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GEOLOGICAL FACTORS IN THE EVALUATION OF VEIN DEPOSITS

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CONTENTS

INTRODUCTION

Vein-type mineralization, particularly gold, copper, tin and tungsten has provided a source of metal to man for over 2000 years. These deposits are usually small but in some cases are of very high grade. Prior to 1940 veins were extremely important sources of metals because of their relatively high-grades. However, with improved mining, smelting and concentrating methods, much lower grade material became economic, hence these smaller deposits could no longer hold their dominance. Recently the energy crisis and escalating costs of capital for large projects has made smaller deposits attractive once more (Temblay and Descarreaux, 1978). At the present time gold, tin and tungsten command high prices on world markets. It is for these reasons that a study of the evaluation techniques pertaining to these deposits has been undertaken. In this review the geological factors which influence the evaluation are stressed. In particular, emphasis is placed on the emplacement of vein deposits, and the subsequent chemical and structural modifications of these deposits. The latter part of the review concentrates on the limitations of the sampling and ore reserve techniques that can be applied to the evaluation of mineralized veins. In the conclusion those techniques that are most applicable are stressed, and an evaluation model is outlined.

It is hoped that the review will be of particular use to geologists involved in the evaluation of vein-type tin, tungsten and gold deposits, where it is predicted the bulk of the exploration effort will be concentrated in the near future. For the sake of brevity discussion is confined mainly to these elements. However, in many cases the concepts employed are sufficiently general to be applied to other metals occuring in veins.

Vein deposits have been defined by McKinstry (1948) as "extramensurate" or "difficult to explore and measure in advance of mining". The definition adopted for vein deposits in this dissertation is as follows:
"Irregular, tabular, ore bodies which are characterized by thin and

erratic mineralization. They may be of considerable lateral extent and vary in dip from horizontal to vertical. The deposits require close sampling and therefore extensive valuation development. This development then serves for stope preparation. The available ore reserves are usually only adequate for several years and the mining techniques employed are commonly labour intensive and highly flexible."

Evaluation begins after a "discovery" has been made. In this review a discovery is defined as "vein material containing some evidence of mineralization."

THE ORIGIN OF VEIN DEPOSITS

The origin of vein deposits has been a source of conflicting opinion for almost 200 years. Most of the divergence has centred on whether the metal and gangue are the products of magmatism or whether they are derived from leaching of the country rock. Gerhard (1781) is quoted in Park and MacDiarmid (1975) as considering vein deposits as open fissures filled by minerals leached from the country rock, whereas Hutton (1788) thought that the deposits were derived from molten magmas at depth and were transported in a liquid state to their present position. Today this argument has not yet been resolved and, as a result, discussion on genesis, in this review, is limited to the following established facts.

1) The deposits are secondary (epigenetic) features that are younger than the country rock in which they are found.

2) Structure has played a vital role in ore localization by guiding ore-bearing fluids into areas favourable for their deposition. Usually favourable areas are those areas of low pressure and temperature.

3) The ores are hydrothermal in that they are deposited from water-rich fluids.

4) The amount of water or steam required to transport the metals indicates that conveyance as simple dissolved solids would require unrealistic volumes of fluid. (Reasonable explanations of how these metals may be transported in geologically acceptable amounts of solutions, centre around transport as complex compounds (Barton, 1959)).

5) The mechanism whereby the ore-bearing fluids migrate is unresolved but it is generally accepted that the migration of fluids through rock is greatly facilitated if the rock is in a state of stress (Park and MacDiarmid, 1975), and that long distance migration of ore-bearing fluids is controlled by the relatively open channel ways provided by fault systems and joint systems.

6) Deposition takes place owing to a decrease in temperature and pressure and from chemical reactions resulting from interaction with the enclosing wall rock.

7) The fluids may contain more than one metal, in which case deposition is in a definite order which may result in a zoned deposit.

Bearing in mind the seven points listed above, a general model for the development of a vein deposit may be summarized as follows.

1) Development of an open space.

2) Infill of this space with gangue material derived from a hot aqueous solution.

3) Infill of some or all of the remaining space with metal, with most of the metal being concentrated in structural traps. (Some of the metal may move out of the open space and replace adjacent rock).

4) Reactivation with the resultant reopening of the open space.

5) Repeat of 2 and 3, with some of the earlier mineralization being remobilized into the new structural openings.

The final result is a well fractured vein quartz with the locality of the bulk of the mineralization being determined by the youngest disturbance (Mawdsley, 1938).

CLASSIFICATION OF VEIN DEPOSITS

There are several classifications of hydrothermal ore deposits, most of which are based on the classification originally proposed by Lindgren (1928). Minor additions and modifications to this classification have been made by Niggli, 1941; Stanton, 1972; Park and MacDiarmid, 1975 and others. Lindgren's classification consists of dividing hydrothermal deposits into classes depending on the depth and temperature of formation. Other classifications, notably those adopted by the Russians (Smirnov, 1976) depend on the stage of development of the magma chamber, or on the composition of the hydrothermally altered wall rocks.

A vast critical literature discusses the principles of these classifications. From the point of view of evaluation the main shortcoming is that these classifications give no indication of the morphology of the deposits, the distribution of the mineralization or the geological environment of the mineralization; factors which would benefit the evaluater. For this reason the classification as proposed by Smirnov (1976) is preferred. This classification is based on the spatial position of the vein deposit relative to igneous and enclosing rocks. Smirnov (op cit.) divides hydrothermal ore deposits into three classes, plutonogenic, volcanogenic and telethermal. A slightly modified account of the main features of these three classes is briefly discussed here.

1) PLUTONOGENIC HYDROTHERMAL DEPOSITS

This class incorporates those vein deposits which occur close to a granite (pluton) contact. As such this division embraces most of Lindgren's mesothermal and hypothermal deposits. In these deposits quartz predominates as the main gangue material, followed by carbonate. The sequence of mineral associations varies for different deposits, but there is a general tendency for oxides (wolframite, cassiterite etc.) to be precipitated closest to the granite, and sulphides to occur further away. Repeated precipitation of ore minerals may result in different generations having different crystal habit, crystal size and trace element content. This may affect the metallurgy. Usually the ore minerals are coarse grained.

The ore may be precipitated by open space filling in voids, or deposited in solid rock by a process of metasomatism or it can be deposited by a combination of these processes. The shape of the ore body will depend on the shape of the spaces available for filling and the chemistry of the enclosing rock. This can later be modified by tectonism. As a result the ore bodies might be flat, pipe-like, tabular, concordant or crosscutting. The size of the ore bodies may also vary within broad limits from a few metres long to tens of kilometres. Some common mineral assemblages and examples of ore deposits in this class are:

1) Quartz-gold (Kolar goldfield)
2) Quartz-arsenopyrite-pyrite-gold (Rhodesian deposits)
3) Quartz-scheelite-gold (some Rhodesian deposits)
4) Quartz-cassiterite-wolframite-arsenopyrite (Kranzberg deposit, S.W.A.)
5) Quartz-antimony-arsenic.

Figure 1 after Smirnov (1976) shows the frequency of occurrence of the principal ore forming minerals in this class. From the figure it is clear that gold, tungsten, molybdenum and copper are the most typical.

2) VOLCANOGENIC HYDROTHERMAL DEPOSITS

The second class of deposit incorporates those veins that occur within the volcanic rock or granitic stocks to which they are genetically related. This distinguishes this class from plutonogenic veins which occur peripheral to the stock. Typically, these deposits occur as veins, pipes and stockworks in the fractured host. The ore bodies of this class are mainly small and pinch out rapidly with depth but not infrequently are composed of very rich ore. Usually this rich ore occurs as sporadic concentrations.

Examples of this type of deposit are:

1) Cassiterite (Zaaiplaats Tin Mine, N. Transvaal)
2) Cassiterite-wolframite-argentite-bismuth (Bolivian tin deposits)
3) Gold-silver tellurides (Cripple Creek, Colorado)

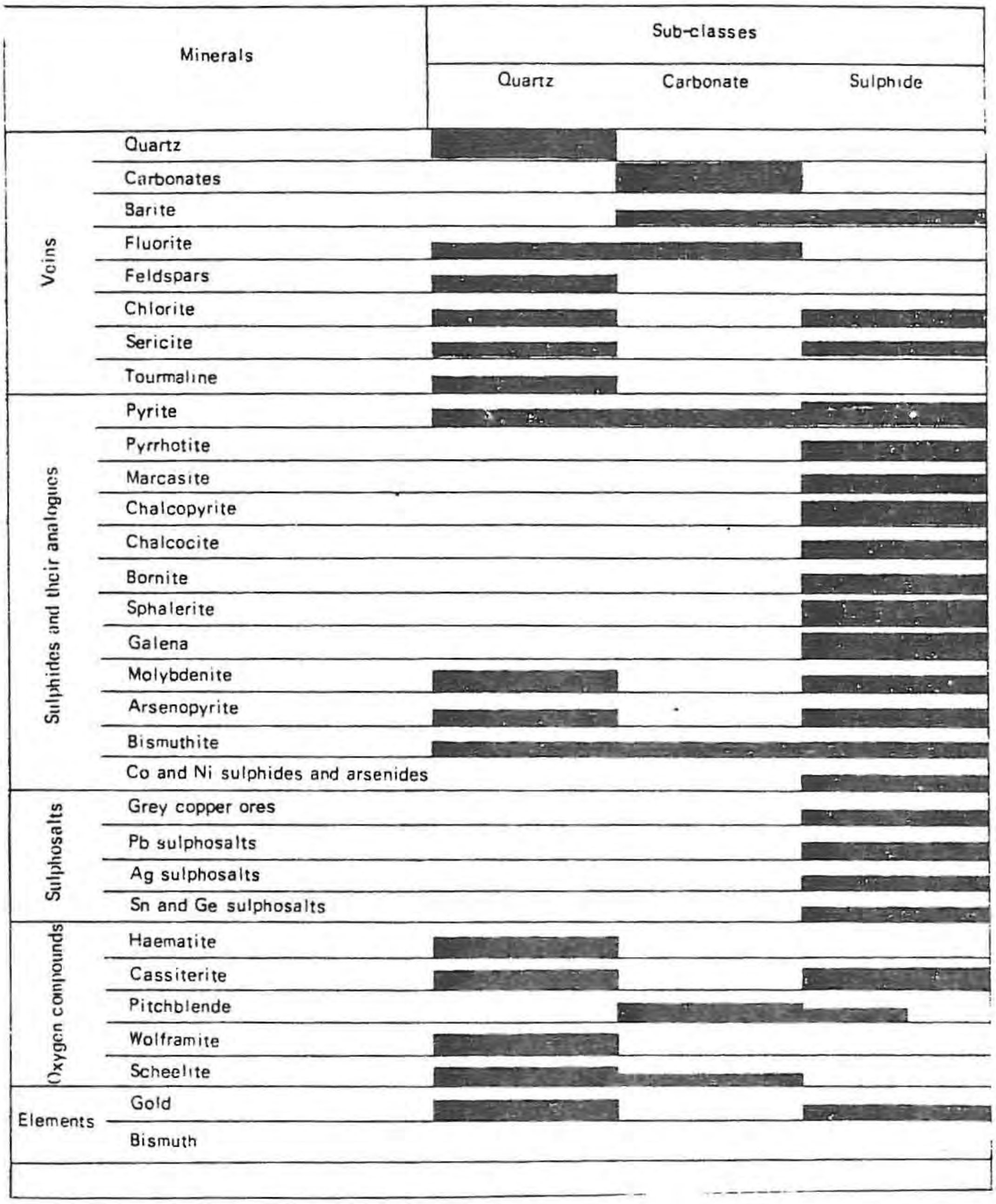


FIGURE 7 Occurrence of the principal ore-forming minerals among the sub-classes of endothermal deposits (broad black bands indicate that occurrence is very common, medium bands common, and narrow bands infrequent).

(After Smirnov, 1976)

4) Copper-molybdenum-silver-gold (Porphyry deposits of Chile and western U.S.A.).

The ores in these environments were rapidly deposited resulting in complex and varied mineral associations and local, highly concentrated zones. Usually the individual ore-bearing pipes or veins are extremely irregular in shape.

3) TELETHERMAL DEPOSITS

Telethermal deposits are those hydrothermal ore deposits that occur in sedimentary rocks, often far from any igneous activity. These ores are typically bedded and may occupy a strict, persistent, stratigraphic position in a thick sedimentary pile. Distinct control of the mineralization by faults is not specific but in some deposits fissures may guide the mineralization from one stratigraphic horizon to another. Telethermal ores have a comparatively simple mineralogy and are usually very fine-grained. Being basically stratiform they are typically thin and tabular. In some localities the ore-bodies may be repeated in section, thereby forming multistorey deposits.
Examples of these deposits are

1) Cassiterite-chalcopyrite-fluorite (Rooiberg Tin Mine, N. Transvaal).
2) Cassiterite-hematite (Union Tin Mine, N. Transvaal).
3) Chalcocite-Chalcopyrite-pyrrhotite (Artonvilla ore body, Messina Mine, N.Transvaal).

WALL ROCK ALTERATION

The country rocks bordering ore deposits of hydrothermal origin are generally altered by the hot ore fluids that have passed through them. The actual process of alteration is not fully understood and what is observed adjacent to the vein is the final result of a very complex series of events. The type of alteration that is associated with vein deposits depends partly on the original composition of the rock that is to be altered, and partly on the class of deposit. The most typical alteration of granitic rocks and sedimentary sandstones and shales is albitization, greisenization, silicification, chloritization, sericitization and argillitization. Hydrothermal mineralization in basic and ultrabasic rocks is accompanied by propylitization, by assemblages consisting of quartz-carbonate and sericite, by ferromagnesian minerals, by serpentinization and by talcose alteration. Basic volcanics undergo propylitization, alunitization and zeolitization, whilst in limestones, silicification and dolomitization are the most typical forms of hydrothermal alteration. Plutonogenic and telethermal deposits may exhibit any of these forms of alteration whilst volcanogenic alteration,because these deposits occur within the confines of the granite, usually have a halo of greisenization, albitization, silicification, chloritization, sericitization and argillitization.

Alteration is useful in guiding the initial exploration effort. However, its use in evaluation is limited to the following.

1) It provides a bigger target zone. This is because the increase in the number of fractures and veins as a lode is approached is rendered more obvious by altered borders (and more significant if the alteration is similar to that accompanying lodes in known mines in the vicinity). This larger target may also be valuable in diamond drilling if the core recovery was poor.

2) The type of alteration controls the various hanging and footwall conditions, which may be advantageous or detrimental to the project. The likely conditions must be considered during evaluation.

3) Smirnov (1976) claims that in some areas a "proportionality is observed between the scale of the mineralization and the extent of the

alteration of the wall rock." In any particular deposit where this does occur, this would be extremely important in guiding the evaluation programme. However, in general, the ratio of the area of altered rock varies within wide limits for different deposits and is not a reliable indication of metal content.

4) Sales and Meyer (1949) found that the alteration products at Butte, Montana were an integral part of the mineralization and that successive zones of sericite and argillite lay adjacent to every ore-bearing fissure. This type of relationship is obviously of use in evaluation and particularly in areas where barren and ore-bearing fissures may exist. Unfortunately, Lovering (1949), reached a different conclusion in the study of the alteration at Tintic, Utah. He found that the alteration was independent of the ore.

5) Most forms of wall-rock alteration, especially silicification, increase the brittleness of the country rock. This makes the rocks more susceptible to fracturing. Fracturing, in turn, increases the permeability of the rock and guides later mineralizing fluids into the previously fractured areas. Also, because of the hardness of silicified material, siliceous alteration leads to clean faulting with a minimum of powdering (Gunning, 1948). Silicification is seldom uniform and for the reasons outlined above the more highly silicified areas may indicate the more highly mineralized portions of the vein.

As stated at the outset, wall rock alteration is more important in the exploration than in the evaluation stage, though it may play a significant role in the local rock mechanics and may guide drill target selection and borehole core interpretation. Schwartz (1959) provides a useful list of alteration guides to ore exploration.

STRUCTURAL CONTROL ON MINERALIZATION IN VEIN DEPOSITS

INTRODUCTION

A voluminous literature is devoted to the geological structure of hydrothermal deposits, elucidation of which is of tremendous significance for determining the environment of emplacement of ore-bodies and for their exploration and evaluation. Structural control on mineralization determines

1) where the various veins are formed,
2) the length and thickness of these veins,
3) the overall distribution of the mineralization in these veins,
4) the final shape of the vein, and of particular importance,
5) the position of ore shoots within the vein.

Structural controls can be subdivided into primary structures which affect the development of the vein, secondary structures which affect the development of the vein and secondary structures which determine the final shape of the vein. These controls influence the tonnage of the deposits. Primary and secondary structures are distinguished according to whether they were formed at the same time as the rock mass or later. Secondary structures are subdivided into constructive structures and destructive structures, depending on whether they duplicate or disrupt the economic horizon. It should be recalled that accurate calculation of the volume of ore-grade rock is critical to ore reserve estimation. This volume is usually strongly influenced by the structure controlling the ore occurrence. Furthermore, only once the geometry of a highly folded ore-body is understood can detailed stope planning be undertaken. The unravelling of the structure is therefore fundamental to ore evaluation.

CONTROL OF PRIMARY STRUCTURES ON THE DEVELOPMENT OF VEINS

Primary structures are those which were formed at the same time as the rock mass in which they occur. Of particular importance are primary structures which control the distribution of fluids, because these will control the localization of ores. Since any textural or structural feature that influences porosity and permeability may control deposition, the variety of primary controls is practically unlimited and examples occur in all types of vein deposit. One form of primary control is the presence of an impermeable cap rock. This will prevent the escape of mineralizing fluids thereby causing them to concentrate,cool and precipitate. An example of this occurs at the plutonogenic Kranzberg Tungsten Mine, Omaruru, South West Africa. Here a capping of biotite schist halted the migration of mineralizing solutions. From the evaluation point of view this is important because most of the mineralization occurs in a restricted, vertical interval below the cap rock.

A second example is the Zaaiplaats Tin deposit near Potgietersrus, N. Transvaal. This is a "volcanogenic" vein deposit; in which the ore occurs within the parent granite. The high grade ore occurs as pipes which mushroom out below an extensive, stratabound, pegmatite horizon. The pipes are restricted to the first 250 metres below the pegmatite but the mushroom, top portion of the pipe, known locally as a "lenticular ore body" , occurs in the first 2 metres below the pegmatite (Groeneveld, 1964). This restriction gives a good control to the estimation of tonnage.

An illustration of a primary control on a telethermal deposit is afforded by the Rooiberg "C" tin mine. Here the cassiterite is concentrated along flat fractures along bedding planes. This results in thin, tabular, but persistent ore horizons. Lastly, Mills and Eyrich (1966), recognise that hydrothermal ore deposits are often associated with unconformities. This is probably because of a change in porosity and permeability across the contact. Igneous rocks may be intruded along this contact and because of the contrasting physical properties of the rock units on either side of the unconformity surface, openings created by folding and faulting may be localized along these surfaces. These openings could be of huge dimensions and the best development of mineralization could be expected near irregularities in the old erosion surface (for example southeast Missouri lead deposits). Detailed studies of the effect of primary structures on the final character of the ore body may therefore aid in the delineation of tonnage and grade in an evaluation programme.

CONTROL OF SECONDARY STRUCTURES ON THE DEVELOPMENT OF VEINS

Secondary structures are those structures that are superimposed on the rocks and which make the country rock more receptive or more reactive to ore bearing solutions. Before proceeding with this section, it is useful to define the terms competent and incompetent. "Competent" as the term is used here, refers to rocks that are relatively strong but when they do fail, break as though they were brittle material. "Incompetent" refers to rocks which are weak and have a tendency to deform plastically or by flow. Competent rocks, in addition to their tendency to fail by fracture rather than by shear, have the advantage that they yield to fracture in such a way as to provide permeable channelways (Newhouse, 1942). Their strength tends to prevent the fractures from closing and subsequent failure is often limited to spalling of the sidewalls to produce a jumble of fragments which presents large surfaces to ore depositing solutions. Furthermore, since the shearing angle decreases with increasing brittleness a shear fracture passing from semiplastic into brittle rock is deflected toward the plane of maximum normal stress and therefore to an attitude more favourable to opening by the movement that initiated the stress. Incompetent rocks deform by plastic flow and do not normally give rise to permeable channelways. For example in Canada and Rhodesia most of the economic vein deposits occur in a series of competent greenstone flows, each thin enough to fail but sufficiently brittle to form good fractures. Unfortunately, exceptions are so numerous that generalizations are not of much help in evaluation. However, if statistical analysis of all available data, including that from similar adjacent properties, indicates a preferred host, then this would be useful in evaluation, in determining limits to the ore body.

Control of Secondary Structures On the Development of Plutonogenic Deposits

Faults and folds are the most common secondary structures in which plutonogenic deposits occur, though joints, pipes and breccia

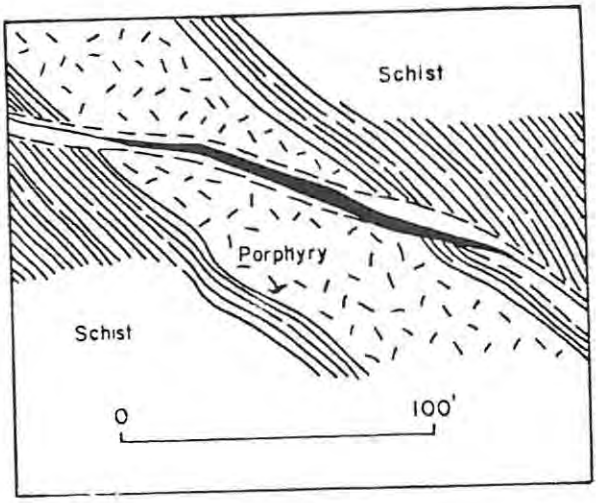
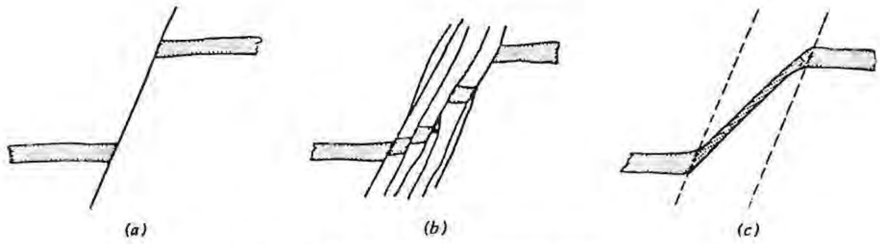


FIGURE 2 Showing how a vein thickens and deflects when passing from incompetent schist into competent porphyry. The vein shown is the Main Vein at Silver Plume, Colorado.
(After Lovering, 1942)

zones are locally of great significance.

Faults

A fault is defined by Hobbs, Means and Williams (1976) as a planar discontinuity between blocks of rock that have been displaced past one another. A fault zone is a tabular region containing many parallel or anastomosing faults, whilst a shear zone is a zone across which blocks of rock have been displaced in a faultlike manner, but without prominent development of visible faults. Shear zones are thus regions of local ductile deformation and are more typical of incompetent rocks in contrast to fault zones that are regions of local brittle deformation more typical of competent rocks. The three terms are diagramatically illustrated below (Figure 3)*.



(*a*) Fault. (*b*) Fault zone. (*c*) Shear zone.

FIGURE 3 (After Hobbs et al., 1976)

There are several types of faults and several classifications of faults. These are given in standard texts of structural geology (e.g. Price, 1966; Ramsay, 1967; Hopwood, 1974; Hobbs et al., 1976) and are not repeated here.

The type of fault and the amount and direction of movement are important because, as a general rule, in faults having a strong horizontal component of movement, a swing of the fracture plane in a direction away from that of apparent movement will favour the creation

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* The term shear or shear zone is very loosely used in the literature and in most cases the term fault zone would be more appropriate. A shear zone, in the sense used here, is a zone of ductile deformation with a large normal component of displacement parallel to the plane of tension. In its simplest form a shear zone would not be sufficiently open to allow significant circulation of ore bearing fluids. In the older literature faults and fault zones are often collectively referred to as fissures.

of an open space. That is, when viewing the fracture plane along strike if the right hand wall has apparently moved ahead, then open spaces are most likely where the fracture plane swings to the left (Figure 4). Similarly the steepening of the walls of a normal fault

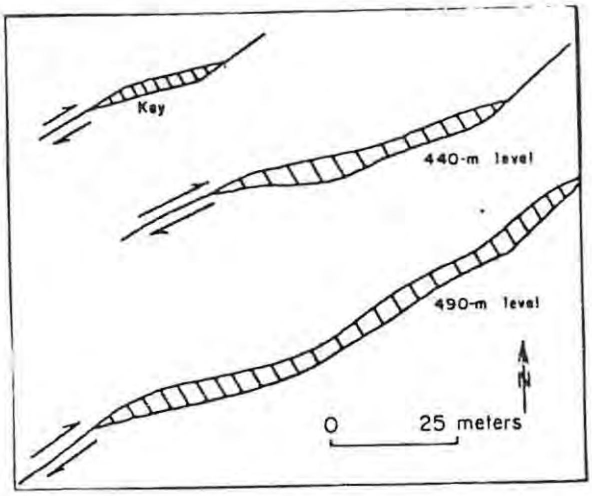


FIGURE 4 Here the left hand sidewall, when viewed along strike has moved ahead and therefore an opening occurs where the strike deflects to the right.

(Taken from Newhouse et al., 1942)

and the flattening of the walls of a reverse fault favour the creation of open spaces (Figure 5). Normal faults steepen as competency increases

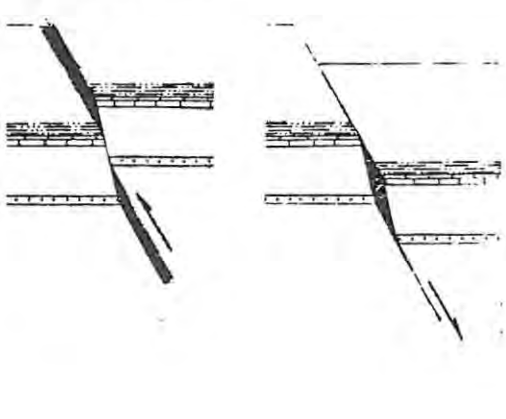


FIGURE 5 Openings caused by reverse and normal movements along veins. Note pinches and swells.

and reverse faults tend to flatten when traversing competent horizons. These relationships occur along strike and down dip and hence a change in competency across contacts provides prime positions for openings if movement takes place. The direction of movement can often be gauged from slickenslides, fault drag (the curvature of layering adjacent faults) or, which is by far the most useful indicator, from two originally contiguous points on either side of the fault, such as a fold hinge, dyke or stratigraphic horizon, which have been offset by the fault.

Openings are important because ore bearing solutions migrate through the more open part of the fissures and are deflected around the tighter zones. Consequently rolls or changes in strike and dip commonly mark the beginning or end of highly mineralized zones (Figure 6). As a result these irregularities are highly significant in evaluation.

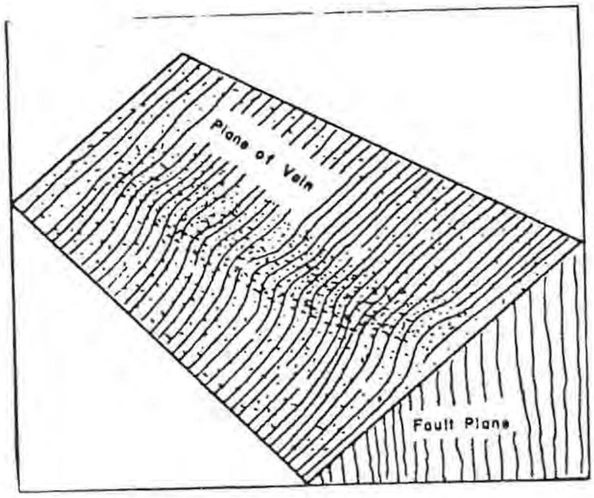


FIGURE 6 Ore shoot related to a "roll" in the plane of a vein. The vein illustrated is the Missouri vein northwest of Grant, Colorado.

(After Lovering, 1942)

Chinnery (1966a) has shown that although after faulting the initial stress is reduced over most of the length of the fault, there are strong concentrations of stress along the ends of the fault. This results in a tendency for the fault to extend itself. This extension can be achieved in several ways but mostly it occurs by secondary faulting. Secondary faulting usually consists of a series of curved splay faults or horsetails in regions close to the end of the major fault. The different modes of secondary faulting are shown approximately in order of likelihood of occurrence in Figure 7.

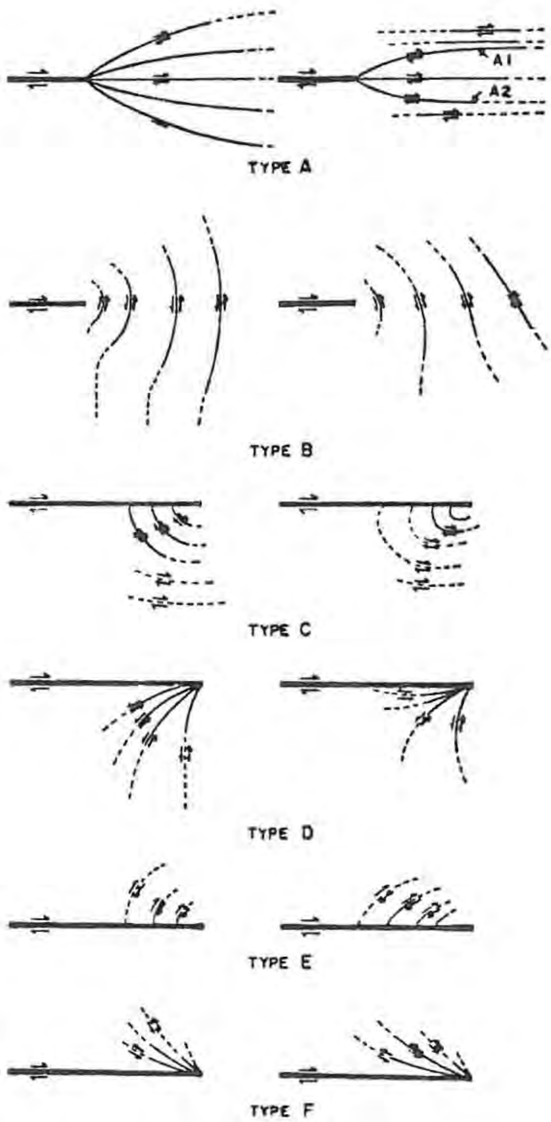


FIGURE 7 Different modes of secondary fault, in approximate order of likelihood of occurrence.

