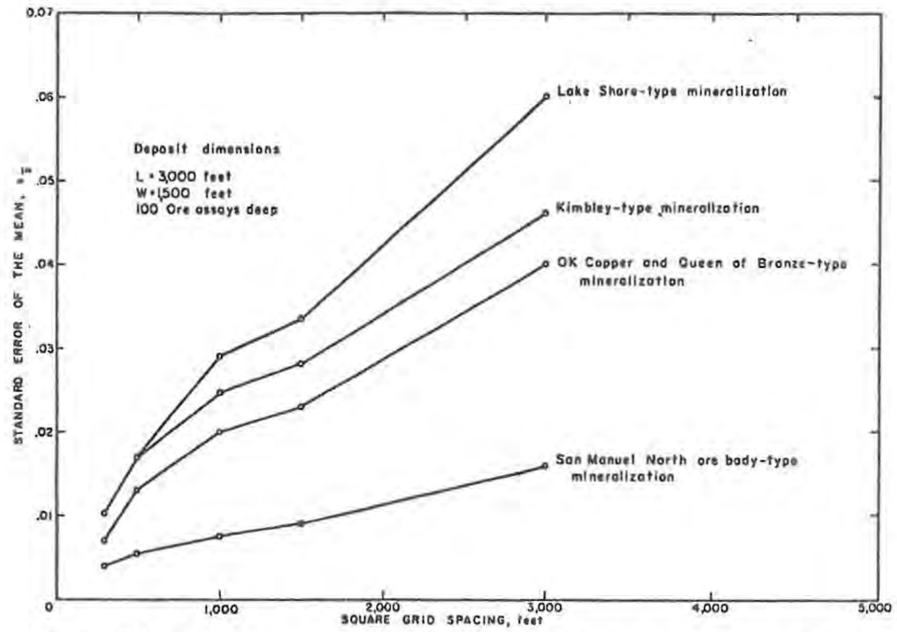


FIGURE 38. Plot of Standard Error of the Mean and Grid Spacing for Various Types of Low-Grade Copper Mineralization (Deposit Length of 3,000 Feet).

(Taken from Hewlett, 1965.)

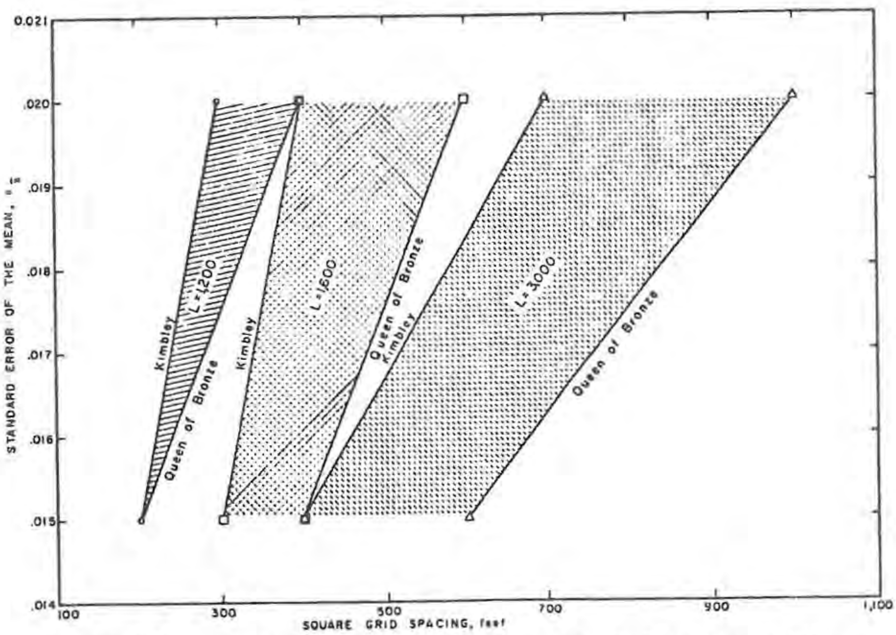


FIGURE 39. Summary of Drilling Requirements for a Deposit for Three Estimated Shapes and Two Mineralization Types.

(Taken from Hewlett, 1965.)

2.2.4 Delineation Investment Decisions

Exploration is a sequential information gathering process that starts with primary regional exploration programs. The most favourable of the mineral discoveries provides economic justification for a delineation drilling program, which should provide enough information for estimating relevant physical characteristics of the deposit, e.g. tonnage and grade characteristics, amenability of the ore, etc. These estimates are used to evaluate the exploration project and, eventually, the assessed economic characteristics of the delineated occurrence provide justification for mine development and production.

Subsequently, a development drilling program, and occasional bulk sampling program, are initiated to obtain the necessary data for detailed mine planning. The delineation drilling stage is in itself a sequential investment process. As delineation information accumulates, the reliability of geological estimates is improved (see chapter 2.2.3) and thus the uncerttainty associated with the assessment of profitability criteria is reduced. Delination drilling should continue as long as the marginal benefits which results from improving the reliability of profitability criteria exceed the marginal costs associated with the additional drilling (Mackenzie, 1979).

The following two criteria are used by Mackenzie (1979) for the evaluation of delineation investment:

i) Expected profitability : the decision maker, motivated by the desire for profitable investment, prefers an investment alternative with a higher expected profitability. Expected net present value or rate of return (average percentage return that an investment opportunity is expected to yield over its life) are used to measure the expected value criterion.

ii) Risk criterion : the decision maker prefers an investment alternative with less economic risk. Risk aversion results in a concern for the insurance of some minimum level of profitability with a given degree of confidence. Thus, while economic uncertainty is measured by the variance of the probability distribution of possible profitability values, economic risk is typically valued as an insurable lower limit of this distribution or by the probability that the profitability will be less than some predetermined rate (e.g. the cost of capital). The 90 or 95% lower limit net present value or rate of return is used to measure the risk criterion. A relative weighting between these expected value and risk criteria (conceptually expressed as "utility") provides the basis for the delineation investment decision, and for any eventual decision, to develop and mine a deposit (Mackenzie, 1979).

Although investment in a more detailed delination program and the gathering of additional information reduces the economic uncertainty associated with mine development, this investment must be economically justified, i.e. the marginal benefits obtained as a result of the additional information must exceed the respective marginal costs. Mackenzie (1979) quotes the following:

"On termination of the initial delineation program, one of four courses of action or stategies are possible. Figure 40 illustrates the decision-maker's alternatives. First of all, the project can be immediately abandoned in view of the information gathered up to that point, either because the project's expected profitability and economic risk are unacceptable or because, under conditions of limited investment capital, a better investment opporrunity can be found elsewhere. Secondly, any further investment in the project can be postponed if future technological and market conditions are projected to become more favourable. On the other hand, the project's expected profitability may be found acceptable to the organization. In this case, there are two possibilities:

i) although economic uncertainty still exists, there is a negligible amount of economic risk associated with mine development and production, i.e. the probability of financial loss is of no significance.

or

ii) there is a significant amount of economic risk associated with development and production. This level of economic risk can either be acceptable or unacceptable to the decision-maker but in any case, it can be reduced by obtaining additional delineation information.

If no significant economic risk is involved (an unlikely possibility in practice), a mine development may be taken. On the other hand, if the risk of financial loss if significant, then the decision-maker must determine whether investment in a more detailed delineation program is economically justifiable. The costs of a proposed delineation program must be weighted against the reduction in economic uncertainty which will result from the additional information. If the new investment is justified, the proposed delineation program is undertaken. If the proposed program is not justified, delineation is terminated. If the project has an accpetable level of economic risk, it is then compared to other investment proposals competing for funds. If the level of economic risk is unacceptable, the project is either abandoned or postponed, as previously described.

In the event that an additional delineation program is implemented, the decision process described above is repeated. The option of abandoning the project remains an alternative in subsequent stages of this decision process. Unless the deposit is evaluated as marginally profitable and the additional information obtained in the last delineation program significantly reduces the deposit's geological estimates or unless a better investment opportunity is found during the carrying out of the last delineation program, the chances of abandoning the project decrease with time.

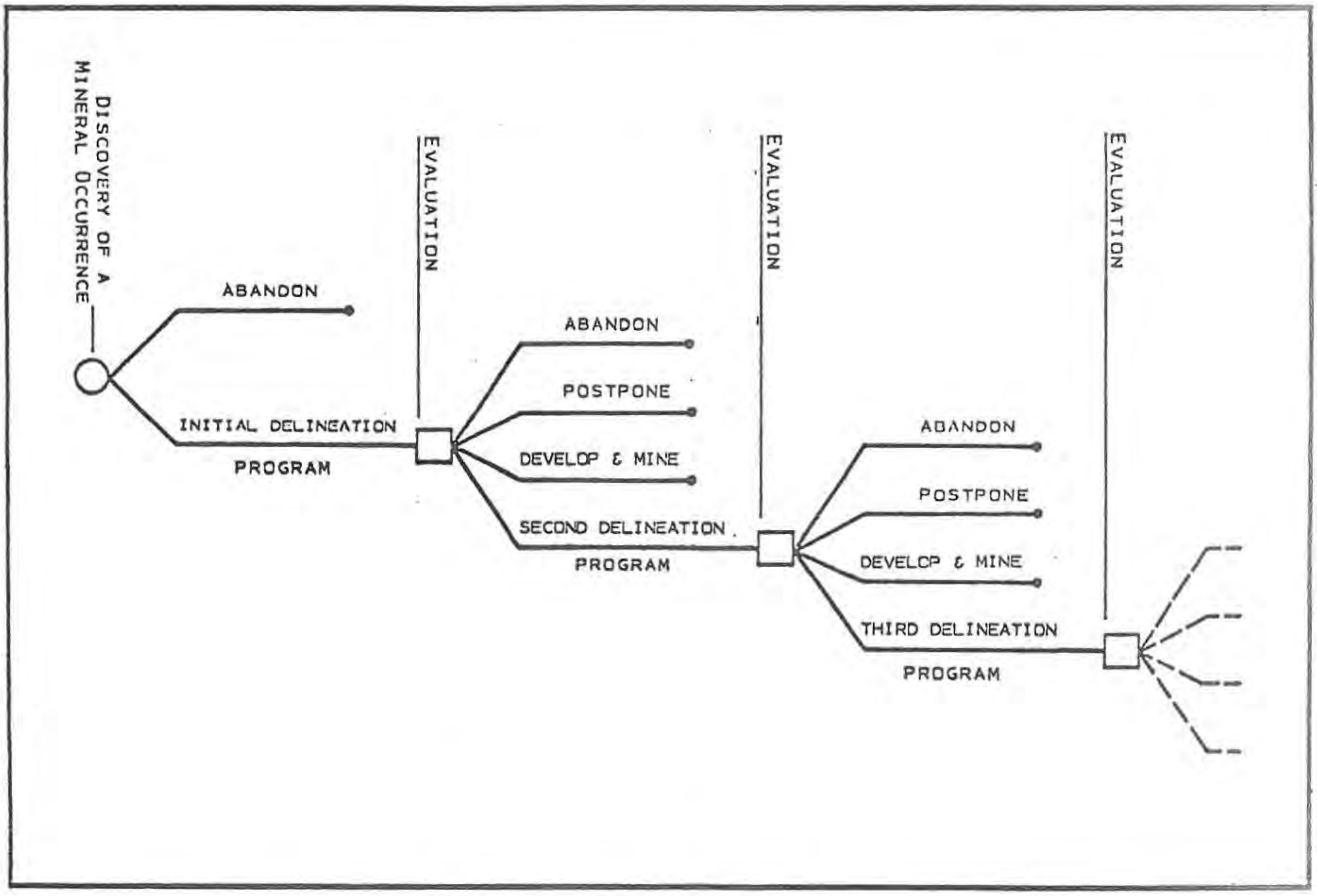


Fig.40 : The sequential delineation process.
(Taken from Mackenzie, 1979.)

As additional delineation information reduces the economic uncertainty associated with the evaluation of mine development and production alternatives, the dispersion of possible profitability values about the expected value estimate is also reduced or, in other words, the variance of the profitability distribution is decreased. As a result, the minimum insurable profitability is drawn in towards the expected profitability. This concept is illustrated in Figure 41. As more information is obtained, the absolute distance between expected profitability and minimum insurable profitability decreases (i.e., D_2 is smaller than D_1).

However, the benefits of additional information in reducing economic uncertainty are subject to diminishing returns. As delineation information accumulates, the reliability of the profitability estimate improves, but at a decreasing rate. This characteristic is comparable to the behaviour of the standard error of the mean in classical statistical theory. The improvement in the reliability of the estimate of the true mean is proportional to the square root of the sample size. For example, the following table shows the percentage increase in reliability of an estimate produced by an increment of 10 observations at several sample size levels.

Initial Sample Size	Sample Size Increased to	Percentage Increase in Reliability
10	20	29.3
30	40	13.4
50	60	8.7
100	110	4.7
250	260	1.9

Costs are incurred to improve the reliability of the profitability estimate. These include the direct drilling, assaying and data analysis costs as well as the time cost of delaying production revenues if the deposit is subsequently developed and mined. These costs decrease expected profitability and shift the entire profitability distribution towards lower values.

Initially, benefits will exceed costs, and minimum insurable profitability will increase with further delineation. The two upper distributions in Figure 41 show how the minimum insurable profitability increases even though expected profitability decreases. In this case, marginal benefits exceed marginal costs. At some point however, the marginal benefits will just balance the marginal costs. At this point, the minimum insurable profitability value is maximized. Beyond this point, further delineation proposals will lower both expected profitability and minimum

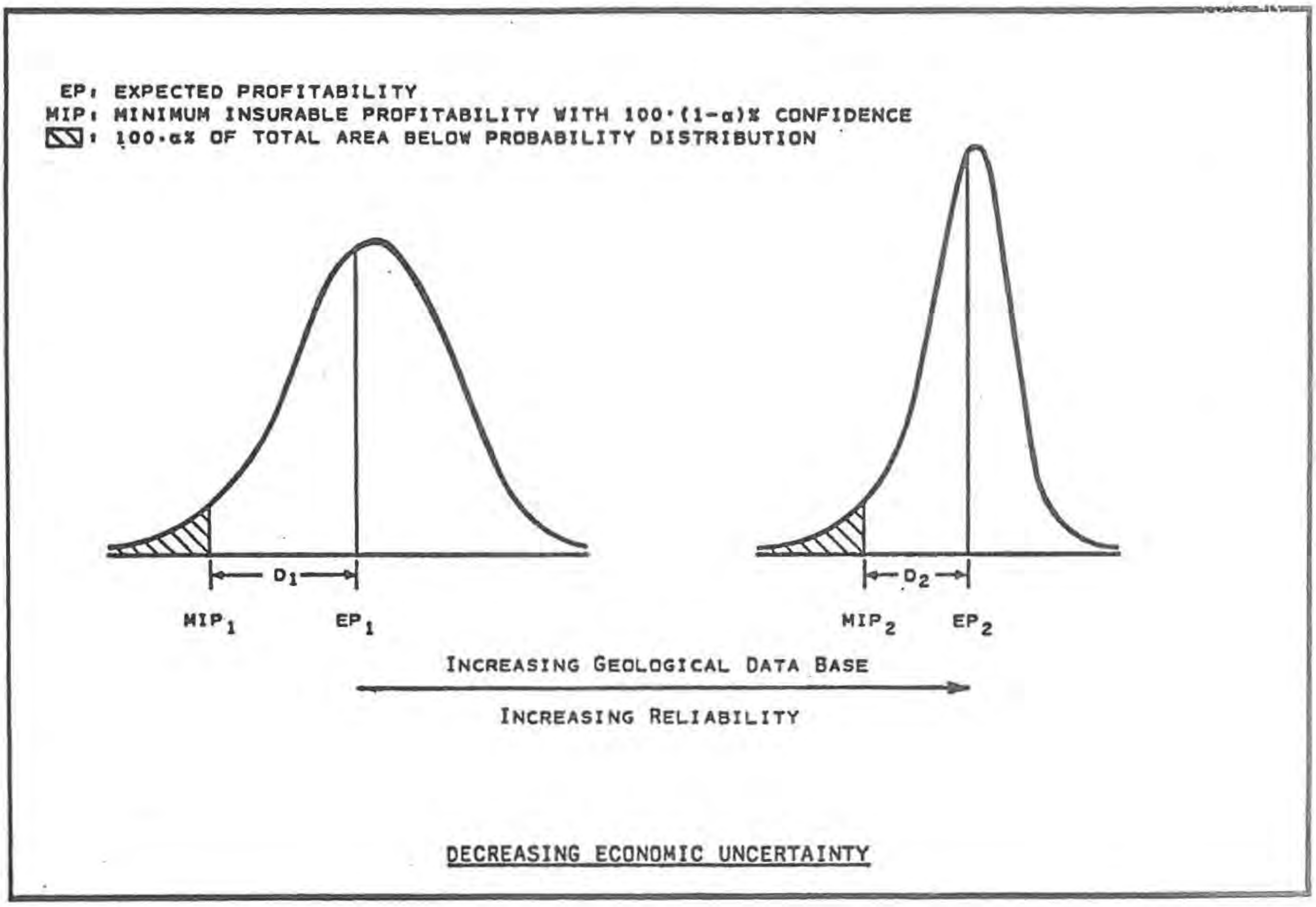


Fig.41 : (Taken from Mackenzie, 1979.)

insurable profitability. As the marginal costs outweigh the marginal benefits, the rate ofdecrease of the expected profitability value exceeds the rate of convergence of the minimum insurable profitability with it. The bottom distribution in Figure 42 shows that the net effect is an overall decrease in minimum insurable profitability. A decrease in minimum insurable profitability represents an increase in economic risk, i.e. the probability of financial loss increases.

The desired economic amount of delineation is typically achieved short of the point where minimum insurable profitability is maximized, depending on the mining organization's relative preference for expected profitability and economic risk criteria. In decision-making terms, the desired amount of delineation should maximize the organization's utility, for example in this case a certain drill-grid spacing may be the most favourable from the economic and geological-statistical point of view (refer to chapter 2.2.3 and to the following paragraph). The amount of delineation economically justified represents the economic sampling limit. Figure 43 shows the conceptual determination of the amount of delineation justified to maximize utility (A) and to minimize economic risk (B). This illustrates the optimization of delineation investment at a given point in time. As the investment in a larger and more costly delineation program decreases expected profitability, the minimum insurable profitability value increases at a decreasing rate eventually reaching a maximum as shown. The consideration of any delineation program requiring a larger investment than the one indicated for the above maximum, will increase economic risk. The evaluation results (expected profitability values) showing the effect of an improved geological data base do not really coincide with the actual evaluation results (true profitability values) obtained after the additional delineation program because as delineation costs become sunk, they are excluded from the evaluation.

