

A Thesis Entitled

GEOLOGICAL CONTROL AND THE APPLICATION OF
GEOSTATISTICS TO MINERAL EVALUATION

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in the

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by

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Last, but not least, sincere thanks are due to Dr. J. O'Leary for introducing me to geostatistics and without whose help and guidance this project would not have been made possible.

CHAPTER 1

INTRODUCTION

The purpose of this study is not simply to provide a mineral valuation from a series of drill hole data but to try and discover how an estimation procedure may be improved by considering the geological factors controlling the location and formation of the deposit. In many cases such factors are not considered, with the result that large errors of estimation are produced which lead to gross misclassifications of material once mining commences. It is believed that the errors of estimation may be reduced if a little thought and consideration is taken regarding the geological processes affecting the sample values contained within the deposit.

It is essential to have a series of good computer programs available for use before any mineral evaluation program can be attempted. This may lead to purchasing a series of programs or to writing them oneself. Money was not available for this study. This resulted in the establishment of a series of programs known as the 'GEOSTAT' Library (a more detailed description is given in the accompanying appendix) which took approximately one year to write. One of the fundamental problems with many geostatistics programs is that they are designed for use around a specific type of data set. 'GEOSTAT' was written to try to allow it to be used with any type of data set and to allow the user a greater control of the program by removing most of the machine dependent decisions. The content of the library is such that it is possible to employ many of the geostatistical techniques including statistics, variograms, kriging and grade tonnage curves. All of the programs were tested against standard programs to ensure that they

produced the correct results. Where this was not possible they were run with simulated data sets with known properties.

A period of eight months was spent working with the technical services group of Rio Tinto Services (RTZ) during 1980 in preparation for a feasibility study for the Cerro Colorado deposit, Panama. This provided an ample opportunity to use the 'GEOSTAT' library under testing conditions and also to try and impose a geological control upon the process of mineral evaluation, since this was a prime objective of the study. A total of three ore reserves were produced for copper, and one for molybdenum during the given period. These compared favourably with the previous estimates.

It was pointless to return to Leicester University to repeat the whole procedure and so it was suggested that it may be worthwhile to produce a reserve based upon sample values composited over a distance of 15 metres as opposed to 3 metres. Attempts were made to produce estimates for copper, molybdenum, silver and gold using similar methods to those used in the previous study in order to allow direct comparison to be made between them.

It is worthwhile to give a brief description of the Cerro Colorado deposit and its history of exploration before describing the geological and geostatistical problems involved within the mineral evaluation programme.

1.1 The Cerro Colorado deposit

The Cerro Colorado porphyry copper deposit lies approximately 250 kilometres to the south-west of Panama City in the province of Chiriqui (Fig. 1.1). The surrounding topography is very immature, rising to an elevation of approximately 1500 metres above sea level. Cerro Colorado itself is a very steep-sided ridge extending