(After Chinnery, 1966b)

Thus, types A and B occur in regions of high shear stress and are common. Types C and D occur in regions of tension and E and F occur where shear stress is quite high but also where compressive stress is high and friction, in many cases, inhibits their formation. Secondary faulting is important in evaluation for the following reasons.

1) It results in a duplication of the ore horizon towards the end of the major fault; these secondary faults usually have curved fault planes with abundant open spaces.

2) The characteristic pattern of secondary faults can be followed into the major fault which may contain the bulk of the ore.

3) Conversely, the major fault may have been too open to retain the ore forming solution, or it may have been too full of fault gouge to permit the circulation of ore forming fluids, and in these cases the major fault should be traced to its extremities where secondary faults may have been better hosts to the mineralization.

4) Secondary faults may themselves generate secondary faults. This is the case at the Messina ore deposits in N. Transvaal (Mr. D. Dicks, pers. comm., 1979). Here the Messina fault is a secondary splay off the major Dome-Tokwe fault and has in turn generated its own secondary faults. The Dome-Tokwe fault is not mineralized but several ore bodies occur along the secondary Messina fault, and the Spence ore body is situated in a secondary fault off the Messina splay. The possibility of multiple ore bodies in secondary faults both down dip and along strike is, therefore, an important consideration in evaluation.

5) The amount of displacement along these secondary faults may be a guide to the tenor of the ore. This is the case in the Barberton goldfield (Gribnitz, 1964) where, as the relative displacement across the fracture decreases so the ore body peters out. If the fracture displacement remains constant then an impoverishment of an ore body may be just a local feature and better ore could be expected away from this point. However, the coincidence of diminishing ore and lessening displacement is considered by Gribnitz (op. cit.) as an "unmistakable danger signal".

In any area of stress there are three attitudes of potential rupture; two of shear stress, which make an angle of approximately 45° with the direction of principal stress and one of tension which is parallel to the principal stress and which bisects the acute angle between the planes of shear failure (Figure 8).

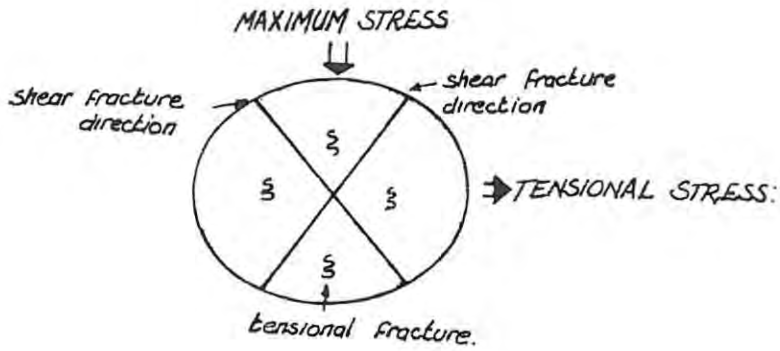
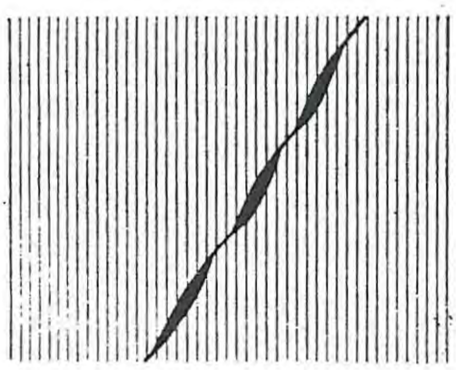


FIGURE 8 Strain ellipse depicting principal stress, tensional stress and the planes of fracture.

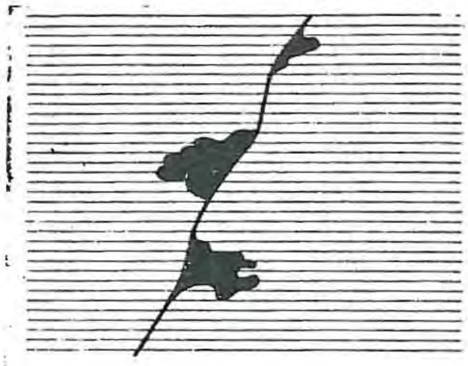
In general, faults which are parallel to the direction of principle stress give rise to large tonnage, low grade ore bodies, whereas ore bodies which occur in tensional fractures, because of their very open nature, are usually highly mineralized but of small tonnage. Mehliss (1964) lists the Dawn Mine and Queens Mine near Bulawayo as examples of large tonnage, low grade vein deposits and the Queen Alice, Hyperion and Reward mines as examples of small, high grade deposits in tension fractures.

Faults are found nearly everywhere and many plutonogenic vein deposits occur within them. Because fault surfaces are uneven, movement along the fault produces breccia and gouge. The fine-grained gouge frequently hinders the circulation of the fluids, thereby preventing the formation of rich ore deposits. Gouge is most typical of large, low angle faults and accordingly minor, secondary faults may be much better hosts for ore solutions. Rolls or changes in strike or dip of a vein commonly mark the beginning or end of an ore shoot, hence such irregularities are significant in evaluation and must be distinguished from rolls developed after the emplacement of the ore. Also, veins are more productive within walls of competent rocks than within walls of incompetent rocks because here the fractures are more likely to be open, and for the same reason, veins are more highly mineralized when the fault has a large displacement and when it crosses (rather than follows) the foliation. These are important aspects to consider when evaluating a vein deposit. A structural analysis can give some indication of the maximum length of any fissure and as a result, the maximum length of any associated vein. Also, it can give some indication of the possible grade of a vein. However, because there are so many exceptions, any further generalizations of expected grade and tonnage would be meaningless. A few examples may illustrate this point; the Pfal vein in the Precambrian rocks of Bavaria is 140 kilometres long, whilst the Halsbrucker Spat vein of Freiberg, Saxony is 8 kilometres long. The extremely rich Cam and Motor veins of Rhodesia had a strike length of 1 kilometre whilst the more weakly mineralized Dalny mine has a recorded strike length of 1.5 kilometres, a similar strike length is reported for the important Queens Mine. Also, there is no definite relationship between depth and length of fissure though generally speaking, the longer veins have the greatest depth.

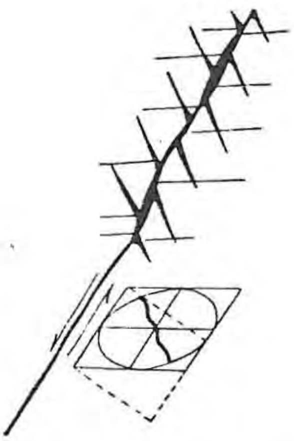
In the discussion so far a vein has been considered a tabular feature formed by progressive filling in an open space. It should be borne in mind that a vein is not always simple but may contain large inclusions of barren wall rock and may have highly irregular contacts owing to selective replacement of certain minerals, or lithologies or through variations in the permeability of the host lithologies. A few self-explanatory illustrations of the morphology of some fault-contained plutonogenic veins is presented below (Figure 9).



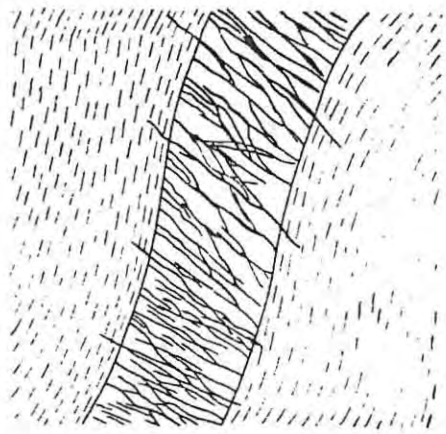
Well-defined vein or lode



Chambers along a vein



Finned vein and diagram of the tectonic dislocation along the pillar of the vein (with an indication of the position of the parallelepiped and ellipse of deformation)



Ladder veins

Plutonogenic Veins in Joints

Jointing is a brittle fracture phenomena which forms under conditions of low confining pressure and relatively high strain rates. Although the origin of joints is a vexed question, three types of joint systems are commonly recognised.

1) An imposed set related to folding and faulting.
2) A set due to unloading.
3) A set due to cooling of igneous bodies.

Of the imposed set, there are four different joint orientations that can form as a result of folding. The first two are a set of joints which make an angle of 45^{o} with the fold axis. Then there is a set of joints which parallel the fold axis and the fourth direction are those joints perpendicular to the fold axis. In any field example this relationship may require modification in that one or more sets of joints may fail to develop. The length of the joint and the distance between joint intersections depends upon the size and type of fold, the relative competence of the rock unit in which the structures are formed and the thickness of the rock unit. Usually joints are confined to particular lithologies and therefore the depth of the joint is equal to the thickness of the jointed horizon. Often only one joint direction is mineralized (Dr. A. Mehliss, pers.comm., 1979). This results in a series of en echelon pods of discontinuous mineralizations (Figure 10). Because of the restricted length of joints, it is important in evaluation to distinguish a joint from a fault. This is usually easily done because joints have smooth surfaces which are not filled with gouge, and often joints that have not been weathered have, along the joint plane, faint ridges or rays which form a barb, plume or feather-structure, the axis of which is commonly parallel to upper and lower surfaces of the rock unit.

In addition to the points mentioned previously, the following aspects of joint frequency, as listed by Price (1966), are worthy of note and may have economic implications.

1) Joint frequency is commonly many hundreds of thousands of times greater than fault frequency.

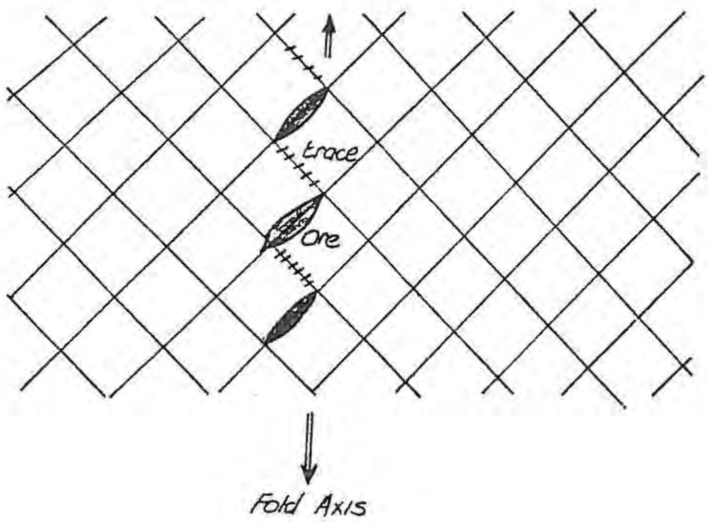


FIGURE 10 Discontinuous mineralization in joints.

2) Joint frequency in any ore locality is not constant but varies with the lithology of the rock type.
3) For any single lithological type, joint frequency is related to the dimensions of the rock unit, thinner rock units having more closely spaced joints of shorter vertical extent.
4) Joint frequency is influenced by the degree of tectonic deformation.

En echelon tension gashes are another form of an imposed joint system. In some areas (commonly associated with gold mineralization) these fracture systems may be abundant and filled with quartz or carbonate (Hopwood, 1974). The conventional interpretation (after Cloos, 1932) is that these fractures are due to a shear system inducing tension as shown in Figure 11. According to Ramsay (1967) zones of en echelon quartz or calcite filled tension fractures form on the flanks of fold structures where the dips attain the highest value (Figure 11) It should be borne in mind that these structures only develop in competent lithologies and their dimensions are controlled by the width and thickness of the host lithology. According to Dr. A. Mehliss (pers. comm., 1979) a rule of thumb, based on wide experience in Rhodesia, is that these tension gashes are crescent shaped and have

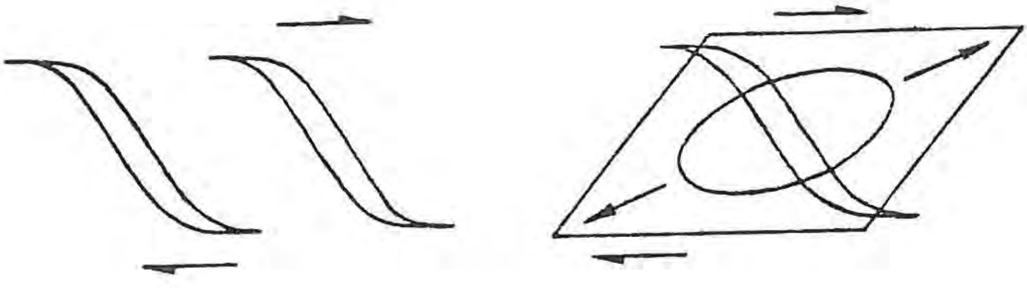


FIGURE 11 The conventional interpretation of the origin of _en echelon_ tension gashes

(After Cloos, 1932)

depth extensions of only one third their strike length.

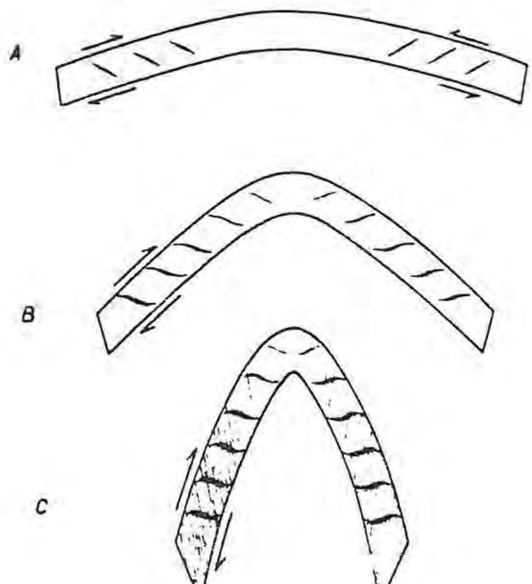


FIGURE 12 Progressive development of sigmoidal tension fissures and slaty cleavage as a result of progressive fold development with internal deformation by flexural flow.

(After Ramsay, 1967)

The best known examples of joints related to faulting are pinnate fractures (Hobbs, et al., 1976). These occur preferentially in the immediate vicinity of a fault plane and intersect the fault in an acute angle pointing in the direction opposite to the relative movement of the block containing the fracture. These fractures can be of any scale, but usually are small (Figure 13). Because these

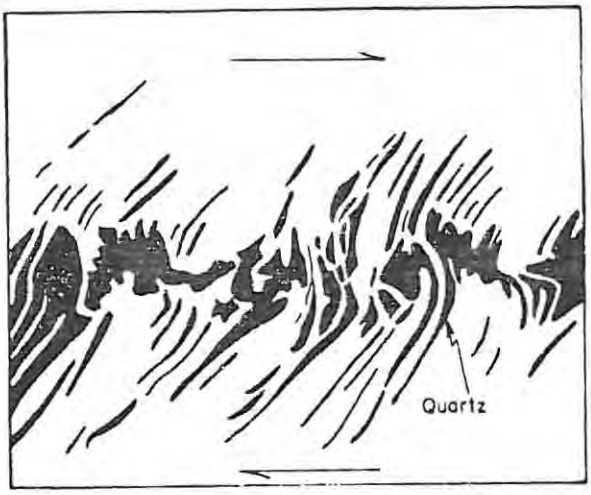


FIGURE 13 Tension gash fractures formed by movement along a fault. The fractures illustrated are gold-bearing quartz veins at the Hollinger mine, Timmins, Ontario, Canada.

(After Newhouse, 1942)

fractures are formed contemporaneously with, and intersect, the main fault, they are usually mineralized, along with the parent structure, should a mineralizing event occur. This can result in a local thickening of the economic horizon at the point of intersection of the two lineaments and, if encountered in diamond drill core away from the main fault, may give the impression of a second ore horizon. These structures are only of limited lateral extent and in many respects are similar to the splay faults that form at fault termini (Hobbs et al., 1976).

Joints that occur as a result of unloading do not normally contain vein material and will not be discussed here. They are treated in detail by Hobbs et al., (1976).

Contraction joints are formed when an igneous rock cools. According to Lindgren (1928) the so called ladder veins are deposits which occur in these joints (Figure 14). These are short transverse

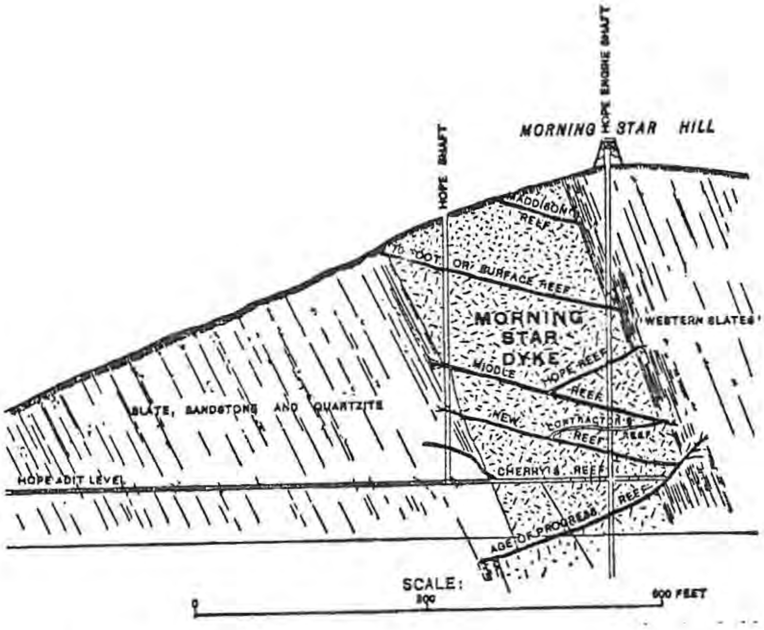


FIGURE 14 A section of the Morning Star dyke, Woods Point, Victoria, showing ladder veins believed to have formed in contraction joints.

(After Lindgren, 1928)*

joints with a maximum length equal to the dimension of the igneous rock mass. If a sufficient number of joints are formed then a stockwork ore body may develop.

Joints can result from cooling of igneous rocks, unloading, or from faulting and folding. They are typical of competent rocks where they are abundant and closely spaced. The tensile stresses responsible for jointing lead to open spaces which may guide metal-

* Not all authors agree that the Morning Star veins occur in contraction joints. Threadgold (1958) interprets the fractures as a conjugate set of reverse faults.

bearing solutions. However, unlike faults, joints seldom cross lithological boundaries and, if filled, result in short transverse veins to give a series of discontinuous mineralized pods each of limited tonnage.

Plutonogenic Vein Deposits in Pipes and Breccias

Pipes, breccias and chimneys are bodies that are relatively short in two dimensions and long in the third. The origin of pipes is uncertain, but they are commonly formed at the intersection of two tabular features such as faults, dykes, bedding, lava flows or joints (Figure 15). Where the tabular features are fault-like,

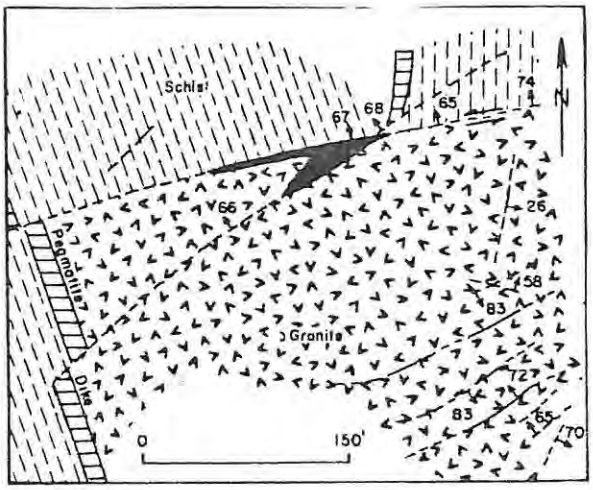


FIGURE 15 Ore shoot localized by the intersection of fractures. Lily tunnel level of the Rakeoff Mine, Nederland, Colorado.

(After Lovering, 1950)

brecciation is likely to be most extensive if the fractures intersect at small angles (Park and MacDiarmid, 1975). Pipes are also formed at the crests of folds, especially where permeability has been increased by strata sliding over each other leaving areas of reduced pressure at the crests. Where several pipes are situated above one another the resultant ore bodies are called saddle reefs (Figure 16). Diatremes or volcanic explosive vents form pipes in areas of igneous activity and represent highly porous avenues for the escape of hydro-

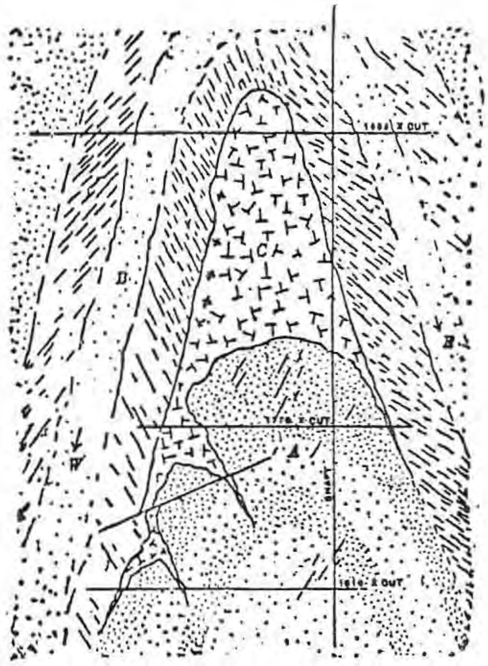


FIGURE 16 Section through a saddle reef, Bendigo, Victoria. A, Sandstone; B, shaly sandstone; C, gold-bearing quartz.

(Taken from Lindgren, 1928)

thermal fluids. Here brecciation may be the result of the explosive escape of magmatic fluids or the breccia may be formed by escaping, superheated, meteoric waters. These pipes are funnel shaped, with diameters that range in thickness from 1000 metres at the top to 100 metres at depth (Hack, 1942).

An excellent review of the origin of breccia pipes is given by Mitcham (1974), who concludes that most breccia pipes develop from movement along irregular fault surfaces. This is diagrammatically explained in Figure 17. It is important in evaluation to recognise that the mineralized body is of pipe-type. These carrot-shaped bodies pinch at depth and this must be appreciated in tonnage estimates. When dealing with mineralized saddle reefs it should be borne in mind that these reefs have no vertical or down dip continuations.

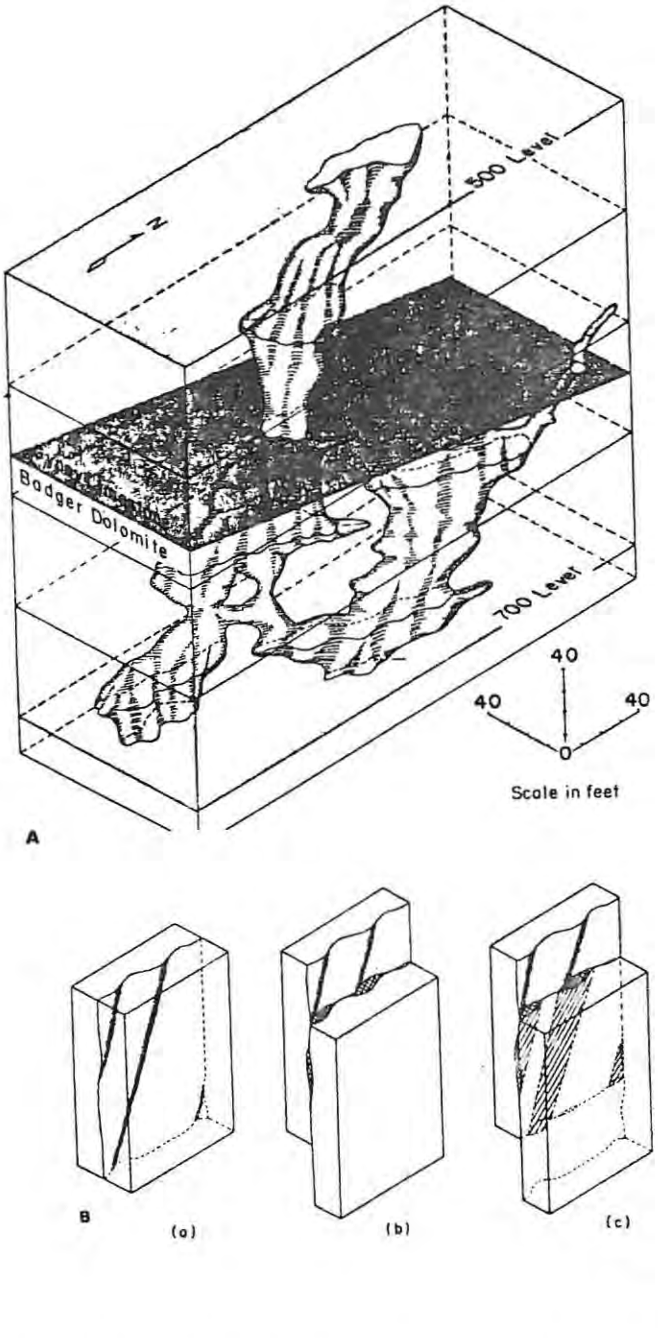


FIGURE 17 (A) Isometric drawing of an ore pipe, Bristol mine, Nevada. (B) Probable development of breccia pipes at Bristol mine, Nevada: (a) irregular fault surface before displacement; (b) after vertical displacement, showing development of swells: (c) pipes along which collapse breccias would form.

Plutonogenic Vein Deposits Associated with Folding

Open spaces generated by folding may give rise to saddle reefs and deposits in joints. These have already been described. Further sites may be provided by the shattering of brittle lithologies along the fold axis and, according to Hopwood (1974), by faulting along the hinge of early formed folds. The latter is called upon by Barker (1977) to explain the gold mineralization in the How Mine near Bulawayo. Generally, because folding is characteristic of ductile rather than brittle deformation, folds do not provide open space for ore deposition and are not as important as faults for vein mineralization.

Control of Secondary Structures On the Development of Volcanogenic Vein Deposits

Volcanogenic vein deposits are, by definition, those vein deposits which occur within the host to which they are genetically related. Secondary structures are, therefore, not as important as primary structures in the localization of these deposits. During crystallization, metals which commonly occur as vein deposits are concentrated in the residual fluid. The pressure in this fluid may build up until it is in excess of lithostatic pressure, whereupon it would crack the walls of the containing chamber and metalliferous veins could be deposited in the ensuing breccia. It is probable that part of this breccia would occur within the solid, outer portion of the parent magma, in which case the veins located within these confines would be regarded as volcanogenic veins in secondary structures. To a certain extent this is the case in some of the Bolivian tin deposits (Sillitoe et al., 1975). The following hypothesis on the origin of these tin deposits is summarized from Sillitoe (op. cit.). A body of undersaturated magma of quartz-latitic to dacitic composition became saturated with aqueous fluids as it intruded into higher level environments. Ascent was halted by the consolidation of the upper and marginal parts of the body. The fluids continued to accumulate in the top part of the still molten magma and were expelled when fluid pressure exceeded load pressure. The escaping fluids brecciated the already solid outer parts of the same magma generating a stockwork. These fluids resulted in alteration and accompanying mineralization (Figure 18).

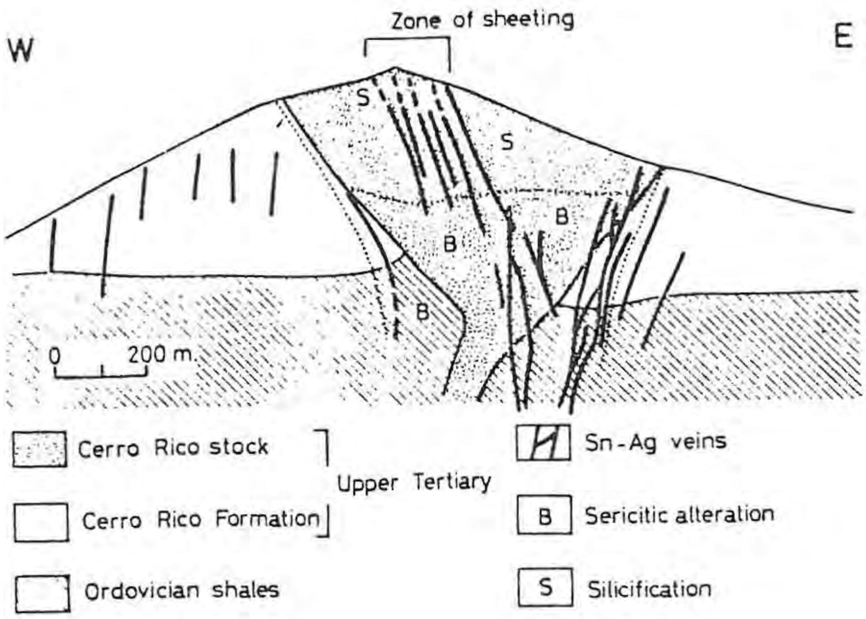


FIGURE 18 Generalized east-west section of the Cerro Luo stock at Potosi, Bolivia. Cassiterite occurs as disseminations and in a clearly visible stockwork of irregular veins 1-5 cms wide, both in the stock and to a lesser extent, in the lutites beneath the wide upper portion of the stock.