Finally the delineation investment decision are computed by combining the geostatistical tonnage-grade estimates and development parameters with the expected profitability and risk criteria. As already indicated, the reliability of a mean or expected value estimate is a function of the variability of the deposits characteristics and the sample size (see chapter 2.2.3, e.g. at the Liberty porphyry copper deposit the optimum economical and statistical-geological grid spacing is between 600-800 ft). As the estimation variances of the possible values of the tonnage-grade distributions are a function of both the amount of information and the position of that information within the mineral occurrence, these variances can then be assessed at the end of each delineation drilling stage. The reduced variance - or uncertainty - estimates to be obtained at each successive delineation stage may then assist in determining the economic justification of further drilling. In order to determine this economic justification, the simulation of profitability

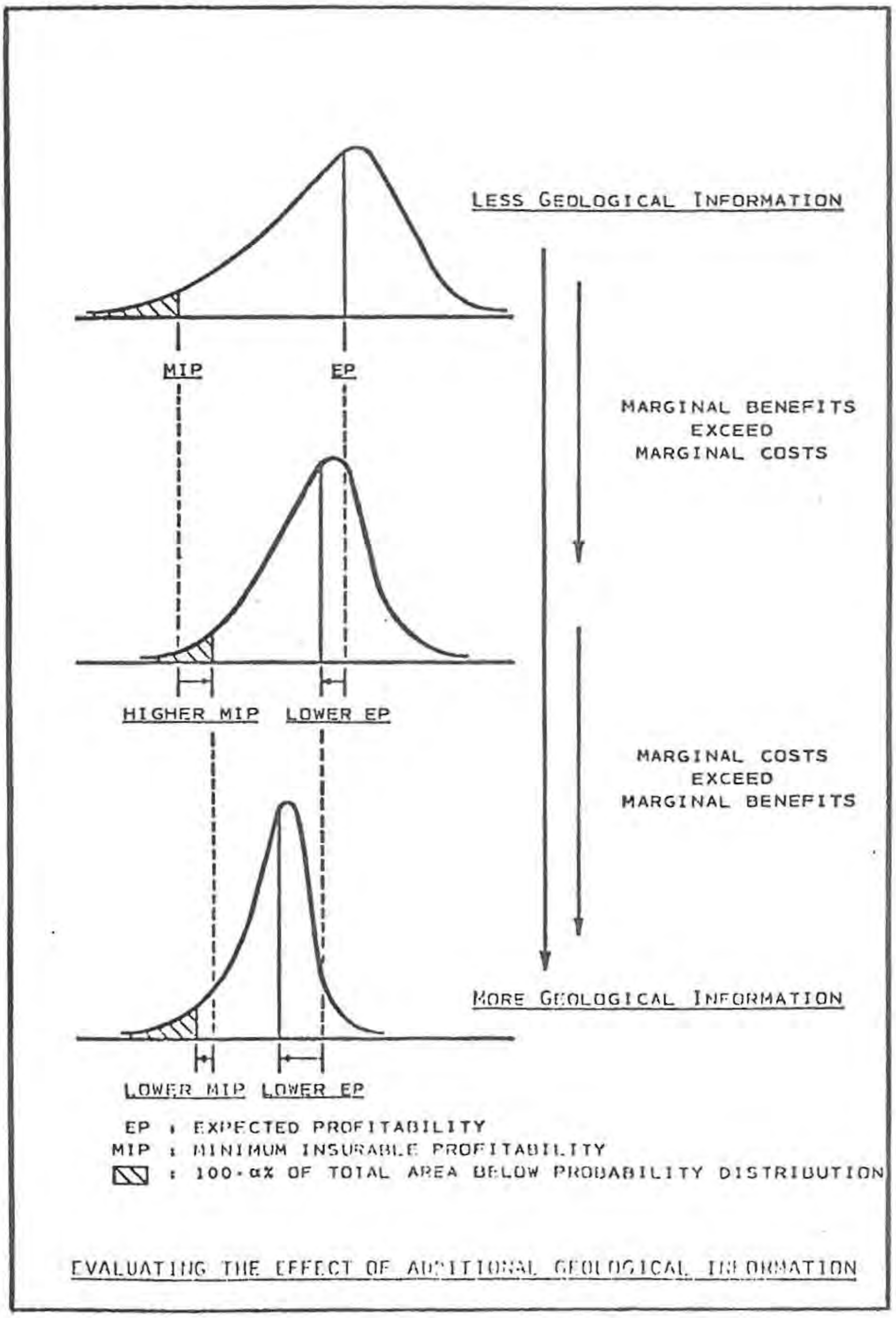
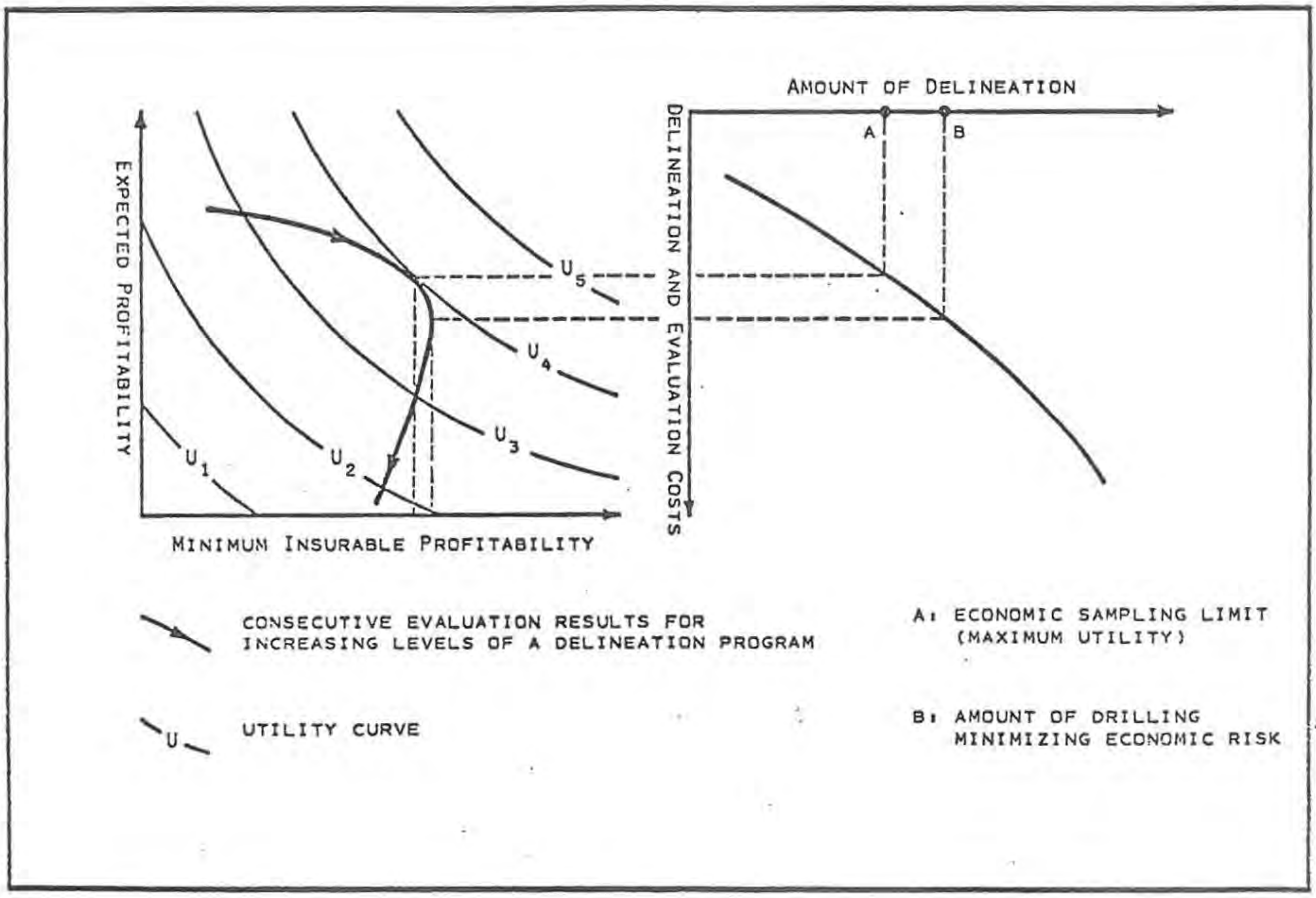


Fig.42 : (Taken from Mackenzie, 1979.)

Figure 43.
DETERMINING THE ECONOMIC SAMPLING LIMIT



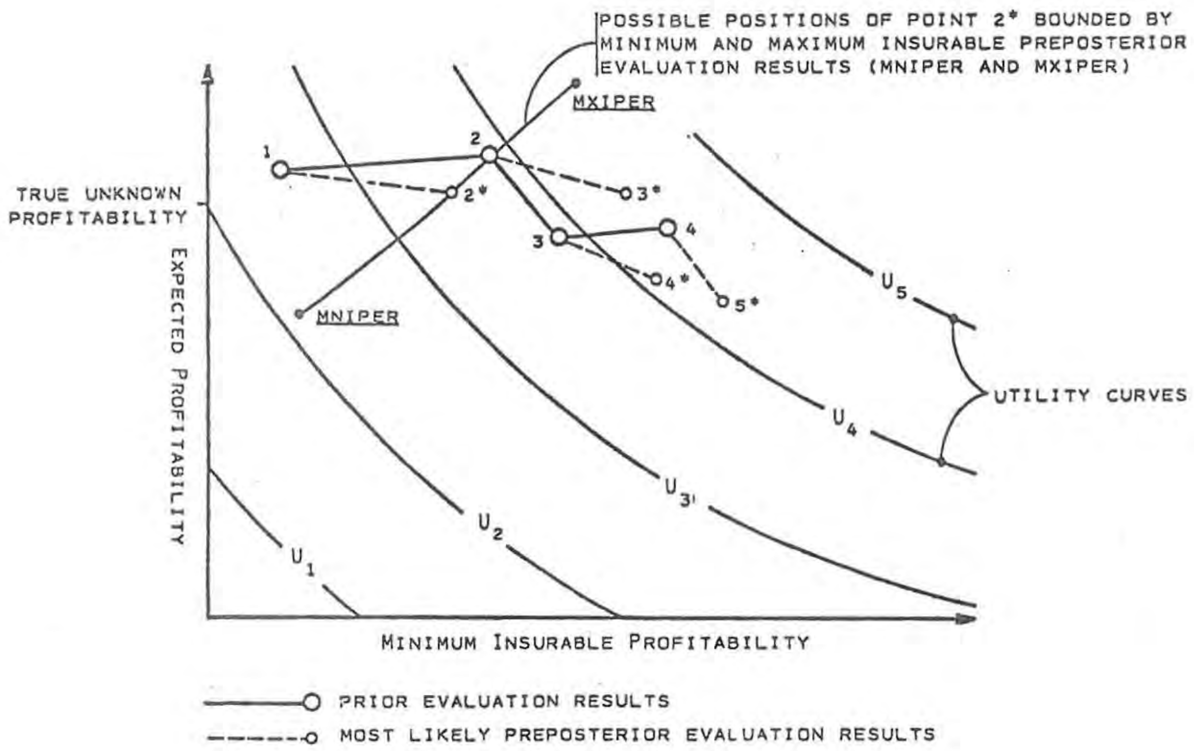
(Taken from Mackenzie, 1979.)

values is used to assess the expected profitability and economic risk resulting from tonnage and grade uncertainties. Such a similuation model generates a probability distribution of profitability values. Such a distribution is characterized by an expected profitability and a variance. Appropriate techniques are then used to estimate the insurable lower limit profitability. Expected and insurable lower limit profitability values have been plotted for a hypothetical case in Figure 44. Point 1 represents the expected result based on prior estimates. Point 2 represents the estimated expected result of undertaking the next delineation stage. The estimated posterior estimates are combined with the time and drilling costs to compute the investment criteria for this latter point.

Utility curves, expressing the decision-maker's relative preference for the two investment criteria are superimposed on the actual and estimated expected results to determine whether or not the increased data base resulting from the next drilling stage would improve the overall economic characteristics of the investment alternative. In this case, the estimated expected result (2) indicates a higher utility than the actual result (1), thereby providing economic justification for the next delineation drilling stage.

The delineation of a mineral deposit is carried out in discrete steps. Consequently, the continuous process depicted in Figure 43 is not encountered in practice. Points 1,2,3 and 4 in Figure 44 represent the actual path followed by the delineation drilling process. Drilling terminates at stage 4 (Fig.44) because the estimated result after the next stage (5) indicates a lower utility than the actual result. At this point, the economic sampling limit (or grid spacing) has been reached and the mine development decision - and/or the bulk sampling program - should be considered.

Figure 44.



ECONOMIC SAMPLING LIMIT - HYPOTHETICAL EXAMPLE

(Taken from Mackenzie, 1979.)

2.3 Assessment of Ore Reserves

Analysis of the sampling data by conventional, statistical or geostatistical computation techniques provides a basis for estimating the tonnage and grade characteristics of a deposit. Initially, the tonnage-grade function of a porphyry deposit establishes the relationship between the cut-off grade variable and the available mineral inventory. These estimates are established for the deposit as a whole. Different types of ore reserves (proven, probably and possible) are defined according to the level and reliability of the sampling information, the economic conditions, mine and plant recovery and the dilution factors. An outline of the different steps and elements involved in the assessment of ore reserves is given in Figure 45.

These ore reserve assessments, together with estimates for possible revenue, costs and government policy conditions, are used for the economic evaluation of the technically feasible alternatives. The alternative embodying the optimum cut-off grade and ore reserves for a certain plant capacity is then finally chosen.

2.3.1 Definitions of Ore Reserves

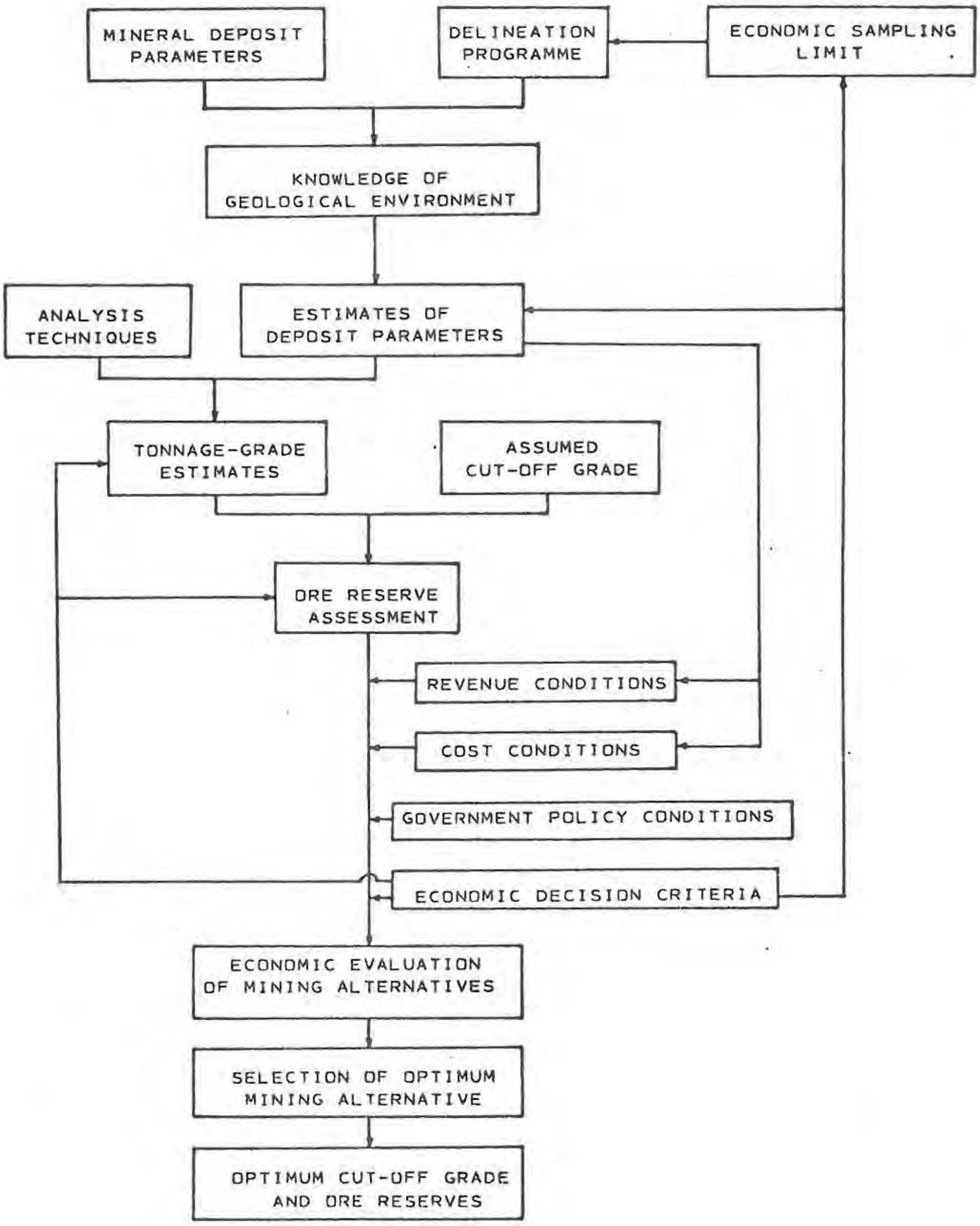
The normal procedure used to define ore reserves is outlined in Figure 46.

Different classes of *in situ* ore reserves, defined by the United States Geological Survey and the United States Bureau of Mines, have been used since 1943. This Bureau of Mines classification is based on the relative amount and type of delineation and on the distances over which sample values are projected:

"Proved" or "Measured" *in situ* ore for which tonnage is computed from dimensions revealed in outcrops, trenches, workings, and drill holes and for which the grade is computed from the results of detailed sampling. The sites for inspection, sampling, and measurement are so closely spaced and the geologic character is so

Figure 45

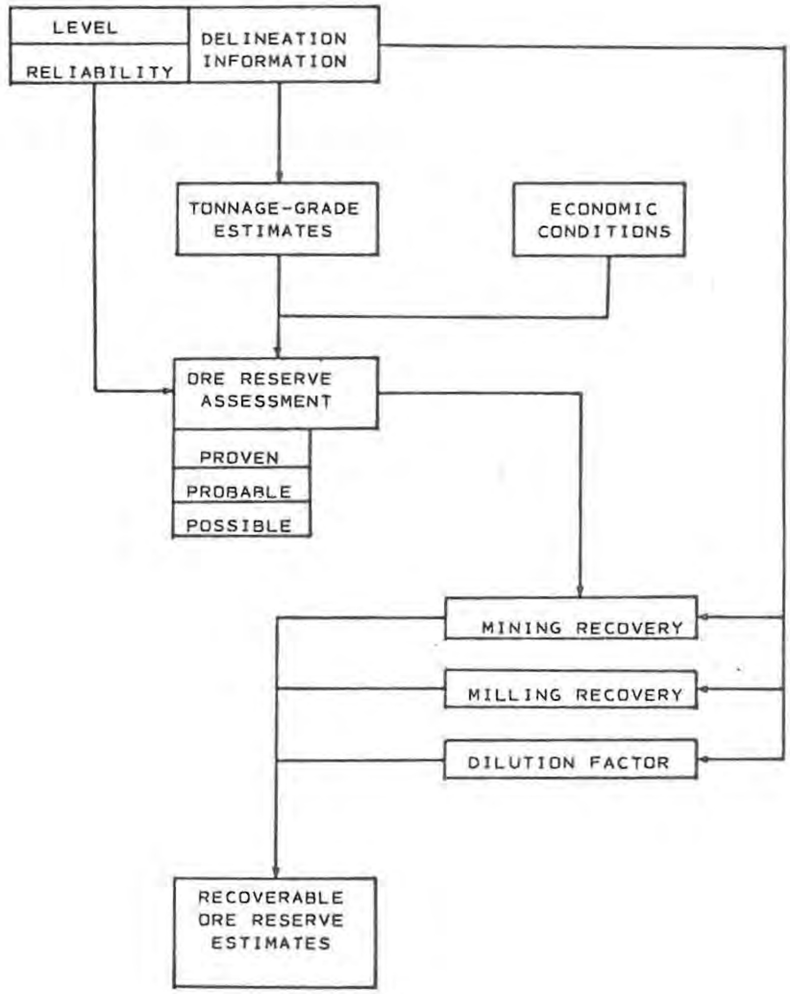
TONNAGE-GRADE ESTIMATION AND ASSESSMENT OF ORE RESERVES



(Taken from Mackenzie, 1979.)