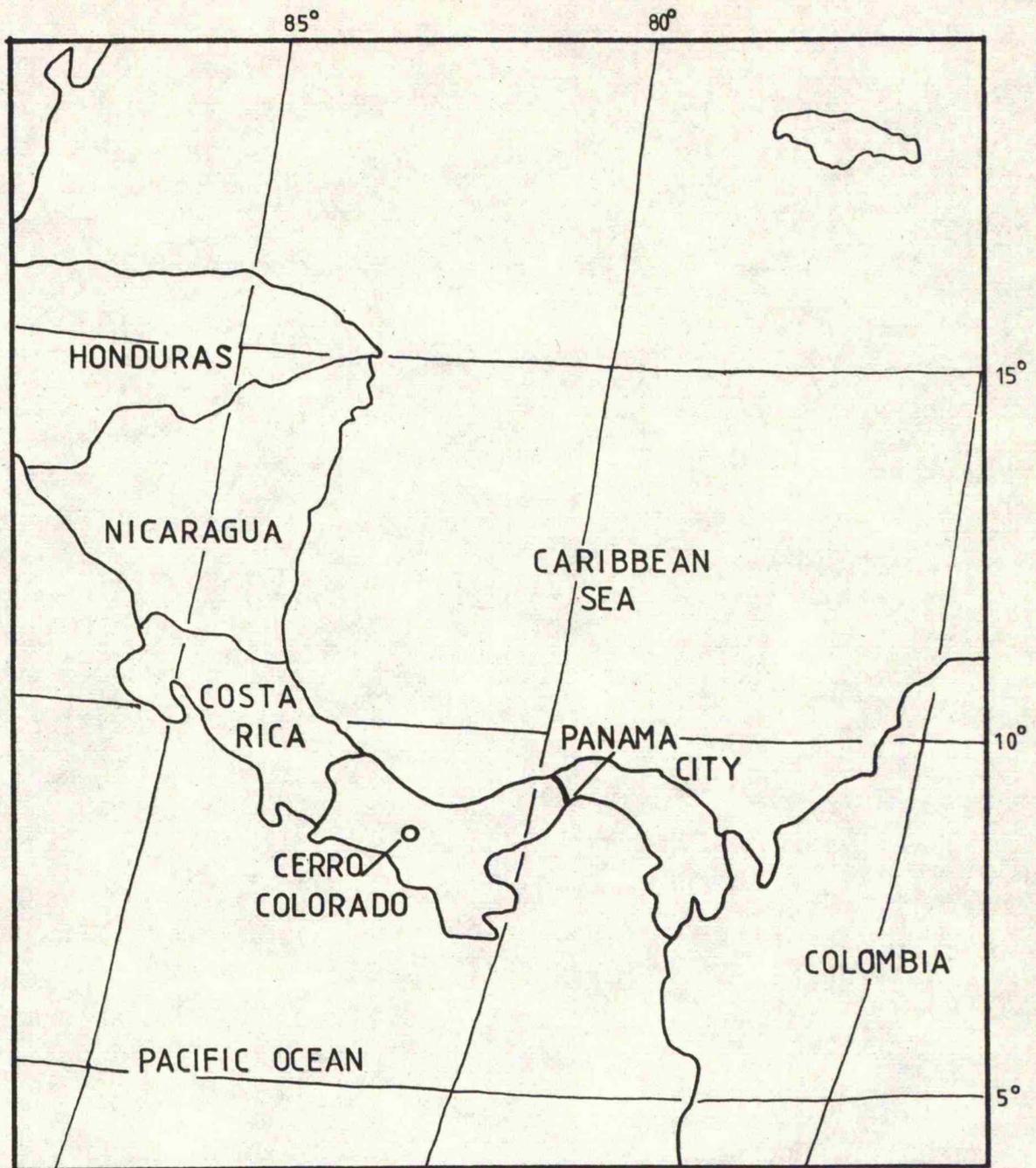


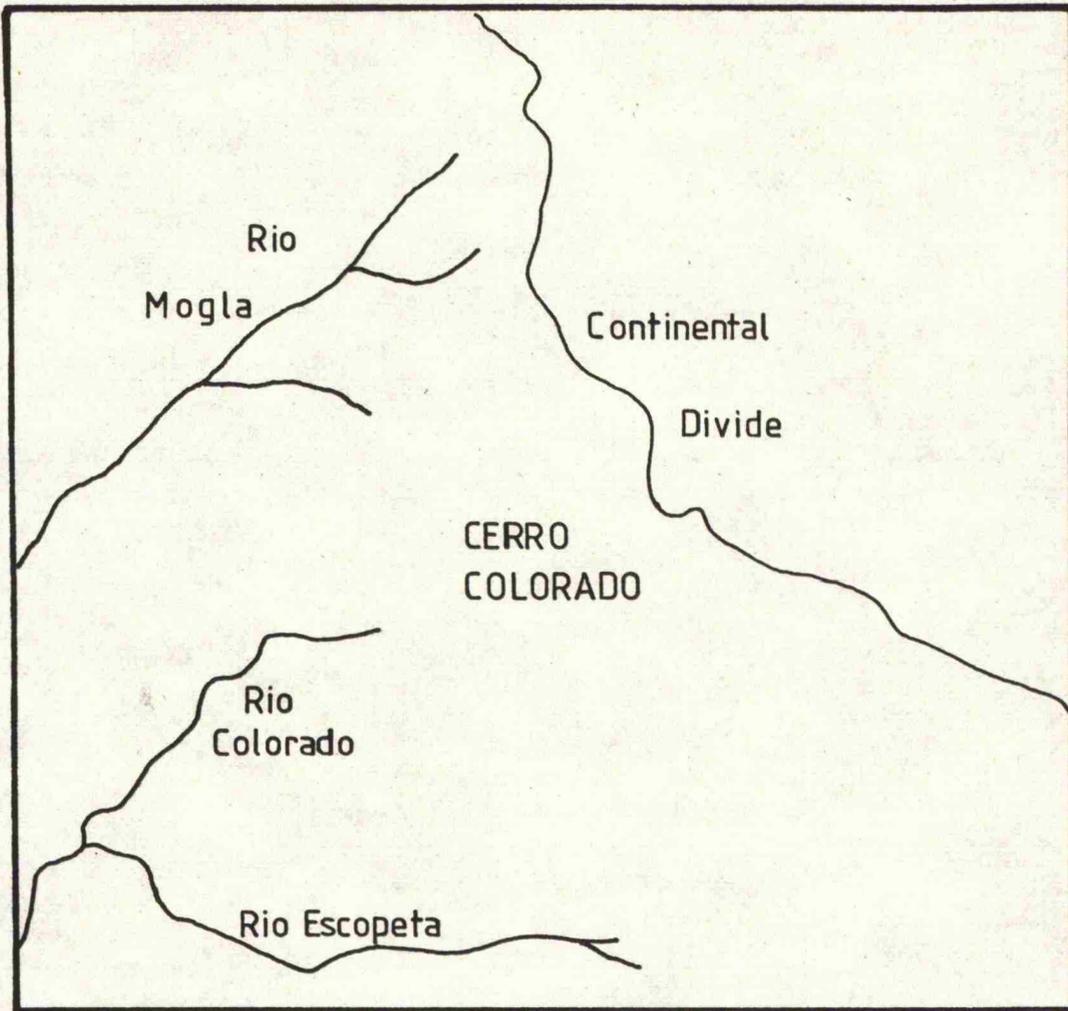
Fig. 1.1 Location of Cerro Colorado

approximately 2 kilometres in a south-westerly direction from the continental divide. It is bounded by the Rio Colorado to the north and by the Rio Escopeta to the south. Thick jungle is present on the Atlantic side (northern side) of the continental divide. The valleys are steep-walled and suffer considerable erosion during the rainy season, making exploration very difficult. To the south the mountains give way to rolling hills with wide valleys and grassland as opposed to jungle. The climate is pleasant with an average annual temperature of 23° C. and an average rainfall of approximately 3500 millimetres per year. It is less humid than in Panama City, though in the dry season it can be very windy.