(After Sillitoe et al., 1975)

Faulting within a mineralized region may provide channelways and barriers to mineralization. According to Hosking (1973, p.24), "the unusual richness of the top-of-cusp Mawchi (tungsten) deposits may well have been due in part to canalisation of the ore forming agents which were liberated, during the consolidation of the deep granitic magma, into the cusp by the impounding action of the pre-lode wrench (?) faults which bound the ore zone to the north and south".

In evaluating volcanogenic vein deposits the following features are important.

1) The economic horizons are restricted to the chill zone of a high level, metal bearing, wet, granitic magma.
2) The mineralizing event is a unique event (rather than multiple as is the case in most plutonogenic deposits) and as a result the grades are usually highly erratic.
3) The deposits display considerable variation in size, shape, and mineralogy.
4) They often consist of subparallel vein swarms which commonly dip steeply, are rarely above 4 centimetres wide and are usually bordered by alteration.
5) Where this alteration consists partly of kaolinization a high recovery of tin or tungsten is difficult to achieve. This must be taken into consideration in evaluation.
6) Pre-ore faults may be important.

Control of Secondary Structures On the Development of Telethermal Vein Deposits

Telethermal vein deposits are those deposits that occur far from any igneous activity. These deposits may occur in faults and joints but usually primary structures, such as bedding, exert the most important control on locality and morphology. This is because the deposits are typical of stable environments. Some secondary features which increase rock permeability, and which may be important, include the dolomitization of limestone and sedimentary collapse breccias. Telethermal vein deposits are usually thin, tabular and fine-grained. Because they are deposited far from their source they usually have simple mineralogy.

CONTROL OF SECONDARY STRUCTURES ON THE FINAL SHAPE OF THE VEIN

Postmineralization warping and dislocation of a vein is extremely important in mine valuation; both in operating mines and in new prospects or re-evaluation of old workings. The premineralization -postmineralization choice may be difficult because movement may have preceded, accompanied and followed ore mineralization. A situation where fault movement occurred repeatedly in the principal ore bearing structure is shown in Figure 19, and described by Spurr (1923).

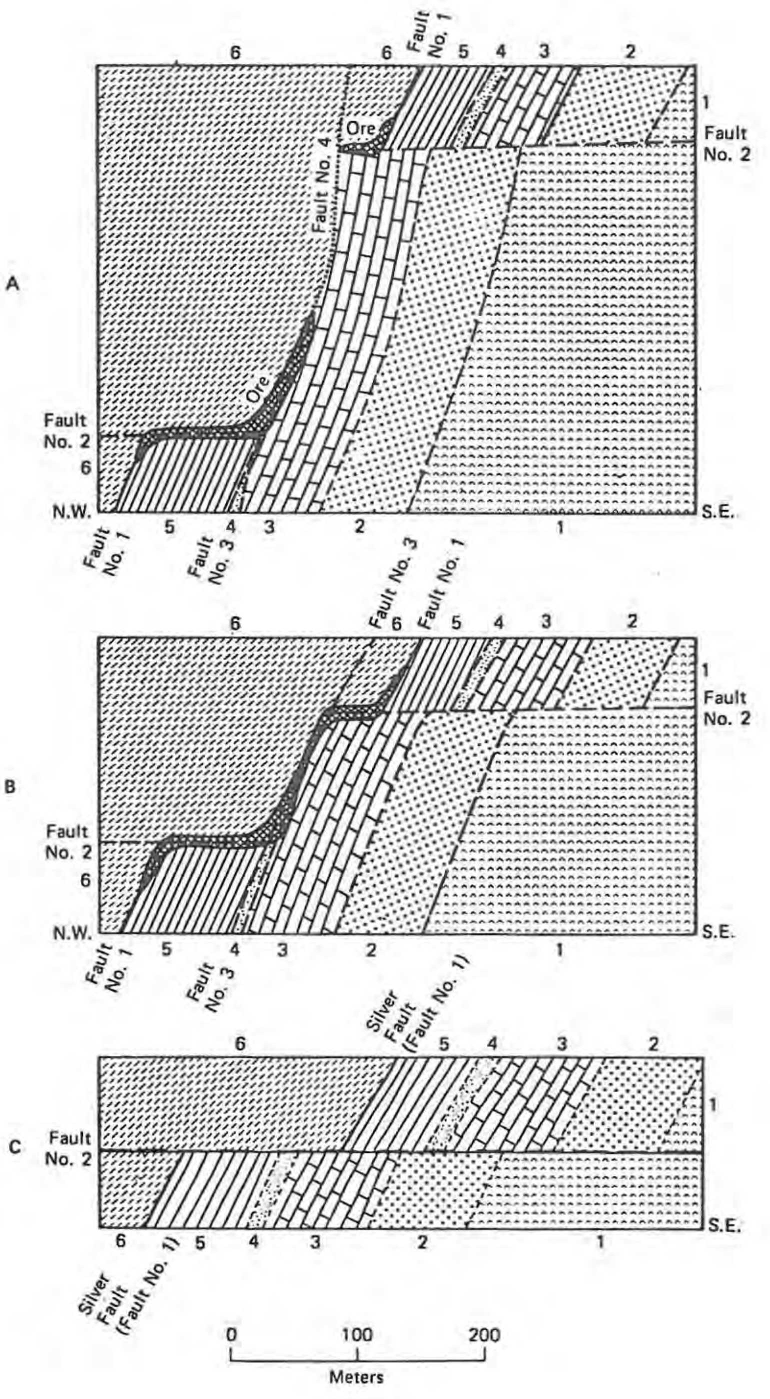


FIGURE 19 Cross sections through the Smuggler shaft, Aspen, Colorado. A= present structure; B=structure before latest movement on fault number 4 (nearly the same fault plane as fault number 3); C = structure before movement on fault number 3 and subsequent deposition of the ore.

(After Spurr, 1923)

Here figure A is the present structure, whilst figure B shows the structure before the latest movement on fault number 4 (nearly the same fault plane as fault number 3). Figure C shows the structure before the movement on fault number 3 and subsequent deposition of the ore. The solution of this type of problem could lead to the discovery of additional ore. It is obviously important to know if the disrupting structure was pre or post mineralization. If the major movement on a truncating fault is pre mineralization then this would discourage the thought of a lost segment of the ore body but it does allow for the possibility of similar controls (and hence similar ore bodies) having existed in the displaced zone. Analysis of the fault may then still be rewarding. This is what occurred at Guanajuato, Mexico and resulted in the discovery of an entire series of important ore bodies beneath exhausted mine workings.

The manner in which post mineralization disturbances affect the shape of the vein is the subject of this next section. The structures under consideration can be broadly subdivided into constructive and destructive structures depending on whether they duplicate or disrupt the economic horizon.

Post Mineralization Folding

During deformation a quartz vein may experience different kinds of strain at different stages in the deformational history, the general principles of which may be represented by two strain ellipsoids. One strain ellipse describes the strain up to that time and may be called the finite strain ellipse. The second describes the amount of strain that is about to be imposed in the next instant and is known as the incremental strain ellipse. In each ellipse three fields may be defined: a field in which all lines retain their initial length, one in which all lines have been shortened and one in which all lines have been extended. This is illustrated in Figure 20. During most geological deformation the principal axes of strain rotate progressively; hence, the principal axes of the finite and incremental strain ellipsoids, at any instant, will not coincide. The superposition of these six fields yields four asymmetrically arranged fields labelled

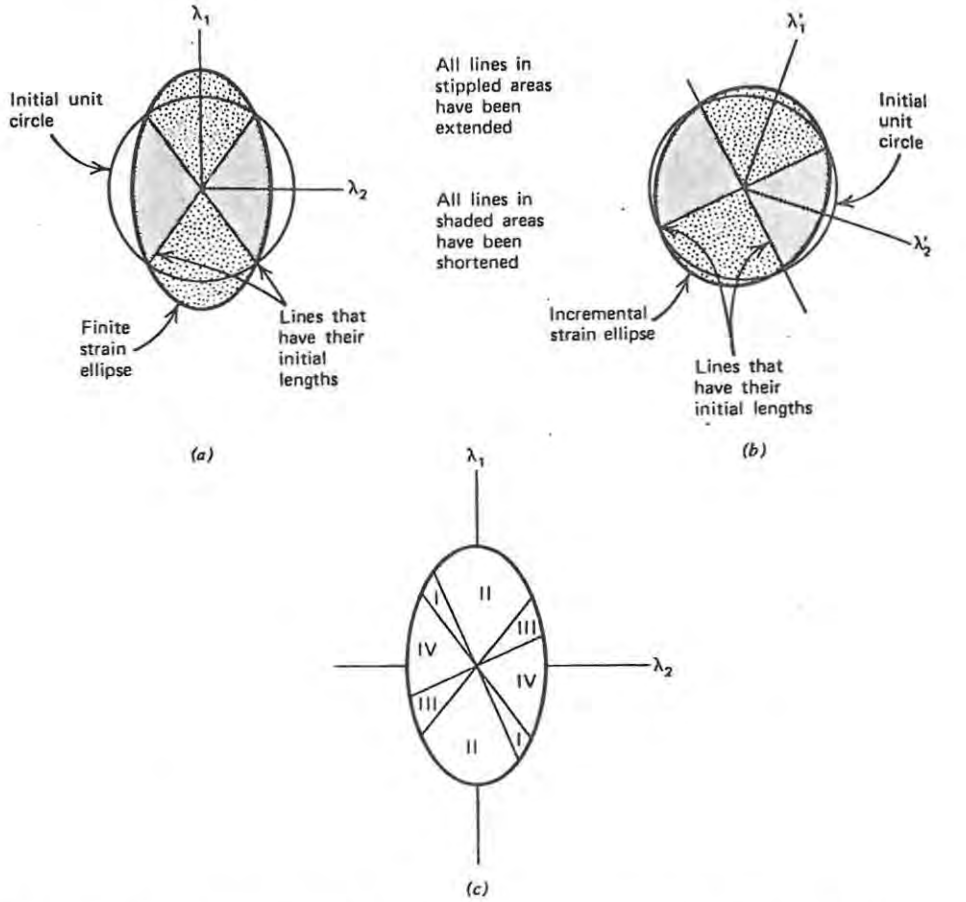


FIGURE 20 Incremental deformation involving constant volume. *(a)* The finite strain ellipsoid with principal axes of strain λ_1 and λ_2. An initial unit circle intersects this ellipse along two lines that have their initial lengths. The stippled area includes all lines that have been extended and the shaded area includes all lines that have been shortened. *(b)* The incremental strain ellipsoid; this is almost a circle and represents the increment of strain that will be added to the finite strain ellipsoid in *(a)* during the next increment of deformation. Notice that the principal axes of the incremental strain ellipsoid λ_1' and λ_2' are not parallel to the finite axes of strain. Again the stippled areas and shaded areas have the same significance as in *(a)*. *(c)* The four different fields that are marked out by the intersection of the four fields in *(a)* and *(b)*. In field I, all lines have been extended but will be shortened in the next increment. In II, all lines have been extended and will continue to be extended in the next increment. In III, all lines have been shortened but will be extended in the next increment, and in IV all lines have been shortened and will continue to be shortened in the next increment.

I, II, III, IV, in the diagram. Extended discussions of progressive strain are given by Ramsay (1967), Elliot (1972) and Hobbs, Means and Williams (1976). A brief summary of the main effects as pertaining to vein deposits is given below.

In field I all lines have been extended and will be further extended in the next strain increment; veins orientated in this direction could be expected to display boudinage along one or more directions. The extent of boudinage would be a function of the contrast in competency. If the contrast is low then necking or pinch and swell may result rather than discrete boudins. This is a destructive factor and will result in lenticular ore pods or in extreme cases, chocolate-tablet structure (Ramsay, 1967).

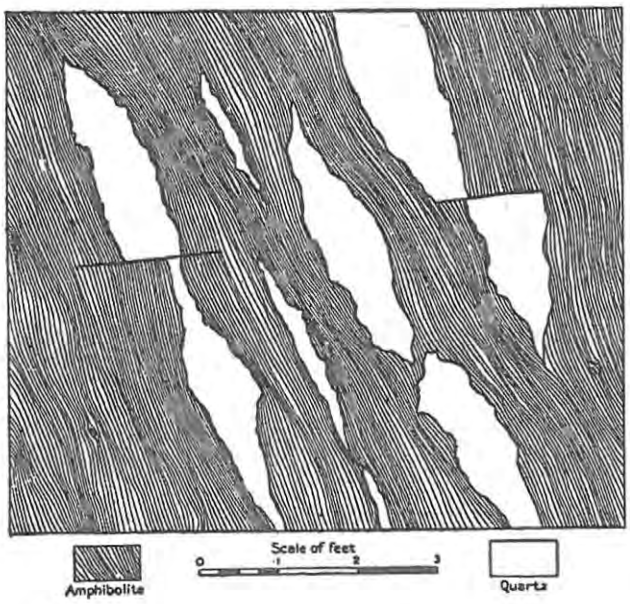


FIGURE 21 Plan of the Schlegelmilch quartz vein, South Carolina, showing lenticular vein structure in incompetent schist.

(After Lindgren, 1928)

In field II all lines have been extended but will be shortened in the next strain increment; a boudinaged vein striking in this direction could be expected to have the boudins folded and the boudinage spacing decreased in the next episode (Figure 21). In this field an initially destructive factor would be partly compensated for by a constructive factor. The ore will still be discon-

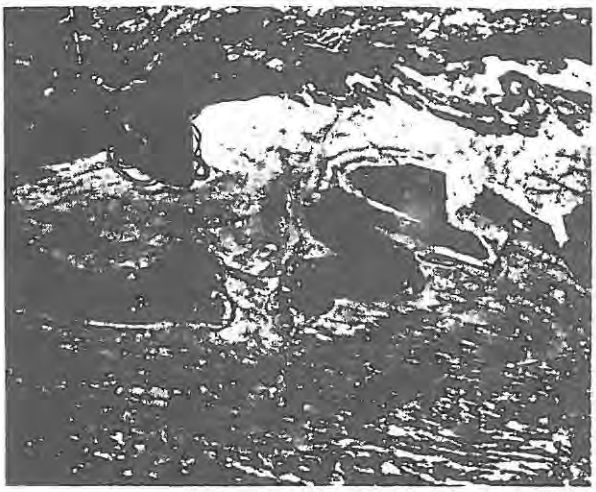


FIGURE 22 Strongly boudinaged folds which indicate an initial contraction along the banding followed by an extension. Khan Gorge, South West Africa.

(After Ramsay, 1967)

tinuous, but the pods will be closer together. In field III all the lines have been shortened but will be extended; these veins will have been folded but will be somewhat unfolded by the next strain increment, and in extreme cases may even be boudinaged (Figure 23).

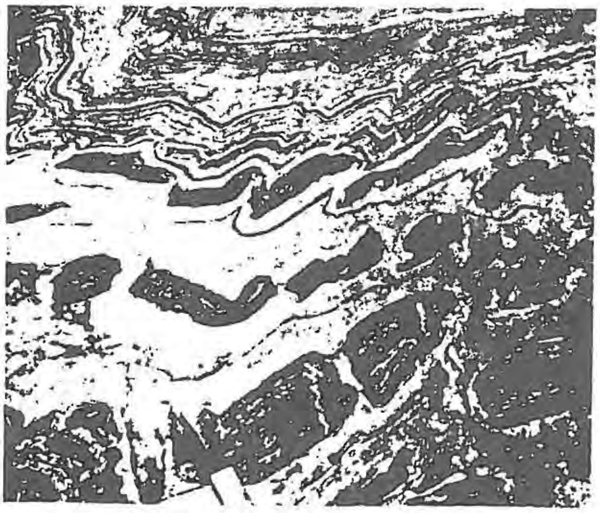


FIGURE 23 Banded hornblendic gneisses which have suffered elongation along the banding and then subsequent contraction. The boudins formed during one stage of the deformation were folded at a later date. Lower Pennine nappes, Ticino, Switzerland.

(After Ramsay, 1967)

Folding is usually a constructive factor as it leads to a duplication of the ore horizon and a thickening of the ore zone in the fold hinge. However if these folds are later boudinaged the net effect will be destructive. In field IV, the previously shortened lines will be further shortened and a vein with this orientation will be more tightly folded. Usually this is a constructive factor leading to the duplication of the ore horizon. However, in heavily deformed areas with a high competence contrast, the continuity of the ore horizon can be lost by gliding occurring along lineations. This may result in the attenuation of alternate fold limbs to leave the competent horizon as isolated hinges or lenticles.

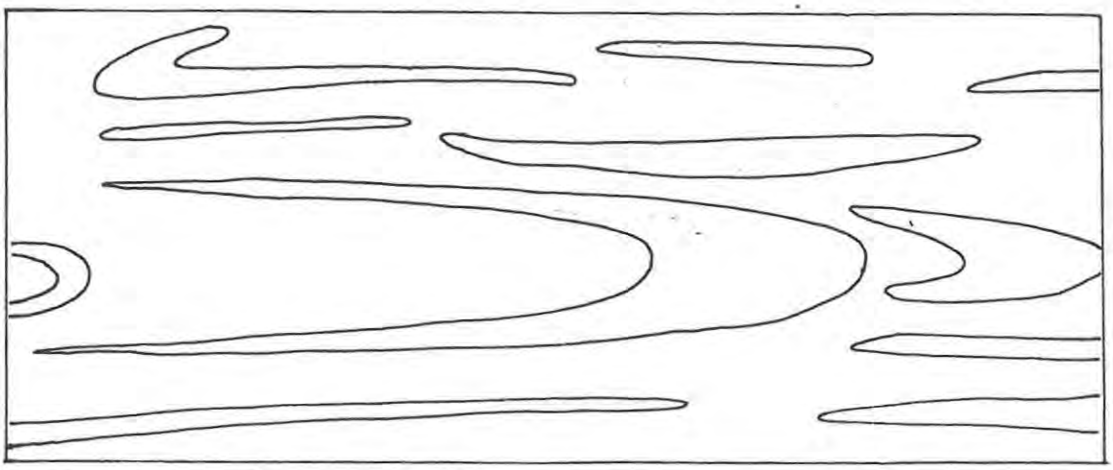


FIGURE 24 Attenuation of the competent horizon by translation gliding during folding.

(Redrawn from Hopwood, 1974)

As the principal axes of strain rotate with respect to one another during deformation, combinations of these deformation effects may occur, resulting finally in some form of boudinaged fold. A detailed examination of small scale structures within the context of the larger structural environment can supply considerable information on the strain history of a deformed vein. Useful markers for analysis of finite strain include

a) initial spherical or ellipsoidal objects (e.g. pebbles)

b) other objects of known initial shape (e.g. fossils)

c) preferred mineral orientation.

The establishment of continuity of the ore horizon between sample points is a fundamental aspect of ore evaluation. This continuity can only be established once the structural history of the environment is unravelled. This aspect is stressed, and will become clear, in the second part of this review.

Finally, we must consider what happens to a vein when it is folded by two discrete events along new axial planes, rather than progressive deformation along the same axes. Ramsay (1967) recognises three basic patterns resulting from two successive foldings, the nature of which depend on the orientation of the axial surface of the first fold relative to the flow direction of the superimposed fold. The angle between the pole to the axial plane of the first fold and the direction of flow of the second fold is termed angle β. The angle between the two fold axes is α, (Figure 25).

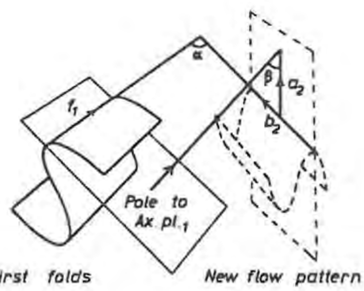


FIGURE 25 Relationship between the two wave forms in a superimposed fold system. As the angles α and β vary, so different types of three-dimensional interference patterns result.

(After Ramsay, 1967)

The first type of interference pattern results when $\alpha >$ zero and $\beta > 70^{\circ}$. This produces eggbox -type, dome and basin structures. Perfect dome and basin structure is achieved when $\alpha = 90^{\circ} = \beta$. From the description by Barker (1977), it appears that the How mine which is situated in a "complicated dome structure, between the axes of two steeply plunging synclines" may occur in an area that has suffered interference folding of this type.

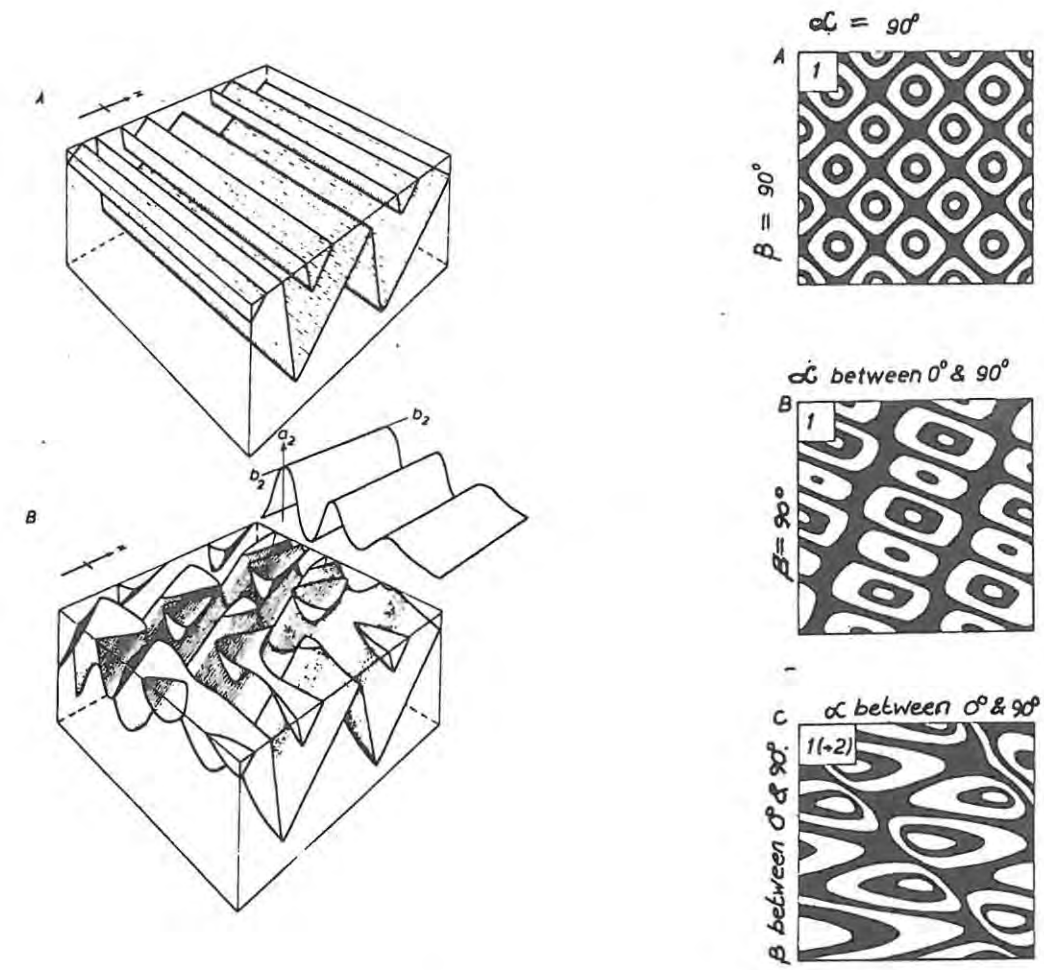


FIGURE 26 Type 1 Interference patterns produced by two successive foldings.

Interference patterns produced by two successive foldings. A illustrates the original form of the first fold, B shows the interference pattern which results from a superimposed fold at approximately 90°.

(After Ramsay, 1967)

A is the surface outcrop pattern that results when $\alpha = 90^\circ = \beta$. B is the pattern when α is between 0° and 90° and $\beta = 90^\circ$ and C results when both α and β are between 0° and 90°.

The second type of interference pattern results when the flow direction of the superimposed movement is at a high angle to the axial surface of the first folds. Here the angle $\alpha > 20^\circ$ and $\beta < 70^\circ$. The fold hinges of the first fold are bowed by the heterogeneous flow and both limbs of any one first fold became refolded upward into a common antiform or downward into a common synform. If this complex form is progressively exposed by erosion, systematically changing outcrop patterns result (Figure 27).

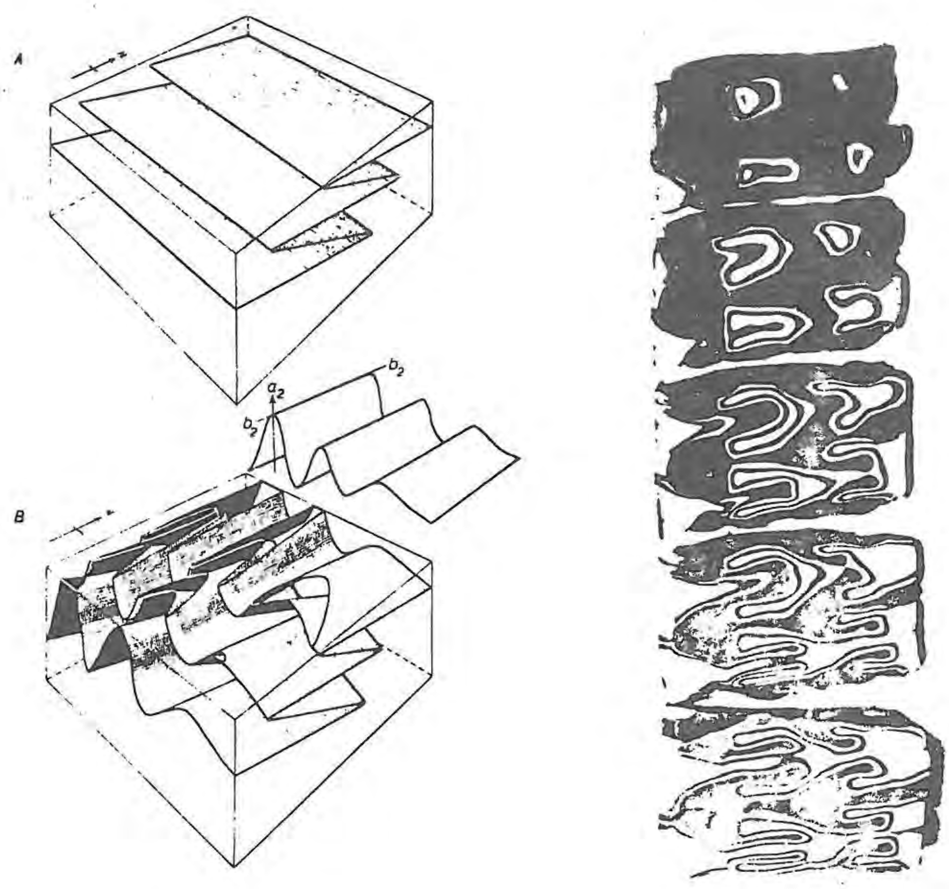


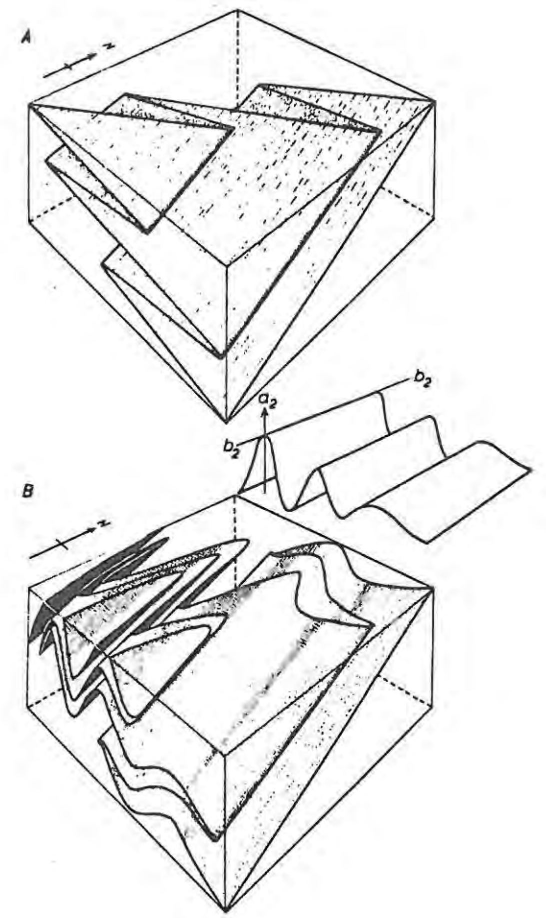
FIGURE 27 Type 2 interference patterns produced by two successive foldings.

A illustrates the original form of the first fold and B the pattern produced by superimposed folding.

Characteristic mushroom outcrop pattern exposed by systematic erosion.

(After Ramsay, 1967)

The third type of interference pattern is commonly developed where recumbent folds are refolded by new structures with steeply inclined axial surfaces. The flow direction of the second structures are at a low angle to the axial surface of the first fold (i.e. α approximately zero, $\beta > 70^{\circ}$). The resulting pattern does not show closed outcrop shapes because the periodic undulations of the first fold hinges are not well developed (Figure 28).



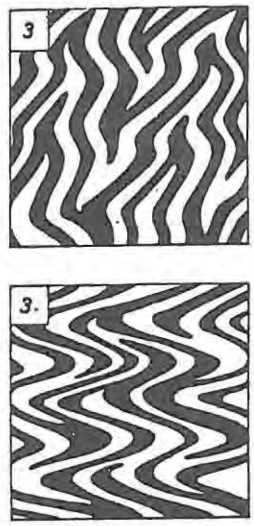


FIGURE 28 Type 3 interference pattern produced by two successive foldings.

A illustrates the original fold. B is the interference pattern that results after superimposition of the second fold.

Typical outcrop pattern of type 3 folds.

Any one of these patterns may develop if a vein deposit is subjected to more than one episode of plastic deformation. Which of the three interference folds had formed, will be revealed by a careful study of the outcrop pattern. All interference folding may be considered as a constructive factor in evaluation because they all lead to shortening of the ore horizon and therefore to a "gain of ground". It is important in evaluation that these structures be identified and not mistaken for irregularities with which ore shoots may be associated. Fortunately, veins are typical of the last phases of deformation when brittle failure is more common than plastic deformation. As a result these complex fold interference patterns are not often encountered in evaluation of vein deposits.