Figure 46

DEFINITION OF ORE RESERVES



(Taken from Mackenzie, 1979.)

well defined that the size, shape, and mineral content are well established. The computed tonnage and grade are judged to be accurate within limits which are stated, and no such limit is judged to differ from the computed tonnage or grade by more than 20 per cent".

"Probably" or "Indicated" in situ ore is ore for which tonnage and grade are computed partly from specific measurements, samples, or production data and partly from projection for a reasonable distance on geologic evidence. The sites available for inspection, measurement, and sampling are too widely or otherwise inappropriately spaced to outline the ore completely or to establish its grade throughout".

"Possible" or "Inferred" ore is ore for which quantiative estimates are based largely on broad knowledge of the geologic character of the deposit and for which there are few, if any, samples or measurements. The estimates are based on an assumed continuity or repetition for which there is geologic evidence; this evidence may include comparison with deposits of similar type. Bodies that are completely concealed may be included if there is specific geologic evidence of their presence. Estimates of inferred ore should include a statement of the spacial limits within which the inferred ore may lie".

Applications of these definitions at the different porphyry deposits vary according to the geologist's appreciation of the characteristics of each deposit, and safety limits imposed by the company involved, for example at Sar Cheshmeh the proved in situ class has been applied to all blocks intersected by a vertical drill hole on the 100 m grid; probable ore is the tonnage from the bottom of a drill hole to the 2.350 m bench elevation, which corresponds roughly to the supergene-hypogene mineralization contact. The probable ore in this case is based on an acceptable projection of the grade trend within the supergene enriched zone. Possible ore in this deposit exists below the abovementioned level, and where the deposit is still open in depth (Selection Trust Ltd., 1970).

Generally, the actual in situ tonnage and grade are expected to be within 90-95% of the estimate for proved ore and within

70-80% for probable ore. Possible ore is considered more in the category of an exploration target.

Economic and mining conditions, in terms of a cut-off grade, are then imposed on the in situ proven and probable tonnage-grade estimates in order to assess the actual "ore reserves" - or mineable ore reserves. Estimates of mine and plant recovery and dilution factors are used to convert the in situ reserve assessment to a recoverable-ore basis. The economic dimension associated with the cut-off grade also involves the relationship existing between cost and revenue. Recoverable reserves are estimated for each technically and economically feasible mining alternative considered.

Changing geological and economic conditions over time shift the ore reserve boundaries, e.g. mine production exhausts ore reserves; exploration results in the discovery and delineation of unknown economic reserves and the "proving up" of known reserves; favourable changes in the cost-revenue relationship reduce cut-off grade and thereby convert known uneconomic resources (e.g. hypogene mineralization zones in porphyry coppers) to ore reserves.

2.3.2 Computation Methods

Conventional statistical and geostatistical methods have been and are being used to compute ore reserves in porphyry deposits. The transformation of the drilling data by means of computation methods into block units, particularly mineable block units, should commence early during the exploration-evaluation stage in order to achieve a better understanding of the geological factors involved in ore reserve estimation and mining of a deposit.

a) Conventional Methods

Estimation of tonnage and grade parameters in porphyry deposits has been made by using cross-sectional, bench by bench, polygonal, isograde surfaces or by moving average methods. The sectional method may be based on the construction of either polygons or triangles (Fig.47).

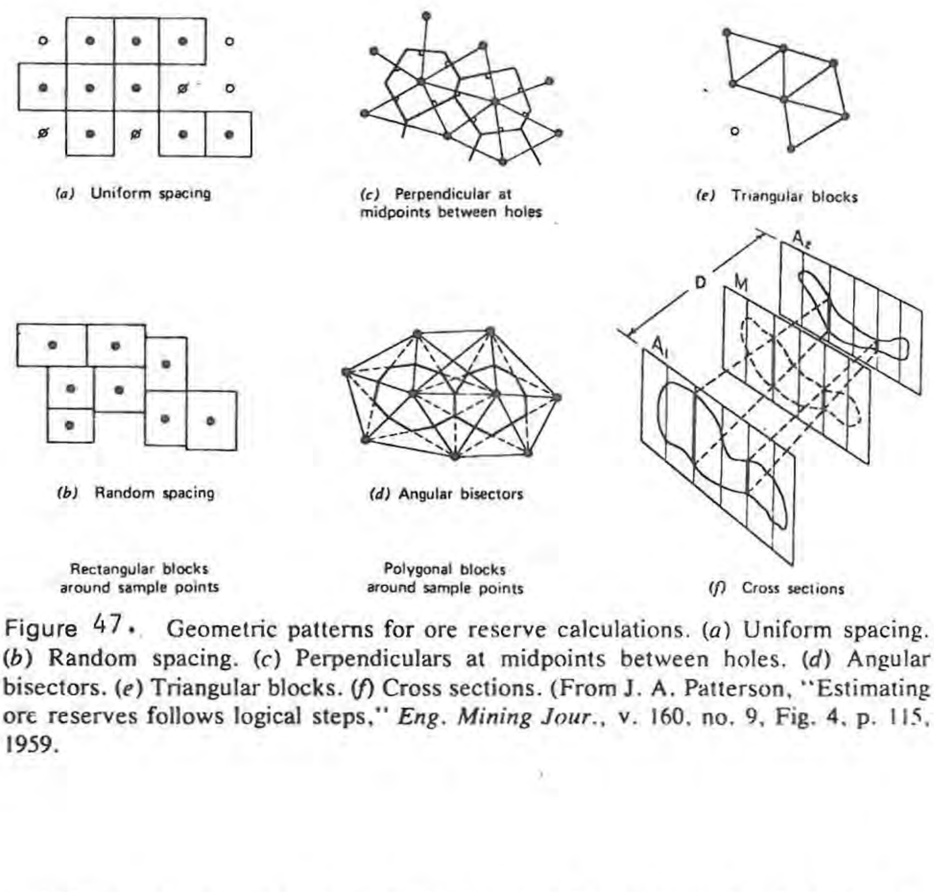


Figure 47. Geometric patterns for ore reserve calculations. (*a*) Uniform spacing. (*b*) Random spacing. (*c*) Perpendiculars at midpoints between holes. (*d*) Angular bisectors. (*e*) Triangular blocks. (*f*) Cross sections. (From J. A. Patterson, "Estimating ore reserves follows logical steps," *Eng. Mining Jour.*, v. 160, no. 9, Fig. 4, p. 115, 1959.

These different computation methods are described in detail by Patterson (1959), Hazen (1968), Raymond (1972), Popoff (1966) and others.

All the conventional methods use basically the same data. After the mineral deposit has been adequately delineated, it is subdivided into blocks of various shapes. These blocks depend on the overall shape and attitude of the deposit, the drilling pattern used for delineation, the minimum size of blocks appropriate for selective mining, and the method of estimation chosen. Each block is approximated by a geometric figure to which the average grade of its samples is given. A volume is computed based on the geometric relationship and then converted to a tonnage by multiplying by the estimated specific gravity of the rock. Block values are combined to assess the weighted average tonnage and grade of the overall deposit. Mineable ore reserves are determined by combining all mineable blocks with grades above the cut-off grade.

Certain block limits are governed by geologic, mining and/or economic conditions. At Chuquicamata, for example, the presence of the West Fault dictates the western limit of the "s" blocks (of either Mo, or As, Cu-MoS, etc. mineralogy) that are emplaced in the geologic unit defined by the sericitic zone (Fig.53). There may be other natural, structural, or economic limits in porphyry deposits - alteration and mineralization zoning, leached zones (e.g. El Salvador, Chuquicamata), water level, property limits or mining limits (stripping ratio, minimum mining widths, depth, etc.)

The attainment of correct tonnage estimations requires the determination of specific gravities for the different rocks (ore and waste) to be mined. The specific gravity of the rocks can be calculated from it mineralogy, as explained by Koch and Link (1971), by determining the specific gravity of samples, or by weighting the rock extracted from measured excavations. The last method, while more time-consuming than the others, is the most accurate if large samples and well-measured sample volumes are used. Variations in specific gravity between these rocks can lead to significant errors in the reserve estimation, e.g. Table II shows the different specific gravities for the rocks at Sar Cheshmeh.

The main attempts and disadvantages of the conventional computation methods are:

1. <u>Direct correlation method</u> : ore grades are classified into regular intervals and correlated with the grades of the same interval of adjacent drill holes. Results of this

TABLE 11 - SPECIFIC GRAVITY AND POROSITY VALUES

Rock Type	Zone	No. of Determinations	S.G.'Dry'	S.G.'Wet'	'Porosity' Percent	'Solid' S.G.
Argillised Porphyry	Leached	16	2.34	2.44	9.4	2.59
	Supergene	20	2.38	2.49	10.2	2.65
	Hypogene	22	2.42	2.48	7.4	2.62
Fine Porphyry	Leached	18	2.20	2.35	15.0	2.58
	Supergene	14	2.36	2.51	9.1	2.71
	Hypogene	18	2.48	2.57	9.5	2.70
Fine Porphyry (Pyrite Halo)	Leached	18	2.15	2.32	15.6	2.61
	Supergene	18	2.48	2.58	9.6	2.76
	Hypogene	17	2.55	2.64	10.7	2.78
Dyke	Leached	19	2.30	2.41	11.7	2.60
	Supergene	13	2.48	2.56	7.8	2.71
	Hypogene	20	2.49	2.56	4.8	2.71

(Taken from Selection Trust Ltd., 1970.)

method in Chuquicamata were in general acceptable but the risk exists of artificially correlating ore grades of different lithologic units, which can lead to false interpretations. At Chuchicamata this system gave the best results in zones drilled by inclined holes (Ambrus, 1973.)

2. Polygonal method : the same area of influence is assigned to all the grades; symmetrals are drawn determining irregular polygons (see Fig.47). At the Chuquicamata, Pampa Norte and El Abra (Chile) porphyries this method gave unsatisfactory results in the higher grade zones. This is probably because the disseminated, veinlet and vein mineralization have different areas of influence and anisotropies (op.cit.). This method also gave unsatisfactory results at the Similkameen open pit porphyry copper mine (B.C.), in which mineralization is extremely erratic and occurs as disseminations, large blebs, as fracture fillings and in discontinuous veinlets. In this deposit

the misplacement of ore-waste boundaries by applying this method(Fig.48) resulted in an underestimation of the ore tonnage by 15% to 45% and an overestimation of ore grade by 20% (Raymond, 1972). This is even considering that the Similkameen porphyry deposit was drilled vertically on a 30 m grid spacing and that a horizontal exploration adit and several horizontal drill holes from it confirmed the initial polygonal estimates based on the drill holes (Raymond, 1972).

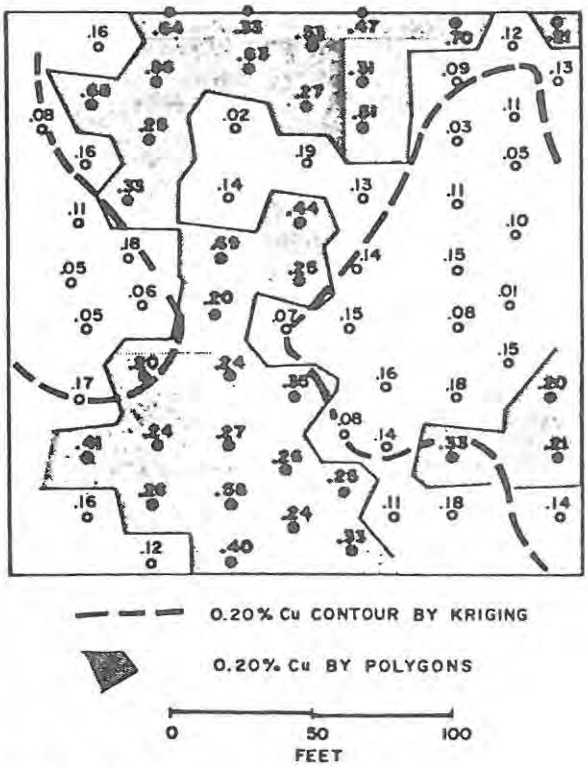


FIG. 48 Kriging from blast-hole assays.

(Taken from Raymond, 1972.)

Similar biased results in the estimation of tonnage and grade have also been obtained by this method (cubic blocks) at the Mantos Blancos porphyry copper mine (Chile), at Cyprus Pima, U.S.A., etc.

At Cyprus Pima it was also recognized that the distribution of ore and waste within each block is the factor that should be considered to reduce the overestimation of grade and underestimation of tonnage. These errors result from the application of such a conventional computation method on the grid drilling. To determine these errors, the number of blast holes in each block having grades above and below the originally estimated cut-off was recorded. The average grade of each subdivision of the block was also computed. The results are shown in Figure 49 for the 0.25% cut-off and for the 0.30% cut-off. Figure 49 is interpreted as following: for a cut-off grade of 0.25% Cu, given a block which is estimated to average 0.30%, one will in fact recover 73% of ore above 0.25% Cu, averaging 0.38% Cu while 27% averaging 0.21% Cu will have to be sent to waste.

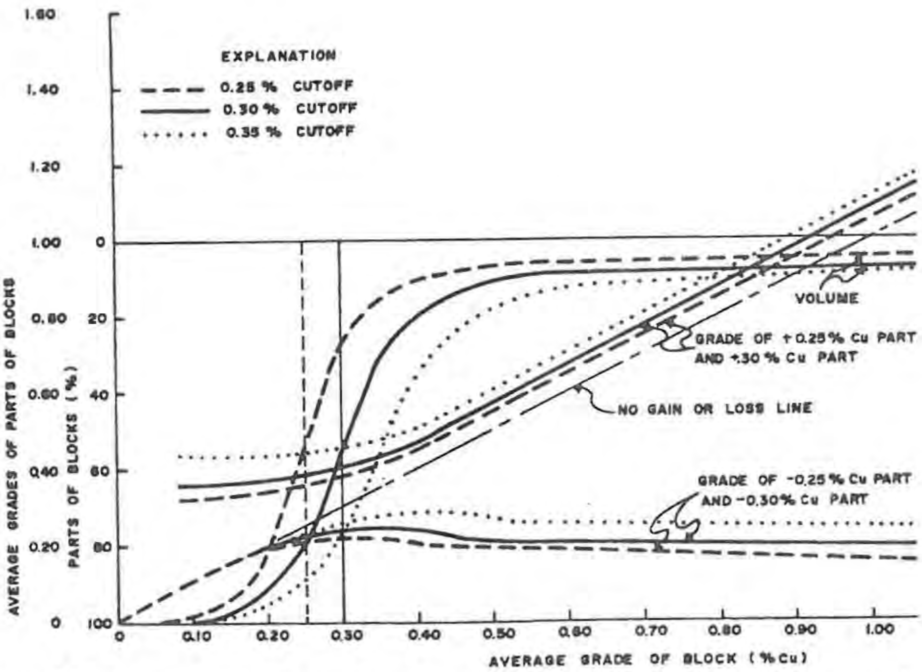


Fig. 49 . Proportion of ore above and below the 0.25%, 0.30% or 0.35% Cu cut-off and average grade of these tonnages given an estimated average grade of the block. After Williamson and Mueller (1976).

(Taken from David, 1977.)

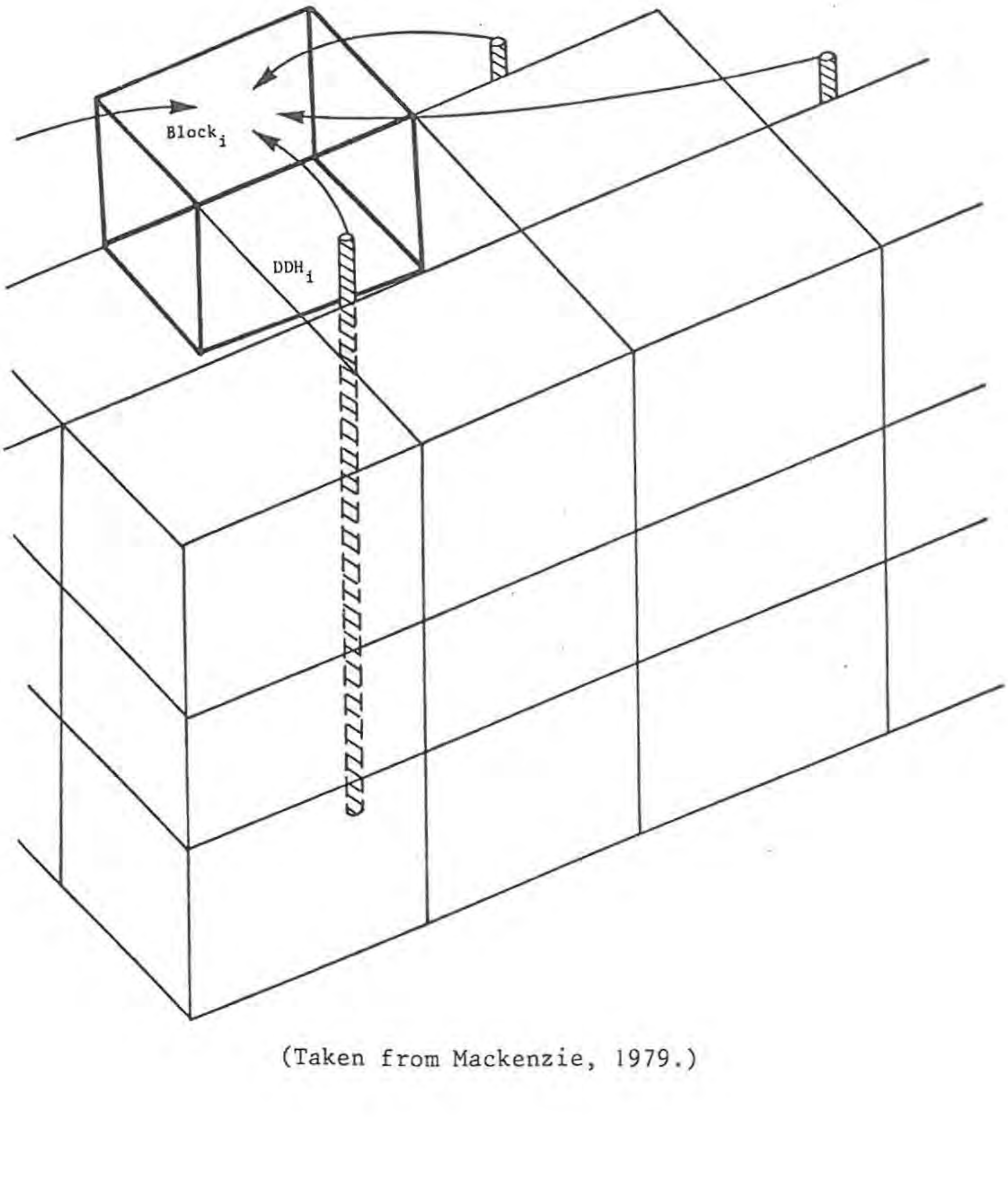
3. Isograde Surfaces : the variations of grade between two points is assumed to be lineal. Surfaces of equal grades are drawn and isograde curves obtained on maps or sections. The isograde curves may reflect weakly the anisotropy of the porphyry deposits, but at the Chuquicamata, El Abra, Pampa Norte porphyries, the grades estimated by this mean were considerably overvaluated (Ambrus, 1973). This overvaluation of grade is produced due to strong to weak grade variations existing because of geological controls such as rock contacts, alteration zoning, veins etc. that are not considered in the application of this method.