1.2 History of exploration

Cerro Colorado is located approximately 30 kilometres from the nearest village of San Felix. Consequently, the only means of access to the deposit prior to its recent development was on horseback or on foot along the Indian trails. As a result, few people visited the area and this is one of the reasons why this deposit has only been recently discovered.

Mineralisation was first noticed in the form of malachite and chalcopyrite veins during a reconnaissance trip through Western Panama in 1936 by Robert Terry - a geologist for Sinclair Panama Oil Company. Many years later a Panamanian prospector contacted Terry about the location of the copper mineralisation in the area both in and around the Rio San Felix. In 1956 after a series of unsuccessful attempts to reach the area, copper mineralisation was found. Interests were quickly gained within the area following the discovery of a large boulder of massive chalcopyrite in the stream bed of the Rio Escopeta (Fig. 1.2) during March 1957. Subsequent visits were made to the area and in a



0 1000

Scale (metres)

Fig. 1. 2. Principal rivers around Cerro Colorado

report issued in June of that year it was suggested that there were approximately 100 million tons of copper ore of a grade of between 0.05 and 0.5 percent copper contained within a volume of 4000 by 300 by 1200 feet deep in the vicinity of Cerro Colorado. Exploration concessions were quickly established though were soon relinquished due to a lack of interest in the area by many mining companies to further the exploration work.

No further work was done until 1965 when the United Nations undertook a joint mineral exploration programme based upon stream sediment geochemistry. As a result of this survey a porphyry copper deposit was discovered 130 kilometres north-east of Cerro Colorado at Petaquilla. Interest was soon regained in the Cerro Colorado deposit and in November 1970 a 759 square kilometre concession was granted to Canadian Javelin Ltd. Test drilling began in 1971, giving encouraging results. Larger drills were soon brought into the area and drill hole number 43 was the first hole to penetrate the leached cap, indicating the possibility of a major copper deposit. Drilling ceased in November 1974 after the completion of 90 holes of a total length of 42,667 metres.

During 1973 the Panamanian Government formed a corporation by the name of Codemin to negotiate with Canadian Javelin the terms for the development of the Cerro Colorado deposit. Both parties failed to reach an agreement with the result that Canadian Javelin were reimbursed for their expenditure. Interests remained dormant until mid 1975 when Texasgulf became involved with the Cerro Colorado deposit. Further exploration work was undertaken to try and establish the limits of mineralisation. This included more drilling and an adit to provide samples for metallurgical test work. The feasibility study was concluded in May 1978.

Rio Tinto Zinc Corporation (RTZ) acquired Texasgulf's interest in the project in May 1980. A further study was undertaken involving drilling, metallurgical test work and engineering studies. The conclusion reached was that there is a reasonable probability that the orebody could be developed on a large scale.

At present the future of the Cerro Colorado project still hangs in the balance and negotiations are still proceeding between Rio Tinto Zinc and Codemin. Hopefully, some agreement may be reached whereby mining operations will commence by 1985.

CHAPTER 2

2. The use of geostatistics in mineral evaluation

Geostatistics is a relatively new method of valuing mineral deposits, developed over the past twenty years, and has been described as 'the application of the theory of regionalised variables to the study of mineralised volumes of rock and all the considerations arising from this' (Royle, 1977A). Prior to the advent of geostatistics and the work of Matheron, geometrical and statistical techniques were used with the assumption that the sample values taken from a mineralised volume were randomly distributed. However, this is very rarely the case. Sample values are directly influenced by geological processes and can thus be considered as regionalised variables whose values are a function of their position within the deposit. Any sample taken at position x within the deposit will therefore have an assay value $F(x)$, where $F(x)$ is a function of (x) . Nevertheless, many mining companies still employ the older, traditional methods which do not take this into consideration. In some cases they provide reliable results for their particular method of operation and therefore have no reason to change to geostatistical methods. It must be emphasised that geostatistics is not the only method that can be used in a mineral evaluation programme and can only be applied where the type of data available suits a geostatistical investigation.

A brief description of the traditional methods is given, not so much to emphasise their pitfalls, but to illustrate how geostatistical methods may be used to overcome some of their problems.

2.1 Statistical methods

Methods using classical statistics assume that the sample values are randomly distributed and are concerned with their magnitudes, not their position. The mean grade of an area is calculated from the mean of the sample values and the average variability of the sample values is given by their variance. Histograms can be drawn to illustrate the frequency distribution of the sample values.