Post Mineralization Faulting

Post mineralization faulting of veins developed in competent horizons is common. It is usually a disruptive feature, the degree of disruption being dependent upon the displacement by the fault. However, in the case of reverse faults there is a net gain of ground and this could be considered a constructive factor (Figure 29).

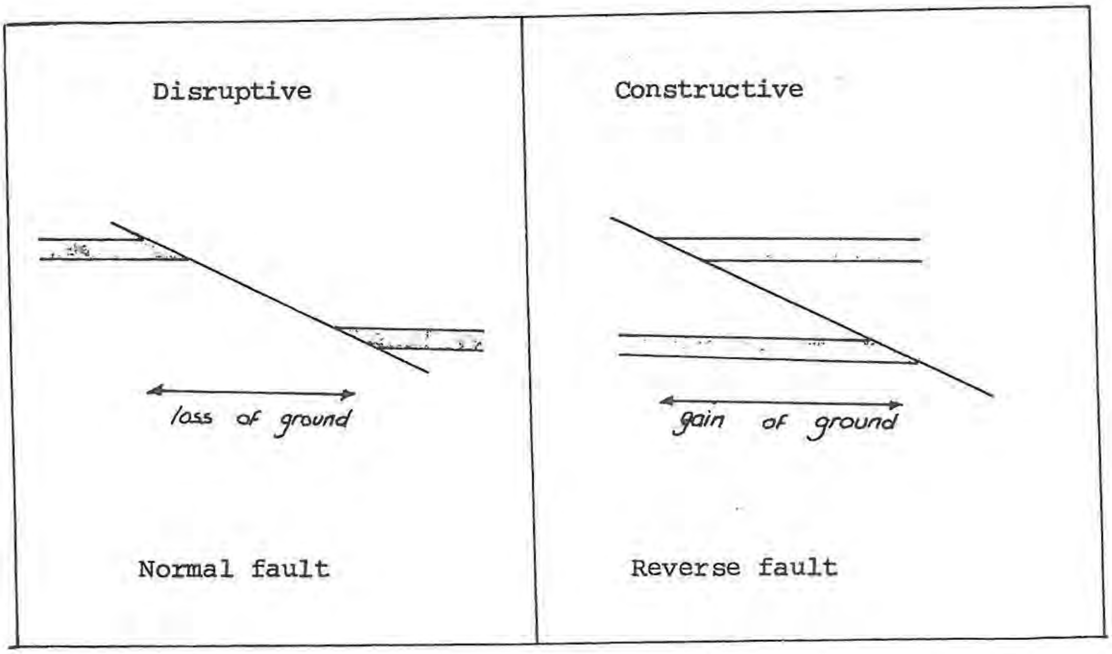


FIGURE 29 Post mineralization faulting.

CONTROLS ON THE POSITION OF ORE SHOOTS AND THE DISTRIBUTION OF MINERALIZATION WITHIN A VEIN

In veins only certain parts can be profitably extracted. These parts are called oreshoots, the outlines of which may be exceedingly variable (Lindgren, 1928). H.G. Hoover, (as quoted in

Lindgren) after examining 70 mines,concluded that oreshoots are generally lenticular and that the probable minimum extension of an oreshoot below any level would be not less than half its strike at that level. The position of oreshoots is determined by a wide variety of geological influences, the most important of which are structure and the chemistry of the wall rock.

The Structural Control On the Position of Oreshoots

Structure may influence the position of oreshoots by determining the width of openings and the surface area presented to solutions for reaction. This repeated emphasis on openings does not imply that all veins were formed by simple filling; indeed many veins were formed by replacement of material in and adjoining the vein-fracture. However, wide openings provide access for a plentiful supply of ore-bearing solutions and as such are potent localizing influences on both open space filling and replacement ores.

New faults often occur along old sutures even when the latter are filled with vein material. The vein may be wholly or largely crushed during renewed movement or it may be broken by a new longitudinal fault or fault zones. In the first case sectors of brecciated ore are formed and in the second case the vein may increase both in thickness and in area through the deposition of additional ore. This is because the new disturbance becomes the conduit for circulation of hydrothermal solutions leading to further ore deposition (Smirnov, 1976). In detail, the distribution of values depends on the availability of microscopic spaces along the fracture plane, at the time of deposition. This normally results in an erratic distribution of values. If successive fluids vary in composition a variation in metal content, metal type and in the colour of the quartz host can result. Therefore the banded or ribboned appearance of a quartz vein is a good indicator of recurring tectonism and filling.

That repeated disturbances play an important role in the final distribution of the valuable metals is illustrated in an example from Mawdsley (1938), on the distribution of gold, at the O'Brien gold mines property, near Quebec. Here gold is concentrated along the centre of the vein and along numerous parallel fractures that lie almost at right angles to the strike of the vein. These transverse films or veinlets of gold must have been deposited in a regular set of fractures whose origin Mawdsley (1938) links with stresses that affected the vein after its consolidation. During later stress the gold may move out from the vein into the country rock, for example at the Siscoe gold mine 40 kilometres east of the O'Brien property, where gold has coated the slickensided face of a mass of talc some distance from the vein. Whenever this occurs an unusually high assay value can be expected (Mawdsley, 1938). The erratic distribution of gold in vein deposits is explained by assuming that the gold is partly remobilized by successive disturbances into new spaces created within and beyond the vein. The quartz vein may be most susceptible to such stress where the contrast in competency between the quartz vein and the host rock is high. Well fractured, laminated vein quartz containing druses and open space filling textures is therefore a pronounced predilection when searching for oreshoots.

Chemical Control On the Position of Oreshoots Within a Vein

The interaction of the mineralizing fluids with the wall rock is important in the localization of some vein ores. Examples include the scheelite deposits of Eastern Siberia (Khrenov, 1959). Here the ore veins traverse alternations of marble and shale and their mineralization varies with the country rock. Those parts of the vein that cut shales, consist mainly of quartz with sparse calcite, pyrite, pyrrhotite, chalcopyrite and galena. Those parts of the vein intersecting marble carry quartz-calcite gangue with a high proportion of scheelite ($CaWO_4$) and minor sericite, garnet, fluorite, galena, chalcopyrite, sphalerite, tennantite and pyrite. That is, in this deposit only those parts of the vein that occur in the carbonate country rock contain commercial tungsten ore.

In addition, ores can be enriched with metals redeposited from the zone of oxidation. Oreshoots of this category occur as sectors of highly concentrated ore that are not connected with changes in the thickness of the vein. The chemistry of the metal and gangue material control the enrichment pattern in the weathering environment. This aspect is covered in more detail in a later section.

It is the general consensus that oreshoot development is not a chance phenomenon but the result of natural controls. These controls can be predicted by detailed geological investigation. One useful technique is to construct a dilation map, by passing a reference plane below and roughly parallel to the footwall of the vein. Distances from this plane to the footwall are plotted and contours are drawn through points of equal distance. Another similar map is made, plotting the distance from the same plane to the hangingwall. The advantage of these maps over the normal isopach map is the depiction of deflections, which may not show as differences in thickness. Also, these maps can be used to ascertain the direction and amount of movement along the fault. This is done by moving the one contour map over the other until the contours agree. The direction and amount of movement required to achieve this is the direction and movement of the fault. The use of these maps allows projection of the widest and more favourable parts of the vein. This map can only be drawn if sufficient carefully surveyed cross sections through the orebody exist to give meaningful contour points. More details on the use of dilation maps are given in Conolly (1936), Roscoe (1951) and Park and MacDiarmid (1975).

ZONING

Once mineralization has been discovered the consideration of zoning patterns is important. The scale of zoning can be extremely variable; from changes as delicate as parts-per-million to coarse changes in gross mineralogy. In terms of distance, zoning can occur on a regional basis of tens of kilometres or on a ore body basis with dimensions of a few metres to hundreds of metres or to actual mineralogical zoning over millimetres. This section is concerned with zoning as it affects evaluation on the scale of an ore body.

The full zonal sequence is given in the following table after Emmons, as cited in Lindgren (1928). No single vein would exhibit the full succession and in some veins the minerals of one zone may overlap those of another. The cause of the paragenetic sequence is uncertain and a review of the conflicting arguments is beyond the present scope. What is important is that this well defined succession allows some prediction of the change in economic numerals that can be expected at different levels in a vein deposit. The deposits of Cornwall, England furnish a classic example of the zonal succession, with tin at depth, copper in the distal areas and lead and zinc still further away (Lindgren, 1928).

It should be stressed that zoning can indicate the true bottoming of the ore; whereas, a decline in productivity as a result of structural factors may not be the base of the economic horizon. Zoning can be important in determining if a fault is pre mineralization or post mineralization. If the zoning is disrupted then it is likely that the fault is post mineralization and the possibility of further ore exists; if the zoning is not disrupted then the fault is pre mineralization and this possibility cannot be considered. Practical application of the zonal theory is not always possible since mineralogical zoning may be so gradational as to be practically imperceptible or may be too rapid and erratic to be predictable, even after statistical analysis of the data.

Zoning was defined by McKinstry (1948), as the mining geol-

A RECONSTRUCTED VEIN SYSTEM FROM SURFACE TO NEAR BATHOLITH ROOF

(After W. H. Emmons)

Surface

Barren, 1.—Barren zone, chalcedony, quartz, barite, fluorite, etc. Some veins carry a little mercury, antimony, or arsenic.

Mercury, 2.—Quicksilver veins, commonly with chalcedony, marcasite, etc. Barite-fluorite veins.

Antimony, 3.—Antimony ores—stibnite often passing downward into lead, with antimonides. Many carry gold.

Gold and Silver, 4.—Bonanza ores of precious metals. Argentite, antimony and arsenic minerals common. Silver minerals, some copper, lead and zinc sulfides, quartz, calcite, rhodochrosite, adularia, alunite, etc.

Barren, 5.—Most nearly consistent barren zone, represents the bottoms of many Tertiary precious metals veins. Quartz, carbonates, etc. with pyrite and small amounts of other sulfides.

Silver, 6.—Argentite veins, complex antimony silver sulfides, stibnite, etc. Galena veins with silver. Commonly silver decreases with depth. Quartz gangue, siderite common, often increasing with depth.

Lead, 7.—Galena veins, commonly with some silver. Sphalerite generally present, increasing with depth. Chalcopyrite common. Gangue is quartz and often carbonates (Fe, Mn, Ca).

Zinc, 8.—Sphalerite veins with some lead and chalcopyrite, quartz gangue.

Copper, 9.—Tetrahedrite veins, commonly argentiferous, chalcopyrite present. Some pass downward into chalcopyrite. Enargite veins generally with tetrahedrite and tennantite.

Copper, 10.—Chalcopyrite veins, generally with pyrite, often with pyrrhotite. The gangue is quartz and in some places carbonates. Some pass downward into pyrite and pyrrhotite with a little chalcopyrite. Generally carry silver or gold.

Gold, 11.—Gold veins with quartz, pyrite, and commonly arsenopyrite and chalcopyrite. At places zones 10 and 11 are reversed.

Bismuth, 12.—Bismuthinite and native bismuth with quartz and pyrite, etc.

Arsenic, 13.—Arsenopyrite with chalcopyrite and often tungsten ores.

Tungsten, 14.—Tungsten veins with quartz, pyrite, chalcopyrite, pyrrhotite, etc. Arsenopyrite is commonly present.

Tin, 15.—Cassiterite veins with quartz, tourmaline, topaz, etc.

Barren, 16.—Quartz with small amounts of other minerals.

In many veins only one of the upper portions (2, 3, or 4) can be observed. Few of these veins have been developed beyond the barren zone, 5. Nearly all of the zones below 5 have been observed at many places grading one into another.[1]

TABLE 1 (Taken from Lindgren, 1928)

ogists "ringed target". This is because if the pattern can be recognized and interpreted it can guide to the core of the target. Peters (1978) cites an example where mineralogical and chemical zoning have been of use in defining the best areas of a tin-tungsten deposit at Altenberg in the German Democratic Republic. The ore body at Altenberg consists of a stockwork of cassiterite-wolframite-molybdenite veinlets in greisen. In exploration, tin, bismuth and molybdenum were good pathfinders whilst gallium indicated the roots of the ore body. In the ore zone the Mn/Fe ratio of wolframite was used to determine the optimum area for tungsten development. The ratio ($MnWO_4/FeWO_4$) was plotted along the course of a small tungsten-bearing vein to find the highest value. Once this had been located it was followed up by drilling which revealed an intrusive cupola, with tin ore, at a depth of 300 metres (Peters, 1978). A good knowledge of zoning is therefore fundamental to evaluation, when searching for high grade zones, and when predicting the possibility of more ore at depth. This possibility could guide the exploration programme and should be considered when evaluating the overall potential of the ore body. It should be borne in mind that during deformation certain minerals may be mobilized and this could disrupt the zonal pattern. Also, in plutonogenic deposits it is unlikely that the mineralizing event was unique and each renewed event may bring with it a full spectrum of minerals. This zoning may be imperfectly imprinted on the earlier mineralizations to blur the overall succession. Therefore, although zoning is potentially useful it is not often that the theory can be applied to advantage in evaluation.

THE BEHAVIOUR OF SOME VEIN-ASSOCIATED MINERALS IN THE ZONE OF WEATHERING AND THE EFFECT OF THIS ON EVALUATION

Evaluation begins at the surface, in an arena of soil and weathered rock where minerals have adjusted, at least in part, to the new environment. The degree of adjustment depends on the climate and the chemistry of the ore and gangue minerals. The effect of weathering on evaluation is that it leads to a decrease in S.G., a change in mineralogy and a change in total metal content. These changes are best described by reference to specific metals.

Physical and chemical weathering operate in a direction of equilibrium. Thus, lower pressures result in an increase in volume. This is accomplished by disintegration and the formation of low density minerals. Lower temperature is accommodated by exothermic reactions. For example bornite (S.G. = 5.06) is replaced by malachite (S.G. = 4.0) with a release of energy. Scheelite and wolframite are fairly insoluble in the pH range of most surface waters. However, acid surface waters resulting from the decomposition of sulphides can dissolve these tungsten minerals, the former being attacked more readily (Campbell, 1920). It is suggested by Haag (1943) that wolframite disintegrates along cleavages and is moved away as tungstic acid, with manganese dioxide and limonite. This may result in cavities in the place of leached scheelite in the oxidized portion of quartz vein deposits or this process may be indicated by the presence of pseudomorphs of quartz and chalcedony (Hosking, 1970). The final result is a decrease in S.G. and metal content no matter which process is followed. Khristoforov (1955) notes that in an acid environment wolframite becomes progressively replaced by a mixture of ferritungstite and $Fe(OH)_3$. Further evidence that under certain conditions tungsten can be dissolved is provided by the presence of up to 70 ppm of the element in the brines of Searles lake. Hoskings (1970) suggests that the degree of attack depends on the texture of the deposit, or the spatial relationship between the wolframite, sulphide and quartz. It will also depend on the extent to which open fractures and joints occur in the deposit, as these provide passageways for the attacking agents.

In the secondary environment tungsten may occur as tungstite ($WO_3.H_2O$), hydrotungstite ($WO_2.2H_2O$), ferritungstite ($Ca_2Fe_2^{2+}Fe_2^{3+}(WO_4)_7.9H_2O$), stolzite ($PbWO_4$) or cuprotungstite ($Cu_2WO_4(OH)$), all of which are commonly inconspicuous and difficult to identify. According to Varlamoff (1971) these secondary tungsten minerals may represent as much as 50-70 percent of all the tungsten bearing minerals in the zone of weathering. This may require a modification of the beneficiation process when dealing with oxidized ore. Certainly oxidized ore should be evaluated separately from fresh ore material.

Gold is often considered inert to natural chemical processes. However, as early as 1931, Freise noted that alluvial gold deposits that had been thoroughly exhausted could after a period of years once more be panned and yield a profitable amount of newly accumulated gold. Also, that the "paying spark" may reappear sooner if the exhausted goldfield occurred in a thickly vegetated area. Lakin _et al_., (1971) reports acidic oxidation of pyritic gold deposits may result in transient mobilization of gold as $AuCl_4$, alkaline oxidation of pyritic gold deposits may result in transient mobilization of gold as $Au(S_2O_3)^{3-}$ and in soils, under native vegetation, gold may be mobilized as $Au(CN_2)^-$. Whatever the circumstances, Lakin (_op_. _cit_.), states that the gold solutions would be transitory. Krauskopf (1951) claims that gold is soluble in dilute alkali sulphide solutions. Therefore, there is evidence that gold can be mobilized in the weathering environment. The resulting change in metal distribution is the most important aspect to consider during an evaluation programme. Once gold has been dissolved, a change in conditions, such as pH, can lead to precipitation of the gold. The pH is liable to change in the vicinity of the watertable and in the humic layers of soils. With a fluctuating watertable the gold could be continuously redissolved, retransported and reprecipitated. These effects are unlikely to be uniform. Consequently, under certain conditions, the first few hundred metres of a gold bearing vein may have been successively leached and enriched leading to the formation of what Mehliss (1964) terms "perched bonanzas", separated by low grade or barren zones. Such conditions of highly erratic metal distribution in gold deposits can extend to vertical depths of 700 metres (Lonely Mine; Mehliss, 1964) but more commonly only extend to a depth of 200 metres.

The upward movement of gold tends to spread the values along the strike of the vein, giving the appearance that the oxidized outcrop is of a long primary shoot, when in fact, deeper explorations may prove only a few narrow primary ore bodies below the continuous surface ore. Finally, oxidized gold is usually free milling whereas the fresh material may occur in sulphides and require roasting. Again, separate evaluation of the oxidized and fresh zones of the ore body, is essential for efficient extraction.

SAMPLING OF VEIN DEPOSITS

Ore valuation is defined by Munro (1966) as the determination of the valuable content of well defined areas or volumes of an ore body. This may be considered under two main headings.

1) The establishment of an ore body.

2) The establishment of tonnage and grade relationships for part or for the whole of the ore body.

Regardless of the stage of ore valuation, sampling is an important operation. A sample is defined by Parks (1957) as a "small portion of an article such that the consistency of the portion shall be representative of the whole". The main problem with ore sampling is the degree of confidence with which the valuable content of a body of ore can be inferred from the valuable content of samples. In this respect mathematical statistics and particularly the theory of probability are useful. This concept will be discussed in subsequent pages. Sampling can be conveniently divided into two aspects.

a) The Sampling Strategy

The choice of sample positions and the decision as to the type and number of samples to be taken.

b) The Sampling

The physical act of cutting or drawing the sample and the assay procedures which follow.

1) Sampling Strategy at The Early Stage of Vein Deposit Evaluation

The sampling programme at this stage is usually determined by the cost effectiveness of obtaining the necessary information. Sampling would commonly include surface soils, outcrop, trenches, pits, diamond drill core, percussion-drill chips, rotary-drill chips or drill sludge or some physical parameter such as fluoresence, magnetics, electrical properties or radiometrics.

A feature of vein deposits is the variability of their values. This is the result of an originally erratic distribution which has been modified by surface leaching and structural disturbances. These effects have been outlined previously. The erratic nature of the distribution introduces a large sampling bias. This bias can be partly eliminated by increasing the sample size. It is for this reason that Cooke (1976) recommends bulk sampling for vein deposits (Figure 34) Also when the sample is large there is a sufficiently high chance of each unit in the population being selected. This means that a large sample can be considered as approximately random (Krige, 1978). Statistically a random sample most closely approximates the characteristics of the parent population. Hence it is desirable that a random sample be obtained in the early stage of exploration from each similar unit within a stratum. However in practice the actual sample interval is seldom random and usually boreholes, pits, trenches etc. are positioned on a regular grid over the geologically determined area of interest. This means that material between sample points has no chance of being selected. A further bias is introduced as the sample interval decreases by virtue that adjacent samples fall within the sphere of influence of their neighbours. That is, these samples are not independent of one another which is a pre-requisite for randomness. Bias may also be introduced by lease boundaries, surface cover, topography etc. limiting the area in which samples may be taken.

The Sampling Procedure in the Early Stage of Evaluation

1) Channel sampling pits and trenches

Pitting or trenching at regular intervals perpendicular to the strike of the vein is the most satisfactory sampling technique at this stage. The usual practice is to dig a pit and then take channel samples of each stratigraphic interval along a sidewall. Channel sampling consists of cutting a groove across the face of exposed ore and collecting the chips. McKinstry (1948) recommends that before sampling the exposure be cleaned to remove contaminants. The next step is to mark out the location of the channel by inscribing two parallel lines on the rock approximately 10 centimetres apart. The sample is then removed as a uniform channel from between these two lines using a hammer and moil or a hand-held diamond saw. A block of wood is useful to gauge the uniformity of the cut. When the block fits into the groove flush with the adjoining rock surface the width and depth of the channel are accurate. Such ideal width and regularity may not always be attainable and the best remedy in this case is to cut a channel so wide and deep that irregularities are small in proportion. A more rapid procedure and therefore a common sampling practice is to take a series of chips from a chip channel (a somewhat broader band of about 0.5 metres) with a field pick. These together form a chip sample. Each chip should be approximately the same size. Smaller chips give a wider cover, but as a rule of thumb, the pieces taken should be ten times the size of the maximum grain size of the element being sampled.

Usually the channel is sampled normal to the strike and dip so that a true width is obtained. However, this is not always possible and sampling at some other angle may be more convenient and is permissible if the angle is maintained. In gently dipping horizons a series of offset, perpendicular samples may be taken (Figure 30). If the ore

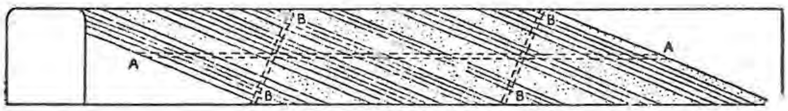


FIGURE 30 Channels for sampling gently dipping formations in a crosscut. (Vertical section) Either the inclined channels, B-B, or the horizontal channel, A-A, would be proper. B-B requires less cutting; A-A may be more convenient.

(After McKinstry, 1948)

horizon has a width in excess of 1.5 metres, or it is banded, then separate samples, of 2-3 kilograms each, should be taken from each band or each 1.5 metre length. McKinstry (1948) lists the following advantages arising from cutting a subdivided channel sample.

I Accuracy of Sampling

1) If a vein consists of bands contrasting in grade, separate sampling of each band evades the possible source of inaccuracy arising from taking too much or too little material from a rich band.

2) If a body consists of hard and soft bands, separate sampling avoids taking disproportionate amounts of the hard and soft material.
3) If for practical reasons the channel does not cut the lode at a uniform angle, subdividing the sample at changes in angle avoids the necessity of varying the weight to correspond to the width represented.

II Accuracy of Information

1) Dividing the sample to accord with bands of differing character provides valuable information on the association of metals with different types of vein matter.
2) Explicit information regarding the distribution of values across the strike may influence methods of mir .ng. For example, in a wide ore body this sampling method will show which parts of the body to mine, while in a narrow ore body the sub-division of samples will indicate the dilution effect of incorporating hanging or footwall material.

Sampling accuracy may be increased by sampling both side walls separately and either combining the material into one sample or averaging the two assay values. The size of the sample will depend on the analytical technique and the accuracy required. Usually a large sample is pulverized and split to the required amount. Aplan (1973) as quoted in Peters (1978) recommends the following degree of pulverization to ensure homogeneity of the sample.

TABLE 2

Weight of Sample kg	Diameter of largest piece cms
1000	7.6
500	5.0
100	2.3
50	1.5
10	0.7
5	0.5
1	0.2

2) Sampling Dumps and Stockpiles of Abandoned Deposits

In several instances the vein to be evaluated may be an abandoned working. In such a case it may be useful to sample old dumps and stockpiles. Here it is difficult to obtain a truly representative sample owing to variations in moisture, temperature and size distributions in the pile. This is because in any dump the angle of repose varies from the top to the base. The net result is that the fine material accumulates towards the top of the pile whilst the larger lumps roll to the base. This means that a representative sample can only be obtained by face sampling pits, sunk at regular intervals through the pile. Temperature and moisture distributions are important because this can lead to local leaching of the valuable constituent, especially in the deposits where the ore is pyritic.

3) The Use of Drilling as a Sampling Technique in the Evaluation of Veins

Drill core and drill cuttings can furnish useful samples at the early stage of evaluation. Since one objective is to obtain sufficient material for accurate ore-grade calculations, larger core sizes are especially preferred for the normally erratically mineralized vein deposits. However the values are usually so discontinuous that little significance can be placed on samples taken from even the largest drill-core sizes. One problem with diamond drilling is the tendency to overvalue high grade blocks and undervalue low grade blocks. This is because the probability of sampling an anomalously high patch of mineralization in a low grade area is significantly less than the probability of sampling an unusually high grade patch in a more highly mineralized area. This concept is dealt with in detail in the section on ore reserve estimation in vein deposits, and arises as a direct result of the erratic distribution of values in these deposits. Over valuation of high grade blocks and under valuation of low grade blocks can also be caused by differential core loss. In diamond drilling the core tends to break along fractures, which in vein deposits is often also the locality of most of the mineralization. Usually the mineralization is softer than the silicate host and as a result some of the valuable material is lost. If the overall grade of the intersected area is low the effect of this differential loss is proportionately much higher than in high grade areas. It is, therefore, good practice in vein deposits to collect and sample drill sludge to check for this.

The major objective when drilling these deposits is to determine the continuity, structure and nature of the mineralized horizon at depth. Of particular importance is continuity. In this respect the total core should be logged with attention being concentrated on any marker horizons that may allow correlation of material from different intersections and with the surface geology. It is important to establish that the vein intersected in the drill hole is the same as that being sampled on surface. It is from drilling that the structural features predicted by surface mapping are established and tested. To this end it is essential to know the exact position of the ore intersection. For example, in Figure 31 careful surveying and the recording of marker horizons, would indicate that borehole number 2 was stopped prematurely and that the low tenor of the intersected vein was not the result of a change in strike but the result of a low angle fault disrupting the mineralized horizon. In this case the plane of the fault is occupied by a barren quartz-vein. In Figure 32a

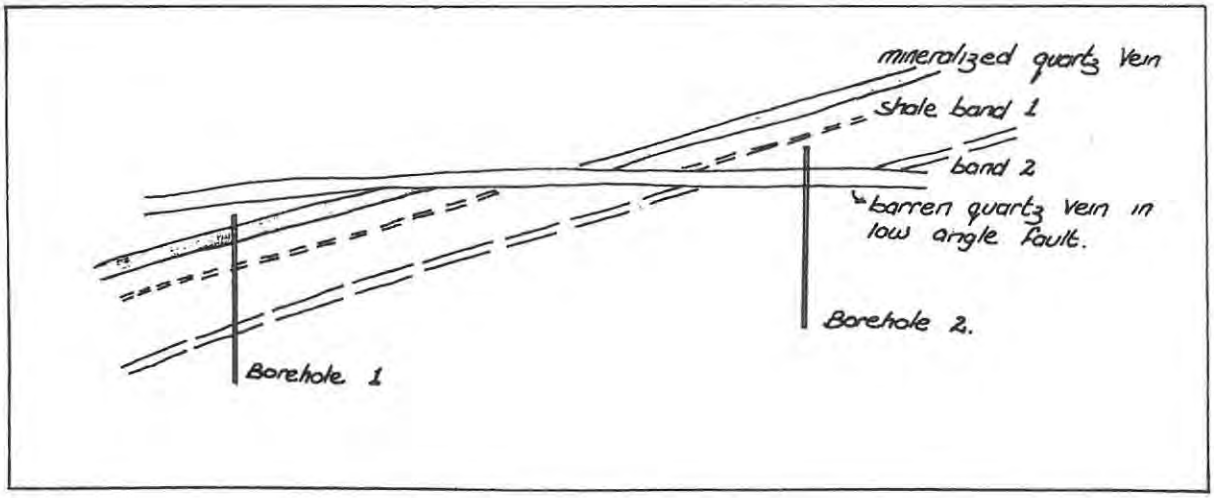


FIGURE 31 The disruption of the ore horizon by a low angle fault.

and b the boreholes have been stopped too soon to reveal the duplication of the ore horizon.

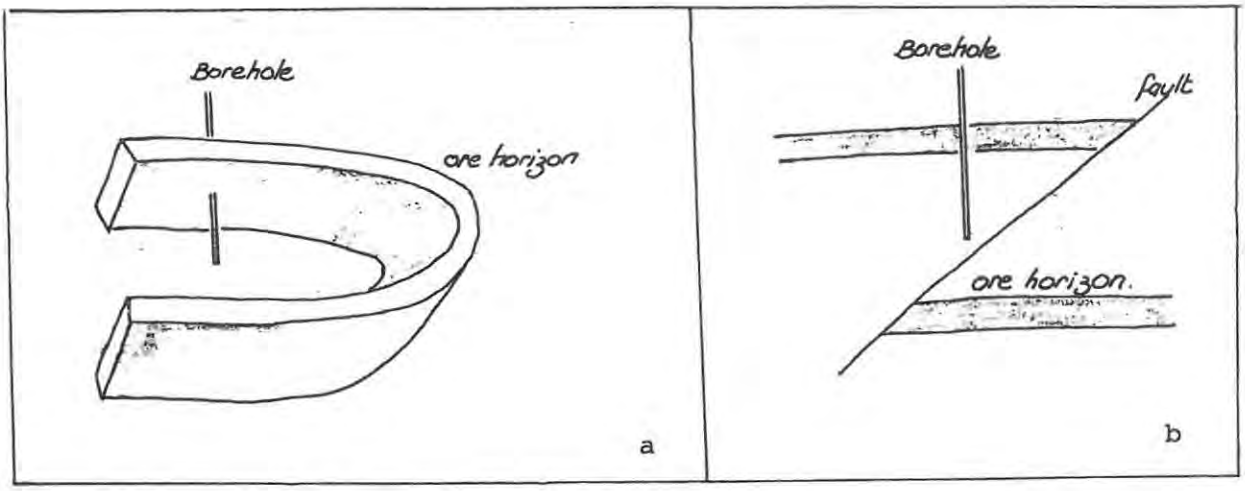


FIGURE 32 The duplication of the ore horizon: a as a result of folding and b due to faulting.