4. Bench by Bench Method : this method uses the composite values of grid-drill hole samples within an open pit bench interval or underground level. At Sar Cheshmeh a bench block is a horizontal slice measuring 100 x 100 m in plan with a drill hole in its centre. The 12.5 m height of the block corresponds to the planned mining benches. Copper assays of the drill hole intersection for each bench at Sar Cheshmeh were weighted by length, without reference to zone or rock type. Total ore reserves were computed by adding all the blocks above an arbitrary cut-off grade of 0.4% total Cu. All these blocks must also be emplaced between the two levels (2600 m and 2350 m) that roughly define the supergene enriched zone in this deposit. Different geological ore zones within a block (e.g. rock types, alteration zones, etc.) were evaluated separately to avoid major estimation errors. The application of this bench by bench method at Sar Cheshmeh cannot be assessed yet because no mining has been undertaken, but it was assumed that losses and gains produced by low-grade blocks that cut into the ore core and blocks at the periphery of the deposit with a decreasing grade towards the deposit's margins will off-set each other (Selection Trust Ltd., 1972). However, a similar assumption made at Similkameen (B.C.) and at Mantos Blancos (Chile), but using the polygonal method, grossly over-estimated grade

and under-estimated tonnage.

5. Ivor Method : this method assumes that the grade at one point is influenced by the adjacent grades according to the reciprocal of the square root of the distance. This moving average method is mainly applicable to porphyry deposits where block values for open pit mining are required. In this case the deposit is divided into blocks of uniform size as shown in Figure 50.

Figure 50.
MOVING AVERAGE METHOD



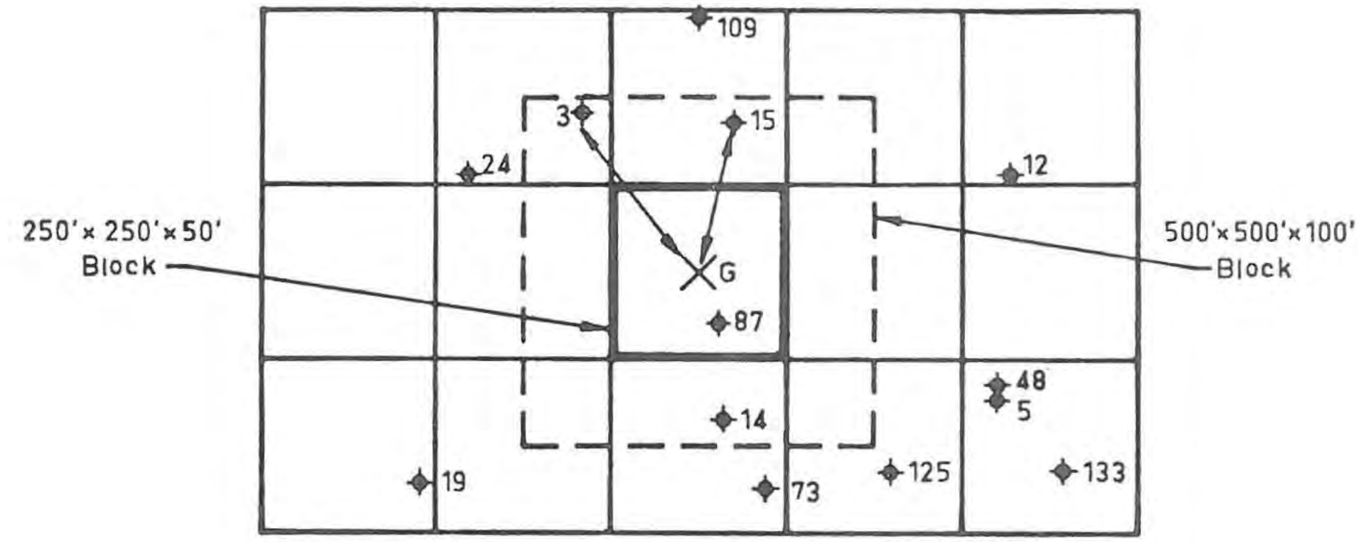
(Taken from Mackenzie, 1979.)

The average grade of each block is computed from the weighted sum of the observations near the block. Some moving average methods use the inverse distance, inverse cubic distance, etc. as weighting factors between the assay values. The advantage of the moving average method is that it reflects increasing or decreasing grades from the geologic contacts. A tonnage-grade curve can also be computed from these results. The moving average method based on the "square root of the distance" weighting gave satisfactory results at Chuquicamata as long as this method was applied independently for each of the geological units distinguished within the deposit (see Fig.54).

At the Panguna porphyry copper deposit (Bougainville), evaluation diamond drill assays on a 122 m grid served as basis data for the zoning program, in which a moving average technique was also used to calculate grade values for node points on a regular three-dimensional grid. The volume scanned for each node value was 150 x 150 x 30 m, with a 50% overlap (Fig.51), on all three axes. This 50% overlap at Panguna was considered because of the fairly wide spacing of the drill grid relative to the manner in which the degree of mineralization changed.

This procedure produced node values on a 75 x 75 x 15 m matrix for part of each mine bench. The node values were than contoured at different class intervals, across geological boundaries to delineate several ore-grade zones (Fig.52).

The proved initial ore reserves at Panguna were calculated by assigning each of the node values to its area of influence (75 x 75 x 15 m) and computing tonnage and grade of ore within economically viable pit limits. After production commenced at Panguna, it was realized that this type of approach had not incorporated geological mineralization controls, especially the contacts between the early-stage - high-grade intrusions with the late-stage intrusions containing low-grade copper (Baldwin, 1978).



$$G = \frac{\sum_{i=1}^{h} g_i \times \frac{1}{d_i}}{\sum_{i=1}^{h} \frac{1}{d_i}}, \text{or } G = \frac{(g_{15} \times \frac{1}{d_{15}}) + (g_3 \times \frac{1}{d_3}) + \ldots\ldots}{\frac{1}{(d_{15})} + \frac{1}{(d_3)} + \ldots\ldots}$$

Fig.51. *Generation of block averages.*

(Taken from Blackwell, 1973.)

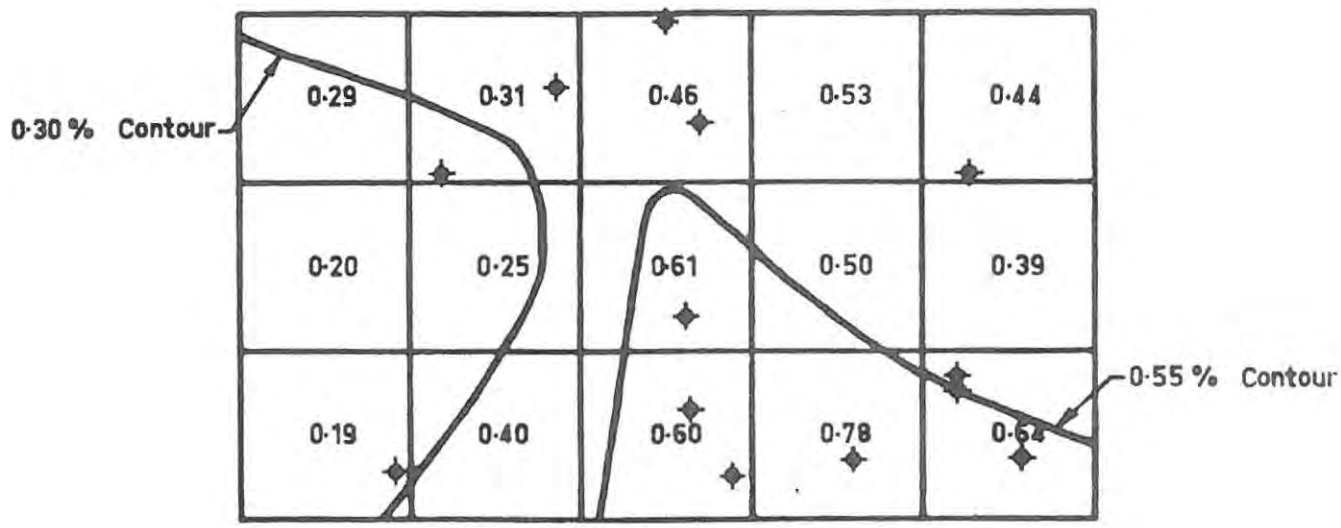


Fig.52 *Interpolation of contours.*
(Taken from Blackwell, 1973.)

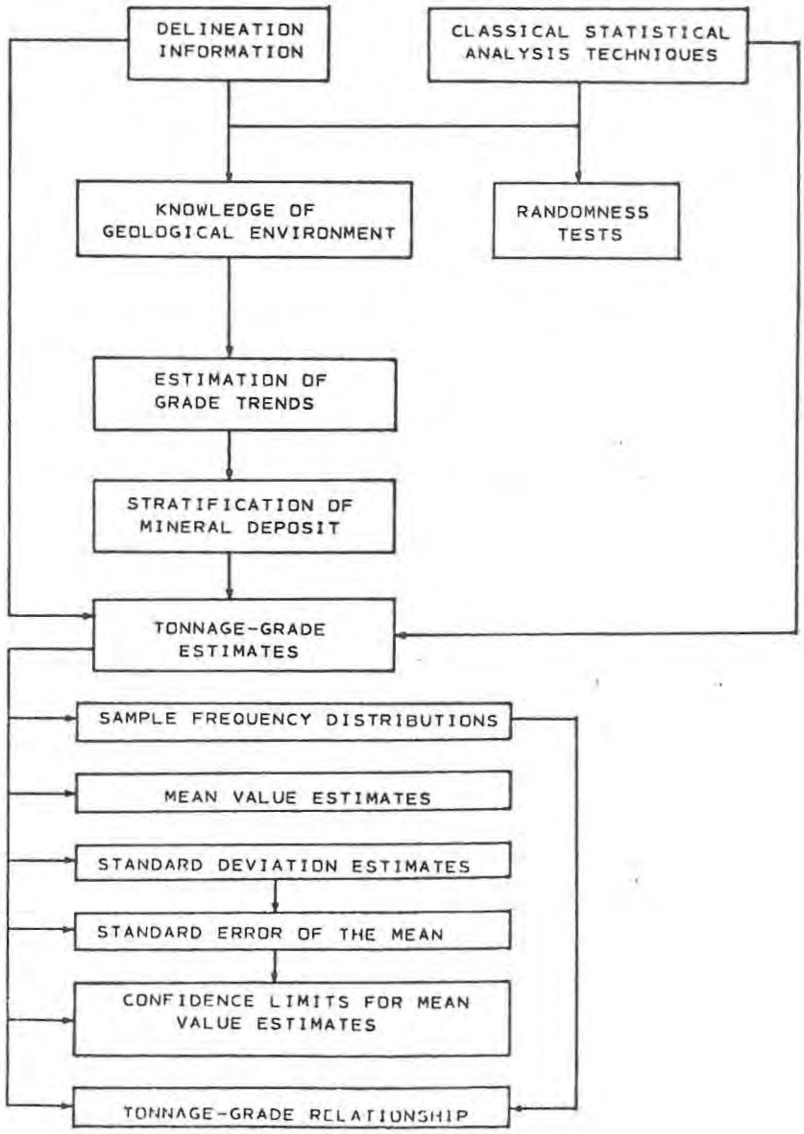
Higher-grade drill hole copper values of the early-stage intrusions were averaged in with the lower values of the cross-cutting late-stage intrusions, thus indicating more ore than was actually present. Ore reserves had consequently to be recalculated by taking into account the geological contacts to describe the limits of the waste blocks.

In general, it can be stated that conventional methods described do not provide a measure of confidence in the overall tonnage and mean grade estimates of porphyry deposits. Usually the area of influence given to individual drill hole assays far exceeds their actual area of influence. The actual mean grade contained within such an area may be quite unrelated to the drill hole assays obtained. Knoweldge of the uncertainty associated with these estimates, particularly grade, is important for evaluating the economics of mine development. Therefore, use of the conventional methods in porphyry deposits precludes risk analysis and the assessment of risk associated with profitability criteria.

6. Statistical Methods : the statistical approach should not be regarded as a substitute for conventional methods, but as an additional method to improve ore reserves estimates (Cooke, 1977). Statistical analysis provides the framework for assessing the uncertainty or reliability of the geological estimates. Classical statistical techniques are based on the assumption that observed variations are due to random fluctuations, i.e. that the sampled data are independent. This means that there are no trends in the data. Trend in a porphyry deposit is commonly a gradual change and is caused by alteration zoning, rock types and structures (see chapter 2.2.1.c, and Fig.17).

An outline of the use of classical statistical analysis techniques for tonnage-grade estimation is given in Figure 53. Results of randomness tests made on the sampling information together with a knowledge of the geological environment are used to estimate grade trends within the deposit and to zone the deposit correspondingly. These zones are usually geologic units characterized by having the same alteration-mineralization and rock characteristics (e.g. Fig.54). Statistical analysis techniques are then

Figure 53.
CLASSICAL STATISTICAL ANALYSIS OF TONNAGE AND GRADE



(Taken from Mackenzie, 1979.)

applied to the sampling information within each zone - or geologic unit - to estimate mean (expected) value, standard error of the mean, and confidence limits for:

i) the mean value estimate
ii) the deposit's tonnage and grade
iii) the tonnage-grade relationship.

The standard error of the mean defines the possible distribution of actual mean values about the mean value estimate, e.g. the reliability or confidence limits of the mean value estimate. For reasonably large sample sizes, this meanvalue distribution will approximate the normal distribution, even if the sample frequency distribution is skewed (e.g.see Fig.20). The mean square successive-difference test is used to assess this randomness. Randomness in the third dimension (within each of the drill holes) may be controlled by adjusting the sample size used, e.g. there will be greater independence or randomness between successive sample values if 3 m sample lengths are taken instead of 2 m lengths.

Classical statistical analysis is therefore a special case, appropriate to apply in the early stages of exploration whendistance between samples is greater than the range of the variogram (see chapter 2.2.1.c(ii)). For more closely spaced samples, it is clear that the use of classical statistical analysis will lead to an overestimation of variability, uncertainty and, ultimately, investment risk. Under these circumstances, geostatistical analysis will be required.

Geostatistical Method

As already indicated, analysis of non-random data and trends by the variogram function (chapter 2.2.1.c(ii)) help in defining homogeneously mineralized units. Classic statistical ore reserve computation methods can then be applied

To evaluate chese homogeneously mineralized units. This combination of geostatistical-statistical methods of ore reserve computation has proved to be adequate in most porphyry deposits (e.g. Chuquicamata, Pampa Norte, El Abra, etc,; Ambrus, 1978).

Nevertheless, the best geostatistical computation method used for the evaluation of porphyry deposits is "kriging". Kriging has the advantage of considering the calculation of the errors of estimation (Royle, 1977). The assessment of this error of estimation is becoming of increasing importance because as lower-grade porphyry deposits are being mined, the margins of allowable error in ore reserve calculations and risk analysis are also decreasing.

Two basic geostatistical concepts are introduced for ore reserve estimation by kriging - extension and error of estimation. Detailed descriptions about the kriging computation method are given by David (1977), Krige (1978), Rendu (1978), and others.

In kriging, the contents of panels of ore are estimated by "extension" from the samples inside and near to them by weighting the sample values in such a way that the errors of estimation are minimized. If Z is called the true grade of a block and Z^* the estimated grade, then the error of estimation is $\varepsilon = (Z-Z^*)$. A good estimation procedure will yield estimates that have an average equal to the mean of the true values (i.e. non-bias and no systematic error) and a small variance or dispersion of errors. The variance of the error of estimation, known as the estimation variance, is expressed by $VAR(Z-Z^*) = VAR(\varepsilon)$. This expands to:

$$VAR(\varepsilon) = VAR(Z) - 2\ COV\ (\ (Z)(Z^*)\ )\ +\ VAR(Z^*)$$

The three terms in this expression are computable from the variogram function. This estimation variance replaces the standard error of the mean used in classical statistical analysis.

The validity of an estimation by kriging is therefore assessed by the magnitude of the error of estimation, e.g. the difference between the true and the estimated grade. The minimum mean squared errors of estimation are called the kriging variances, or σ_k^2.

Ore reserve estimation by kriging involves the following steps:

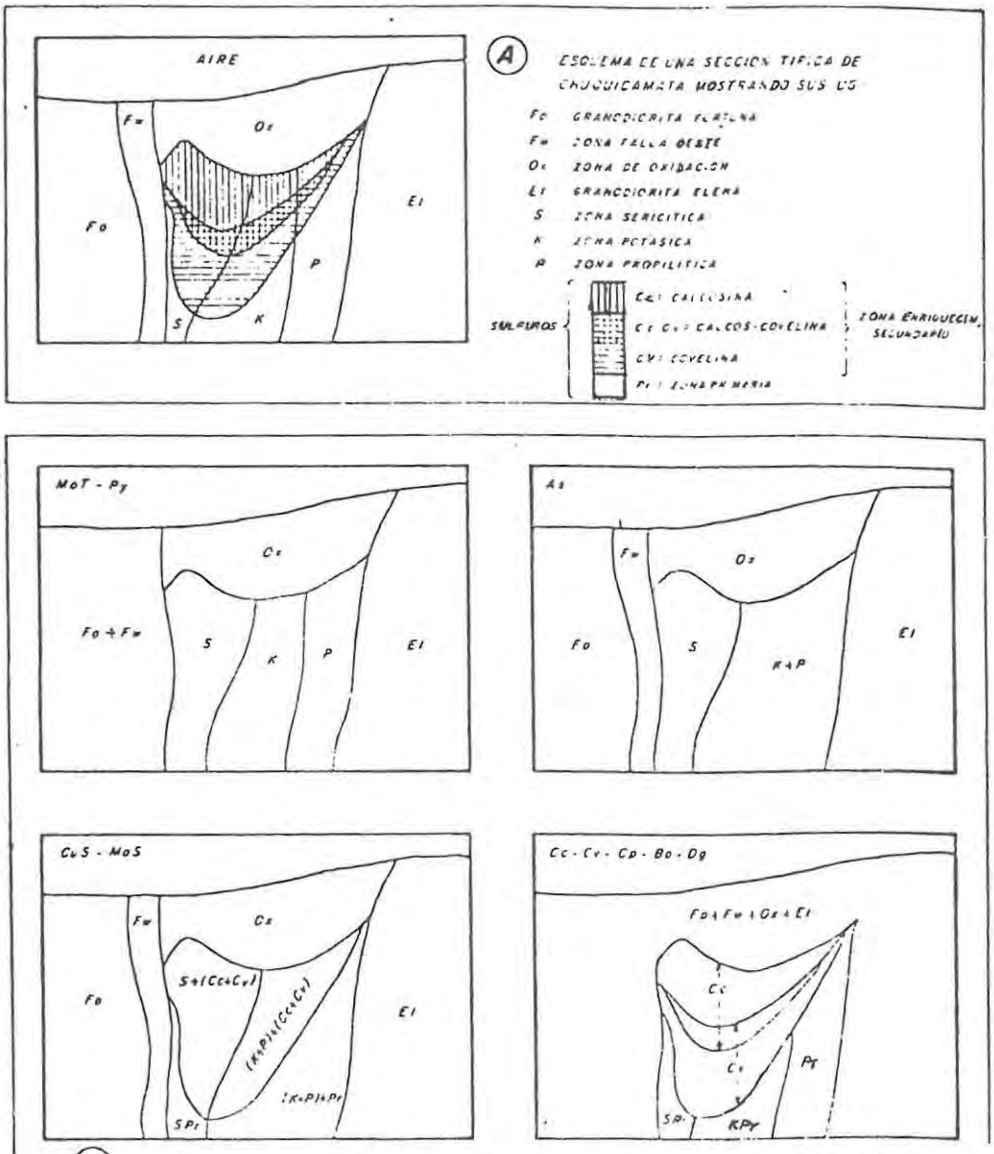
(i) The variograms of the variables of interest are determined and models fitted to them. These variograms should be obtained for each zone of homogeneous mineralization; e.g. at Chuquicamata 16 geological units (Fig.54) have been distinguished after studying the distribution in the deposit of 11 variables such as (Ambrus et al., 1978): copper grade, molybdenum grade, arsenic content, percentage of the different sulphides present because of its influence on the grade of the concentrate, etc.