2.2 Geometrical methods

Geometrical methods are a simple expansion of the classical statistical methods. Basically they attempt to involve the sample locations as well as their sample values within the calculations though in a very simple manner. Numerous methods exist.

2.2.1 The polygonal method

This method is widely used today, often as a check against geostatistical estimates, and is used to estimate the total reserves of a mineralised area. The deposit is divided into a series of polygons centred upon individual samples by the perpendicular bisectors of lines drawn between the sampling points (Fig. 2.1). Each of the polygons is assumed to have an average grade value equal to that of the central sample. The mean grade of the area is obtained by weighting the polygon by its area of influence to the total area of interest. This can be a tedious method, especially if the samples have been taken from an irregular grid, meaning that each polygonal area has to be individually measured. The errors of estimation will vary according to the size of the polygon and a wide range of polygon areas will increase the spread of errors.

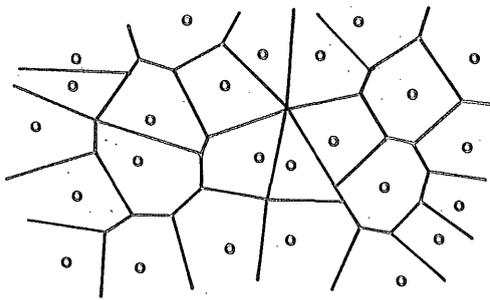


Fig. 2.1 The Polygonal method

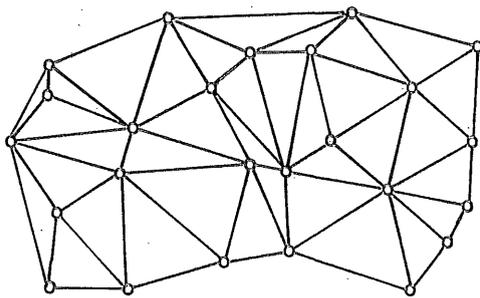


Fig. 2.2 The Triangular method

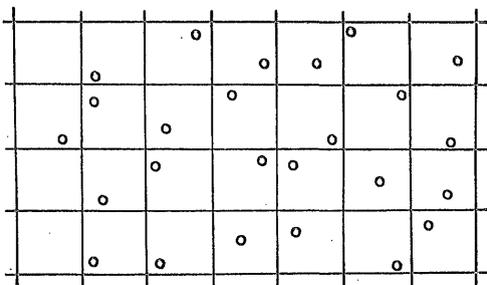


Fig. 2.3 The Random stratified grid

(after Royle, 1977)

2.2.2 The triangle method

This method is very similar to that of the polygonal method and is very simple to use. Sample points are joined together to form triangular polygons (Fig. 2.2) and the value assigned to each triangle is either the mean or a weighted mean of the values at each corner. In this case the individual triangles are valued with less error than the polygonal method since more sample values are involved in the estimation. The average grade of the area is calculated in the same way as for the polygonal method and the degree of error will be reduced if a regular sample grid is used.

2.2.3 Random Stratified Grids (RSG)

This is a more recent development and can be used when a sampling pattern approximates to a regular grid but does not quite attain it due to variations in dip and strike and drill hole deflections (Royle, 1978). A regular grid is fitted to the samples in such a way that each grid panel contains only one sample value which has a random position within the panel when compared with the other grid panels and their sample values (Fig. 2.3). The size of grid can be obtained by dividing the total area of interest by the number of samples contained within it in order to establish the area of one panel. The ratio of the sides of the panel can be obtained by examining the sampling pattern. It is often easier to fit a random stratified grid by eye, the aim being to have only one sample value within each grid panel. If two or more samples exist within a grid panel their arithmetic mean is taken. In some cases panels exist which contain no sample values but are surrounded by panels with samples. It is then supposed that these surrounding values can be projected into the panel and their arithmetic mean assigned to it.

This method differs from the polygonal and triangular method in so much that all of the panels have the same dimensions thus eliminating the need for the numerous tedious area calculations. The degree of error is minimised since the sample values are extended over the same area of influence.

2.2.4 Contours and sections

Sections and plans of the distribution of the sample values can be drawn at regular intervals throughout the area of interest. The sample values are usually contoured by hand or by computer. A planimeter is used to measure the area between the contour boundaries. The value assigned to this area is that of the arithmetic mean of the two contours. This is a very crude method though often effective in simple cases.

2.3 Other methods

The other types of methods usually combine statistical and geometrical techniques and are discussed below.

2.3.1 Distance weighting

This method became popular with the introduction of the computer mainly because a large number of repetitive calculations are required. A block or panel is assigned a combination of the surrounding values, weighted for example by their distance or distance squared, from the block centre. The question now arises as to the maximum and minimum number of samples to be included in the calculation and how far away the furthest sample can be. Generally, as the number of sample values increases, so their influence approximates to $1/n$, where n is the number of sample values. However, there are no real solutions to the problem and the user's experience has to be relied upon.

2.3.2 Trend surfaces

Trend surfaces involve the fitting of a least squares curvilinear surface to the sample data in which the geographical co-ordinates of the sample values can be used to estimate the sample value. Any trends within the data are highlighted and modelled by a simple equation. This technique can be easily applied to relatively simple deposits, e.g. coal.