An important aspect of diamond drilling is that it provides the opportunity to study fresh ore and gangue material. This is useful in order to establish the type of mineralization, the mineralogy and the relationship between the mineralization and the host rock. The association or lithology of the ore should be established because mining to a rock type can often be more efficiently achieved than mining to an assay value. It is important to define the nature and contacts of the hangingwall and footwall for rock mechanic and metallurgical purposes.

The location of drill sites is determined by the shape of the ore body; it is usual to drill normal and parallel to the ore strike. This allows easier sectional calculations. The drill holes should be inclined so as to intersect the mineralized horizon as close to perpendicular as is possible. This general rule may overevaluate deposits where the mineralization occupies more than one fracture orientation, for example, in mineralized conjugate fracture pairs. Here the samples taken from the drill holes may give a good report of the mineralization in the one fracture system but it may be that a portion of this mineralization is contained in other fractures which are orientated very unfavourably in relation to the boreholes. According to Dixon (1979), this is a prevalent problem in such deposits as the stockwork cassiterite-wolframite deposits associated with granites. A careful assessment of even the most subtle geologic controls and the subsequent avoidance of a drill pattern that may coincide with a periodicity in folding or faulting may save the premature abandonment of a project or the costly overdrilling of a poor target area.

In the initial boreholes drilling should be conducted beyond the calculated depth to check for any duplication of the ore horizon. However, once it is established that there is no possibility of ore beyond a certain depth, then the calculated base for the drilling should be adhered to. When calculating drilling depth the structural style should be considered because lateral features observed on surface can be expected to be repeated in the vertical plane. Likewise variability in metal content along the surface can be expected to be equally variable in the third dimension. (However, actual values and ratios of values will differ significantly between the surface samples and fresh core samples because of leaching and enrichment in the near surface environment).

Finally, it must be stressed once more that drilling in vein deposits is a structural rather than a sampling tool, although metal content and the nature of the mineralization should be noted. For this reason diamond drilling is preferred because it gives more geological information than do any of the non-coring techniques. The latter, however, are cheaper to operate and may be used for infill drilling to prove continuity in deformed areas. In percussion drilling, a great deal of significance should not be placed on the valuable metal content of the drill sludge because this technique has all the disadvantages of diamond drilling and in addition, the high S.G. of most vein-associated minerals (Au, W, Sn), means that the valuable minerals concentrate at the base of the hole and are not lifted by pneumatic processes. Although drilling is not particularly useful as an evaluation technique, sufficient boreholes should be sunk to find the area of greatest economic interest on which to plan sufficient development to afford systematic sampling.

Because the sample assay is an unreliable indicator of the overall economic potential of a vein, it may be useful to use the other, more abundant and more uniformly distributed, geological variables as predictors of areas of optimum mineralization. For example, the presence of base metal suphides in a gold deposit could indicate that late, open space filling (with which the gold is associated) had occurred. A further example would be the degree of greisenization in a granite-hosted tin and tungsten deposit. In cases where the deposit is zoned, the use of a negative correlation may be advantageous. For example, tourmaline in a greisen-associated tin deposit would indicate that the cap area had been intersected, whilst in several deposits the presence of pyrite in drill core would indicate the periphery of the deposit.

Evaluation at the initial stage is designed to establish a model of

1) the development of the deposit,
2) the geometry or shape of the deposit,
3) the distribution pattern of the mineralization (that is, are there trends to the mineralization or is it truly erratic?),
4) local factors influencing the metal distribution (that is, what influences the position of pay shoots or the position of high values?),
5) the parameters of these pay shoots and
6) the statistical techniques which may be applicable to the geological parameters to give meaningful patterns (modified from Pretorius, 1966).

From the previous discussion it is obvious that the geological nature of vein deposits severely limits the degree to which any of these models are satisfied at this stage in exploration.

Composite Samples

It is often useful to compile composite samples soon after evaluation commences. These samples normally comprise fresh material collected from pitting or drilling. The composite is made up by weighing out proportionate amounts of various samples according to the tonnage that each sample represents. The composite sample is used for metallurgical testwork and to determine if the product can meet customer specifications. Many lode deposits are polymetallic and composite samples drawn at an early stage should be used to establish a full mineralogical inventory. This will reveal if there are any by-products or co-products(e.g. Ag in Cu) which may make a significant contribution to the success of the operation. Also a check should be made on the presence of any impurities (such as Bi in Pb; Pb in Sb or As in W) which may be detrimental to the product. Because vein deposits are usually zoned, several composite samples should be made up, each from a different part of the body. This should ascertain the significance of the variations in the chemical attributes from different parts of the horizon and how this will effect the metallurgy.

SAMPLING FOR THE DETERMINATION OF ORE RESERVES IN VEIN DEPOSITS

It is becoming increasingly accepted that trial mining is the most satisfactory method for estimating grade and ore reserves, in erratically mineralized deposits. The results, although expensive to obtain, are more reliable than those obtained by other techniques. Some of this expenditure can be recovered, should the property go into production, if the workings are laid out in such a way that they can be used later as development workings, service ways etc. However, should the property prove to be uneconomic the loss from trial mining is invariably less than that which would result if a full scale mining operation was undertaken.

Essentially, trial mining exposes the ore horizon for bulk sampling and close-spaced channel sampling. The benefits of these sample techniques have been discussed.

Dixon (1979) recommends the following procedure for the layout of trial workings.

1) The trial workings should cover all types of ore as distinguished by exploration drilling. This means that in a zoned lode deposit the workings may have to be quite extensive.

2) The workings should be orientated in the direction in which the mining faces are likely to advance. This is important in vein deposits where the bodies are usually anisotropic - see later.

3) The bulk sample should be of a comparable size to the minimum grade-control block size. (This is because selective mining on a scale smaller than this is impossible).

4) It is useful to diamond drill ahead of the working so that a correlation study of drill sample to bulk sample is possible. In some veins it may be possible to establish a factor, which if applied to exploration drilling results, would make these drilling results more accurate.

If bulk sampling is conducted in this manner then some estimate of grade and tonnage is possible and confidence limits can be applied to these estimates. However, in any trial mining exercise it is unlikely that sufficient development will be undertaken to expose a sufficient amount of the ore horizon to establish more than a limited amount of the ore reserves with any high degree of confidence.

SAMPLING TECHNIQUES USED TO DETERMINE ORE RESERVES

Several sampling techniques may be employed in ore reserve estimation. Of these, channel sampling is the most frequently used. At this stage the channel is cut at regular intervals along all drifts and raises so as to surround all exposed ore blocks with periphery samples. Again the samples are subdivided lithologically and are taken across strike. The latter is especially important in telethermal deposits where the mineralization is often concentrated along bedding or other structural planes. Some common geological factors which affect the accuracy of a channel sample and which may be encountered in a vein deposit, include variations in the hardness of the sampled horizon and local duplication of the economic horizon by faulting and intraformational folding.

Diamond drilling may be employed in trial mining to sample ahead of development and to sample the interior of ore blocks. At this stage some degree of confidence can usually be placed on diamond drill results, based on the correlation between previous drilling and the material encountered in mining. Normally the cores are sampled lithologically in the same manner as channel samples.

Blast hole sampling is commonly practised to determine grade in a variety of ore occurrences. Two South African hydrothermal deposits where this technique is employed are the Zaaiplaats and Union Tin mines in the Northern Transvaal. Usually the results obtained are unreliable because of the limited area of influence that can be confidently allocated to the sample value and because the sample may be concentrated or diluted according to the angle of the blast hole. In high angle holes the cassiterite is concentrated in the sample owing to the loss of light material as dust, whereas in down holes the cassiterite remains at the base of the hole and the sample is dilute. The net result is that this technique is a useful indicator of the presence of ore but an unreliable indicator of the value. A correction factor, based on the inclination of the hole, is difficult to apply because the collection of material is sensitive to operator skill.

Another sampling technique that is commonly used is grab sampling. This may be of material from cars, drawpoints or blast heaps. In veins, because the mineralization is usually erratically distributed and concentrated along fractures, the valuable minerals often occur in the fine material. This means that a grab sample from any pile may consist of a barren lump or a concentrated dust, neither of which are representative of the run-of-mill ore. Furthermore, because of the vast differences in S.G. that usually exist in lodes, between the host rock and the valuable minerals, the dust is often stratified. This is particularly true in cars, where a crude panning can occur. In order to draw a representative grab sample under these conditions it is necessary that the sample be sufficiently large to incorporate all variables. This large sample can then be quartered to the required size.

Visual Sampling of Vein Deposits

In some vein deposits where the grain size of the valuable material is sufficiently large and it is possible to easily distinguish ore material from gangue, it may be practical to use a visual assay. A clearly presented example of this technique is given by Soloman and Brooks (1966) for the Story's Creek Mine in Tasmania. They write that point counting techniques have been used to assay wolframite in quartz veins at Story's Creek Mine. Preliminary tests showed that the precision of the estimate of wolframite could be gauged by a formula involving the percentage of wolframite, the grid size, the measurement area and the size of the wolframite grains. The method was introduced because of the expense involved and delay in chemical assaying of wolframite. Three principal errors are inherent in the process

1) Operator variations, such as the misidentification of minerals.
2) Counting errors, involving areal relationships.
3) Sampling errors, involving the determination of volumes from areal analyses.

At Story's Creek Mine, the point counting grid used was 2 foot square and contained 1 inch mesh, to give 576 points. Operation errors were estimated by repeated countings at an identical grid position. Counting errors,calculated on a series of counts on each of several faces, were checked by comparison with a theoretical variance, based on the estimated percent of the numeral (p), grid spacing (a), radius of grains (h), the area covered by the grid (A) and the number of points counted n. Where the grid spacing is more than twice the radius of the grains, the variance is 100 $\frac{pa^2}{A}$ i.e. where

$$\frac{a}{R} > 2, \quad S^2 = 100 \frac{pa^2}{A}$$

After discounting the counting value by 30 percent, for losses in post mining operations, the count and production assays were very close. Soloman and Brooks (1966) conclude that the grid size would have to be adjusted to suit local conditions.

A visual assay technique is also employed at the Messina Copper mine in the Northern Transvaal (Van Graan, 1964). This technique is used for those faults and breccias where the ore is sufficiently massive to be seen easily. The method is preferred to chemical assay because it is cheap and fast, meaning that more faces can be sampled at more frequent intervals and, apart from this, optimum results are not achieved by cutting grooves across massive sulphides. The visual assaying is performed by drawing a 2 metre chalk line across the strike and measuring, with a pair of engineering calipers, the length of copper sulphide along the chalk line. The value of each stretch is read off by placing the points of the open calipers on a calibrated scale. If more than one copper mineral is present, each is measured separately and the values combined. The percentage copper is determined by the caliper formula as follows:

$$\frac{a \times b \times c \times 100}{(a\times b)+(A-a)B} = \%\ Cu$$

where

a = total centimetres of copper sulphide measured

b = specific gravity of copper sulphide encountered

c = percentage copper contained in sulphide (chalcopyrite 34.5%Cu, bornite 55%Cu, chalcocite 79.8%Cu etc.)

A = distance across which sampling was done

B = specific gravity of host rock. (Unpublished company report, Messina Transvaal Development Company Limited).

Visual sampling is also used on some of the Mississippi valley deposits of the viburnum trend and at the Pine Creek tungsten deposit in California (Dr. R.E. Jacob, pers. comm., 1979).

TREATMENT OF SAMPLE DATA

Sampling is conducted in order to classify material into ore and waste. To achieve this the first step is to obtain a mean assay value for each sample point over a particular width. This may be the minimum mining width or the width of the economic horizon, whichever is the greater. To obtain a mean value for subdivided channel samples it is essential that the assay value be weighted against the sample length. This is evident from the following example (from McKinstry, 1948). The sample is a channel sample cut from a gold bearing quartz vein. The example indicates the vast difference between mean value and the weighted mean value. A further weighting may be introduced to allow for differing spheres of influence as a result of an irregular sample interval or because the samples are of different length. (Samples at regular intervals will have similar areas of influence, but the volume of influence will be affected by the total length that is sampled).

TABLE 3

Difference between mean value and weighted mean value.

Length of sample	Assay	Length of sample	Assay	LengthxAssay
5	30	5	30	150
8	15	8	15	120
10	10	10	10	100
7	15	7	15	105
4	60	4	60	240
3	65	3	65	195
2	75	2	75	150
5	50	5	50	250
Total 44	320	44	320	1310

Mean value $\frac{320}{8} = 40$ Weighted mean $\frac{1310}{44} = 29.77$

(Taken from McKinstry, 1948)

Once the mean value for every sample point is known, Rendu (1978) recommends the use of classical statistics for the critical analysis of sample data in the early stages of evaluation. This is because the number of samples at this stage is too small and the distances between them too large to permit the use of spatial statistics.

In geological sampling the statistical distribution of sample values can be random, normal or log-normal. Normal distribution of values is typical of ores of low variability such as bedded iron ore deposits. Vein deposits are typified by an approximately log-normal distribution. To determine the distribution of sample values it is necessary to group the values into a series of classes and then to count the number of samples within each class. Using this data a histogram may be drawn. If the histogram is reasonably symmetrical, the population is considered normally distributed, whereas a histogram of positively skew distribution is typical of log-normally distributed data.

The detailed shape of the histogram depends on the limits of the classes used to group the samples and a less biased method of determination is to calculate the cumulative frequency distribution of the values and to plot this distribution on log-probability paper. Normally distributed data will plot as a straight line on normal probability paper and log-normally distributed data will plot as a straight line on log-probability paper. Alternatively, a X^2 test may be applied to test if the actual distribution of data is significantly different from a theoretical curve representing normally or log-normally distributed data (David, 1977). The samples taken during the initial evaluation might be insufficient to obtain a representative histogram or cumulative distribution. Judgement and past experience may then be used to decide on the distribution.

In hydrothermal deposits mineralogical (zoning), structural and secondary-superficial (oxidation, enrichment or leaching) processes may result in more than one sample population. These multiple populations will plot as a series of crests and troughs on a curve fitted to a histogram of the data. The various populations can then be separated by inspection or by plotting the cumulative frequency of the data on log-normal probability paper, in which case the points of inflection will demarcate the various populations (Sinclair, 1976; Hatherly, 1978). For each population and for the total population, a mean sample value, a variance from the mean, and a confidence interval should be estimated by calculation. These parameters are useful in deciding on the future of the project.

The following discussion on the estimation of mean, variance and confidence intervals is summarized from Rendu (1978) and Krige (1978) but is explained in most standard statistical texts.

The mean ($\bar{x}$) is the average of all the values (xi) in a population consisting of n elements and is defined as follows:

$$\text{Sample mean } \bar{x} = \frac{\Sigma xi}{n}$$

The variance S^2 is a measure of the dispersion of data and is given by the squared total of the difference between all individual values and the population mean, divided by the number of samples in the population

i.e. $S^2 = \frac{\Sigma(xi - \bar{x})^2}{n - 1}$ or by more simple calculation

$$S^2 = \frac{\Sigma xi^2 - n(\bar{x})^2}{n - 1}$$

which reduces to

$$S^2 = \frac{\Sigma xi^2 - \frac{(\Sigma x)^2}{n}}{n}$$

If the variance is large, as in erratically mineralized deposits, then the confidence with which the mean value of the ore body can be considered the same as the mean value of the samples is decreased. The variance is merely the mathematical confirmation of the geological suspicion that if values are erratic in both distribution and metal content, it is unlikely that the mean of the sample values will be the same as the mean of the sampled population. The square root of the variance is the standard deviation and at this stage in evaluation gives an idea of the relative magnitude of the errors between the means. It should be emphasized that the reason for statistical analysis of the sample data is to distinguish the different populations and then to compute the mean metal content for each of these populations to within a certain degree of confidence.

One problem which is characteristic of vein deposits is that of the occasional high value. This problem is illustrated in the following example taken from McKinstry (1948). Consider the following list of assay values of channel samples taken along a drift on a gold bearing vein:

5.25; 4.00; 17.85; 480.10; 49.20; 22.40; 6.00; 10.15; 1.40; 7.00.

The arithmetic mean of these ten sample assays is 59.70, but if the high value is omitted then the average of the remaining nine is 12.99 Obviously the high sample value plays a critical part in determining the average and prompts the question as to whether the sample should be given its full value, a reduced value or simply ignored? De Gast (1968) suggests the rejection of high assay values, whereas Mr. W.R.

Atkinson (written comm, 1979) remarks that it is a common practice with quartz- vein gold deposits in Rhodesia to cut any values considerably higher than average to a value 50 percent above average. Bird (1969) found that at the Migori-Nyanza mine in southwest Kenya the erratic nature of the gold distribution and the considerable variation in the magnitude of the assays was apparent from the surface sampling. The erratic distribution is indicated in Table 4.

TABLE 4

Range dwt/ton	% Frequency
nil - 0.5	33
0.6 - 5.0	27
5.1 - 10.0	8
10.1 - 20.0	10
>20.0	22

Consequently all assays along the entire quartz reef were weighted and treated as follows to establish a down dip grade.

Overall weighted sample grade for outcrop	13.74 dwt/t
less 10% sampling safety factor	12.36 dwt/t
allow 20% wallrock dilution at 0.5 dwt/t	10.30 dwt/t
expected mill-head grade	10.30 dwt/t

In practice the reef down to the 22 metre level has produced a mill-head grade of 14.6 dwt/ton. Therefore, in this case there was no justification for arbitrarily discarding high values or for making grade cuts.

MacKenzie (1979) suggests that the practice of building a conservative bias such as these into mineral evaluation, results from the notion that a reward greater than that expected is preferable to the penalty if the project proves to be less profitable than initially estimated. Obviously these arbitrary adjustments do cut down the likelihood of making a bad investment but at the same time such empirical methods will inevitably result in the rejection of economic investment opportunities. Generally, the rejection or adjustment of information, simply because it is highly uncertain, can lead to serious economic repercussion (MacKenzie, 1979).

Sichel (1966) presents an efficient means of estimating the unknown true average grade from few samples of an erratically mineralized ore body. This is by use of the t estimator, which is unbiased and less sensitive than the arithmetic mean to scattered high values. The first problem is to determine the location parameter (a). This may be estimated from experience (or calculated, if sufficient data exists) and arises if the population distribution is not absolutely log-normal. The departure from the log-normal model is according to a regular pattern (Krige, 1960) and if ignored means that the lower values are lower than those of the theoretical distribution and the final estimate of the mean has a positive bias. Krige (op. cit.) suggests that the actual distribution of values be fitted to the log-normal distribution by the addition of a, to each assay value z. He suggests two methods for estimating the best value of a (which can only be estimated if a sufficiently large data base exists at that stage; Krige, 1960, uses 1000 sample values in his example).

1) Graphical trial and error, using logarithmic -probability paper. Here the distribution of z is first plotted and then the distribution of (z + a), with the constant a being increased in stages until the resultant plot yields the most satisfactory straight line. For all practical purposes this procedure will be found adequate because the fit is not sensitive to small variations in the value of a.

2) A mathematical solution based on the median value m , and any two cumulative frequencies (F_1 and F_2) spaced equidistantly from and on either side of m,e.g. the 10 percent and 90 percent cumulative frequency values. These values are then substituted in the following equation

$$a = \frac{m^2 - F_1F_2}{F_1+F_2-2m}$$

Once a is estimated, the t estimate of the mean can be made. The following example from Sichel (1966) is intended as a guide to the computation process. Assume that the location parameter a was estimated from adjacent areas as 80, and that the sample data consists of 5 borehole samples well spread over the entire geographical area under investigation. The boreholes yielded the following 5 intersections 329, 277, 111, 189 and 5071 in dwt. Table 5 shows the layout of the computations. The derivation of the equations will not be repeated here

TABLE 5

Intersection	z (in. dwt)	$z + \alpha$ (in. dwt + 80)	$x' = \log_{10}(z + \alpha)$	$(x')^2$
1	329	409	2·6117	6·8210
2	277	357	2·5527	6·5163
3	111	191	2·2810	5·2030
4	189	269	2·4298	5·9039
5	5,071	5,151	3·7119	13·7782
Total	5,977	6,377	13·5871	38·2224
Means	$\bar{z}$ = 1195·4	$\overline{z + \alpha}$ = 1275·4	$\bar{x}'$ = 2·7174	m_2' = 7·6445

Computation of t - estimator.

but the final result is $t = e^{\bar{x}} \gamma_5(v)$ where $\gamma_5 = 5.3019$ (from tables),

$e^{\bar{x}}$ is the antilog_{10} of $\bar{x}'$ and $v = m_2' - (\bar{x}')^2$.

Therefore $\gamma_5 v = 5.3019[m_2' - (\bar{x}')^2]$.

In this example $e^{\bar{x}}$ = antilog 2.7174 = 521.7

and $\gamma_5 v = 5.3019[7.6445 - (2.7174)^2] = 1.379$

[Table A (in the Appendix) is used in the same manner as log tables].

For a sample size of n = 5 and v = 1.3, the tabulated value from Table A is 1.807; and for n = 5, v = 1.4 it is 1.884 . Linear interpolation between these figures gives a value of 1.868 for v = 1.379 .

Now t = $(e^{\bar{x}})$(tabulated value for n,v)

which in this example is

$t = (521.7)(1.868)$

$= 974$

This value (974) contains the location parameter a (here a=80) and to obtain an estimate of the mean grade, the value of a must be subtracted from the t value. Here 974-80 = 894 in dwt. Compared to the arithmetic mean, $\bar{z}$=1195 in dwt, this estimate is 301 lower. The actual average grade, of the erratically mineralized Harmony gold mine, in the area from which these 5 samples were drawn is 618 in dwt.

This example shows that the one very high value in the group influences the t estimator much less than it does the arithmetic mean. It will be appreciated that a grade estimate based on five boreholes may deviate considerably from the unknown true average value of the deposit. Sichel (1966) presents confidence limits for the t estimator. These limits indicate with a specified degree of confidence in which grade interval the unknown true mean value is expected. Again, the theory is rather complex but the actual calculations of the confidence limits are extremely simple if tables B and C (reproduced in the Appendix) are used. Table B is used for a 90 percent central confidence limit, whilst Table C provides the lower confidence limit. Again the use of these tables is best explained by means of an example. Take again the example of the five intersections from Harmony (Table 5). n = 5; v = 1.379; t=974 in dwt. From Table B the following factors are obtained

	v = 1.3	v = 1.4
Upper limit	11.75	13.55
Lower limit	0.4210	0.4083

From the linear interpolation for v = 1.379, the upper limit from Table B for 5 samples is 13.17 and the lower limit factor 0.4110 . These factors are then multiplied by the t value (974) and a (80) is subtracted to give the final upper and lower limits i.e.

$[(974)(13.17)] - 80 \quad = 12748$ in dwt

$[(974)(0.4110)] - 80 \quad = \quad 320$ in dwt.

It may therefore be stated with a confidence of 90 percent that the unknown true grade of the area from which the samples were drawn has a mean grade of between 320 and 12748 in dwt. This range is extremely large but can be substantially reduced by increasing the number of samples. For example for 20 intersections the upper and lower limits for the same values of v and t are 1958 and 544 respectively.

In most borehole valuation problems it is the lower limit which is of utmost importance. From this example one can state with a confidence of 95 percent that the minimum grade will be at least 320 in.dwt. If a lower confidence limit of 90 percent is acceptable, then Table C can be used, which for n = 5 and v = 1.379, gives a lower limit of

$$\left[(974)(0.5272)\right] - 80 \quad = 433 \text{ in.dwt.}$$

The probability from Sichel's method is therefore 90 in 100 that the unknown true mean will be at least 433 in.dwt. If a value of 433 in.dwt. is economically justifiable then evaluation can proceed.

Mackenzie (1979) suggests an alternative method for determining the lower limit to within a certain degree of confidence. In this technique, once the data have been transposed into a frequency distribution curve and the various populations have been separated, then the deviation from the expected value of any one population can be expressed as a probability distribution. The expected value of the ore block is indicated by the central tendency of the distribution, while the standard deviation reflects the variability. With flatter distribution there is a greater chance that the actual result on mining will be different from that predicted from sampling. The disadvantage of this technique is that the probabilities have to be estimated from empirical data from operating mines, from historical information, from experience and from questioning experts in the field. The advantage of the technique is that it can be used to justify further investment in information since in most cases increased sampling leads to a steepening of the distribution curve around the expected value and hence increased confidence with which the expected value can be estimated.

This leads on to one of the most important uses of statistics in evaluation, which is to establish the optimum sample density. There are several statistical tests which can be applied to achieve this, one of which involves taking alternate sample results along a drift. Here, two sample populations are made up, one consisting of all the odd numbered samples and one of all the even numbered samples. (This effectively doubles the sample interval). The mean value of these two populations can then be tested, for example by the "t" test of the null hypothesis, to determine if there is any significant difference between the two means. If this test is repeated for several drifts and there is no significant difference in the mean values (within an acceptable confidence limit), then the sample spacing can be doubled. Alternatively a variogram (see p.82) may be constructed to determine the area of influence of any one sample value. The point of inflection of the variogram would be the optimum sample interval.

In veins neither of these techniques is ideal because of the great variability in the distribution of values. This variability can result in the sample interval being decreased to zero with the two samples that are taken from the same locality still containing significantly different amounts of metal. This is known as the "nugget effect" and results from the chance inclusion in one sample of a small highly mineralized veinlet. In vein deposits this nugget effect is usually unavoidable because of the nature of such ore. Therefore, here optimum sample density can only be determined on a historical basis, normally after some mining results are to hand. Once mining results are available statistical tests can be used to determine the degree of difference between the actual mined grade and the grade as predicted by various combinations of samples taken at various different intervals.

GEOLOGICAL FACTORS INFLUENCING THE CALCULATION OF ORE RESERVES IN VEIN DEPOSITS

Cox (1968, p.1) defines an ore as "a natural aggregate of one or more minerals which may be mined and sold at a profit or from which some part may be profitably extracted". That part of this material which is still to be mined is referred to as a "reserve of ore". These ore reserves are traditionally divided into three classes; proven, probable and possible (Figure 33), the precise meanings of which vary from one organization to another. According to Cox (1968)

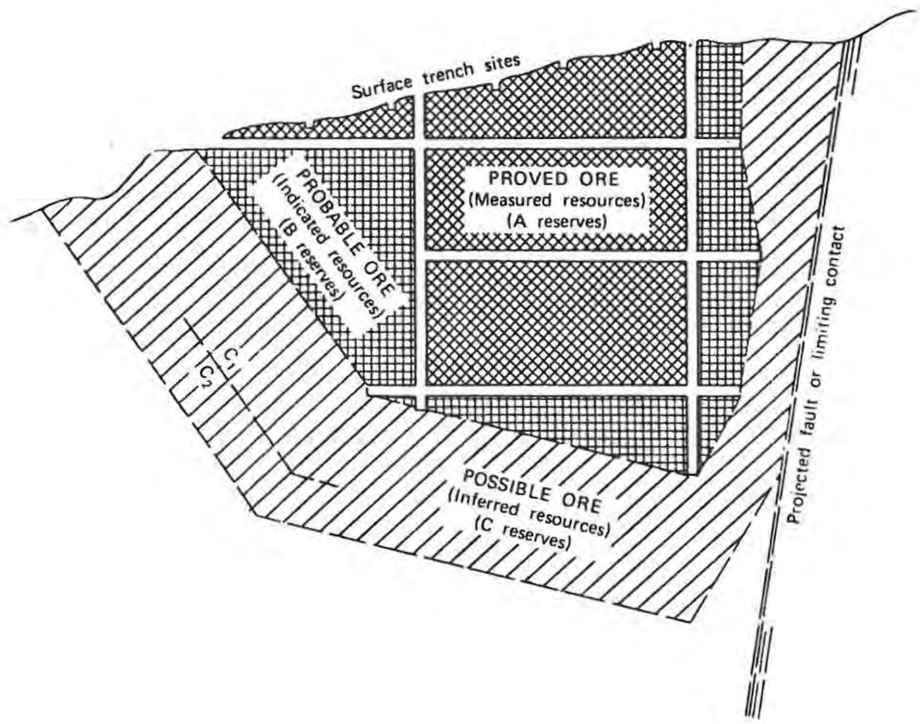


FIGURE 33 Proved, probable and possible ore designations in the conventional sense of the terms, plotted on a longitudinal section of a vein-type deposit.

proven ore is that for which tonnage is computed from dimensions revealed on four sides (in outcrops, drill holes and workings). The sites for inspection, sampling and measurement are so closely spaced and the geologic character so well defined that size, shape and mineral content are well established.

In probable ore, tonnage and grade are computed partly from specific measurement and partly from projection. Together these two classes make up ore reserves. In veins it is seldom that the reserves are ever elevated above the probable status.

The third classification is that of possible ore, for which quantitative estimates are based on the broad knowledge of the geologic character of the deposit and for which there are few if any samples or measurements. However, enough geologic evidence exists to say that a certain amount of ore at a certain grade is possible.

The definition of ore as material that can be sold at a profit automatically introduces an economic bias to reserve determination. This leads to the definition of a "cut off grade", which is the level of mineralization below which mining can no longer be profitably conducted. Cut off grades (and production levels) should be varied through the life of the mine so as to optimize profits (Mackenzie, 1979). The rate of mining and the cut-off grade should initially be set at higher levels than those defined for the fixed optimum and then allowed to decline over the production life, eventually attaining levels lower than the fixed optimum. This means that a tonnage-grade classification is fixed for only one particular point in time. This results in the largest possible discounted cash flow for the property. Obviously, this technique can only be efficiently practised if the potential of the ore body is fully realised at the outset. In vein deposits dynamic optimization of cut off and production rate is difficult because of the uncertainty inherent in the evaluation of these deposits.