(ii) Each of these geological units having homogeneous mineralization is divided into blocks of a certain size (e.g. 20 x 20 x 13 m at Chuquicamata) which is normally the minimum size unity (20 x 20 m) and the most appropriate bench height (13 m at Chuquicamata) for selective mining purposes.

With a regular sampling grid the blocks considered for kriging purposes have usually the dimension of the drill grid spacing (each block with a borehole in its centre) or have the dimension of an integral fraction of it.

(iii) Each panel is then estimated by kriging, a computer almost certainly being needed for this procedure.

Figure 55 shows two sampling patterns in which the central block is to be estimated in each case.



Ⓑ Diagrams showing the regrouping of the geological units at Chuquicamata.

Fig.54 (Taken from Ambrus et al., 1978).

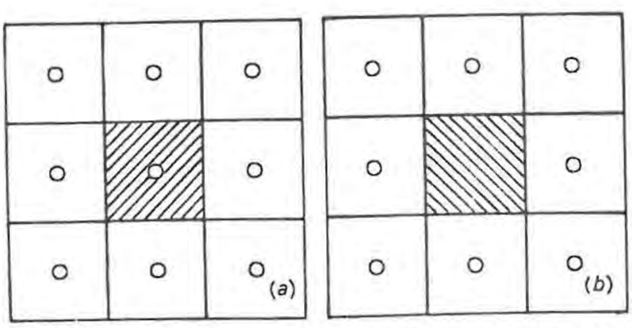


Fig. 55 Sampling patterns from which central block of ore is to be estimated

(Taken from Royle and Newton, 1972.)

Pattern 55(a) has a sample in the central block, and this block was estimated, first by kriging and, secondly by giving the block the value of the sample taken in it, i.e. by the polygonal method. In Figure 55(b) no sample is available in the central block, so its value was estimated first by kriging and then by using the method of weighting by "inverse squares". The best estimator will have the regression slope (of actual values on estimated values) which is nearest to unity. Kriging produced the best forecast of what will be recovered after a selection of payable blocks, and the polygonal method by far the worst (Table 12).

Table 12 Regression slopes of different estimators for sampling patterns of Fig. 5 and data of model no. 8439576

	Regression slope			
	Pattern 5(*a*)		Pattern 5(*b*)	
$\sigma^2(O/P)$	Kriging	Polygonal	Kriging	Inverse squares
40 000	0·983	0·438	0·987	0·890
48 400	0·911	0·469	0·914	0·860
57 600	1·042	0·360	1·049	0·854
67 600	0·933	0·310	0·931	0·799
Means	0·967	0·394	0·970	0·851

(Taken from Royce and Newton, 1972.)

No matter what method is used to determine the selection of payable panels, it will not be completely correct. Some payable panels will be left unmined and unpayable panels will be mined. (Fig.57). Nevertheless, kriging gives the most accurate and unbiased estimate of what will be recovered. Graphs are then drawn of tonnage of ore reserve versus cut-off limits and mean grade of ore reserve versus cut-off limits. From these it is possible to estimate the tonnages available, and the mean grade of what will be mined, at different cut-off limits and for any estimator (Fig.56). For example, from Figure 56, for a cut-off limit of 150 in dwt, the indicated mean grade of what will be mined obtained by the polygonal method, is 490 in-dwt, or 22.5% too high, and the indicated ore reserve is 1215 blocks, which is 19% too low.

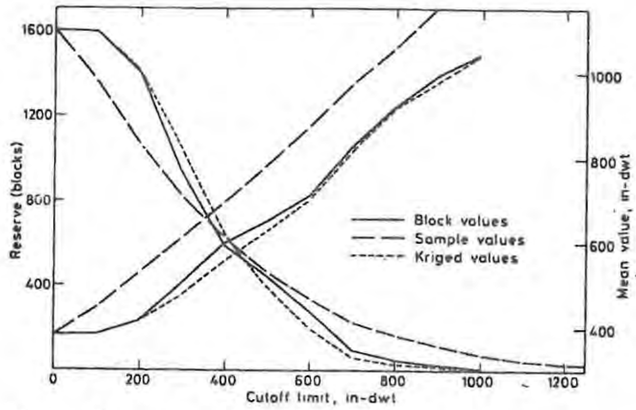


Fig. 56 Ore-reserve/cutoff limit curves

(Taken from Royle and Newton, 1972.)

The kriged estimates agree almost exactly at 150 in dwt, which is very similar to the values of the basic blocks.

Figure 57 shows the elliptical outline of a two-dimensional array of estimated/actual values. Z^* is a cut-off limit based on estimated values and Z is the same numerical value on the actual value axis. The area to the right-hand side of the vertical line through Z^* will contain the blocks selected for mining, as all these blocks have values apparently higher than Z^*. The area above the horizontal line through Z will contain the blocks whose true values are higher than Z . These two lines divide the ellipse into four areas.

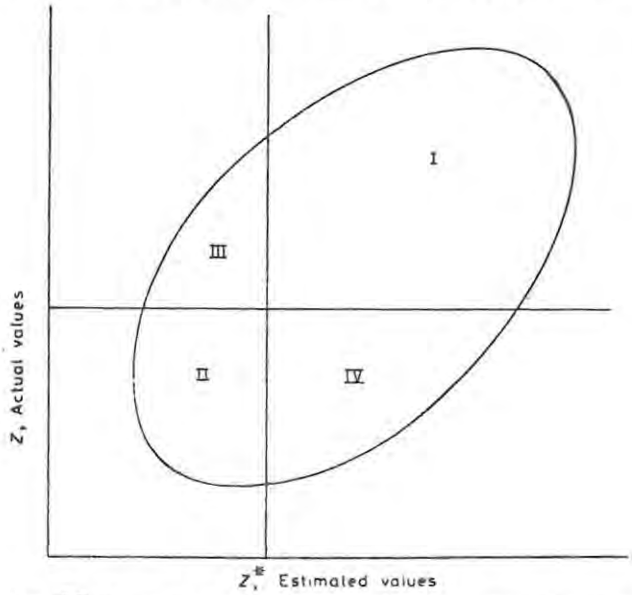


Fig. 57 Allocation of blocks of ore into payable and unpayable groups

(Taken from Royle and Newton, 1972.)

Area I contains the blocks estimated to have values higher than Z. In area II there are blocks estimated to have values lower than Z. Area III contains the blocks estimated to have values below Z, but which, in fact, have higher values. Area IV contains the blocks which were estimated to have higher values than Z, but which, in fact, had lower values. Thus, areas I and II contain the blocks which were assessed correctly and areas II and IV those assessed incorrectly.

The application of geostatistics has given excellent results in the ore reserve estimation of highly erratically mineralized porphyry deposits, e.g. the Similkameen porphyry, B.C. At Similkameen (Copper Mountain), the various intrusive sills, dykes and stocks of different relative ages and mineralization intensity, display complex contact relationships with the rocks of the intruded formation that are also partly mineralized (Raymond, 1972). Comparisons based on 30 million tons of ore indicate that the kriged exploration reserves were within 7% of both tons and grade of mineable ore; as where the original polygon ore reserves under-estimated again ore tonnage by 15% to 45% and over-estimated ore grade by 20%. This kriged ore reserve estimation also improved the pit design and the possibility of a 10% reduction in waste removal at Similkameen. Kriging at this deposit was based on averaging blast hole assays within the 15 x 15 x 12 m blocks and then re-evaluating the block grade as a weighted average to assays within the block itself (Fig.55(a)) and those within the eight or nine surrounding blocks (Fig.58). Each block V is assigned a weighted average of the blast hole values, the weights being chosen to ensure the minimum estimation variance. At Similkameen the best estimate of the block grade was obtained when the 4 or 5 assays (each an average of 4 or 5 blast holes) within each block were weighted by only 40% to 50%. These weighting factors are normally inferred from a series of linear equations.

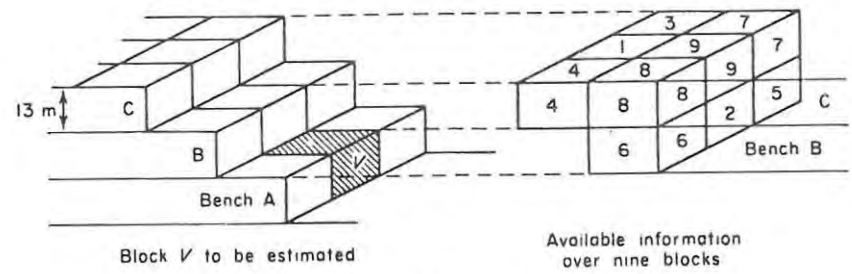


FIG. 58 Short-term kriging at Chuquicamata.

(Taken from Journel and Huijbrecht, 1978.)

Figure 48 shows at Similkameen the ore-waste boundaries at a 0.20% Cu cut-off grade obtained by the polygonal method and by kriging from blast-hole assays from surrounding samples and from adjacent benches. The reasons for the indicated tonnage gain using kriging at Similkameen are obvious from this example, i.e. the established boundaries by kriging are almost coincident with the mining limits (Figs.59 and 60).

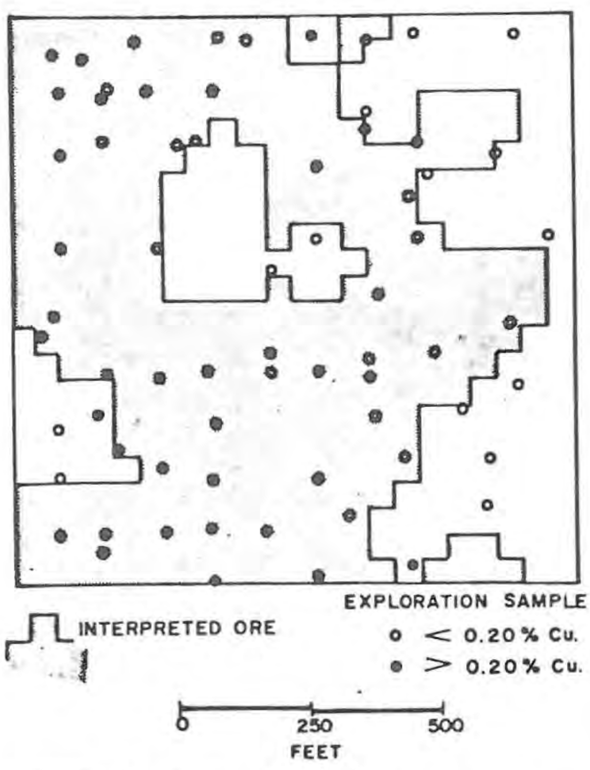


FIGURE 59 Exploration kriged ore within area in Figure 4.

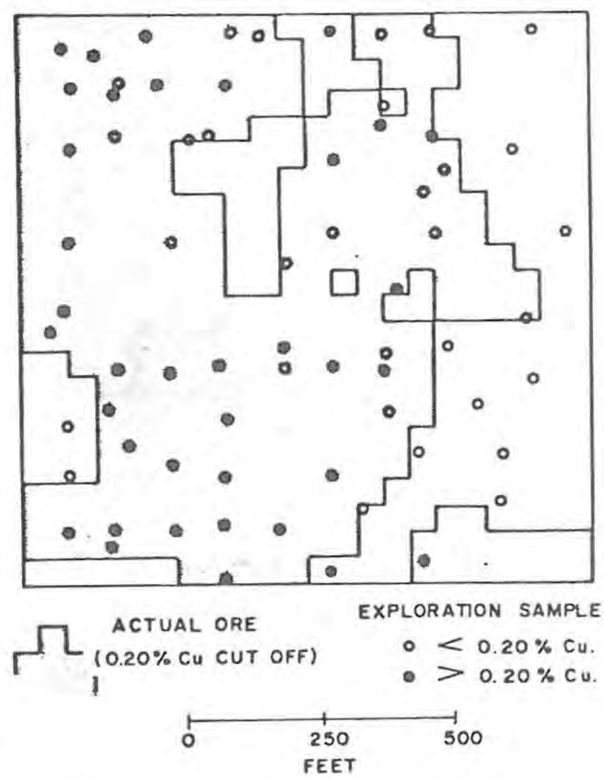
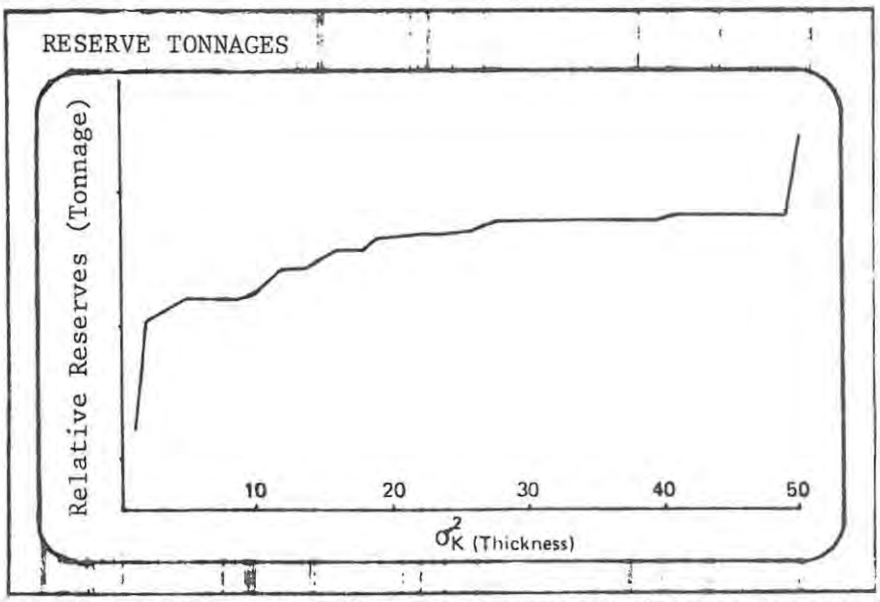


FIGURE 60 Actual ore within area in Figure 4.

(Taken from Raymond, 1972.)

The kriged estimates and kriging variances also serve as basis of a rational system of ore reserve classification; instead of being grouped on a partly subjective basis into proved, probably and possible categories, blocks can be classified continuously according to their kriging variances (Royle, 1977). In the same way as grade and tonnage curves can be drawn up on the basis of grades, similar curves can be determined on the basis of the kriging variances; e.g. by assuming a constant arithmetic variance and a lognormal distribution of errors, a good approximation of the kriging variance was obtained at Similkameen (Raymond, 1972). The top classes of reserves will have the lowest kriging variances, and reserves will be added continuously in ascending order of σ_K^2 (Fig.61). Confidence limits can also be applied if

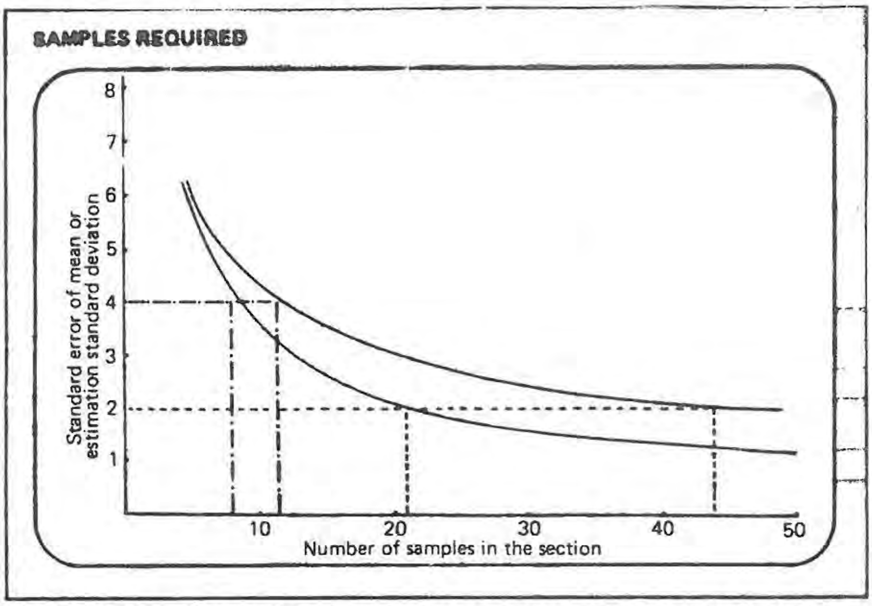


RELATIVE reserve tonnages as functions of mean squared estimation errors. **Figure No.** 61.

(Taken from Royle, 1977.)

the distribution of σ_K^2 is a classical one of errors. The confidence limits for the estimation of individual blocks can be taken as being the sequare root of the kriging variance multiplied by the appropriate figure in 2-tail t-tables for some desired level of probability.

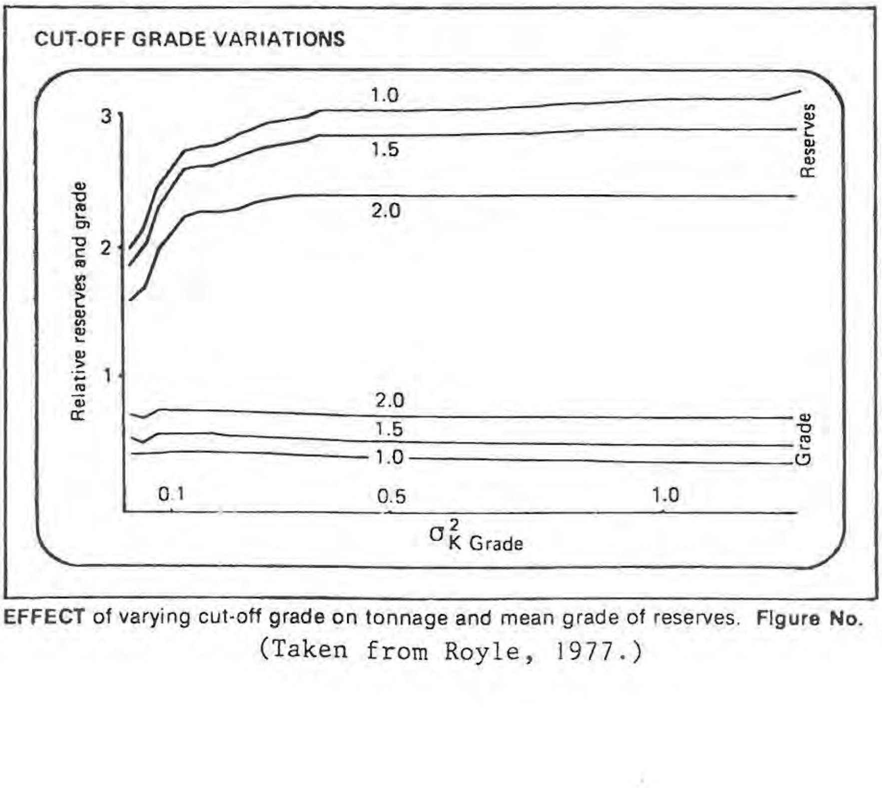
Figure 62 shows a curve in which reserves are separated into two or three main groups according to the degree of accuracy.