2.4 Errors of estimation

All of the methods mentioned so far are able to assign an estimated value of grade to an area, but how accurate is this estimate? The only way to establish this is to actually mine the whole area. Therefore, the optimum method of evaluation will be the one which produces the least error. In many of the techniques a single value is assigned and projected into an area many times greater than itself. No account is taken of the possibilities of sampling errors and if they exist will be greatly magnified. An error of estimation is present which is simply the difference between the actual and estimated values. The estimation variance is the difference in value squared and the smaller this value is, the better the estimate. In practice, the actual error of estimation will never be known, and if it were, it would never exist since it could be adjusted for. Errors of estimation can be minimised by involving more than one sample value in the estimation, to try and eliminate any unforeseen errors such as poor sampling and misclassification of the sample values.

2.5 The geostatistical concept

None of the methods mentioned so far would appear to be an optimum method of estimation since there are numerous ways in which errors can

be introduced. It seems reasonable to try and use a method which is based upon the fact that the sample values are a function of their positions within the deposit. However, the spatial behaviour of the values can be complex and virtually impossible to model exactly, though a close approximation may suffice. Areas exist within the deposit where the sample values are predominantly high. Other areas are low in value suggesting that it is likely that the values of two samples taken close together would be more similar than if they were taken far apart. It would appear that the sample values are correlated to each other by means of a function which is dependant upon the distance apart of the sample values. However, low grade values exist within the high value areas and high grade values exist in the low value areas, suggesting that there is a random or 'local' influence which must also be considered. This means that two samples taken close together will not necessarily have the same value, e.g. two halves of a drill core producing different assay values. The total variation between two sample values can now be considered to be the sum of the local random variation and the spatial variation. Traditional methods do not take these factors into account but simply extend sample values into a given area regardless of any spatial or structural influences.

2.5.1 Calculation of the sample variability - the variogram

The variability between a set of sample values can be measured by the variogram function. Consider two sample values, $Z(x)$ and $Z(x + h)$, located at positions x and $x + h$. The difference in value is $(Z(x) - Z(x + h))$. In practice, it is better to compare the squares of the difference of the values and since many sample pairs will be found at distances of h metres apart the average can be calculated. This forms the basis of the semi-variogram function $\gamma(h)$ and is given by the equation:

$$\gamma(h) = \frac{1}{n} \sum_{i=1}^n [F(x_i) - F(x_i + h)]^2$$

where n is the number of sample pairs available.

A semi-variogram will be produced by plotting the semi-variance $\gamma(h)$ against its respective lag distance h (Fig. 2.4). In most cases the semi-variogram will approximate to a spherical scheme variogram (Fig. 2.5), although other types exist but are less common (Royle, 1977, David, 1977). Since the semi-variogram function directly compares the sample values it will consider both the spatial and local components within the sample variation.

The variogram has three properties:

1. The nugget variance (C_0): in most cases the variogram will not tend to zero at lag 0 but will intercept the Y-axis at some positive value [$\gamma(0) \geq 0$]. This part represents the random component of the sample variation and is often the result of poor analytical techniques, erratic mineralisation or any structures which may occur over a distance smaller than that of the lag interval (Journel).
2. The continuity (C): the continuity of the variogram is given by the rate of increase of the semi-variance with lag distance h. This gives an indication of the rate at which the influence of a sample decreases away from the sample site. No continuity (Fig. 2.6) suggests that the samples are independent of each other whatever their distance apart for that particular lag interval. Reducing the lag interval may eliminate this.