Ore reserve estimates are required to assist with the analysis of mine development and operating decisions. As a result reserves are often determined prematurely on limited information. The confidence that can be placed on the reserve estimate is a function of the geological information available at the time the estimate is made. In veins the erratic nature of the mineralization usually results in a low level of geological information until a mature stage in mine development is reached.

According to Cooke (1976) the accuracy of ore estimation is directly related to the distribution of mineralization between sample points. The main characteristics that influence the confidence of interpolation between sample points are:

1) Distribution of valuable constituent. Evenly disseminated mineralization can be evaluated with greater accuracy by taking a relatively limited number of samples compared to irregular (lode-type) mineralization.

2) Volume content of the valuable constituent. The larger the volume of valuable material to volume of barren rock, the more accurate are grade estimations by normal sampling methods. This characteristic is related to the range of values in individual samples which influence the accuracy of calculating the average value. In vein deposits the content of valuable material in the host rock is normally so low that it can be conveniently expressed in terms of grams per tonne. Obviously under criteria (1) and (2) veins represent unsatisfactory conditions for accurate calculation of average grade.

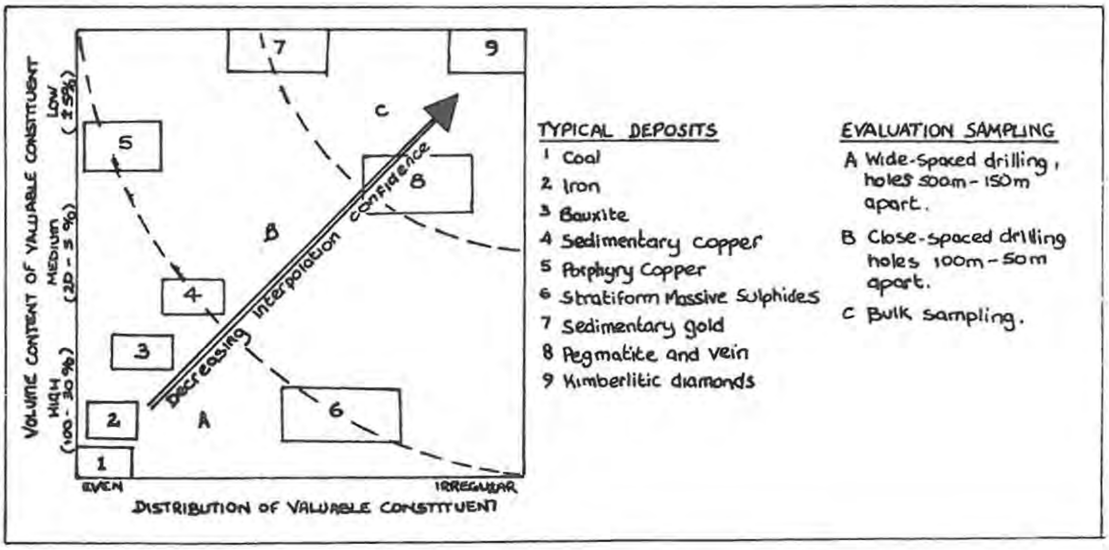


FIGURE 34 Relationship between nature of mineralization and confidence of interpolation between samples.

(After Cooke, 1976)

Figure 34 shows the relationship between the nature of mineralization and confidence of interpolation between sample points. It also shows that this confidence can be improved by increasing the intensity and volume of the samples. The deposits being considered fall into category 8, which according to this diagram are best evaluated by a combination of close-spaced drilling and bulk sampling.

There is no established method whereby the influence of assay values between sampling points can be accurately estimated. Existing techniques are based on geological knowledge and past mining experience of a particular ore deposit. For example Hazen (1968) found, at the Butte Copper (Montana) vein deposit, that the area of influence of an assay appeared to be 4 metres to 9 metres in drifts and 2 metres to 10 metres in raises. In horsetail veins, in the same deposit, in crosscuts, raises and stopes, the area of influence was less than 2 metres. The following diagram illustrates some common methods used to project assay values between boreholes. Although all these methods

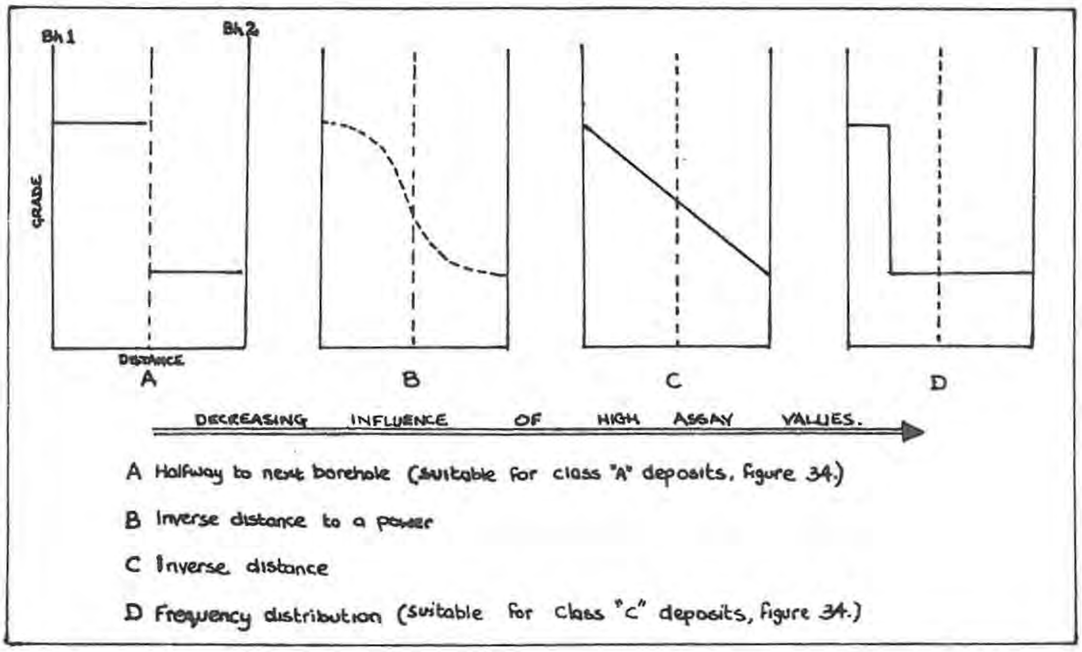


FIGURE 35 Various interpolation methods for projecting assays between boreholes.
(From Cooke, 1976)

have been used for hydrothermal deposits, method "D" is probably the most commonly employed. The result of this technique is the same as cutting the initial assay value, because it decreases the area of influence of the high assay value and gives a greater area of influence to the assay value more close to that of average.

There is also the problem of the extension of the ore body beyond the last borehole (sample) that intersected ore-grade material. Popoff (1966) recommends that the following procedure be used for extrapolation of boundaries of sandstone uranium deposits between drill holes; some of which may intersect ore, good mineralization, poor mineralization or barren rock. The cut-off boundary between two holes is selected as three quarters of the distance between an ore intersection and a position where strong mineralization is intersected, one half if the second hole only intersects weak mineralization and one third if the adjacent hole was barren. Again such empirical rules are not applicable in veins where natural variability is so high that excessive errors may be introduced by using these techniques. In vein deposits, however, there is usually one direction (for example parallel to the foliation) along which extension of sample influence is the most valid. For example at Consolidated Murchison Antimony mines' Monarch shaft the mineralization is elongated in the direction of foliation and therefore the area of influence of a sample value is greatest in this direction (Mr. J. De Vaal, pers. comm., 1979). The structural control exerted on the emplacement of vein deposits has been stressed in earlier chapters where it was emphasised how the mineralization concentrated along the plane of the structure. Therefore, the observation made at Monarch shaft of a structural direction where the mineralization is more continuous, is not unique.

Detailed surface geological mapping and surface trench sampling will give some indication of the area of influence of any sample. Systematic geophysical surveys may also be useful in this regard. For example, should pyrrhotite be associated with the ore mineral,then a magnetometer may be of use in proving continuity of values. Rooiberg Tin mine is currently experimenting with I.P. in an attempt to locate disseminated sulphides associated with the cassiterite (Mr. R. Stuart, pers. comm., 1979). This has potential as a tool to prove continuity of values.

The area of influence of a sample can be determined mathematically using covariograms, variograms (Matheron, 1971; Rendu, 1978), or semi-variograms (Krige, 1978; Rendu, 1978). These statistical methods are based on the fact that the values of two samples taken close together are more likely to be similar than if they are taken far apart. In other words the correlation between sample values is a function of the distance between them (Rendu, 1978). In isotropic ore bodies this correlation is the same in all directions. In vein deposits, as previously mentioned, the correlation between sample pairs and therefore the area of influence of any one sample is unlikely to extend any great distance and is quite likely to extend further in some direction than in others. This direction and the extent of influence can be determined by plotting the semi-variogram. This technique is explained in the following example.
Assume that a series of channel samples were taken at equal intervals (h) along a drive and the assay results were those tabulated below.

Sample Number	1	2	3	4	5	6	7	8	9	10	11	12
Assay Value	5	5	7	9	12	11	8	7	2	4	3	3

This gives 11 pairs of samples taken a distance h apart. That is, samples Number 1 and 2, 2 and 3, 3 and 4 etc., make up the sample pairs at lag 1. For lag 2 those pairs of samples that are a distance 2h apart are considered. These are sample pairs 1 and 3, 2 and 4, 3 and 5 etc. Likewise samples 1 and 4, 2 and 5 etc. make up the pairs for lag 3 and so on. In each of these sets of pairs an average value is found for the left hand members of pairs and for the right hand members of the pairs. Then the variance at lag 1 is determined by the sum of the difference between the left hand value of each pair and the mean of the left hand values multiplied by the difference between the right hand member of each pair and the mean of the right hand values. The total is divided by one less than the number of pairs; in this example 11-1. The process is repeated for the pairs at lag 2h, 3h, 4h etc., until all possible lags have been dealt with. In this example the final lag would be lag 11, which would be the combination of sample 1 and sample 12. This is summarized in the following equation.

$$\gamma h = \frac{1}{n(h)-1} \sum_{i=-1}^{n(h)} \left[x(z_i) - \bar{x}_1\right]\left[x(z_i + h) - \bar{x}_2\right]$$

where n(h) is the number of pairs at that lag, z_i is the value of the left hand member of the pair, $\bar{x}_i$ is the mean of the left hand members, $x(z_i+h)$ is the value of the right hand member of the pair, $\bar{x}_2$ is the mean of the right hand members and h is the sample interval.

From this data the semi-variogram may be drawn, which is a plot of the variation or difference in sample value at that particular lag. An ideal example is shown below. Here the semi-variogram consists of an inclined line, a point of inflection and a sill or plateau.

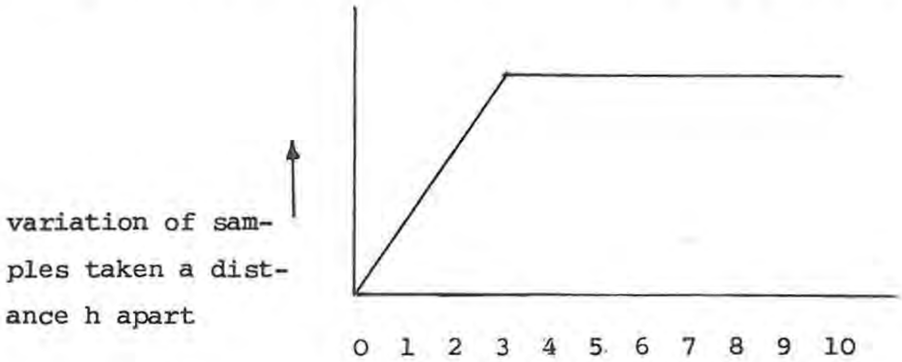


FIGURE 36 An idealized semi-variogram.

Along the inclined line, up to the point of inflection, the sample values are not independent and the value of the sample is influenced by its relative position. Beyond the point of inflection the sample values may be considered random. That is, in this example, those samples taken at intervals of less than 3h are within the area of influence of the previous sample. An interval of 3h is therefore the optimum sample spacing and is also the maximum distance any sample value may be extended, with any degree of confidence, between boreholes or beyond the last ore intersection when determining ore reserves. The semi-variogram is directional and different plots should be made for sample traverses in different directions. Often the data may have to be transformed, for example into log values, before the variogram can be drawn. In deposits with a nugget effect the sloping line will not intersect the axes at the origin; the amount of displacement is equivalent to the nugget effect.

CONVENTIONAL TECHNIQUES FOR TONNAGE-GRADE ESTIMATION

Traditionally, the estimation of tonnage and grade is made using cross-sectional, longitudinal section or moving average methods. All these techniques are basically similar. Once the outline of the ore body is known it is divided into geometric blocks of various dimensions depending on the shape of the ore body, the drilling pattern used for delineation and the method of estimation chosen. The slope of the block and the position of block boundaries are governed by geological conditions such as the presence of a fault, change in dip, thinning of an ore shoot, folding, zoning, weathering and water level. The guiding factor is that the material within any block should be as homogeneous as possible. Each block is then given the average grade of its samples. A volume is computed for each block and converted to a tonnage by dividing by a tonnage factor based on the S.G. Block values are then combined to assess the weighted average, tonnage and grade of the overall deposit.

Whichever of the conventional analysis techniques is used for tonnage-grade estimates, the following assumptions are required.
1) the basic elements of a mineral deposit vary in some geometric manner between two adjacent observations: usually the rule of gradual change or value of the nearest point is accepted for interpretation,
2) there is continuity of the deposit between observations,
3) the actual form of the mineral deposit can be approximated by the combination of several hypothetical geometric shapes,
4) the ore body continues for some distance beyond the last ore-bearing section drilled.

The first step in the application of the cross-sectional method is to construct horizontal or vertical sections at equal intervals. These intervals are often chosen to coincide with the drill sections. The trace of the ore body is then drawn on the sections, usually using the rule of gradual change between data points. Each drill hole is then assigned an area of influence, usually half way to an adjacent point and an average grade and area of the section is then computed. The section values are extended to adjacent sections, usually by the rule of gradual change, to give volumes. These volumes are then combined and (after allowing for the end sections and dividing by a tonnage factor) a final tonnage at a particular average grade is accepted (Patterson, 1959; Popoff, 1966; Hazen, 1968).

In the longitudinal sectional method, the section is usually the projection of the deposit onto a plane approximately parallel to the deposit. The section is then divided into triangular blocks, as shown in Figure 37, by connecting all drill hole intersections

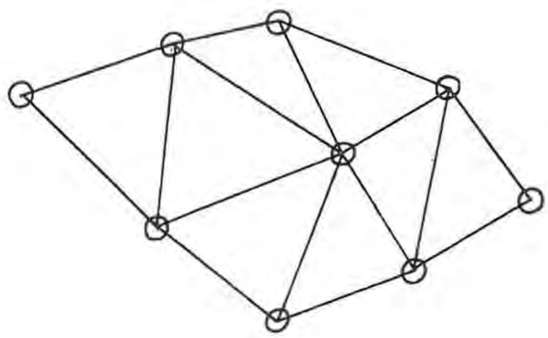


FIGURE 37 The use of triangles for the area of sample influence in ore reserve estimation.

(Redrawn from Patterson, 1959)

with straight lines. Each triangle in the plane is assumed to represent a prism with edges equal to the thickness of the individual drill hole intersection. The volume is computed using the formulae for a truncated triangular prism. The average grade is estimated as a weighted average of the drill hole intersections on the apices of the triangle. Individual block values are combined to derive overall deposit tonnage and grade estimates. Alternatively the longitudinal section may be divided into polygonal blocks as shown in Figure 38, by the construction of medians to the lines joining the drill hole projections. Here each

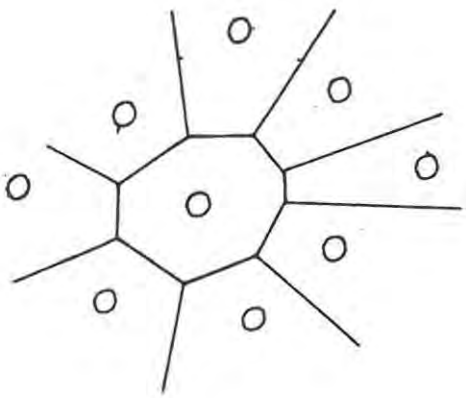


FIGURE 38 The use of polygons for the area of sample influence in ore reserve estimation.

(Redrawn from Patterson, 1959)

block takes the grade and thickness of the central drill hole. Block values are then combined to determine the tonnage and average grade of the deposit (Patterson, 1959; Popoff, 1966).

In the moving average method, the deposit is divided into blocks of uniform size and the average grade of each block is computed from the weighted sum of observations near the block. The assigned weights may be based on various assumptions such as inverse distance, inverse squared distance, inverse cubic distance etc. Ore reserves are then determined by combining all blocks with average grades above a pre-determined cut-off grade (Mackenzie, 1979). With regard to erratically mineralized vein deposits these conventional techniques have four major shortcomings

1) the area of influence given to individual sample values far exceeds their actual area of influence. This means that the actual average grade of an area assigned to a sample assay may be quite unrelated to that assay. (In terms of the overall deposit, this may balance out but it certainly will introduce a bias on a smaller, oreshoot scale).

2) the conventional methods do not provide a measure of confidence in the tonnage and mean grade estimates. A knowledge of the uncertainty associated with these estimates, particularly grades, is important for evaluating the economics of mine development.

3) These techniques not only assume an area of influence for grade values but also assume continuity of the ore horizon between sample points. It is essential that the degree of continuity be established prior to a tonnage calculation. In veins this requirement can usually be met only after extensive underground development particularly in areas that have experienced multiple deformation.
4) These techniques are best suited to ore bodies which are tabular in shape. Vein deposits are usually more complex than this. Often veins are duplicated or boudinaged by folding or may occur as lense-like bodies in 'horsetails' or tension gashes.

Heim (1968) stresses the importance of calculating ore reserves as mining reserves which he defines as the estimated tonnage and grade of ore material which is finally recoverable and delivered to the mill. This is particularly applicable in hydrothermal deposits which often have irregular ore boundaries resulting from replacement mineralization and may contain large areas of internal waste through lack of mineralization where space did not exist, as well as in zones of brecciation. Most mining methods do not allow complete extraction of bodies with irregular boundaries. Problems also occur where the vein suddenly narrows. Here additional material beyond the ore horizon must often be taken so as not to leave overhangs which could later present mining problems. Heim (op. cit.) suggests that ore reserves should be determined on the proposed stope outlines rather than according to the geological outline. Obviously, this suggestion is only reasonable in the more advanced stages of exploration and development when the details of the mining technique to be employed (e.g. stope height, level interval etc.) are known. In some vein deposits, several ore lenses occur parallel to the main vein. In this situation it is difficult, when calculating reserves, to predict whether the lenses will be mined separately or as part of the main stope. In the latter case this may necessitate the mining of the intervening waste as well.

One mine where this type of ore reserve is calculated is at the Oamites copper lode in South West Africa (Yaldwyn, 1979; written comm.). Here a total in situ mineral inventory is calculated twice a year. This includes economic and sub-economic blocks and is divided into proven, probable and possible classes using the guide lines set out previously. The advantage of this system is that the total tonnage and grade is not altered for changing copper prices. From the in situ mineral inventory, the technical staff then "take out" (1) uneconomic blocks due to low copper grades, (2) areas of poor ground conditions, (3) pillars which are to remain for stability, (4) oxidized zones and (5) areas of the ore body which are geometrically uneconomic to mine using the current mining method. That is, the final reserve is adjusted according to local geological conditions. In blocks, where poor hangingwall exists, more ore may have to be left as support than elsewhere. As a result each block is treated separately. These "taken out" tonnages are put into the "not in reserves" category and may become economically minable at a later stage. Heim (1968) looks at poor hangingwall conditions in the North Coldstream mine from the opposite point of view. Instead of subtracting tonnage for support pillars he increases the tonnage to allow for dilution. This mine, which is located 98 kilometres west of Port Arthur, Ontario, has a hangingwall of competent chert and incompetent chloritic schist. These varying conditions result in varying degrees of sloughing of the hangingwall which, according to Heim (op. cit.) can add a further 25 percent, at zero grade, to the calculated tonnage. Poor hangingwall conditions are characteristic of vein deposits because of the frequent development of argillic alteration along the ore boundaries.

At the Anglo-Rouyn property in Saskatchewan, Skopos and Lawton (1968) report that a different degree of dilution is calculated for different blocks, depending on relevant geologic factors such as ground conditions, stope width and grade of the material on either side of the oreshoot.

The ore reserves at Anglo-Rouyn and the North Coldstream mine were calculated using the conventional longitudinal method. Parfitt (1968), the chief mine engineer at McIntyre mine, Porcupine district, suggests a more conservative technique for ore reserve estimation in vein deposits. At the McIntyre mine, gold occurs in numerous steeply dipping quartz-carbonate lenses. The gold distribution is extremely erratic and because of this the arithmetic mean of the underground samples is grossly in excess of the mill-head grade. The assays are therefore adjusted to agree with the mill-heads, by reducing all high assays to an arbitrary maximum. The cutting figures are adjusted according to sample width because experience has shown that in this body higher-than-average assay values are more likely to be valid if they are taken from narrower widths. The sampling system has also been changed from conventional channel sampling to chip sampling. By collecting large chip samples the number of assays above the arbitrary cutting point was reduced by 30 percent. The mine has been developed by levels that are usually either 38 metres or 46 metres apart. "The veins are followed by drifting on each level and two sets of samples are taken on each face. In estimating the average value of an ore section, the widths used are either the minimum mining width of 1.2 metres or the width of the vein. The grade is adjusted by a percentage correction factor equal to the average discrepancy between the sampling and the recovery for the period and then reduced by 10 percent to allow for dilution. If the ore sections on adjacent levels are sufficiently continuous and there is no geological evidence to the contrary, the values on the two levels are averaged and a reserve is calculated between them. Reserve blocks are calculated on isolated sections of drifts above and below the level to a height of half of the ore length at that level, up to a maximum of half the level interval. This is, of course, only a probable reserve. However, as the ore lenses tend to have greater vertical than horizontal dimensions, this practice is on the whole, conservative." (Parfitt, 1968, p.198).

A similar technique of ore reserve estimation is employed at Messina mine, in the Northern Transvaal. However, at Messina the average for the drift is only extended 5 metres above the hanging-wall and 5 metres below the footwall, regardless of the strike length of the shoot intersected in the drift (Mr. D. Dicks, pers. comm., 1979). These areas are then classified as probable reserves. Inter-levels are then developed at 10 metre vertical intervals and if ore is intersected again in these interlevels, then the ore is upgraded to the class of proven reserves (Figure 39). This technique is deemed

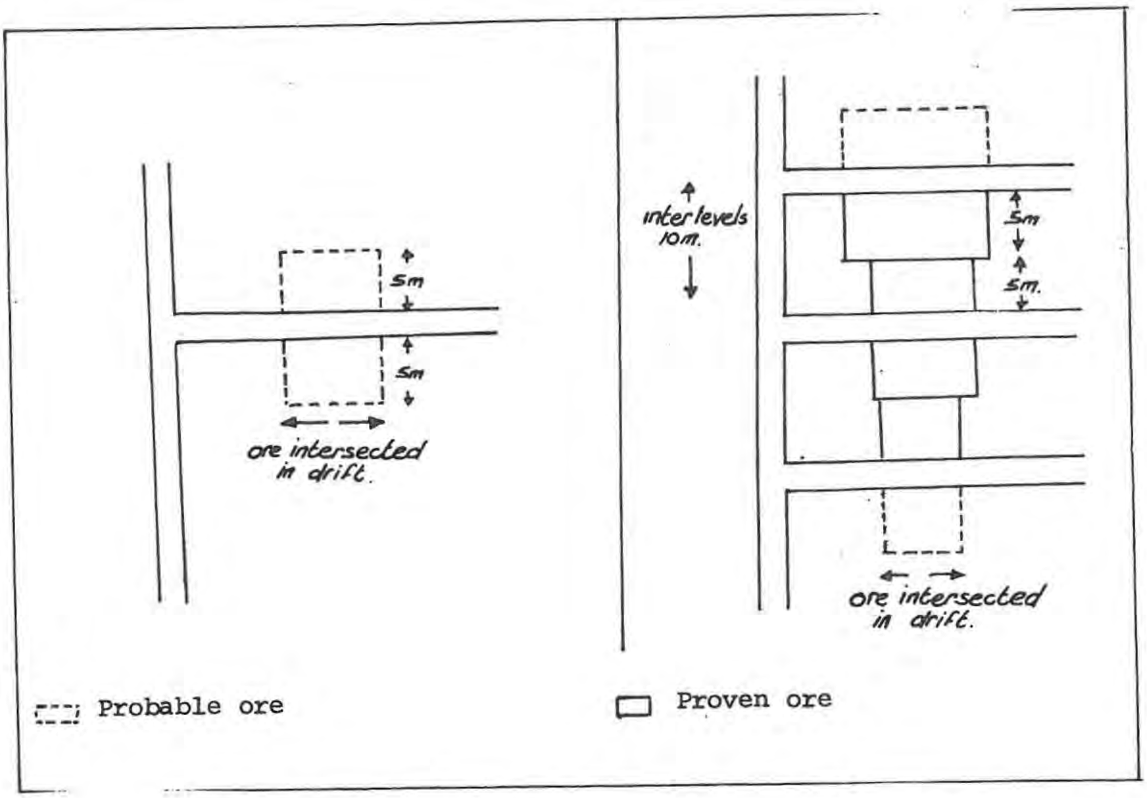


FIGURE 39 Method of determining ore reserves at the Messina mines.

necessary because of the discontinuous nature of the Messina ore pods. The blocks are given a mean grade by averaging the sample results obtained by the visual sampling techniques described earlier (supplemented by any internal information from diamond drilling or sludge sample results).

A feature of the Messina ore body is the mineralogical zoning. This affects the degree of correlation between the predicted ore grade and the mill-head assay values. In general this correlation is good but where chalcocite is mined, the predicted values are invariably lower than the mill feed.

The Use of Isometric Projections in Ore Reserve Estimation

Badgley (1959) recommends the use of isometric diagrams as an aid to ore reserve estimation in deposits which have a complex ore outline. These are block diagrams in which all the distances in the block diagram are equal to the measurement along the corresponding directions in the maps and cross sections from which the block diagram is made up. They are particularly valuable in demonstrating the complex and often non-continuous relations of structurally controlled hydrothermal ore deposits and are of great value in predicting ore extensions during the development stages of a mine. In this respect these diagrams are useful in establishing "possible" ore reserves and for extending "probable" reserves beyond the last ore intersection.

Isometric diagrams are extensively used at the Messina deposits and they have been used by Koch and Link (1970) to plot a three dimensional trend surface (see later) for each metal in the Frisco veins. Here the diagram clearly revealed the steep plunge of the shoot and the zonal decrease in lead content downwards. The construction of isometric diagrams is reviewed in Badgley (1959), shading techniques are given by Ives (1939) and a list of devices designed to take much of the drudgery out of constructing the diagrams is provided by Johnston and Nolan (1937).

Statistical Analysis for Tonnage-Grade Estimates of Vein Deposits

Statistical techniques have made significant contributions to the improvement of ore reserve estimations. The main advantage of statistical analysis is that it provides a framework for assessing the uncertainty of geological estimates. The main disadvantage is the frequent lack of recognition and understanding of the influence of geological control by the statistician on the one hand, and on the other hand, the difficulty experienced by many geologists and mining engineers in fully understanding the complex mathematical statistics. In vein deposits where natural variability is high, this variability tends to mask the significance and relationships that are present in the information gathered on the mineral deposit. It is a function of statistical analysis to remove this mask and to assess these relationships (Popoff, 1966).

Statistics can be considered under two main headings; classical statistics and geostatistics. In classical statistics the assumption is made that the samples are independent of one another. However, in geostatistics it is accepted that adjacent samples are related to one another by an amount proportional to the distance between them. Classical statistics are only recommended for the early stages of exploration, when samples are widely spaced and each sample is beyond the range of influence of its neighbours. With more detailed exploration, the sample density is increased and spatial or geostatistics is recommended (Rendu, 1978).

Blais and Carlier (1968) explain the fundamental differences between classical statistics and geostatistics by means of the following simple example (Figure 40). They assume that 100 equidistant channel samples of identical volume have been taken at 5-foot intervals, along a drift 500 feet long, from a copper-bearing vein having a constant thickness of 5 feet. The imaginary assay values from this vein are plotted as silhouette diagrams in relation to distance along the drift and to an arbitrary cut off grade of 1.5 percent Cu. In the two cases shown in Figure 40 the 100 assay values are the same - namely two equal series of values ranging from 0.1 to 5.0 percent Cu. Obviously, population I is identical to population II because the assay values are the same in both cases. Furthermore, the average grade, standard error of the mean and frequency histograms of assay values are the same. The only difference is that in case I the values are distributed randomly and in case II the values are symmetrical about a high-grade centre. Classical statistical techniques will not reveal this difference which will be revealed in geostatistical analyses.