NUMBERS of samples needed to produce some desired standard error. **Figure No.** 62.

(Taken from Royle, 1977).

A range of cut-off grades is then tried to see their effects on reserves, as a cut-off grade is not static and can vary throughout a deposit. An example of altering the cut-off grade is shown in Fig.63.



EFFECT of varying cut-off grade on tonnage and mean grade of reserves. **Figure No.** 63.
(Taken from Royle, 1977.)

2.3.3 Economic and Mining Factors influencing Recoverable Ore Reserves

Parameters involved in correct estimations of recoverable ore reserves are (Fig.46):

- an accurate and adequate sample population and assays (refer to chapters 2.2.1 and 2.2.2).
- the application of appropriate computation methods, dimensional measurements and geologic projections (refer to chapters 1.3 and 2.3).
- the determination of specific gravities for each of the rocks to be mined (refer to chapter 2.3).
- cut-off grade: the lowest grade that can be mined economically.
- mining methods and their effect on the potential ore recovery - percentage extraction - and dilution - percentage increase in tonnage during mining with little or no increase in total ore mineral content.
- metallurgical recovery; its effect on the recoverable reserves.

The last three kinds of information are related to economic and engineering factors, although they have fundamental geologic elements:

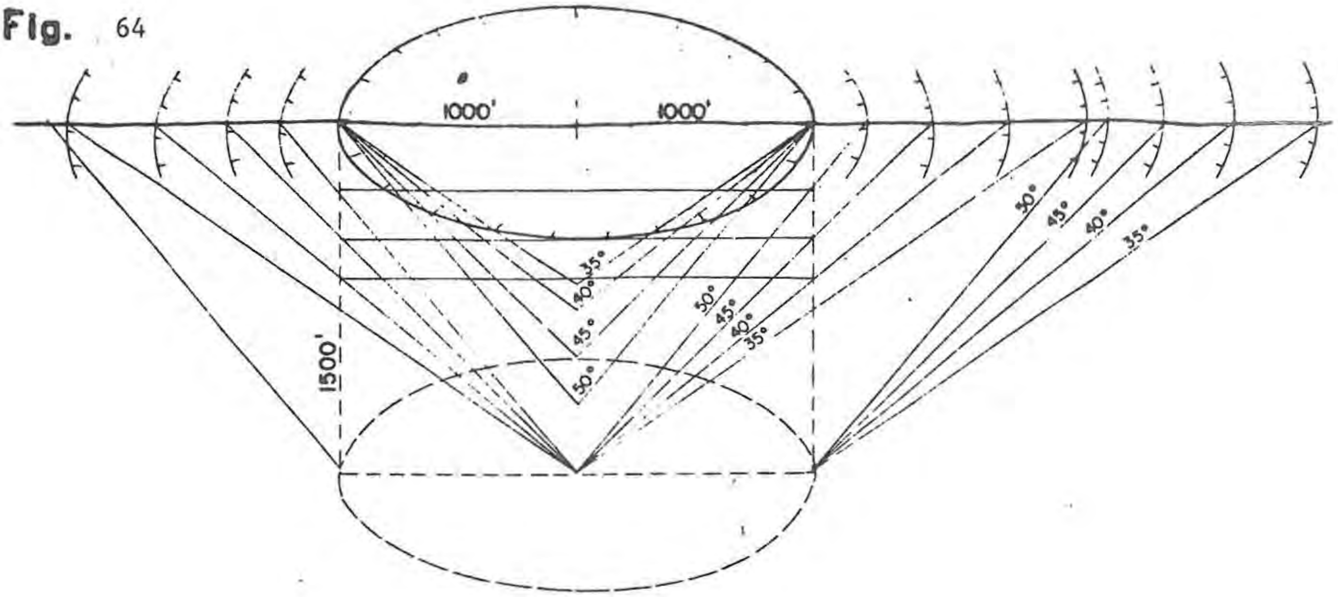
a) Cut-off grade : depends on several factors, i.e. geological, mineralogical-metallurgical and economic. Errors in the cut-off grade estimate due to the application of inappropriate computation methods are also common, e.g. at the Cyprus Pinna open pit porphyry copper mine (U.S.A.) the use of conventional computation methods for the 100' x 100' x 40' grid drilling resulted in the under-estimation of tonnage and over-estimation of grades (average and cut-off grade). However, the blast hole assays in this case consistently detected these errors.

The estimation based on the grid drilling tends to overestimate the cut-off grade of poor blocks, but these blocks were then identified by blast hole sampling and dropped as waste (David, 1977). Hence the usual loss in tonnage and gain in grade when conventional computation methods are applied (see also chapter 2.3.2 and Figure 49).

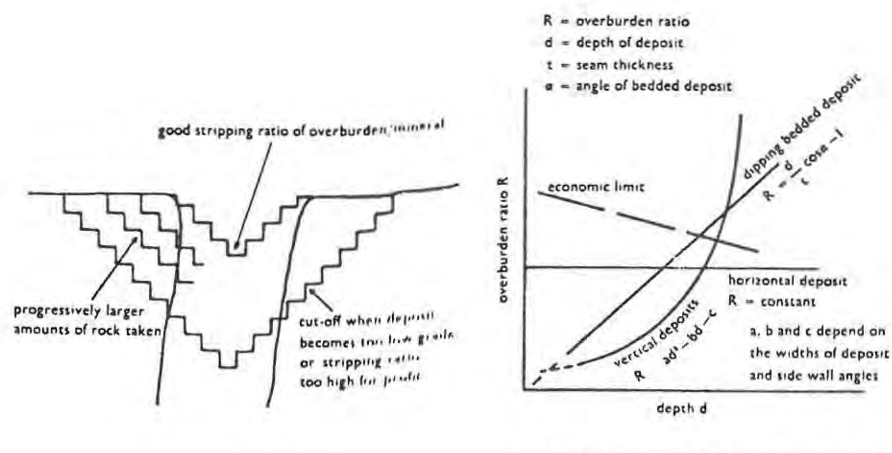
Generally, the cut-off grade of each block considered for mining is determined by analysing its profitability as follows. The expected metal content for a block in conjunction with forcasted sales prices and estimated mine and plant recoveries and dilution, is used for determining the revenue of its block. Production costs through to sales and stripping costs are estimated. Such a reserve calculation for the different tonnage-grade alternatives, each with its respective cut-off grade and stripping ratio, gives a much better appreciation of the economic value of the deposit under changing costs and prices (Seraphim, 1973). The stripping ratio of a pit is the tonnage ratio of over burden plus barren rock volumes within the deposit (waste) to be removed to ore to be mined.

Figure 64 makes it obvious that the stripping ratio (considering overburden only) becomes astronomical for the increments of mineralization near the lower parts of the perimeter of the cylinder. Therefore, for each tonnage-grade alternative a different stripping ratio and cut-off grade is obtained (Fig.65A). The general relationship with depth and stripping ratio (considering overburden only) is shown in Figure 65B. Thus the necessary cut-off grade of these tonnage increments also increases with depth until a point is reached where the cut-off grade surpasses the average grade of the respective blocks and no further profitable open pit mining can be undertaken. Pit profit is obtained by adding the profit of each block that has to be mined to produce the required pit. Underground mining may eventually be consid-

Fig. 64



(Taken from Seraphim, 1973.)



A. OPEN PIT LIMITS

B. RELATIONSHIP OF CUT-OFF CURVES

Fig. 65

(Taken from Thomas, 1973.)

-ered in depth when high stripping ratios and enough mineralized tonnage warrant underground development and preparation works. In open pit mining the cut-off-stripping relationship is dependent on the pit wall slopes (Fig.64). It is also obvious that the pit slope becomes an increasingly important factor in depth, i.e. during the last years of an open pit mine. Pit wall-slopes will depend on geological and hydrological factors, e.g. density and attitude of the fracture system, the degree of alteration and its effect on the mechanical behaviour of the rock, the presence of water or greasy minerals in the fracture planes, etc. Hence, ore reserve alternatives and their related cut-off-stripping relationships in porphyry deposits are triply dependent on the knowledge of the fracture systems; they have an important effect on (i)metallogenesis and grade of these deposits (see chapter 1.3.3.b), (ii) the drilling results (see chapter 2.2.1.b) and (iii) on the relationship between the pit slope-the stripping ratio and the cut-off grade.

In addition to the production, sales and stripping costs, the amount of depreciation and a minimum profit after tax, related to a minimum acceptable return on investment, must also be assigned to each block for its respective cut-off grade determination. Finally, the grade of material which produces a revenue equal to the production costs through to sales, together with depreciation and the minimum profit, is the "cut -off grade" and is used to separate ore from waste. It is, however, difficult to estimate unit depreciation and required minimum profit per ton accurately before the ore reserve is known, because both will vary with plant size and the tonnage-grade alternative chosen. The selection of the optimum tonnage-grade alternative is normally achieved by incremental financial analysis (Phillips, 1973). This technique compares the financial returns of alternative ore inventories and associated plant sizes, e.g. by proceeding from the smallest tonnage ore inventory and plant size each progressively larger tonnage ore inventory is considered as a possible expansion. The additional cash flow developed from each expansion is determined as a return on the additional (incremental) capital required for such an expansion. The largest tonnage and, consequently the lowest-grade ore inventory in which every increment of invested capital yields at least the minimum desired corporate return is the optimum ore reserve for the assumed life (op.cit., 1973). However, economic benefits usually result from sequencing mining so that cut-off grade is reduced over mine life. This is if favourable geologic and mining conditions (e.g. stripping ratios, grade, etc.) allow for such a reduction in cut-off grade. Blackwell in Mackenzie (1979) describes such a dynamic optimization technique for a hypothetical open pit porphyry copper mine development, similar to Panguna. The sample assumes fixed mine and mill capacities and does not consider uncertainty. Thus, the net present value criterion may be utilized. The assumed mine development conditions are as follows:

" mine capacity - 40 million tons per year
mill capacity - 15 million tons per year
mill recovery - 85 per cent
unit pit costs per ton mined - $0.25
unit mill costs per ton ore - $0.50
selling costs per ton copper - $50
copper price - $0.39 per pound recovered.

Tonnage and grade are specified by tonnage estimates over a range of grade categories. Economic results for assumed cut-off grade strategies are shown below:

	Cut-off Grade (% Cu)	Net Present Value (millions of dollars)	Total Cash Flow (millions of dollars)	Mine Life (years)
	Fixed: 0.2	255	910	25
	0.3	276	860	21
	0.4	294	819	18
	0.5	287	717	15
	0.6	236	548	15
Variable :	optimum	305	797	16

From the results it can be seen that the optimum variable cut-off grade strategy has an advantage of \$9 million over the highest present value obtainable by a fixed-cut-off grade strategy. With this strategy, the cut-off grade is reduced from 0.54 per cent Cu in the first year to 0.17 per cent Cu at the end of mine life."

For the mining operation, investment in exploration and mine development is sunk and decisions are of a tactical nature. Operational decisions embody such elements as cost control, production planning, productivity, the development and implementation of new mining and processing technology and equipment replacement decisions. The effectiveness of these policies will be reflected in cost trends which in turn will determine changes in cut-off grade, ore reserves and mine life. A primary objective of the mining organization with respect to a mining operation is maximization of total profit (cash flow) over mine life. This objective will be achieved by minimizing economic cut-off grade and maximizing ore reserves and mine life.

For operational conditions where the primary economic criterion is maximization of total profit over mine life, cut-off grade is determined by the equation of marginal cost and marginal revenue. At the margin, grade is just sufficient to offset cost, that is,

$$MC = MR = PGR/(1 + D)$$

$$G = (MC)(1 + D)/PR$$

where

MC = cost of mining and processing marginal ton of ore (in dollars);

MR = revenue realized from marginal ton of ore (in dollars);

P = net price of metal recovered (in dollars per ton);

G = cut-off grade, percentage content of mineral product in marginal ton of ore;

R = mill recovery factor (metal recovered/metal content of ore reserves);

D = dilution factor (waste mined/ore mined).

b) Mining Methods : the large, low grade porphyry deposits are of a size and shape that lend themselves to low cost bulk mining methods. Since ore boundaries in porphyry deposits are determined by the economic cut-off grade, a careful analysis of contained mineral values (Mo, Au, Ag, W, Sn, U, etc. in addition to Cu) is essential. Grade trends within the deposit are important because they present the opportunity of mining higher grade areas in the early years of mine life, e.g. the supergene enriched zone (e.g. Chuquicamata, El Teniente, Chile); higher grade mineralized breccia pipes (e.g. Panguna, El Teniente). These higher grade areas allow for selective mining and will have a significant effect on the profitability of investment (Mackenzie, 1979). Selective mining in porphyry deposits usually also considers the different types of ore present, such as those that must be treated by different benefici-ation processes, e.g. sulphide and oxide copper ore types at Chuquicamata, El Abra, Mantos Blancos porphyry deposits, Chile. In this case the cut-off parameter is also based on the value parameter but the alteration in the mineral ratio is comm-only associated with a change in the performance of the met-allurgical plant. For example, lowering the molybdenite cut-off in response to an increase in its market price, will increase the Cu/Mo ratio of the mill head.

Geological and geotechnical features of the rock such as the strength and competency of the ore in the deposit and in the adjacent rock formations also play an important role in deciding the most favourable mining method and plan, and in determining the size and location of the mine's primary development.

The amount and quality of ground water that is in and around the ore deposits is also a major factor affecting the cost required to develop the ore deposit (Pillar and Drummond, 1975).

- Open pit mining : is an automatic choice for porphyry deposits lying near to surface, the only question being that of the economic cut-off limit where a change may be made to underground mining. The advantages of open pits (Thomas, 1973), are:

1. higher producitivity and output.
2. greater concentration, safety and control of operations.
3. lower capital and operating costs per ton mined.
4. greater geological certainty and easier exploration.
5. less limitation on size and weight of machines.
6. better recovery of mineral.
7. simplified engineering and planning.
8. closer prediction of the economic outcome because less mining variables and uncertainties are involved in comparison to underground mining.

The reasons for choosing open pit mining must be a matter of overall economics determined by a feasibility study. Factors affecting the choice are:

1. relative thickness of overburden and orebody. Depending on the costs, richness of the ore, nature of overburden and extent of deposit, the stripping ratio in porphyry deposits may go up to 5.5/1 (e.g. Mantos Blancos, Chile). In steep and rather narrow porphyry deposits the open pit mine may extend very little beyond the 3/1 stripping ratio before going underground.

2. size of orebody.

3. relative costs of mining by open cut and applicable underground method.
4. relative dilution and loss of mineral in both cases.
5. relative costs of development.
6. climate (e.g. rain, snowfall, etc.)
7. topography - a very irregular topography will favour underground mining.
8. continuity of projected operations.
9. availability of skilled labour; underground mining requires a more highly skilled labour force than open pit mining.
10. capital available.

The major factors affecting the layout of a pit are:

1. the shape and depth of the deposit. This is of particular importance since the selection of a transport system is restricted by the distance and vertical height through which overburden and mineral must be moved.
2. the properties of the mineral and overburden:
   (i) slope angle limitations
   (ii) degree of hardness and abrasiveness, which influences the selection of the excavating and transport equipment
   (iii) variations in grade which may require selective mining to be practised and affect the shape of the pit due to cut-off grade limits
   (iv) competence of the various rocks to support the equipment.

3. The geometry and size of the excavating machinery, particularly digging height or depth, dumping heights, and reach.

4. the drainage requirements; saturated slopes are more likely to collapse than are well drained slopes.

Mining recovery and dilution also depend on the structural competence of the ore and host rock, the depth and nature of the overburden and the more or less selective methods of sequencing mining. Since the aim is to determine the effect of selective mining to some cut-off grade, the basic mining unit is a block of dimensions which can be mined out with the particular equipment in use. To study the distribution of mining units at Panguna, Blackwell (1973) compiled a histogram of 50 ft. lengths of core from all portions of each hole passing through a zone. A frequency distribution was fitted to the histogram and from this a similar distribution for mining units was derived (see Fig.20). In large-scale open pit mining the degree of selectivity is low; with this mining method individual lengths of diamond drill core can not be mined selectively and therefore a bias arises when estimating the effect of applying a cut-off grade based on the original drilling results.

The effect of considering larger units is to squeeze up the distribution (Fig.20) so that the drill core curve has a significantly longer tail on either side. Thus, whatever cut-off grade is applied, the head grade will lie between the cut-off and the end of the right hand tail. The drill core curve, therefore, has a high grade tail inflating the value above a cut-off relative to the value from the tighter mining unit curve.

- Underground mining : is applied to porphyry deposits of a significant vertical dimension, e.g. high stripping ratios and/ or that have other features that do not warrant open pit mining. Wall rocks, and/or the rocks capping an orebody (e.g. leached zone) tend to be weak in porphyry deposits and in this case caving mining methods can be considered. Geological processes involved in the genesis of porphyry deposits, such as the stockwork fracture system, has generally imprinted a favourable geotechnical behaviour on these deposits for the subsequent application of the "block caving" method. The geologic factors that affect block caving are essentially those which pertain to rock strength (Wilson, 1957); e.g. the principal factors affecting block caving in the San Manuel mine are structure, rock types, alteration, mineralization, and presence of water. The caving rock tends to break or to spall along definite fracture surfaces. The density of fracturing constitutes an index of the zones of weakness and is the reason why the parameters fracture frequency (FF) and rock quality determination (RQD) are being measured during core logging in many porphyry deposits (e.g. El Teniente, Rio Blanco, El Salvador - Chile). The density of low-angle fracturing is prticularly significant. The rock of blocks having a greater FF tends to break into boulders small enough for passage through chutes and grizzlies. At San Manuel the youngest fractures exert the most influence upon subsidience. Intense sericitization and/or argillization also contributes to weaken the rock in most porphyry deposits resulting in ghih productivity rates (e.g. more than 50 ton/man/shift). Nevertheless, this weakened rock may also create caving of the development and preparation workings, requiring therefore costly reinforcement (e.g. El Salvador).

Sublevel caving is applied when the ore is relatively strong and needs to be blasted. A good example of this is encountered at the El Teniente porphyry deposit. The presence of anhydrite in the stockwork system within the primary ore zone has given the host rock a stronger mechanical competence and, therefore, does not break as easily as the more brittle rock of the supergene enriched zone, where anhydrite has been leached out. The highly productive block caving method used for the mining of the supergene ore zone can not be applied in the hypogene ore zone. This zone will have to be mined by sublevel caving, a method which yields about half the productivity of the block caving method.