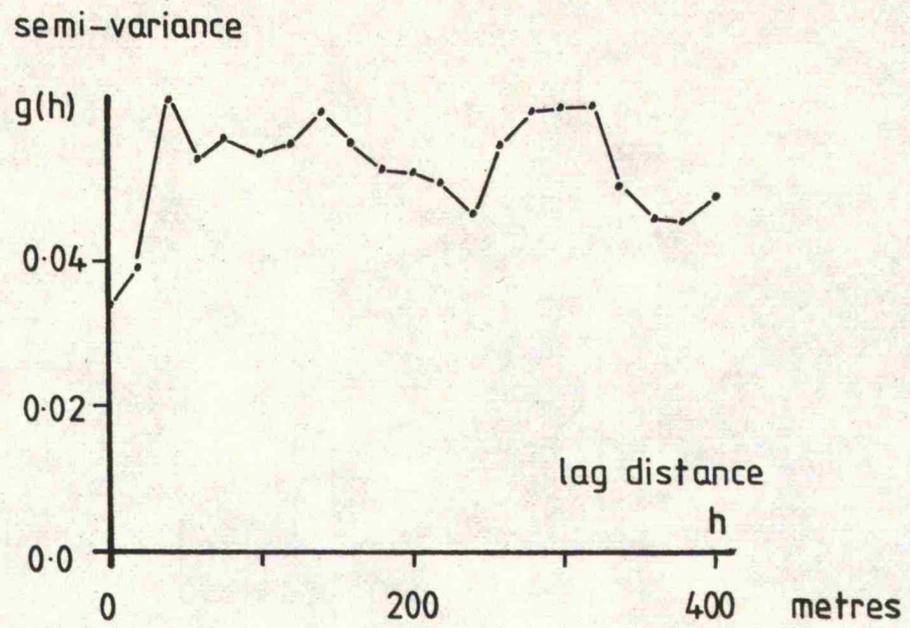


Fig. 2.4 The Semi-variogram

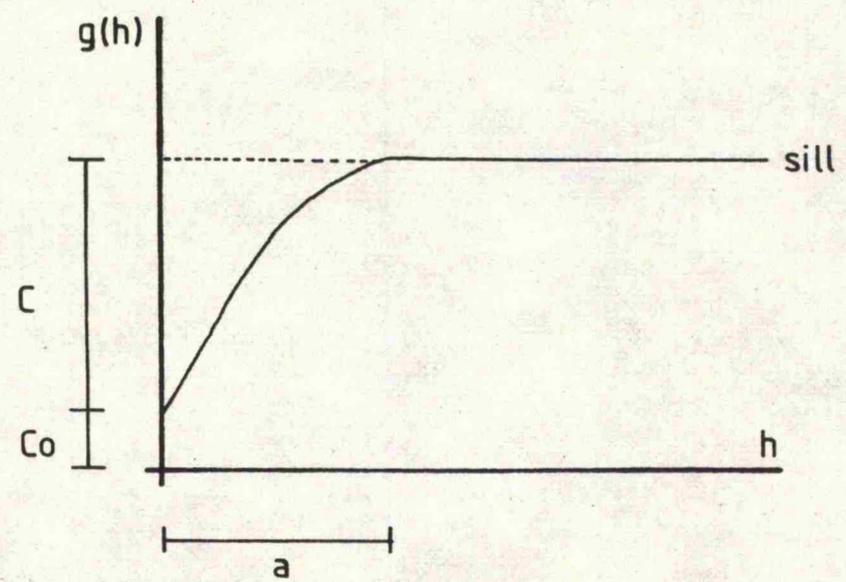


Fig.2.5 The Spherical scheme variogram

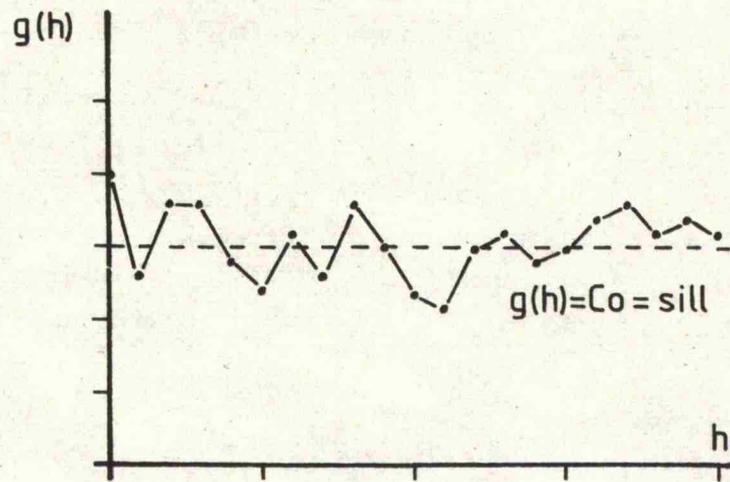


Fig. 2.6 Pure nugget effect

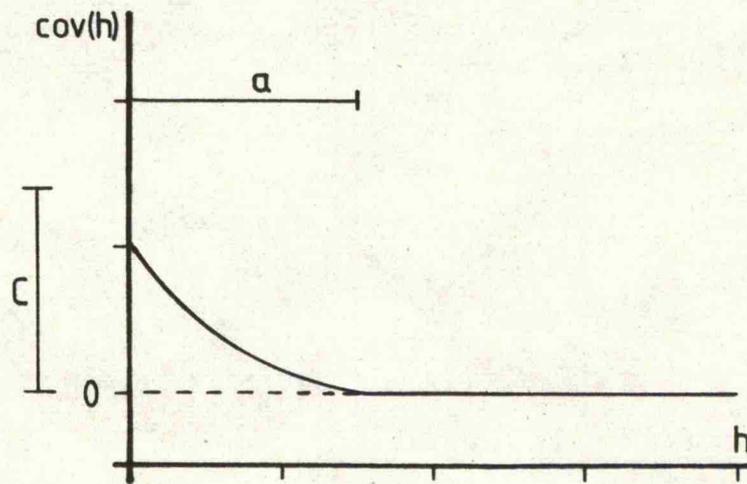


Fig. 2.7 The Covariogram

3. The range of the variogram (a): a point is reached in the variogram where the continuity is zero and reaches a plateau. The distance at which this is seen to occur within the variogram is an indication of the distance beyond which the sample values are no longer related to each other.

A directional semi-variogram can be produced by using only the sample values which lie in a specific direction with respect to each other. An angle of regularisation can be specified as an error margin to ensure that sufficient sample pairs are available for use. If semi-variograms are calculated in more than one direction, anisotropy within a data set may be detected. The variogram is said to be isotropic if the range of influence of the sample values is constant for each direction.