The Use of Geostatistics to Determine Grade

One application of geostatistics is in the selection of blocks to include in a reserve calculation. In conventional ore reserve estimation techniques, if the average of the peripheral sample values is below the cut-off grade then that block is not considered as forming part of the reserves. This can lead to an underevaluation of the

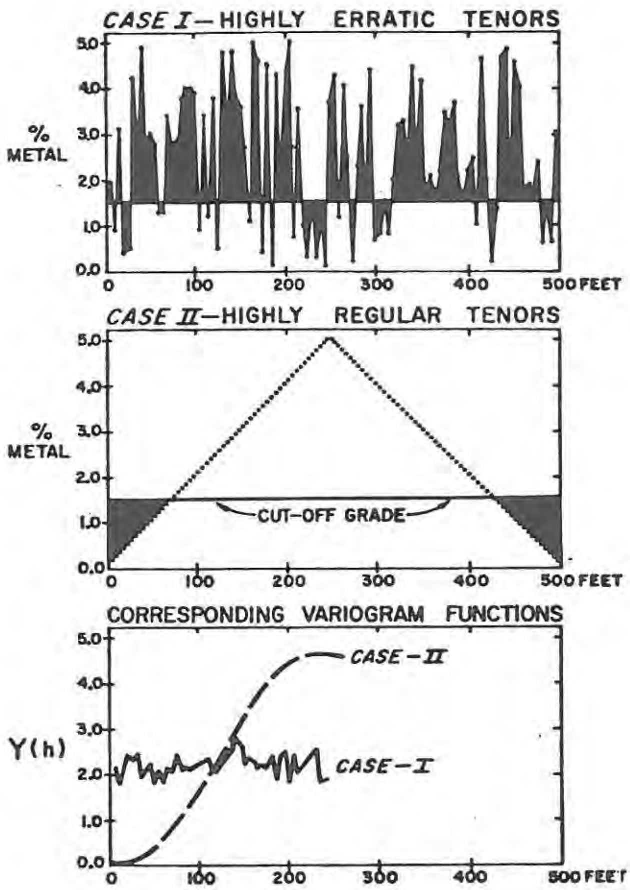


FIGURE 40 Silhouette diagrams of two vastly different populations of identical assay values, with corresponding variogram functions.

(After Blais and Carlier, 1968)

property. Krige (1962) has shown that in erratically mineralized ore bodies, the block expectation from a series of peripheral samples may be anything from twice to half such sample average. For this reason, in these bodies, personal discretion has had to play a major role in selecting blocks for inclusion in the reserves. Krige (1962) suggests that this personal selection be replaced by the application of the principles of correlation and regression. These principles require a measure of

1) The variation between the actual value of the ore parcel concerned and

2) the accuracy of the valuation of the individual ore parcels.

The technique is based on a lognormal distribution of values (this is the usual case in veins) and the accuracy of the valuation of the ore parcel is accepted as being directly proportional to the average values of the samples along the exposed face. It must be recalled that in conventional estimation, the ore along the thin exposed skin is seen as being representative of the intact ore lying beyond this sampled skin. At Messina the sampled face is regarded as being representative of the material 5 metres ahead of the face while at McIntyre mine a height equal to half the strike length of the mineralized intersection is considered to be of a grade equal to that calculated along the exposed face. Obviously in erratically mineralized bodies these assumptions can often lead to errors, the result of which is that, relative to the cut-off, the low grade blocks are under valued and the high grade blocks are over valued. This is statistically unavoidable. Krige (1962) explains this in a simple example:

Consider 100 ore blocks consisting of 50 blocks all at an average grade of 8 dwt/ton, and 50 blocks at 2 dwt/ton. The maximum error in the valuation of an individual block is 2 dwt/ton. If the pay limit is set at 5 dwt/ton, all the 8 dwt blocks will still be valued as payable (6 to 10 dwt) and all the 2 dwt blocks as unpayable (trace to 4 dwt). If, however, these errors applied to a set of 50 blocks at an actual grade of 4 dwt/ton and 50 blocks at 6 dwt/ton, it is evident that some of the 6 dwt blocks will show average grades as valued below 5 dwt/ton and similarly some of the actually unpayable 4 dwt/ton blocks will be valued as payable.

The general tendency which leads to an under valuation of low grade blocks and an over valuation of high grade blocks in erratically mineralized bodies, once a cut-off is imposed, is illustrated in the following diagram taken from Krige (1962). Here, the x axis represents the value of samples taken along the face which is exposed at time 1. These are the values which are used to predict the grade of the intact area ahead of the face. On the y axis are the values of the samples taken on the face once it had advanced 30 feet. The 45° line is that of perfect correlation and represents the expected values of the face 30 feet ahead. The flatter line (correlation

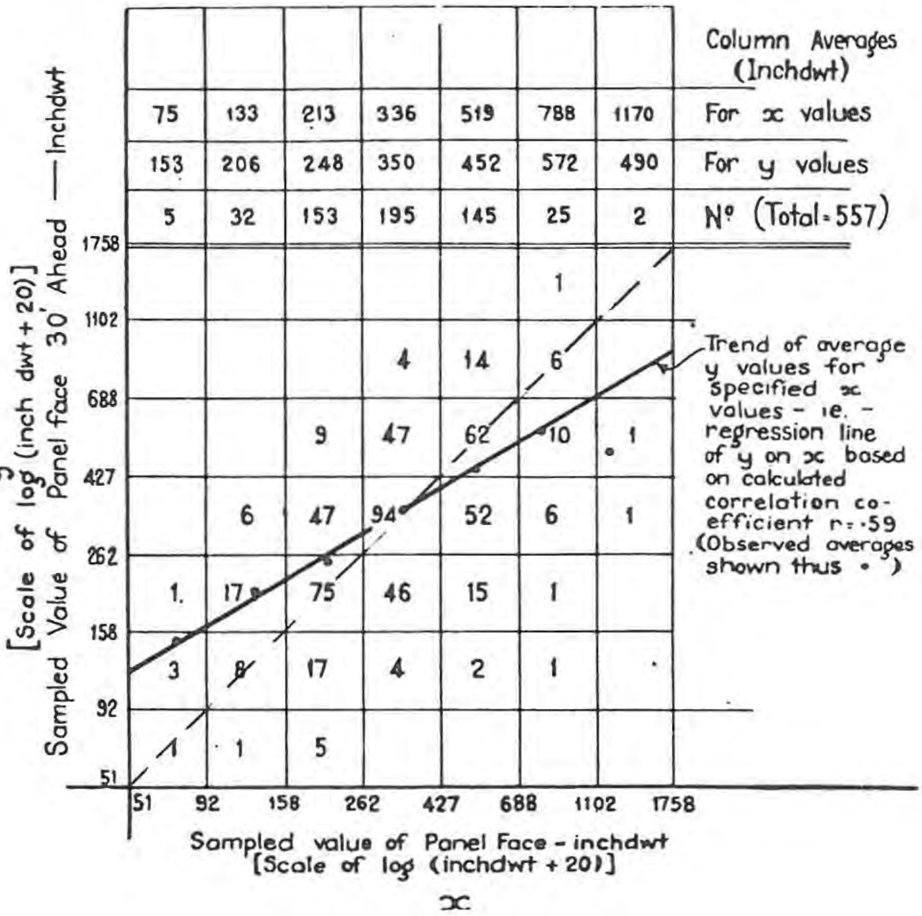


FIGURE 45 Correlation between panel face values 30 ft apart, and hence the value on average of the ore 30 ft ahead of a panel face of any specified value.

(After Krige, 1962)

coefficient 0.59), is a plot of the actual values recorded at that position against the actual value recorded at the same relative position 30 feet previously. The over valuation of faces in the high grade categories and under valuation in the low grade categories is clearly apparent.

In conclusion, Krige (1962) suggests that, in erratically mineralized bodies, whose values have a lognormal distribution, an "effective" pay limit should be introduced. The effective pay limit is defined as the current face value which on statistical average indicates that the ore a specified distance ahead of the face would have an actual value just on the pay limit. The pay limit will be lower than the apparent pay limit when dealing with low grade blocks and higher than the apparent pay limit in those blocks in the higher grade categories. There will, however, be one position where the effective and apparent pay limits will be the same. The net result of the application of this technique will be to increase the reserves because the marginal blocks will not be under valued and rejected, as well as providing a more accurate estimate of mill head grades.

Geostatistics and Trend Surface Analysis in Ore Reserve Estimation

Krige (1966) advocates the use of trend surface analysis to predict the continuity of ore in erratically mineralized ore bodies. This is a smoothed surface based on applying a moving average to the points of the discontinuous valuation surface. For the purpose of ore reserve estimation a trend surface can be visualized as being similar to a contour plan depicting a topographical land surface (Krige, 1966). However, in the trend surface the contour links areas of equal assay value rather than equal elevation.

An immediate objection to the use of this technique in vein deposits is the "nugget effect" which means that ore samples cut in or immediately adjacent to the same groove are known to differ substantially even when perfect samples are available. In addition the surface will consist of a mixture of holes (representing areas where no values exist) and needle-like peaks (where isolated extremely high values occur). By using the average of a sufficient number of points the surface can be smoothed. This smoothed surface may exhibit certain trends which could be useful in establishing the continuity and pitch of oreshoots. The main problem is the optimum number of samples which should be used to obtain the average for use in the analysis. In this respect it is important that a balance be attained between the loss of detail arising from an average of too many samples and the random appearance that can exist if too few samples are used to establish an average value. Krige (1966) considers that the minimum size of an ore unit that can be selectively mined is the smallest area that should be taken as being a representative block. The average of this block is then combined with the average of the three blocks to the side of it and the mean plotted in the central position (Figure 46). The grouping is then shifted sideways

Block A mean value a	Block B mean value b	Block C mean value c	Block D mean value d
contour point $a+b+e+f \div 4$	$b+c+f+g \div 4$	$c+d+g+h \div 4$	
Block E mean value e	Block F mean value f	Block G mean value g	Block H mean value h

FIGURE 46 A suggested matrix for trend surface analysis.

an amount equal to the length of a block and the process is repeated. It must be stressed that the final contour is not representative of any individual value but merely emphasizes a trend.

A logical step from trend surface analysis is kriging, which combines the concepts of trend analysis with the important problem of the accurate grade estimation of individual blocks, for which there are limited samples available. In kriging, each sample is assigned a weighting factor based on the location of the sample relative to the geometry of the block. It is common sense that those values closest to the point of interest should carry maximum weight and those values further away progressively less weight. In a typical case, values on the block periphery would carry 60 percent of the weight and values outside the periphery, up to 100 metres away, would carry 35 percent of the weight and the remaining 5 percent would be allocated to other values still further away. When a body has a definite trend, such as an oreshoot in a vein, a modification to the weighting factor should be applied. This modification should be applied in the first periphery which carries the most weight.

The actual procedure to be followed in establishing a weighting table is given in Krige (1966) annexure 3. It is based on a multiple regression coefficient as determined by correlogram or semi-variogram analysis. A matrix notation and an example in which optimum weights are established is given in Krige (1978), and is not repeated here. Although the concepts embodied in kriging are simple, the calculation of the optimum weighting "a" is outside the scope of any manual effort by desk calculator and should only be attempted on a modern, large computer. David (1977) sets out the basic structure of a kriging program.

The main advantages of kriging are that it gives an accurate estimate of the thickness and grade of the ore, it provides a measure of the uncertainty of the result and it leads to improved grade control. Trend analysis has also been used (Cook and Munro; in Krige, 1966) to apply a crude weighting to a moving average technique. Here a weighting is applied to peripheral samples according to the general trend of the contours in that part of the property.

One vein deposit where geostatistics have been successfully employed is at the Eagle copper vein in Northern British Columbia (Sinclair and Deraisme, 1974). The vein occurs in a tightly folded sequence and contains chalcopyrite as the most abundant sulphide. The mineralization is present as massive patches of anhedral grains and, less commonly, as fine fracture fillings. Oxidation has occurred at and near the surface and along some fracture zones but no oxidized material was included in the study. The average vein width is 4 feet but considerable variation exists in the range 1 to 9 feet giving the vein a complex lenticular form.

The data for the geostatistical study consisted of 693 channel samples from three levels, taken at an average interval of 8.5 feet. The study was concerned with two regionalized variables, vein thickness and accumulation. It involved a series of well defined steps listed as

1) critical analysis of data,
2) construction and interpretation of experimental variograms,
3) global estimation of ore reserves (for the whole deposit) and

4) local estimates of ore reserves (kriging).

Semi-variograms were the fundamental tool for the study and were determined for both variables (Figures 47 and 48). Because the

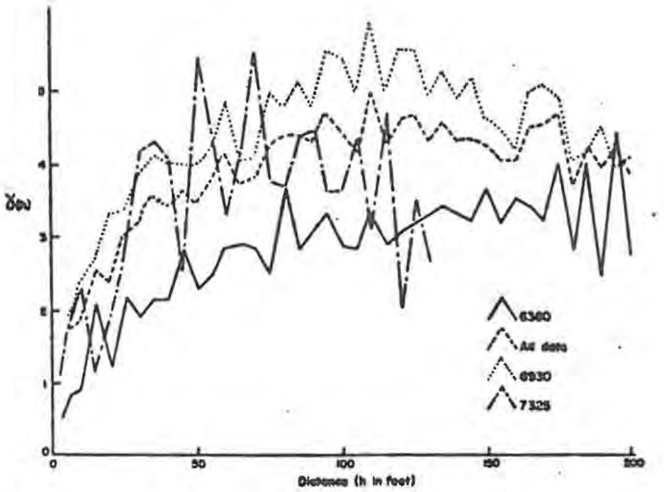


FIGURE 47 Experimental variograms for vein thickness.
(After Sinclair and Deraisme (1974)

data are not contained on a perfectly regular grid it was necessary to group sample pairs into distance classes as follows

h = 5 ft $\pm$ 2.5 ft, 10ft $\pm$ 2.5ft, 15ft $\pm$ 2.5ft etc.

The experimental variograms were prepared for a planar vein by calculating the true distances between paired samples along the vein to produce the effect of unfolding.

Sinclair and Deraisme (1974) found that in the variogram for vein thickness there was a break at a lag of 20ft and a general levelling of the values beyond a sample spacing of 100ft. In physical terms these ranges refer to average lengths of lenses and indicate the presence of two groups of lenses with average lengths of 20ft and 100ft respectively. In practical terms this means that a value for vein thickness can only be estimated 100ft beyond the last sample.

At the Eagle vein mine the level interval is 395ft. Because sample data can only be estimated 100ft below the one level and 100ft above the next level this leaves a vertical gap of 195ft between levels to which no estimate can be applied. One aspect of the importance of geostatistics in ore reserve calculations thus becomes apparent.

Variograms for accumulation (Figure 48) have a pattern similar

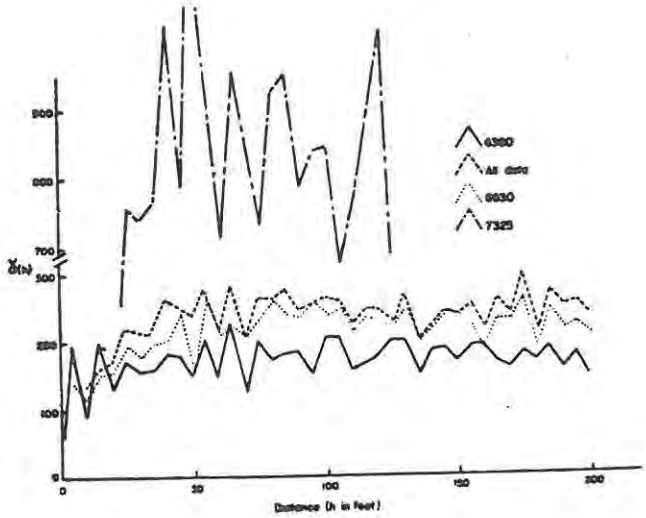


FIGURE 48 Experimental variograms for accumulation.
(After Sinclair and Deraisme, 1974)

to those for vein thickness, except that local fluctuations are more pronounced and a discontinuity (nugget effect) exists at the origin. Sinclair and Deraisme (op. cit.) calculated the grade of each block by kriging the arithmetic mean of the contained data. From the variogram it was obvious that the blocks to be kriged should not have dimensions greater than 100ft. Hence it was apparent that local estimates of ore reserves could be made for less than one half of the known ore body. Furthermore, if estimates were desired for those zones between levels, additional sampling drifts would be required.

One further advantage of kriging is that it provides a measure of uncertainty of the result. This uncertainty depends on the sample variance and can be reduced by increasing the total number of samples. For example, in order to halve the uncertainties of local grade at Eagle vein, drifts would have been required at intervals of 200ft and raises at intervals of 100ft (or vice versa). Obviously the cost of such workings must be weighed against the necessity of attaining increased precision.

The main reason why kriging is not in common use for the estimation of ore reserves in vein deposits is because kriging can be carried out only if there are fairly large amounts of closely spaced data. Since veins are usually of limited tonnage and irregular shape, these deposits suffer in comparison with large, regular ore bodies in that the mining cost per ton is generally higher. Furthermore, the erratic nature of the mineralization in vein deposits means that the exploration and development costs per ton are orders of magnitude larger than in the case of regularly mineralized bodies, in order to reach the same accuracy of ore evaluation. These costs may be too much of an expense for the small mine. As a result sample information in veins is usually limited in the early stages. In more advanced stages of development, trend analysis may be useful. Again it should not be applied if data are too scanty as this will lead to incorrect trends. Trend analysis is proposed because

1) it can define areas of uncertainty where further samples may be useful to distinguish between pay and unpay,
2) it allows visual extrapolation of ore grades for long term forecasts,
3) it can be a useful guide in the exploration effort,
4) it can be used to establish any correlation between two valuable constituents such as Cu and Ag; or values and a structural feature, such as mineralization extending in the direction of the foliation,
5) it is useful in dividing up the property into blocks for ore reserve estimation,
6) it may be used to apply a crude weighting to the peripheral samples for the determination of ore reserves by a technique of moving averages.
7) it may be used to establish if there are any trends to the density,
8) it smoothes the data by eliminating the effect of erratic high values thereby emphasizing the trends in data which initially may

appear random,

9) it is cheap and easy to apply.

The final technique chosen for ore reserve estimation in vein deposits depends on the character of the mineral deposit, the mining technique to be employed, the exploration data available and the accuracy required. The polygonal and cross-sectional methods have been successfully used in computing reserves of vein deposits (Popoff, op. cit.). At times more than one method may be used to check results but the accuracy of the final result usually depends more on the geologic interpretation and assumptions, than on the method used. The geologic interpretations are, in turn, influenced by the number of blocks, the density of the grid of workings and the density of the drill holes.

The need for improved ore valuation techniques in the Witwatersrand gold mining industry was only fully recognised once the order of errors committed in the valuation of ore blocks, and the effect of such errors on the mines' profit were appreciated (Krige, 1966). This stage is still to be reached in many vein deposits, where the old axiom championed by McKinstry (1948) that veins are "extramensurate" or "difficult to explore much in advance of mining" still prevails. Some of the statistical techniques outlined above and an appreciation of the pitfalls of conventional techniques may aid in establishing the magnitude of inherent error. This may lead to a progressive improvement in ore grade and tonnage prediction.

It should be borne in mind that an incorrect estimate of the average dip, thickness or S.G. of any ore block, can lead to vast inaccuracies in the estimate of tonnage. Any inaccuracies in the plotting of the ore body outline or a poor sample or diamond drill core loss can also result in gross inaccuracies in the final figures. In these cases no amount of statistical sophistication can make up for inadequate input data.

THE PROBLEM OF SPECIFIC GRAVITY

In all ore reserve estimation techniques a tonnage factor, based on the S.G. of the ore, is applied to convert volumes of ore to tonnes. The cubic metres per ton factor is usually calculated from an average specific gravity estimate. This average is usually determined empirically from a few samples or by guess. However, it must be emphasized that metalliferous bodies are heterogeneous and their S.G. varies. This means that the paymetal content per unit volume varies within the ore body even though assay values might be identical. This is shown in the following illustrative table and example taken from Dadson (1968).

Sp. G.	lbs Ore per cu. ft	lbs BM in 1 cu. ft Ore	oz. PM in 1 cu. ft Ore	Cu. Ft per Ton of Ore
2.5	156	1.56	0.078	12.82
3.0	187	1.87	0.094	10.69
3.5	218	2.18	0.109	9.17
4.0	250	2.50	0.125	8.00
4.5	281	2.81	0.140	7.12
5.0	312	3.12	0.156	6.41
5.5	343	3.43	0.171	5.83
6.0	374	3.74	0.187	5.35
6.5	406	4.06	0.203	4.93
7.0	437	4.37	0.218	4.58
7.5	468	4.68	0.234	4.27

Range of specific gravity of ores. For purpose of comparison, the pay-metal content is taken as 1 percent base metal (BM), and 1 oz. per ton of precious metal (PM). One cubic foot of water weighs 62.4 lbs.

TABLE 6 (After Dadson, 1968)

Example: Assume a deposit which contains a lead-silver ore in a quartz gangue. Also contained in the gangue are some barren base-metal sulphides. A series of borehole samples were taken, and the intersected width, assay value and S.G. of each sample were recorded. The percentage lead and the oz/ton silver were then calculated; "a" using the specific gravity in the calculation and "b" not using the specific gravity.

"a"

Using Specific Gravity

				Assay			
W	G	WxG	lbs Ore WxGx 62.4	% Pb	oz/t Ag	lbs Pb	oz Ag
2.0	3.5	7.0	436.8	2.0	10.0	8.736	2.18
0.5	7.5	3.75	234.0	70.0	100.0	173.800	11.70
2.0	4.0	8.00	499.2	4.0	20.0	19.968	4.99
4.5	4.167	18.75	1170.0	16.45	32.6	192.504	18.87

Av. lead = 16.45 percent

Av. silver = 32.26 oz/ton.

"b"

Not Using Specific Gravity

	Assay			
W	% Pb	oz/t Ag	Wx % Pb	Wx oz. t Ag
2.0	2.0	10.0	4.0	20.0
0.5	70.0	100.0	35.0	50.0
2.0	4.0	20.0	8.0	40.0
4.5	10.44	24.44	47.0	110.0

Av. lead = 10.44 percent.

Av. silver = 24.44 oz/ton.

(After Dadson, 1968)

The example set out above may be extreme, but it does illustrate the fact that assay results should not be applied directly to linear measures when there is appreciable density contrast in the samples. When dealing with typical vein-deposit minerals such as gold, tin, tungsten etc. the high S.G. of these minerals when compared with the S.G. of the gangue means that, at the very least, the tonnage factor should vary with the assay result. This would require the assumption that the S.G. of the gangue material remains constant. In vein deposits the gangue often contains varying amounts of base metal sulphides and hence this assumption is not normally valid. A further problem is that these deposits are often porous.

These types of problems are documented for the Tsumeb Mine in South West Africa. Although it is not a vein that is mined at Tsumeb, the principles can be adapted to the deposits under consideration. The geology of the Tsumeb mine is described by Söhnge (1964). The Tsumeb ore body occurs as a flat-cylindrical pipe in karsted dolomite. Here the variation in the grade of the three main metals, copper, lead and zinc is wide. Fluctuations range from 0 to 55 percent for copper, 0 to 75 percent for lead and 0 to 45 percent for zinc (Söhnge, 1966). The various metal ratios vary with depth and horizontally and the metals are contained in over 100 different minerals.

Widespread oxidation has occurred to 300 metres below surface and again along the North Break fracture zone, from 850 metres depth to 1000 metres depth (Söhnge, 1964). The gangue may be any combination of quartz, calcite, dolomite and pseudoaplite . Subterranean water channels lead into as well as away from the ore pipe and the ores thus modified are cavernous and more often leached than enriched.

The density for use in the ore reserve estimation can therefore vary according to metal content, metal ratio, degree of oxidation density of the metal bearing mineral, the type of gangue and the presence of vugs. This results in a range of ore S.G. from 6.0 down to 3.0. It is obvious from the example cited from Dadson (1968), that the S.G. must be known in order to accurately calculate the tonnage and grade. At Tsumeb it is determined as follows

$$S.G. = \frac{3870}{138.2 - .874X_c + X_l + .619X_z}$$

where X_c = Cu percent in ore
X_l = Pb percent in ore
X_z = Zn percent in ore.

The S.G. of the gangue material is assumed to be constant at 2.8, which is the S.G. of dolomite and no allowance is made in the formula for oxidation or cavities. However, a cut of between 5 and 10 percent may be made to the volume in areas where the ore is obviously modified and vuggy.

The derivation of the constants in the formula is given in Söhnge (1966) and is based on the galena equivalent for chalcocite, sphalerite, bornite and tennantite. At Tsumeb the S.G. is combined with each single assay to arrive at a properly weighted average for any group of assays before then applying a weighting for the area of influence.

CONCLUSION

The conclusion is intended to serve as a model for the evaluation of vein deposits as well as a summary of the main points of this review.

1) Before evaluation can commence it is important to develop a conceptual model of the ore deposit. This is normally aided by detailed surface mapping and by channel sampling of surface pits and outcrop. This indicates the deformational history of the deposit and the distribution of the mineralization. Emphasis should be placed on the distribution rather than actual value of the mineralization because the latter can be modified by surface leaching and enrichment. Particular note should be made of the continuity of the ore horizon, any deflection in the trend of the horizon and any duplication by reverse faulting, secondary faulting or folding. A uniform, continuous, duplicated horizon is preferable to a discontinuous horizon, whilst the best mineralization is often associated with an area of deflection. If the structural history of the area can be unravelled, by mapping and strain analysis, then it is useful to ascertain whether the ore horizon occupies a zone of tension or compression. Ore bodies in tension zones are usually smaller but of greater tenor than those occupying zones of compression.

2) Diamond drilling is necessary for testing the continuity of the ore-zone at depth and as an aid in structural analysis. Drill holes should be concentrated in structurally distrubed zones, and all marker horizons should be carefully noted for structural purposes. Any mineralization intersected by diamond drilling should be noted, but the actual values should be treated with caution. The nature of the mineralization is important. Disseminated mineralization is more likely to be continuous than mineralization occurring as fracture infill. Any open space textures such as comb structures, druses or brecciation would give greater impetus to the project than would massive host rock. Banded or ribboned vein material is promising as this suggests multiple addition of vein material and associated mineralization.

3) Once several diamond drill results are available Sichel's (1966)

"t" estimate should be employed to determine the minimum grade to within 90 percent confidence limits. If this minimum grade is acceptable, then further drilling should be initiated with a view to proving continuity and to establish the optimum area for trial mining. Several composite samples should be made up for metallurgical testwork and to obtain a full mineral inventory of the deposit. This would indicate the potential for co-products or by-products and the presence of deleterious elements. A mineralogical examination should be undertaken with particular attention being devoted to the texture of the ore minerals, as this will affect the metallurgical recovery.

4) The accurate evaluation of vein deposits is impossible by diamond drilling. Hence it is recommended that limited underground development, from a series of prospect winzes, be conducted. The siting of these winzes should be guided by the diamond drilling results. The development should be designed to test the grades determined by previous sampling. Oxidized zones should be treated separately from fresh material, as should areas of differing deformation and mineralogy. All development, raises and winzes should be channel sampled at close intervals, each lithology being sampled individually. All samples should be weighted according to their length and S.G., and mean grades for each block should be calculated, from the channel sampling, using Sichel's "t" estimator. These values should be checked against those obtained on mining in order to determine an effective cut-off. Variograms should be constructed to test the area of influence of any sample in each direction. This will determine the sample pattern. Block values may be discounted to allow for dilution and the porosity of the ore horizon but it is not recommended that high assay values be cut by any arbitrary technique.

5) Finally, it is necessary to calculate the full mineral inventory of the deposit. This should be calculated under the headings proven, probable and possible ore reserves. In areas where sufficient data exist kriging may be used to determine the local proven reserves to a high degree of accuracy. Normally, the data will be too uncertain to justify the use of this time consuming technique and for areas that are not well documented, conventional techniques of ore-reserve estimation are recommended. These reserves can only be considered

probable. Tonnages should be calculated on unfolded, homogeneous blocks and block dimensions should take the mining technique into consideration; the minimum sized block being the smallest area that can be selectively mined. Areas of the ore body that are to be left as support should be placed in the "not in reserve" category. Because these deposits are usually structurally complex, the estimation of the volume of ore-grade rock may require the use of an isometric projection or an isopach map. The area of influence assigned to any assay value for the purpose of determining grade and S.G. may be calculated by use of a variogram. Trend analysis may also be useful for projecting grades beyond the area of influence of a variogram into zones where "possible ore" may exist. On mining an 'effective', dynamic cut-off should be employed so as to maximize the cash flow.