For more detailed descriptions of these underground methods refer to Thomas (1973).

Thomas (1973) listed the following advantages and disadvantages of the block caving method:

(i) Advantages:

1. Lowest mining cost of the underground mining methods.
2. A high rate of production.
3. The accident rate is fairly low.

(ii) Disadvantages:

1. The capital expense is large and the development time is long.
2. The ore is diluted with waste and some loss of ore may occur.
3. There must be careful supervision of ore drawing.
4. Low grade ore in the capping and at the margin of the orebody is lost or there is excessive dilution if caving is uncontrolled.
5. There is no possibility of selective mining of high and low grade ore.

c) Metallurgical Amenability of the Ore : ultimately, information is needed on potential recovery in metallurgical processing as geological and engineering estimates of reserves have no significance unless they refer to what can be extracted and marketed.

The degree of recovery of each metal during the metallurgical processes is a function of many variables, e.g. the recovery of molybdenite varies with the abundance of molybdenite, the molybdenite grain size and composition of associated sulphides. Gold in porphyries presents even greater recovery problems because it is usually present as very fine particles of native metal or as solution in sulphide minerals (Kesler, 1978). Sulphide grains in the supergene enriched zone may also have an oxidized plating (e.g. oxidized copper, hematite, limonite, etc.) that inhibits their flotation and subsequently decreases their recovery. More expensive processing techniques like finer milling, may increase the recovery in this case.

2.4 Trial Mining and Bulk Sampling

Trial mining and bulk sampling of porphyry deposits is increasingly used because it is the most effective way to check drilling results and of obtaining trustworthy geological information, needed for the estimation of:

- the grade-reserve relationships
- the recoverable reserves and dilution estimates
- the mechanical behaviour of the host and wall rocks
- planning of grade control
- metallurgical amenability of the ore and recovery by means of a pilot plant.

However, trial mining and bulk sampling must be justified, because they can be expensive and time-consuming operations.

One reason why bulk sampling is necessary in evaluation of porphyries is appreciated when common sampling patterns are analysed. If a porphyry copper deposit is sampled by vertical holes at a 100 m spacing (e.g. Sar Cheshmeh) and the core is sampled every 1,5 m a great deal of information about vertical but little information about the horizontal variation is obtained. The grade control in an eventual later mining operation will probably be done by classifying horizontal runs along benches as ore or waste. Thus the abovementioned exploration data will not contain the information necessary for grade control planning. In this case there should be two exploration programs side by side, one aimed at defining the resources and the geologican structure of the deposit, the other the grade-reserve relationship and other factors affecting mine and grade control planning.

In chapter 2.3.3. the derivation of the grade-reserve relationship from the cut-off/average grade and cut-off/tonnage relationships has been briefly described. These relationships are normally estimated from the available bore hole sample data. The theoretical uncertainty of these estimates based on normal sample data is also clearly confirmed by practical experience in some operating mines where the method of mining, transportation network and plant processes allow a comparison of production records with specific blocks of ore. In this case the errors of these relationships can be estimated. But in the case of still unmined deposits other less precise methods like trial mining and bulk sampling, e.g. Sar Cheshmeh, Panguna, Quebrada Blanca (Chile), etc. are used. As low grade-large tonnage operations like porphyry deposits are more sensitive to grade variations the essential objectives of bulk sampling-trial mining should be accurate sampling of representative parts of the deposit and processing the bulk sample through a pilot plant. Bulk samples may either comprise large diameter drill core, the cuttings of raise-boring holes drilled along exploration holes, or the contents of underground blasts.

The following procedures are generally followed if bulk sampling seems justified, (Dixon, 1979):

1. A series of places are selected in the deposit in order to cover the different metallurgical ore types present (e.g. sulphides, oxides, different proportion of sulphide minerals, different grain sizes, etc.) and the total range of grades that could be mined.
2. A series of large diameter boreholes and/or mine workings is planned in order to cover the selected places and blocks defined by the grid drilling.

   (i) Raise boring has been used at Sar Cheshmeh in order to check that the core recovered during the grid drilling provided representative samples. The choice of holes to be raised bored by reaming out to 15 inches in diameter was influenced by a number of factors in this deposit:

   a) the borehole must be accessible from a tunnel to allow the reaming bit to be attached underground;
   b) the diamond drill holes selected must be uncased;

c) a minimum thickness (15 m) of supergene ore above the tunnel was desired;

d) there was a preference to check holes with low core recoveries.

The diamond drill holes checked by this method cover the lower end of the core recovery range with above average copper values, i.e. those most likely to be incorrectly evaluated by diamond drilling (Selection Trust Ltd., 1970). Although characteristic local variations in grade appear in the individual results, differences in averages for each hole were very close and did not exceed 0.11% Cu and these include any errors arising in core-splitting, sample preparation and assaying. The weighted average of all the results for the much larger vertical raise-bore samples at Sar Cheshmeh is 1%, higher than that for the original core and this close agreement confirmed in this case that the core provided an accurate and unbiased sample on the vertical dimension. Figure 66 is an example of a comparison of assays of the cores of two diamond drill holes and their respective cuttings.

ii) When underground workings are considered for bulk sampling and trial mining, they are planned so that they can eventually be used later as development workings, service ways, drainage ways, etc. The orientation of the workings is commonly related to those in which mining face advances are likely to take place.

At Panguna bulk sampling of a preproduction exploration adit and raises along previously drilled diamond boreholes confirmed that the estimation based on these boreholes upgraded the copper values and over-estimated the ore tonnage. This probably is because no geological contacts were considered during the ore reserve evaluation in this case (see chapter 2.3.2.-5).

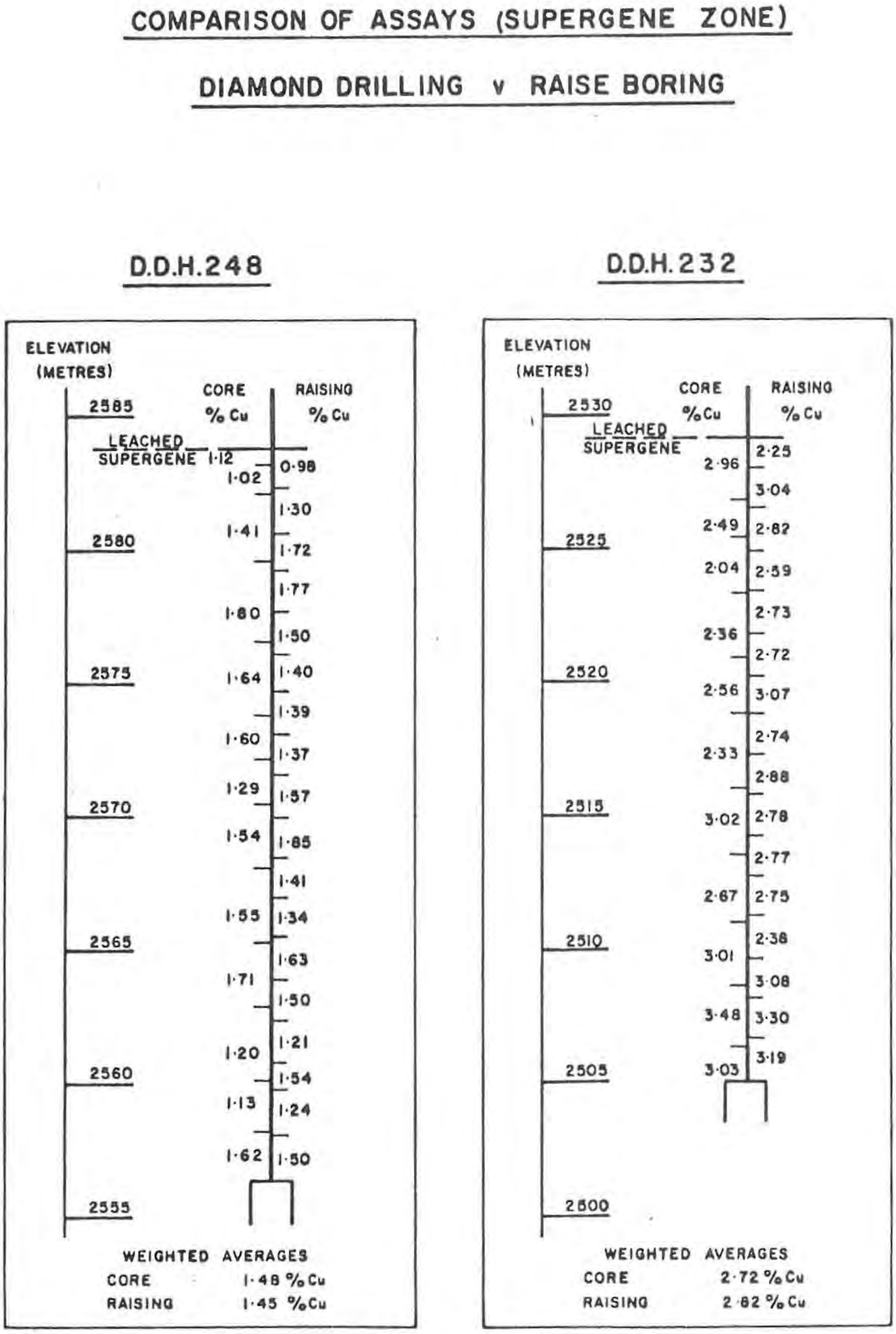


Fig.66 (Taken from Selection Trust Ltd., 1970.)

Later mining and blast hole sampling of this orebody also indicated that the dimond drill assays upgraded on an average of 0.20% to 0.30% Cu the lower grade mineralization, but there was no significant change in the higher grade zones, i.e. plus 1% Cu mineralization (Baldwin et al., 1978).

At Sar Cheshmeh a total of 8700 m of preproduction underground development was completed mainly on a 200 to 100 m grid and fitted according to the diamond drill grid. This underground development was sampled along 3 m sections by randomly chipping small pieces of rock over the area of both side walls and the back of the tunnel. The sample size in this case was 30 kg. This sample size was selected statistically using Gy's equations, the main assumptions being that:

a) the mineralization is uniformly distributed;
b) liberation is complete at 100 mesh;
c) 95% of the sample was -25 mm.

Two types of tests were also instituted at Sar Cheshmeh to establish the effects of varying the sample weight and sampling method:

a) Chip samples weighing 5 kg and 30 kg. The average assay of both types of samples, for a total of 27 sample lengths, did not vary more than 1%.

b) Channel sampling on the same 27 sample lengths averaged 5% less than the average for the chip samples.

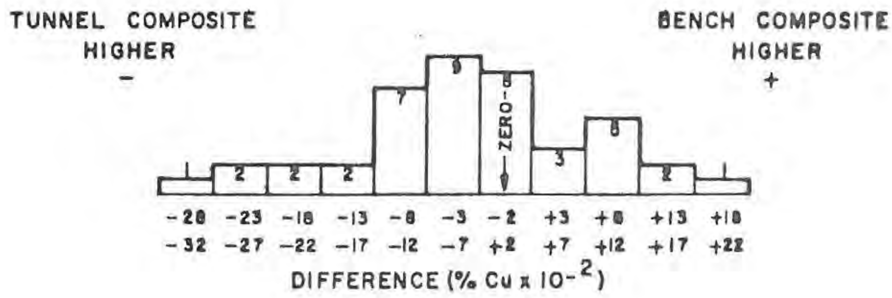
The objective of the underground development program at Sar Cheshmeh was to investigate the horizontal continuity of values between the vertical drill holes. This would show that the drill holes had not by mischance passed through a number of isolated pockets of mineralization, and provide justification for the assumption that drill hole values extend to the edge of the block through which they pass. Underground sampling results showed values in accordance with the rock and zone type. To check the overall assumption, a comparison has been made between bench composite values from 46 drill holes and block averages derived from tunnel assays within the same mineral zone and rock type. The average of each set of 46 results agreed closely (Fig.67):

Average of bench composite values from drill holes: 2.01% Cu

Average block values for tunnels: 2.03% Cu.

HISTOGRAM OF 46 COMPOSITE VALUE DIFFERENCES

TUNNEL COMPOSITE VALUE v BENCH COMPOSITE VALUE



AVERAGE VALUES

TUNNEL COMPOSITE	2·03% Cu
BORE HOLE COMPOSITE	2·01% Cu
AVERAGE DIFFERENCE	−0·02% Cu
PERCENTAGE DIFFERENCE	− 1·0 %

EXCLUDED FROM HISTOGRAM:

DRILL HOLE No.	DIFFERENCE (% Cu)
306	− 0·38
129	+ 0·89
243	− 0·50
29	+ 0·40

Fig.67 (Taken from Selection Trust Ltd., 1970.)

As with the raise-boring, the check result (from the tunnels) is 1% higher than the core values (bench composite values) used for ore reserve estimation.

Throughout the Sar Cheshmeh investigation a comprehensive series of system checks was also in operation to ensure that:

a) samples sent to the sample preparation laboratory were truly representative;
b) sample preparation procedures were accurate;
c) assaying procedures were accurate.

The results of some of the more important quantitative checks are presented in the following table (Table 13).

Parameter Checked	Method of Checking	Difference * (%)
Core sample validity	Raise boring	-1.00
Core splitting accuracy	Assaying residual core	+0.47
Assay accuracy	External assaying of 10% of core samples	-0.60

* Differences are expressed as a percentage of the Sar Cheshmeh assays used in calculation of the Ore Reserves, not as % Cu. Positive differences indicate that the Sar Cheshmeh values are higher.

(Taken from Selection Trust Ltd., 1970.)

3. Each selected area or block is mined out so that each individual sample is of a scale at least as small as the possible minimum grade control block-size. This allows study of the effect of scale on the uncertainty of the expected grade as well as the degree of mining dilution and mechanical behaviour of the rock under the selected mining method and scale.

4. After each blast the workings should be accurately surveyed and geologically and geotechnically mapped in detail. The volume of each is computed and weighted for specific gravity determinations.

5. Each bulk sample is passed through a sampling plant designed in such a way that uncertainty introduced by sample reduction is within the limits over which grade control must operate. One part of the bulk sample is assayed; the other part, the reject from the sampling plant, is assigned for processing in the pilot plant for the determination of the optimum extractive processes, recovery and for obtaining engineering and metallurgical criteria necessary for the design of a full-scale plant. Usually the parts of the bulk sample destined for the pilot plant may add up to 20,000 to 30,000 tons; e.g. at the Berkely porphyry deposit (Waterman and Hazen, 1968).

6. Finally, the results of the expected grades (based on the grid boreholes) and the actual results (from the bulk sampling), can be represented in a diagram like the one shown in Figure 68. This figure shows a series of elliptical contours each of which encloses a certain proportion of the points. This can be done empirically or by fitting the data to a statistical model (e.g. the bi-variable Gaussian distribution fitted to the logarithms of the data). The outer contour is a line that

can be expected to enclose 99% of the points. If a particular cut-off is chosen parts of the depoit can be classified as "ore" and the rest as "waste", and by integrating under the bi-variate Gaussian surface the proportion of reserve of the total deposit is determined. The grade of the ore and waste are easily calculated. But as the classification is based on expected (borehole) grades, there are four possible outcomes (Fig.69):-

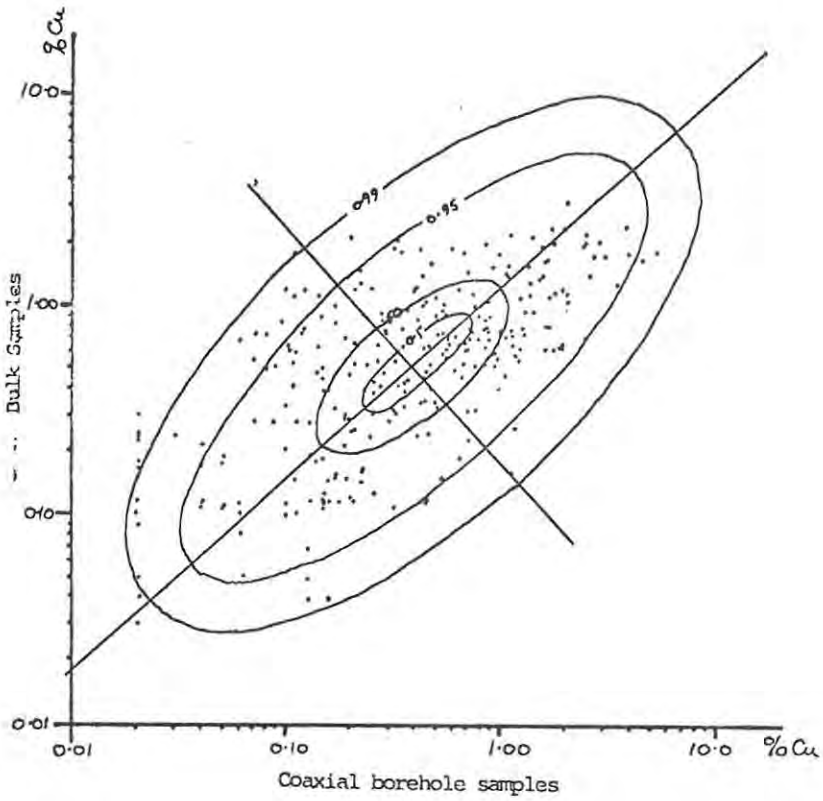


Figure 68

Bi-variate Gaussian model of log copper values(%) for an underground borehole/bulk sample survey.

(Data from Pryor, Rhoden & Villalon; Trans.Inst.Min.Metall. 1974)

(Taken from Dixon, 1979.)

1. Ore will be correctly classified as such.
2. Waste will be correctly classified as such.
3. Actual ore will be misclassified as waste.
4. Actual waste will be misclassified as ore.

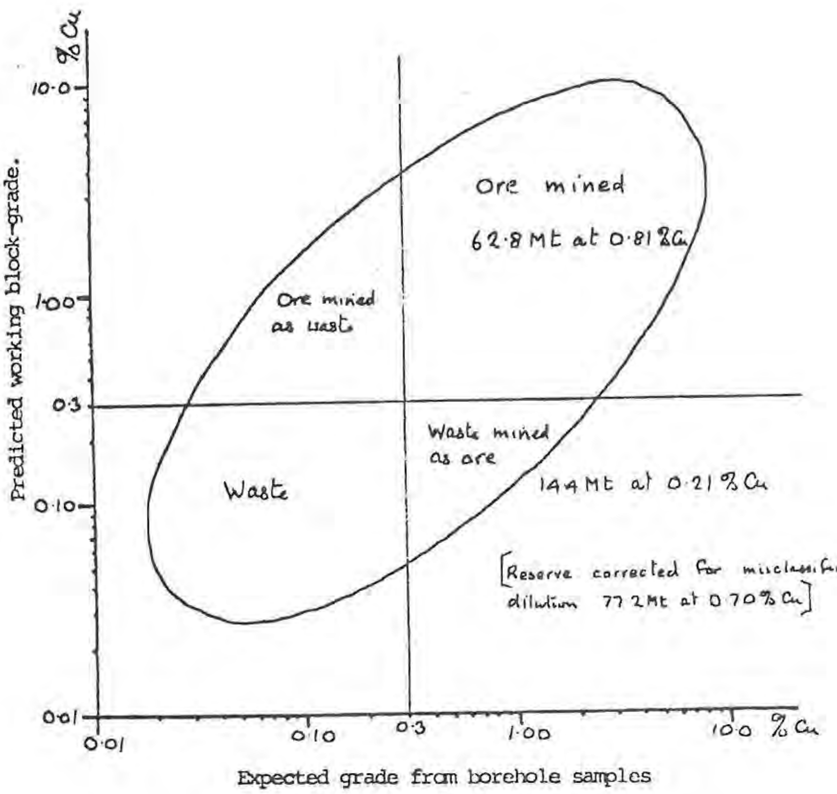


Figure 69

Classification based on bulk sampling, representing selective mining of 30t blocks to a cut-off of 0.30% Cu.