2.5.2 The covariogram

The covariogram reflects the mean variability of the sample values about their common mean value as given by the equation:

$$\text{cov}(h) = \frac{1}{n} \sum_{i=1}^n \left([F(x_i) - m] [F(x_i + h) - m] \right)$$

where n is the number of sample pairs and m is the mean of all the sample values.

Like the variogram, the covariogram illustrates the degree of continuity and range of the sample values (Fig. 2.7) but does not have a nugget effect. The range of influence of the sample values occurs where cov(h) is zero. The disadvantage of the covariogram is that it requires an estimate of the mean value of the deposit and it assumes that it is the same as that of the sample values. It is often unstable and difficult to model.

2.5.3 Drift

The drift between two sample values separated by a distance of h metres is given by the equation:

$$D(h) = \frac{1}{n} \sum_{i=1}^n [F(x) - F(x + h)]$$

where n is the number of sample pairs.

Drift is present if the value of D(h) is seen to increase either positively or negatively with distance h and represents the mean difference in sample values with distance. If a drift is present, a technique known as Universal Kriging (UK) may be employed to overcome this. Drift may be removed by fitting a trend surface or polynomial equation to the data and producing a variogram of the residuals. Simple kriging may then be performed upon the residuals.

2.5.4 The cross-variogram

A cross-variogram measures the spatial continuity between two variables, A and B, for increasing distance apart. The cross-variance can be calculated from the equation:

$$\gamma_{cv}(h) = \frac{1}{n} \sum_{i=1}^n \left[[F(x_{1i}) - F(x_{1i} + h)] [F(x_{2i}) - F(x_{2i} + h)] \right]$$

$F(x_1)$ = value of the first variable
where n is the number of sample pairs. $F(x_2)$ = value of the second variable

If the two variables are spatially correlated with increasing distance h, the resulting cross-variogram will have a similar appearance to that of the variogram.

2.5.5 Summary

To summarise, the variogram, covariogram and cross-variogram give an indication as to the extent of the local and spatial variation

between the sample values which can be considered when valuing a mineral deposit.

2.6 Kriging

Kriging is a method for evaluation which makes use of the variogram to calculate a series of weighting coefficients for the sample values which surround a block or panel whose mean value is to be estimated.

2.6.1 Producing a kriged estimate

Consider the sample pattern in Figure 2.8 in which the value of point A is required. All of the samples will have an influence on each other and will also exhibit different sample variations when compared with each other. The variogram demonstrates the average variability between each of the samples and is given by the equation:

$$\gamma(h)_{ij} = C_0 + c(h)$$

Similarly, the semi-variance between each of the sample values and the point A can be calculated. Kriging minimises the variance of estimation, i.e. the weighting coefficients are arranged in such a way as to obtain a weighted average which will have the smallest estimation variance.

In order to be unbiased, $E(z^*) = m$, which means that

$$E\left[\sum_{i=1}^n a_i z(x_i)\right] = m \quad \Rightarrow \quad \sum_{i=1}^n a_i E[z(x_i)] = m$$

and as $E[z(x_i)] = m$ then $\sum_{i=1}^n a_i = 1$ (David)

where a = the weighting coefficients
 m = the mean of the deposit
 z^* = the estimated value
 z = the actual value.

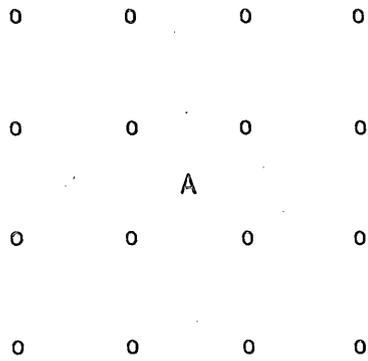


Fig. 2.8 Relationship between the known sample values (o) and the unknown sample value (A)

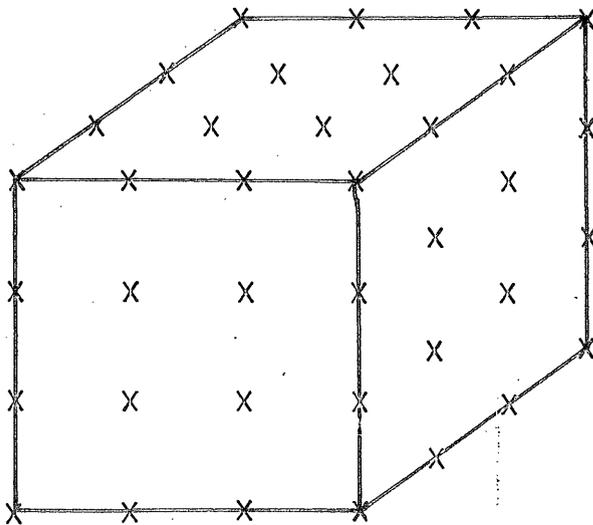


Fig. 2.9. Location of the theoretical points to be used to calculate the variance of the block