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APPENDIX

TABLE A

FACTOR Υ_n (V) FOR ESTIMATION OF MEAN OF LOGNORMAL POPULATION

V	n = 2	n = 3	n = 4	n = 5	n = 6	n = 7	n = 8	n = 9	n = 10	n = 11	n = 12	n = 13	n = 14	n = 15	n = 16	n = 17	n = 18	n = 19	n = 20	n = 50	n = 100	n = 1.000
0·00	1·000	1·000	1·000	1·000	1·000	1·000	1·000	1·000	1·000	1·000	1·000	1·000	1·000	1·000	1·000	1·000	1·000	1·000	1·000	1·000	1·000	1·000
0·02	1·010	1·010	1·010	1·010	1·010	1·010	1·010	1·010	1·010	1·010	1·010	1·010	1·010	1·010	1·010	1·010	1·010	1·010	1·010	1·010	1·010	1·010
0·04	1·020	1·020	1·020	1·020	1·020	1·020	1·020	1·020	1·020	1·020	1·020	1·020	1·020	1·020	1·020	1·020	1·020	1·020	1·020	1·020	1·020	1·020
0·06	1·030	1·030	1·030	1·030	1·030	1·030	1·030	1·030	1·030	1·030	1·030	1·030	1·030	1·030	1·030	1·030	1·030	1·030	1·030	1·030	1·030	1·030
0·08	1·040	1·040	1·040	1·040	1·040	1·041	1·041	1·041	1·041	1·041	1·041	1·041	1·041	1·041	1·041	1·041	1·041	1·041	1·041	1·041	1·041	1·041
0·10	1·050	1·051	1·051	1·051	1·051	1·051	1·051	1·051	1·051	1·051	1·051	1·051	1·051	1·051	1·051	1·051	1·051	1·051	1·051	1·051	1·051	1·051
0·12	1·061	1·061	1·061	1·061	1·061	1·061	1·061	1·061	1·061	1·062	1·062	1·062	1·062	1·062	1·062	1·062	1·062	1·062	1·062	1·062	1·062	1·062
0·14	1·071	1·071	1·071	1·072	1·072	1·072	1·072	1·072	1·072	1·072	1·072	1·072	1·072	1·072	1·072	1·072	1·072	1·072	1·072	1·072	1·072	1·072
0·16	1·081	1·082	1·082	1·082	1·082	1·082	1·082	1·083	1·083	1·083	1·083	1·083	1·083	1·083	1·083	1·083	1·083	1·083	1·083	1·083	1·083	1·083
0·18	1·091	1·092	1·092	1·093	1·093	1·093	1·093	1·093	1·093	1·093	1·094	1·094	1·094	1·094	1·094	1·094	1·094	1·094	1·094	1·094	1·094	1·094
0·20	1·102	1·102	1·103	1·103	1·104	1·104	1·104	1·104	1·104	1·104	1·104	1·104	1·104	1·104	1·104	1·104	1·104	1·105	1·105	1·105	1·105	1·105
0·3	1·154	1·156	1·157	1·158	1·158	1·159	1·159	1·159	1·160	1·160	1·160	1·160	1·160	1·160	1·160	1·160	1·160	1·160	1·161	1·161	1·162	1·162
0·4	1·207	1·210	1·212	1·214	1·215	1·216	1·216	1·217	1·217	1·217	1·218	1·218	1·218	1·218	1·219	1·219	1·219	1·219	1·219	1·220	1·221	1·221
0·5	1·260	1·266	1·269	1·272	1·273	1·275	1·276	1·276	1·277	1·278	1·278	1·278	1·279	1·279	1·279	1·280	1·280	1·280	1·280	1·282	1·283	1·284
0·6	1·315	1·323	1·328	1·332	1·334	1·336	1·337	1·338	1·339	1·340	1·341	1·342	1·342	1·343	1·343	1·343	1·344	1·344	1·344	1·348	1·349	1·350
0·7	1·371	1·382	1·389	1·393	1·397	1·399	1·401	1·403	1·404	1·406	1·406	1·407	1·408	1·409	1·409	1·410	1·410	1·411	1·411	1·416	1·417	1·419
0·8	1·427	1·442	1·451	1·457	1·462	1·465	1·468	1·470	1·472	1·473	1·475	1·476	1·477	1·478	1·478	1·479	1·480	1·480	1·481	1·487	1·490	1·492
0·9	1·485	1·503	1·515	1·523	1·529	1·533	1·537	1·540	1·542	1·544	1·546	1·547	1·549	1·550	1·551	1·552	1·552	1·553	1·554	1·562	1·565	1·568
1·0	1·543	1·566	1·580	1·591	1·598	1·604	1·608	1·612	1·615	1·618	1·620	1·622	1·623	1·625	1·626	1·627	1·628	1·629	1·630	1·641	1·645	1·649
1·1	1·602	1·630	1·648	1·661	1·670	1·677	1·682	1·687	1·691	1·694	1·697	1·699	1·701	1·703	1·705	1·706	1·708	1·709	1·710	1·723	1·728	1·733
1·2	1·662	1·696	1·718	1·733	1·744	1·752	1·759	1·765	1·770	1·774	1·777	1·780	1·782	1·785	1·787	1·789	1·790	1·792	1·793	1·810	1·816	1·822
1·3	1·724	1·764	1·789	1·807	1·820	1·831	1·839	1·846	1·851	1·856	1·860	1·864	1·867	1·870	1·872	1·874	1·876	1·878	1·880	1·900	1·908	1·916
1·4	1·786	1·832	1·862	1·884	1·900	1·912	1·922	1·930	1·936	1·942	1·947	1·951	1·955	1·958	1·961	1·964	1·966	1·969	1·971	1·995	2·004	2·014
1·5	1·848	1·903	1·938	1·963	1·981	1·996	2·007	2·017	2·025	2·032	2·037	2·042	2·047	2·051	2·054	2·058	2·060	2·063	2·065	2·095	2·106	2·117
1·6	1·912	1·975	2·015	2·044	2·066	2·082	2·096	2·107	2·116	2·124	2·131	2·137	2·142	2·147	2·151	2·155	2·158	2·161	2·164	2·199	2·212	2·226
1·7	1·977	2·049	2·095	2·128	2·153	2·172	2·188	2·201	2·212	2·221	2·229	2·236	2·242	2·247	2·252	2·256	2·260	2·264	2·267	2·308	2·323	2·340
1·8	2·043	2·124	2·177	2·214	2·243	2·265	2·283	2·298	2·310	2·321	2·330	2·338	2·345	2·352	2·357	2·362	2·367	2·371	2·375	2·422	2·440	2·460
1·9	2·110	2·201	2·260	2·303	2·336	2·361	2·382	2·399	2·413	2·425	2·436	2·445	2·453	2·460	2·467	2·473	2·478	2·483	2·487	2·542	2·563	2·586
2·0	2·178	2·280	2·347	2·395	2·431	2·460	2·484	2·503	2·519	2·533	2·545	2·556	2·565	2·574	2·581	2·588	2·594	2·599	2·604	2·668	2·692	2·718
2·1	2·247	2·360	2·435	2·489	2·530	2·563	2·589	2·611	2·630	2·645	2·659	2·671	2·682	2·691	2·700	2·707	2·714	2·721	2·726	2·800	2·827	2·858
2·2	2·317	2·442	2·526	2·586	2·632	2·669	2·698	2·723	2·744	2·762	2·778	2·791	2·803	2·814	2·824	2·832	2·840	2·847	2·854	2·937	2·969	3·004
2·3	2·388	2·526	2·618	2·686	2·737	2·778	2·811	2·839	2·863	2·883	2·900	2·916	2·929	2·942	2·952	2·962	2·971	2·979	2·987	3·082	3·118	3·158
2·4	2·460	2·612	2·714	2·788	2·846	2·891	2·928	2·959	2·986	3·008	3·028	3·045	3·060	3·074	3·086	3·098	3·108	3·117	3·125	3·233	3·274	3·320
2·5	2·533	2·699	2·812	2·894	2·957	3·008	3·049	3·084	3·113	3·138	3·160	3·180	3·197	3·212	3·226	3·238	3·250	3·260	3·270	3·391	3·438	3·490
2·6	2·607	2·789	2·912	3·003	3·073	3·128	3·174	3·213	3·245	3·274	3·298	3·320	3·339	3·356	3·371	3·385	3·398	3·410	3·420	3·557	3·610	3·669
2·7	2·682	2·880	3·015	3·114	3·191	3·253	3·304	3·346	3·382	3·414	3·441	3·465	3·486	3·505	3·522	3·538	3·552	3·565	3·577	3·730	3·791	3·857
2·8	2·759	2·973	3·120	3·229	3·314	3·382	3·437	3·484	3·524	3·559	3·589	3·616	3·639	3·661	3·680	3·697	3·713	3·727	3·740	3·912	3·980	4·055
2·9	2·836	3·068	3·228	3·347	3·440	3·514	3·576	3·627	3·671	3·710	3·743	3·772	3·799	3·822	3·843	3·862	3·880	3·896	3·911	4·102	4·178	4·263
3·0	2·914	3·166	3·339	3·469	3·570	3·651	3·718	3·775	3·824	3·866	3·902	3·935	3·964	3·990	4·013	4·035	4·054	4·072	4·088	4·301	4·387	4·482
3·1	2·994	3·265	3·453	3·593	3·703	3·792	3·866	3·928	3·981	4·028	4·068	4·104	4·136	4·164	4·190	4·214	4·235	4·255	4·273	4·510		
3·2	3·075	3·366	3·569	3·721	3·841	3·938	4·018	4·086	4·145	4·195	4·240	4·279	4·314	4·346	4·374	4·400	4·424	4·446	4·465	4·728		
3·3	3·157	3·469	3·688	3·853	3·983	4·088	4·176	4·250	4·314	4·369	4·418	4·461	4·500	4·534	4·566	4·594	4·620	4·644	4·666	4·956		
3·4	3·240	3·574	3·810	3·988	4·129	4·243	4·338	4·419	4·489	4·549	4·603	4·650	4·692	4·730	4·764	4·796	4·824	4·850	4·875	5·195		
3·5	3·324	3·682	3·935	4·127	4·279	4·403	4·506	4·594	4·670	4·736	4·794	4·846	4·892	4·933	4·971	5·005	5·037	5·065	5·092	5·445		
3·6	3·409	3·792	4·063	4·270	4·434	4·568	4·680	4·775	4·858	4·929	4·993	5·049	5·099	5·145	5·186	5·223	5·258	5·289	5·318	5·706		
3·7	3·496	3·903	4·194	4·416	4·593	4·738	4·859	4·962	5·052	5·130	5·198	5·260	5·315	5·364	5·409	5·450	5·488	5·522	5·554	5·980		
3·8	3·583	4·017	4·329	4·567	4·757	4·913	5·044	5·156	5·252	5·337	5·412	5·478	5·538	5·592	5·641	5·686	5·726	5·764	5·799	6·266		
3·9	3·672	4·134	4·466	4·721	4·925	5·093	5·234	5·355	5·460	5·552	5·633	5·705	5·770	5·829	5·882	5·930	5·975	6·016	6·054	6·566		
4·0	3·762	4·252	4·607	4·880	5·099	5·279	5·431	5·562	5·675	5·774	5·862	5·940	6·011	6·074	6·132	6·185	6·234	6·278	6·319	6·879		
4·1	3·853	4·373	4·751	5·042	5·277	5·471	5·634	5·775	5·897	6·004	6·099	6·184	6·260	6·329	6·392	6·450	6·502	6·551	6·596			
4·2	3·946	4·496	4·898	5·209	5·460	5·668	5·844	5·995	6·127	6·242	6·345	6·437	6·519	6·594	6·662	6·724	6·781	6·834	6·883			
4·3	4·040	4·622	5·049	5·380	5·649	5·872	6·060	6·223	6·364	6·489	6·599	6·698	6·788	6·869	6·942	7·010	7·072	7·129	7·182			
4·4	4·135	4·750	5·203	5·556	5·843	6·081	6·283	6·458	6·610	6·744	6·863	6·970	7·066	7·154	7·233	7·306	7·373	7·435	7·493			
4·5	4·231	4·881	5·361	5·736	6·042	6·297	6·513	6·700	6·863	7·008	7·136	7·251	7·355	7·449	7·536	7·614	7·687	7·754	7·816			
4·6	4·328	5·014	5·522	5·921	6·247	6·519	6·750	6·950	7·126	7·281	7·419	7·543	7·655	7·756	7·849	7·934	8·013	8·085	8·152			
4·7	4·427	5·149	5·687	6·111	6·457	6·747	6·995	7·209	7·397	7·563	7·711	7·845	7·965	8·075	8·175	8·266	8·351	8·429	8·502			
4·8	4·527	5·288	5·856	6·305	6·674	6·983	7·247	7·476	7·677	7·855	8·014	8·157	8·287	8·405	8·512	8·611	8·703	8·787	8·865			
4·9	4·629	5·428	6·029	6·505	6·896	7·225	7·507	7·751	7·966	8·157	8·328	8·481	8·620	8·747	8·863	8·970	9·068	9·159	9·243			
5·0	4·732	5·572	6·205	6·709	7·124	7·474	7·774	8·036	8·265	8·470	8·652	8·817	8·966	9·102	9·227	9·341	9·447	9·545	9·636			
5·1	4·836	5·718	6·386	6·919	7·359	7·731	8·050	8·329	8·574	8·792	8·988	9·164	9·324	9·470	9·604	9·727	9·841	9·946				
5·2	4·941	5·866	6·570	7·134	7·600	7·995	8·335	8·631	8·893	9·126	9·335	9·524	9·696	9·852	9·996	10·13	10·25					
5·3	5·048	6·018	6·759	7·354	7·847	8·266	8·628	8·944	9·222	9·471	9·695	9·897	10·08	10·25	10·40	10·54						
5·4	5·156	6·172	6·951	7·579	8·102	8·546	8·930	9·265	9·563	9·828	10·07	10·28	10·48	10·66	10·82							
5·5	5·266	6·329	7·148	7·811	8·363	8·833	9·240	9·598	9·914	10·20	10·45	10·68	10·89	11·08								
5·6	5·376	6·489	7·350	8·048	8·631	9·129	9·561	9·940	10·28	10·58	10·85	11·09	11·32									
5·7	5·489	6·652	7·555	8·290	8·906	9·433	9·890	10·29	10·65	10·97	11·26	11·52										
5·8	5·603	6·818	7·766	8·539	9·188	9·745	10·23	10·66	11·04	11·38	11·68											
5·9	5·718	6·987	7·980	8·794	9·478	10·07	10·58	11·03	11·44	11·80												
6·0	5·834	7·159	8·200	9·054	9·776	10·40	10·94	11·42	11·85													

TABLE B

FACTORS $\Psi'_{\cdot 95}(V,n)$ AND $\Psi'_{\cdot 05}(V,n)$ FOR THE ESTIMATION OF CENTRAL 90 PER CENT CONFIDENCE LIMITS OF THE MEAN OF LOGNORMAL POPULATION

V	Limit	$n=5$	$n=10$	$n=15$	$n=20$	$n=50$	$n=100$	$n=1.000$
0·00	Upper	1·000	1·000	1·000	1·000	1·000	1·000	1·000
	Lower	1·000	1·000	1·000	1·000	1·000	1·000	1·000
0·02	Upper	1·243	1·117	1·085	1·069	1·038	1·026	1·007
	Lower	0·8978	0·9331	0·9458	0·9530	0·9697	0·9782	0·9927
0·04	Upper	1·364	1·171	1·123	1·100	1·055	1·037	1·011
	Lower	0·8592	0·9068	0·9243	0·9342	0·9573	0·9692	0·9895
0·06	Upper	1·467	1·216	1·154	1·124	1·069	1·046	1·013
	Lower	0·8309	0·8872	0·9080	0·9199	0·9478	0·9622	0·9872
0·08	Upper	1·562	1·255	1·131	1·146	1·080	1·053	1·015
	Lower	0·8078	0·8709	0·8945	0·9080	0·9398	0·9564	0·9852
0·10	Upper	1·653	1·292	1·206	1·166	1·091	1·060	1·017
	Lower	0·7878	0·8567	0·8826	0·8975	0·9328	0·9512	0·9833
0·12	Upper	1·741	1·326	1·230	1·185	1·100	1·066	1·019
	Lower	0·7703	0·8440	0·8720	0·8881	0·9264	0·9464	0·9817
0·14	Upper	1·828	1·360	1·252	1·202	1·109	1·072	1·020
	Lower	0·7545	0·8324	0·8622	0·8794	0·9204	0·9420	0·9801
0·16	Upper	1·915	1·392	1·274	1·219	1·118	1·078	1·022
	Lower	0·7400	0·8218	0·8532	0·8713	0·9149	0·9380	0·9787
0·18	Upper	2·001	1·423	1·295	1·235	1·126	1·084	1·023
	Lower	0·7267	0·8118	0·8447	0·8638	0·9097	0·9341	0·9773
0·20	Upper	2·088	1·454	1·316	1·251	1·135	1·089	1·025
	Lower	0·7142	0·8024	0·8367	0·8566	0·9048	0·9304	0·9760
0·3	Upper	2·535	1·604	1·413	1·326	1·172	1·113	1·031
	Lower	0·6618	0·7620	0·8019	0·8253	0·8828	0·9139	0·9701
0·4	Upper	3·023	1·754	1·508	1·398	1·207	1·135	1·037
	Lower	0·6202	0·7287	0·7728	0·7989	0·8639	0·8996	0·9648
0·5	Upper	3·567	1·906	1·602	1·469	1·240	1·156	1·042
	Lower	0·5855	0·6999	0·7472	0·7756	0·8470	0·8867	0·9600
0·6	Upper	4·182	2·065	1·698	1·540	1·273	1·175	1·047
	Lower	0·5556	0·6743	0·7242	0·7545	0·8313	0·8741	0·9554
0·7	Upper	4·879	2·232	1·796	1·612	1·306	1·196	1·052
	Lower	0·5294	0·6512	0·7033	0·7350	0·8168	0·8632	0·9511
0·8	Upper	5·673	2·409	1·898	1·686	1·338	1·215	1·057
	Lower	0·5063	0·6302	0·6839	0·7170	0·8030	0·8525	0·9470
0·9	Upper	6·584	2·596	2·004	1·762	1·371	1·235	1·062
	Lower	0·4855	0·6107	0·6659	0·7001	0·7899	0·8421	0·9429
1·0	Upper	7·622	2·796	2·114	1·840	1·404	1·254	1·067
	Lower	0·4670	0·5928	0·6490	0·6841	0·7774	0·8322	0·9389
1·1	Upper	8·816	3·010	2·230	1·921	1·437	1·274	1·071
	Lower	0·4501	0·5760	0·6331	0·6690	0·7654	0·8226	0·9351
1·2	Upper	10·18	3·238	2·351	2·005	1·471	1·294	1·076
	Lower	0·4349	0·5604	0·6181	0·6547	0·7538	0·8133	0·9313
1·3	Upper	11·75	3·484	2·478	2·092	1·506	1·314	1·080
	Lower	0·4210	0·5458	0·6039	0·6410	0·7426	0·8042	0·9276
1·4	Upper	13·55	3·747	2·612	2·183	1·540	1·334	1·085
	Lower	0·4083	0·5321	0·5904	0·6280	0·7318	0·7954	0·9240
1·5	Upper	15·61	4·029	2·753	2·278	1·576	1·354	1·089
	Lower	0·3968	0·5192	0·5776	0·6156	0·7214	0·7868	0·9203
1·6	Upper	17·98	4·332	2·901	2·376	1·613	1·374	1·094
	Lower	0·3863	0·5071	0·5655	0·6036	0·7112	0·7784	0·9168
1·7	Upper	20·70	4·659	3·058	2·479	1·650	1·395	1·098
	Lower	0·3766	0·4956	0·5539	0·5922	0·7014	0·7702	0·9133

TABLE B—*Continued*

V	Limit	$n=5$	$n=10$	$n=15$	$n=20$	$n=50$	$n=100$	$n=1{,}000$
1·8	Upper	23·83	5·010	3·223	2·586	1·688	1·416	1·103
	Lower	0·3678	0·4848	0·5428	0·5813	0·6918	0·7622	0·9098
1·9	Upper	27·41	5·389	3·396	2·698	1·728	1·438	1·107
	Lower	0·3598	0·4746	0·5323	0·5708	0·6825	0·7544	0·9064
2·0	Upper	31·51	5·796	3·580	2·815	1·767	1·459	1·112
	Lower	0·3525	0·4650	0·5222	0·5607	0·6734	0·7466	0·9030
2·1	Upper	36·21	6·236	3·774	2·938	1·808	1·481	1·116
	Lower	0·3458	0·4559	0·5126	0·5509	0·6646	0·7391	0·8996
2·2	Upper	41·60	6·709	3·980	3·066	1·850	1·504	1·121
	Lower	0·3397	0·4472	0·5033	0·5416	0·6560	0·7317	0·8962
2·3	Upper	47·77	7·220	4·197	3·201	1·893	1·526	1·125
	Lower	0·3342	0·4391	0·4945	0·5325	0·6476	0·7245	0·8929
2·4	Upper	54·83	7·771	4·427	3·342	1·937	1·549	1·130
	Lower	0·3292	0·4314	0·4860	0·5238	0·6394	0·7173	0·8896
2·5	Upper	62·92	8·365	4·670	3·488	1·982	1·572	1·134
	Lower	0·3246	0·4241	0·4779	0·5155	0·6314	0·7104	0·8864
2·6	Upper	72·16	9·006	4·928	3·642	2·029	1·596	1·139
	Lower	0·3206	0·4172	0·4701	0·5074	0·6236	0·7035	0·8831
2·7	Upper	82·73	9·698	5·200	3·804	2·076	1·620	1·144
	Lower	0·3169	0·4107	0·4627	0·4997	0·6160	0·6967	0·8799
2·8	Upper	94·80	10·44	5·488	3·974	2·125	1·645	1·148
	Lower	0·3137	0·4046	0·4556	0·4992	0·6085	0·6901	0·8767
2·9	Upper	108·6	11·25	5·794	4·151	2·175	1·670	1·153
	Lower	0·3108	0·3988	0·4488	0·4849	0·6012	0·6836	0·8736
3·0	Upper	124·3	12·12	6·118	4·337	2·226	1·695	1·158
	Lower	0·3083	0·3933	0·4422	0·4780	0·5941	0·6772	0·8704
3·1	Upper	142·3	13·06	6·460	4·532	2·279		
	Lower	0·3062	0·3881	0·4360	0·4712	0·5872		
3·2	Upper	162·8	14·08	6·824	4·737	2·333		
	Lower	0·3043	0·3832	0·4300	0·4648	0·5804		
3·3	Upper	186·2	15·18	7·209	4·952	2·388		
	Lower	0·3028	0·3786	0·4243	0·4585	0·5738		
3·4	Upper	212·8	16·36	7·617	5·177	2·445		
	Lower	0·3016	0·3743	0·4188	0·4525	0·5673		
3·5	Upper	243·1	17·64	8·050	5·413	2·504		
	Lower	0·3006	0·3702	0·4135	0·4466	0·5609		
3·6	Upper	277·6	19·03	8·509	5·662	2·564		
	Lower	0·3000	0·3664	0·4085	0·4410	0·5547		
3·7	Upper	316·9	20·52	8·996	5·922	2·626		
	Lower	0·2996	0·3628	0·4037	0·4356	0·5486		
3·8	Upper	361·6	22·14	9·512	6·196	2·689		
	Lower	0·2994	0·3595	0·3990	0·4304	0·5427		
3·9	Upper	412·4	23·89	10·06	6·483	2·754		
	Lower	0·2995	0·3564	0·3946	0·4253	0·5369		
4·0	Upper	470·1	25·78	10·64	6·785	2·821		
	Lower	0·2999	0·3535	0·3904	0·4205	0·5312		
4·1	Upper	535·7	27·82	11·26	7·102			
	Lower	0·3004	0·3508	0·3864	0·4158			
4·2	Upper	610·1	30·03	11·91	7·435			
	Lower	0·3013	0·3483	0·3826	0·4113			
4·3	Upper	694·6	32·41	12·61	7·784			
	Lower	0·3023	0·3460	0·3789	0·4069			
4·4	Upper	790·4	34·99	13·35	8·152			
	Lower	0·3036	0·3439	0·3754	0·4027			

TABLE B—*Continued*

V	Limit	n = 5	n = 10	n = 15	n = 20
4·5	Upper Lower	899·1 0·3050	37·78 0·3419	14·13 0·3721	8·539 0·3987
4·6	Upper Lower	1022 0·3067	40·80 0·3402	14·96 0·3689	8·945 0·3948
4·7	Upper Lower	1162 0·3086	44·06 0·3386	15·85 0·3659	9·372 0·3910
4·8	Upper Lower	1320 0·3108	47·59 0·3372	16·79 0·3631	9·822 0·3874
4·9	Upper Lower	1499 0·3131	51·40 0·3360	17·79 0·3604	10·29 0·3840
5·0	Upper Lower	1701 0·3156	55·53 0·3349	18·85 0·3578	10·79 0·3806
5·1	Upper Lower	1930 0·3184	59·99 0·3339	19·97 0·3554	
5·2	Upper Lower	2189 0·3213	64·81 0·3332	21·17 0·3531	
5·3	Upper Lower	2482 0·3245	70·03 0·3326	22·44 0·3510	
5·4	Upper Lower	2812 0·3279	75·68 0·3321	23·80 0·3489	
5·5	Upper Lower	3186 0·3315	81·78 0·3318	25·24 0·3470	
5·6	Upper Lower	3608 0·3353	88·39 0·3316		
5·7	Upper Lower	4084 0·3393	95·53 0·3315		
5·8	Upper Lower	4621 0·3436	103·2 0·3316		
5·9	Upper Lower	5227 0·3480	111·6 0·3318		
6·0	Upper Lower	5910 0·3527	120·6 0·3322		

TABLE C

FACTOR $\Psi_{\cdot 10}(V,n)$ FOR THE ESTIMATION OF THE LOWER 90 PER CENT CONFIDENCE LIMIT OF THE MEAN OF A LOGNORMAL POPULATION

V	$n = 5$	$n = 10$	$n = 15$	$n = 20$	$n = 50$	$n = 100$	$n = 1{,}000$
0·00	1·0000	1·0000	1·0000	1·0000	1·0000	1·0000	1·0000
0·02	0·9190	0·9466	0·9568	0·9626	0·9760	0·9828	0·9943
0·04	0·8882	0·9256	0·9396	0·9476	0·9662	0·9757	0·9918
0·06	0·8656	0·9098	0·9266	0·9362	0·9587	0·9702	0·9900
0·08	0·8470	0·8967	0·9157	0·9266	0·9523	0·9656	0·9884
0·10	0·8310	0·8853	0·9062	0·9182	0·9467	0·9614	0·9870
0·12	0·8169	0·8751	0·8976	0·9106	0·9416	0·9577	0·9857
0·14	0·8041	0·8657	0·8897	0·9037	0·9369	0·9542	0·9845
0·16	0·7923	0·8571	0·8825	0·8972	0·9325	0·9510	0·9834
0·18	0·7816	0·8491	0·8756	0·8911	0·9284	0·9479	0·9823
0·20	0·7715	0·8415	0·8692	0·8854	0·9244	0·9450	0·9813
0·3	0·7289	0·8088	0·8411	0·8602	0·9070	0·9319	0·9766
0·4	0·6950	0·7817	0·8175	0·8389	0·8918	0·9205	0·9725
0·5	0·6667	0·7583	0·7968	0·8200	0·8782	0·9102	0·9688
0·6	0·6423	0·7374	0·7781	0·8029	0·8657	0·9000	0·9652
0·7	0·6211	0·7186	0·7611	0·7871	0·8540	0·8915	0·9618
0·8	0·6023	0·7014	0·7453	0·7724	0·8429	0·8829	0·9585
0·9	0·5856	0·6856	0·7306	0·7587	0·8324	0·8746	0·9554
1·0	0·5708	0·6710	0·7168	0·7457	0·8223	0·8667	0·9523
1·1	0·5574	0·6574	0·7039	0·7334	0·8126	0·8590	0·9492
1·2	0·5455	0·6448	0·6917	0·7218	0·8032	0·8515	0·9463
1·3	0·5348	0·6330	0·6802	0·7106	0·7942	0·8442	0·9434
1·4	0·5252	0·6219	0·6692	0·7000	0·7855	0·8371	0·9405
1·5	0·5167	0·6116	0·6588	0·6899	0·7770	0·8302	0·9376
1·6	0·5091	0·6022	0·6490	0·6802	0·7688	0·8235	0·9348
1·7	0·5024	0·5930	0·6396	0·6710	0·7608	0·8169	0·9321
1·8	0·4965	0·5843	0·6307	0·6621	0·7530	0·8104	0·9293
1·9	0·4913	0·5763	0·6222	0·6535	0·7454	0·8041	0·9266
2·0	0·4869	0·5689	0·6141	0·6453	0·7381	0·7978	0·9240
2·1	0·4831	0·5619	0·6064	0·6375	0·7309	0·7918	0·9213
2·2	0·4800	0·5553	0·5990	0·6299	0·7239	0·7858	0·9187
2·3	0·4774	0·5492	0·5920	0·6226	0·7171	0·7799	0·9160
2·4	0·4755	0·5436	0·5854	0·6157	0·7104	0·7741	0·9135
2·5	0·4741	0·5383	0·5790	0·6090	0·7039	0·7685	0·9109
2·6	0·4732	0·5334	0·5730	0·6026	0·6976	0·7629	0·9083
2·7	0·4728	0·5288	0·5673	0·5964	0·6914	0·7574	0·9058
2·8	0·4729	0·5246	0·5618	0·5904	0·6853	0·7520	0·9033
2·9	0·4735	0·5208	0·5567	0·5848	0·6794	0·7467	0·9008
3·0	0·4746	0·5173	0·5518	0·5793	0·6736	0·7416	0·8983
3·1	0·4761	0·5140	0·5471	0·5740	0·6680		
3·2	0·4781	0·5112	0·5427	0·5690	0·6624		
3·3	0·4804	0·5085	0·5386	0·5642	0·6570		
3·4	0·4833	0·5062	0·5346	0·5596	0·6517		
3·5	0·4865	0·5042	0·5310	0·5552	0·6466		
3·6	0·4902	0·5024	0·5275	0·5509	0·6416		
3·7	0·4942	0·5009	0·5242	0·5469	0·6366		
3·8	0·4987	0·4997	0·5212	0·5430	0·6318		
3·9	0·5036	0·4987	0·5184	0·5393	0·6271		
4·0	0·5089	0·4980	0·5157	0·5358	0·6225		
4·1	0·5147	0·4976	0·5133	0·5325			
4·2	0·5208	0·4973	0·5110	0·5293			
4·3	0·5274	0·4973	0·5090	0·5263			
4·4	0·5343	0·4976	0·5071	0·5234			
4·5	0·5417	0·4981	0·5054	0·5207			
4·6	0·5496	0·4988	0·5039	0·5182			
4·7	0·5579	0·4997	0·5026	0·5158			
4·8	0·5666	0·5009	0·5014	0·5135			
4·9	0·5758	0·5022	0·5004	0·5114			
5·0	0·5854	0·5039	0·4996	0·5095			
5·1	0·5955	0·5057	0·4989				
5·2	0·6061	0·5078	0·4984				
5·3	0·6172	0·5100	0·4981				
5·4	0·6288	0·5125	0·4979				
5·5	0·6409	0·5152	0·4979				
5·6	0·6536	0·5182					
5·7	0·6668	0·5213					
5·8	0·6806	0·5247					
5·9	0·6949	0·5283					
6·0	0·7098	0·5321					

(Tables taken from Sichel, 1966)