(Taken from Dixon, 1979.)

Consequently, the misclassification of rock can be reduced if the zone of uncertainty in Figure 69 can be predicted by trial mining. Another aspect to be considered here is that an increase in cut-off grade also requires a compensatory reduction in the uncertainty of the expected grade and this is associated with an increasing cost of trial mining and bulk sampling. Additional trial mining and bulk sampling may not be economically justifed from a certain stage onwards (refer to chapter 2.2.4).

2.5 The Economics of Mine Development

Once a potentially economic porphyry deposit has been discovered, economic evaluation takes place. Estimates are made of the deposit's characteristics and environmental parameters as this will serve as a basis for assessing a future stream of economic benefits associated with the investment. An optimum future mining plan for the deposit's operational life is then fitted to these estimates. In reality, mine development decisions are made under conditions of uncertainty.

Uncertainties arise because the porphyry deposit characteristics, while existing at the time of the decision, cannot be fully examined and because the environmental parameters will only become fully known with time. Under conditions of uncertainty, the economic attractiveness of developing and mining such a deposit is subject to those conditions which are expected. In general, it can be said that porphyry deposits are mined with a low profit margin and require a high capital investment based on high manpower porductivity and large ore reserves. Risk criteria are used during such an evaluation to translate the perceived deposit and environmental encertainties into a probability distribution of possible values around the expected value investment criterion (Mackenzie, 1979).

Priot to commencing even the most preliminary economic analysis, certain information is required. The following list outlines the geological factors which are normally of economic significance in the evaluation of porphyry deposits:

- location of the deposit with respect to transportation, water, sources of power;
- topographic location of the deposit;
- nature and proportion of minerals, grain size and mineralogical complexity;
- presence of clays or other constituents which could affect the mining and milling of the ore;

- strength and competency characteristics of deposit and host rocks;
- amount and quality of subsurface waters;
- the shape of the deposit;
- overall tonnage and grade of the deposit;
- the range of grade-tonnage combinations which exist within the deposit;
- intangible potential;
- metallurgical test results.

The basic geological information guides the selection of possible development, mining and processing methods. In making this selection, it is not a specific method which is important but the recovery and dilution factors, and costs associated with it.

The possible range of mine capacity is based primarily on the size and physical characteristics of the deposit and the cut-off grade alternatives. Rates of output will be limited by practical problems, of which one of the most important is working-space in both underground and open pit mines. In underground mines, capacity tends to vary with strike and width, but not depth. Similarly, in an open pit the working space for equipment tends to vary with area while tonnage varies with volume.

The economic optimization of each project is normally studied by the discounted cash flow technique. The mining plan is linked closely to the search for the best possible discounted cash flow. This makes the short-term profits much more desirable than the long-term profits.

The economics of a mining operation may be thought of as juxtaposition of incoming and outgoing cash flows. Negative cash flows are frequently considered in three parts:

(i) the primary investment, which comprises the cost of all works necessary to commence production (preproduction stripping in case of an open pit operation, development work in an underground operation, purchase of machinery, construction of plant, etc.). This initial capital investment expenditure is commonly considered and justified in the long-term (10-30 years?) planning of an operation . This long-term mining plan also attempts to maximise the exploitation of resources available.

(ii) The working capital necessary to maintain and, if necessary, expand or make more efficient the mining during production. The medium-term mine plan (1-5 years?) normally justifies this expenditure.

(iii) The operating cost, or the money necessary to pay labour, energy, and consumable materials during the operation. Short-term planning (1 shift to 1 month) is carried out at the level at which income is generated and working cost is incurred and controlled.

Associated with these three elements of negative cash flow are additional charges, such as interest charges, taxes, royalties and so on.

2.6 Grade Control

Operational decisions relating to ore reserve estimation fall into the category of grade control. The time domain for this type of decision varies from the daily control required to minimize grade variability in mill feed, to production planning on quarterly, annual and five-year basis.

Grade control is central to the success of mining ventures. Good grade control is a matter of rigorous scientific investigation and good engineering design, but depends also on skill and good communications between the mining, metallurgical, engineering and geological departments of a mining operation.

According to Mackenzie (1979, page 548) the following types of grade control are of possible concern:

" i) daily grade control of mine production to minimize grade variability in mill feed, thereby optimizing mill efficiency and recovery;

ii) cut-off grade decisions relating to closing of old working places and new stope development;

iii) production planning, including cut-off grade considerations, on quarterly, annual and five-year bases;

iv) control and economic limitation of dilution. "

For any given mining plan there is a certain size of ore block at which a practical distinction can be made between ore and waste. Once the method of mining has been chosen and the equipment purchased the grade-control block size is usually either fixed or can only be varied within narrow limits. The determination of a minimum practical grade-control block size is an essential pre-requisite for any grade control plan. In an open pit porphyry operation, the size of the equipment will determine the bench height, and the width will be decided by the safe slope, so that the grade control block size is represented by a length along the bench at which it is feasible to separate broken rock to the crusher or to the dump, e.g. at Chuquicamata these blocks measure 20 x 20 m and 13 m in height.

A plan for grade control must integrate the geological properties of the deposit with the mining plan and must be capable of fulfilling the two objectives of feeding the treatment plant with material within grade and tonnage limits, and be able to respond to changes in the effort/reward relationshp (Dixon, 1979). The essential information required in making a grade control plan is:

Economic

1. The management strategy of the mine, e.g. maximization of profit, maximization of the resource, making a constant return on the investment, etc.

2. The range of grade and production rates, etc. at which the mine is expected to operate.

3. The customers requirement for the product.

Mine Design

4. The number of working faces and their productive capacity.

5. The degree of mixing in transportation.

6. Geotechnical constants (pit slope, size of safe workings, groundwater control).

Mineral Resources

7. The toleration of the processing plant to variations in grade at a fixed scale of operation.

8. Factors affecting the recovery and product quality, e.g. grain size of the sulphides, proportion of pyrite to copper sulphides in the ore, presence of slime producing clays, presence of oxidized coatings on the sulphide grains, etc.

Geological

9. The classification of the deposit into types of mineralization and definition of the limits of the deposit.

10. The distribution of grade at possible grade control block sizes and hence the reserve/grade relationship at a range of block sizes.

11. The nature of grade variation at the scale of mining.

In most open pit porphyry deposits the sampling of blast holes together with the detailed geological bench mapping is sufficient to program the daily loading and other aspects of grade control.

In some irregularly mineralized porphyry deposits a monthly to six-monthly planning phase also exists. In this case more detailed information is required for the short-term, but at this stage the blast holes have not yet been drilled. This problem is commonly solved by additional drilling in a closed spaced grid. This drilling is commonly done by the more economic percussion drilling where either cuttings or the sludge is sampled.

In most porphyry deposits characterized by a rather uniform distribution of ore grades and mineralogy at the bench scale, this monthly to six-monthly planning stage is accomplished without further drilling or sampling. In this case the planning is based on a statistical or geostatistical method that makes use of the great amount of information collected on the adjacent zones that have already been mined (see chapter 2.3.2.c.).Taking into consideration the trends of continuity of the mineralized structures these data are then extrapolated towards the face of the pit. Empirical studies of more than 300 mined blocks in the Chuquicamata pit have resulted in different weighting factors for these adjacent blocks.

At Panguna the blast hole block values (individual blast hole assays averaged within 30 x 30 x 15 m blocks) falling within a particular zone are being used for grade control purposes, and have indicated the necessity of upgrading the lower grade mineralization, ranging from 0.20 to 0.30% Cu from the diamond drill assays to 0.31 to 0.35% Cu in the blast hole assays.

In general, grade control means finding a series of compromises between properties of the deposit and of the engineering design. Usually the smaller the number of working faces, the larger the equipment that can be used, the higher the production of the work force and the lower the unit operating cost. But as the number of working faces becomes smaller the opportunity to mix ore of different grades also decreases. At the same time the risk of grade variation and of increased dilution becomes higher due to mis-classification or misplacement of the rock to be mined (e.g. the Rössing open pit in South West Africa). This type of scale-mining problem commonly arises when engineers consider bigger mining equipment because of their lower unit costs.

The economic criteria applied for tonnage and grade control decisions are relatively simple. Generally, marginal analysis may be used. Since investment in mine capacity is sunk, time value and profitability are less important than in the mine development decision situation.

There are two kinds of dilution to be considered in grade control planning:

(i) Physical dilution - this dilution is produced by hangingwall overbreak, upper bench collapses, etc. This type of dilution can be controlled and its effects measured. If the dilutant is sub-cut-off mineralized rock then the assumption that it is barren leads to a loss of reserve, i.e. - too high a cut-off may be chosen. Underground porphyry mines developed beneath a leached capping may present dilution by hangingwall failure. This dilution is time - dependent and structurally controlled.

(ii) Dilution caused by mis-placement, e.g. a truck driver who dumps waste into the primary crusher. This type of dilution can be minimized by good design and management.

For purposes of grade control, tonnage and grade estimates are required for blocks associated with individual stopes, and mining faces, and for stope-size blocks along development headings. The basis for these estimates is detailed stope and development sampling supplemented by mine car sampling programs. It is also quite common for mines to study expected grade/mill grade correlations and to derive from this a simple factor which is then used to discount the mine expected grade, e.g. at the Mantos Blancos porphyry deposit the following correction factors are used (expected grades are based on Churn Drill holes):

- <u>For open pit ore reserves:</u>
  assay grades are reduced by 20% to compensate for dilution with waste and possible sampling errors. Ore tonnage in place is increased by 10% on account of dilution with waste.

- <u>For underground reserves:</u>
  a 10% assay reduction plus the following additional corrections according to the type of mining project:
  a) chambers of the Room and Pillar and Sub-Level Caving methods: - dilution of 5% with waste averaging 0.42% Cu without varying ore tonnage in place. This represents correction for dilution with waste because of irregular ore outlines within blocks and partially from incidental caving of walls.
  b) sub-level caving and pillar recovery: dilution of 30% with waste averaging 0.42% total Cu without varying ore tonnage in place. Represents correction for mechanical dilution with caved in waste from above and wall rocks.

However, the use of blanket mine/mill correction factors tend to obscure the real causes of the mine/mill differences. These differences could be caused by:

1 - Bias in expected grades.
2 - Errors in the estimation of the different types of dilution (or no estimate at all).

3 - Errors in the recording of production rates.

4 - Errors in the estimation of the grade received by the processing plant.

At the Similkameen porphyry, the overall effect of the mining dilution was to increase the $\pm$ 0.30% Cu in situ ore reserve tonnage (estimated by the polygonal method and confirmed by underground development and diamond drilling) by 14% and to reduce the grade by 9%.

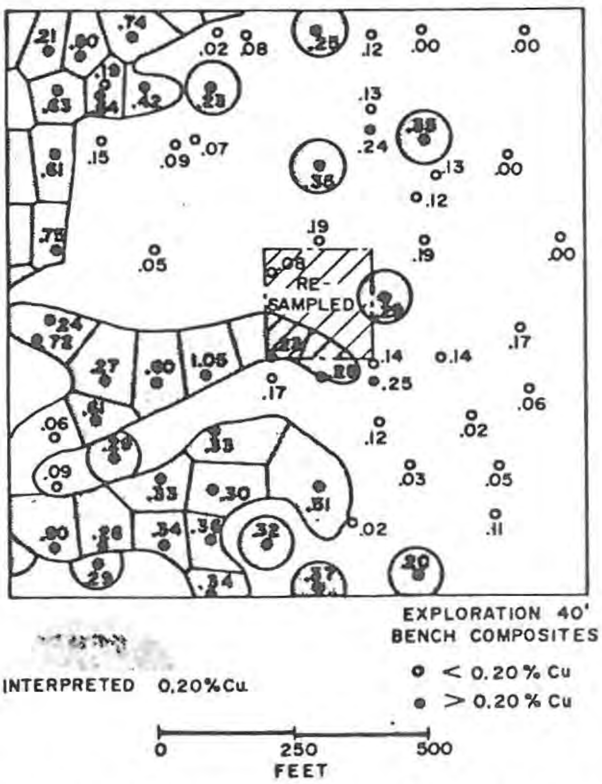


FIG. 70. Typical exploration bench plan showing polygon interpretation.

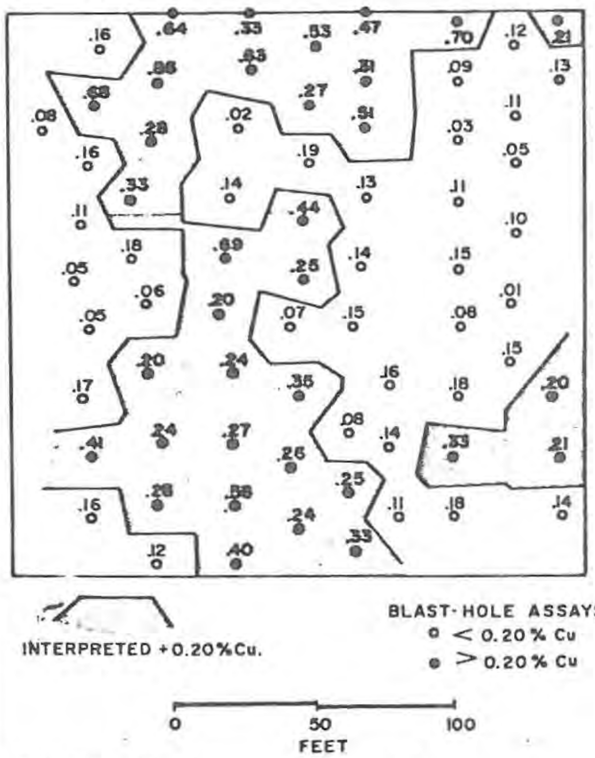


FIG. 71. Blast-hole assays in resampled area.

(Taken from Raymond, 1972.)

However, the blast-hole assays did not show the expected patterns of large, irregular ore blocks with sharp ore-waste boundaries. Figure 71 shows the blast-hole samples within the hatched area of Figure 70 but on a larger scale. The shaded area indicates ore as defined by polygons around blast-hole assays. The shaded area is similar to the mining limits that would have been used by 7.6 m^3 shovel working on a 24 to 30 m face width while loading trucks on both sides. Despite efforts to flag the small ore and waste zones at the Similkameen pit, the production showed a 15% ore tonnage loss and a 12% grade loss compared to diluted polygon ore reserves at a 0.30% cut-off. Analysis of grab samples (1 grab sample was taken from the production shovels for every five or six truck loads) showed that only blocks with kriged grades of 0.15 to 0.29% Cu could be effectively sorted by grab sampling. At the same time a weighting of 50% to the block grade and 50% to the grab-sample grade was introduced to further limit the influence of the grab sample at Similkameen. This resulted in an immediate increase in ore tonnage recovery and no further grade loss at the 0.20% Cu cut-off. The success of using kriging for ore control is obvious from Table 14, which shows a comparison between mined ore and diluted polygon ore reserves at both the 0.20% and 0.30% Cu cut-offs.

Table 15 shows comparisons between mined ore and ore reserves based on the distribution of blast-hole assays (polygonal estimation). The figures indicate that ore tonnage recoveries using kriging are about 20% higher than those achieved using grab sampling alone.

Table 16 shows comparisons between mined ore and the tonnage and grade indicated by kriging the 15 x 15 x 12 m block grades from blast-hole assays. A 38% improvement in recovered ore tonnage is indicated using kriging as compared to the original manual outlining. Mining strictly to kriged ore-waste boundaries at Similkameen over an 11-month period yielded a milled tonnage and grade within 2% of the kriged estimates. This proved that the kriged grades are correct on average and that no dilution has to be taken into account in this estimate. The resulting estimates are nearly unbiased. In practice, however, mining at Similkameen was not determined by the ore-waste boundaries from exploration drilling, but rather to boundaries that are re-evaluated from blast-hole assays.

Figure 72 shows a plot of kriged grades from exploration versus kriged grades from blast-hole assays for individual 15 x 15 x 12 m blocks.

The shaded area indicates all blast-hole block grades for which the grade estimated from exploration assays falls between 0.20% and 0.30% Cu. With a 0.20% Cu cut-off, all of these blocks are estimated to be ore. However, there is still about a 30% chance that the eventual blast-hole block grades will be waste.

TABLE 14 Mined ore versus exploration polygon reserves

Period	0.20% Cu Cut-off Tons	0.20% Cu Cut-off Grade	0.30% Cu Cut-off Tons	0.30% Cu Cut-off Grade
1971-1973 Manual Outlining			- 15%	- 12%
1973-1975 Grab Sampling	+ 13%	- 18%	- 5%	- 11%
1976-1978 Kriging	+ 59%	- 13%		

+ indicates mined ore is greater than estimated.

TABLE 15 Mined ore versus blast-hole assay distribution

Period	0.20% Cu Cut-off Tons	0.20% Cu Cut-off Grade	0.30% Cu Cut-off Tons	0.30% Cu Cut-off Grade
1972-1973 Manual Outlining			- 12%	- 23%
1974-1975 Grab Sampling	+ 2%	- 17%	- 4%	- 19%
1976-1978 Kriging	+ 20%	- 17%		

+ indicates mined ore is greater than estimated.

TABLE 16 Mined ore versus kriged grades from BH assays within 50- by 50- by 40-ft blocks

Period	0.20% Cu Cut-off Tons	0.20% Cu Cut-off Grade	0.30% Cu Cut-off Tons	0.30% Cu Cut-off Grade
5 Upper Benches, Mainly Manual Outlining			- 26%	+ 4%
1976-1977, Kriging and Grab Sampling	- 7%	+ 5%		
1977-1978, Kriging Alone	+ 2%	+ 2%		

+ indicates mined ore is larger than estimated.

(Taken from Raymond, 1972.)

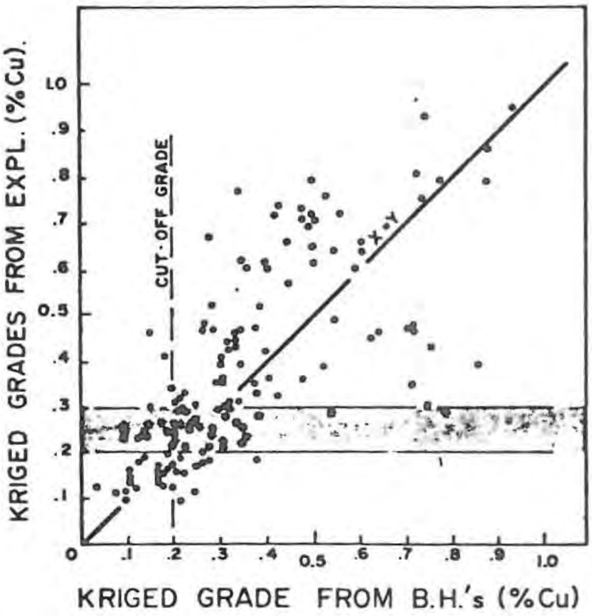


FIGURE 72 **Block-by-block comparisons.**

(Taken from Raymond, 1972.)

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