A series of simultaneous equations can now be established to calculate the weighting coefficients (a) and will have the form:

$$\begin{bmatrix}
 \gamma_{11} & \gamma_{12} & \gamma_{13} & \gamma_{14} & 1 \\
 \gamma_{21} & \gamma_{22} & \gamma_{23} & \gamma_{24} & 1 \\
 \gamma_{31} & \gamma_{32} & \gamma_{33} & \gamma_{34} & 1 \\
 \gamma_{41} & \gamma_{42} & \gamma_{43} & \gamma_{44} & 1 \\
 1 & 1 & 1 & 1 & 0
 \end{bmatrix}
 \begin{bmatrix}
 a_1 \\
 a_2 \\
 a_3 \\
 a_4 \\
 \mu
 \end{bmatrix}
 =
 \begin{bmatrix}
 \gamma_{A1} \\
 \gamma_{A2} \\
 \gamma_{A3} \\
 \gamma_{A4} \\
 1
 \end{bmatrix}$$

where the s-matrix contains the expected semi-variance between each of the samples

t-matrix contains the expected semi-variance between each of the samples and point A

l-matrix (lambda matrix) contains the solution for the respective weighting coefficients (a).

A series of 1's are placed in the final column and row of the s-matrix to ensure that the sum of the weighting coefficients is unity. The diagonal consists of a series of 0's because the expected semi-variance between a sample point and itself is zero. A multiplier (μ) must be added to the l-matrix in order to minimise the kriging variance. This is known as the Lagrange multiplier.

From this the value of A is:

$$z^* = \sum_{i=1}^n a_i z(x_i)$$

and the kriging variance:

$$\sigma_{kr}^2 = \sum_{i=1}^n a_i \gamma_{Ai} + \mu$$

2.7 The estimation of volume

The previous example demonstrates the valuation of a point value. In real terms this is of limited use since volumes of ground and not points are mined. Kriging must be able to estimate a block or panel of ground in order to have any real value.

If every sample possible were taken from within a block of ground and valued, then the mean grade of the block would be the mean of all the sample values. This mean value could then be considered to be representative of a single sample value situated at the centre of the block. This has to be assumed in order to value the block by a geostatistical method. Thus, instead of producing a t -matrix of the semi-variance between all points and the unknown point, a matrix of the semi-variance between each point and the block centre is used. The kriging process remains the same except for the calculation of the kriging variance.

Again, if every sample possible were taken from within the block it would be possible to calculate its mean semi-variance. This can be done by hand using tables (David, 1977) or by a computer using a series of theoretical samples at regular distances apart (Fig. 2.9). For each theoretical pair, their distance apart can be established and hence their expected semi-variance. The mean of the semi-variance for all sample pairs is that of the mean semi-variance of the block. Since every block is of the same dimensions each will have the same expected mean semi-variance or extension variance (λ), so called because point values are extended into a volume.

The kriging variance of the estimate for the block values must not only consider the variance of estimation of the block value but also variance of estimation of the actual samples within the block.

The kriging variance of the block is given by the equation:

$$\sigma_{kv}^2 = \sum_{i=1}^n a_i Y_{A_{iv}} + \mu - \lambda$$

It is now possible to produce an estimate and assign a degree of confidence to that estimate.

2.8 The effects of bias

It has been stated that the aim of kriging is to produce an unbiased estimate (David, 1977). This can only be achieved if the mean of the kriged estimates is the same as the mean of the actual values. A perfect estimator will produce a linear regression of actual against estimated values with a slope of '1' with all points lying on the line (Royle, 1978). In practice this can never be achieved due to irregularities within the data. The fact that blocks can be either under or over-valued means that they can be misclassified between ore and waste when applying a cut off grade. Obviously, these must be classified accurately for mine planning purposes and therefore have a minimum bias. In this respect, kriging is a better estimator than the traditional methods.

2.9 Advantages and disadvantages of kriging

Kriging is mistrusted by many people because its concepts are considered to be too mathematical and incomprehensible. This is unfortunate in so much that many people do not use the technique because they believe that they do not understand it and prefer to remain with the older, well established methods. Basically, the word 'Kriging' frightens some people. Compared with traditional techniques, kriged estimates are generally better since they are based upon the

actual sample values and their relationships with other samples. Similarly, considerations are taken to account for the extension of points into volumes many times greater than themselves. The greatest asset to kriging is that each estimate is assigned an error of estimation and when the volumes of ground remain constant the induced error by the method remains constant, making it a more meaningful estimate. Traditional methods are unable to assign an error of estimation. The advantage of the traditional methods is that they do not require extra, expensive equipment such as a computer for example, but just a ruler, a pencil and possibly a hand calculator. Computers are essential in order to be able to produce a kriged estimate and since many simultaneous equations have to be established and solved, computer time can be expensive. However, it is felt that the accuracy and speed at which results can be obtained justify the cost and, in general, these are small when compared with costs incurred during evaluation.

CHAPTER 3

3. The interface of geological processes and mineral evaluation

3.1 The need for a geological input

It must be remembered that geological processes are fundamental in the formation of a mineral deposit. These provide the means for allowing mineral or other matter to be concentrated in such a way that it may be economically viable to mine or remove it. Drill hole information is very costly to obtain and it is essential that as much information as possible is extracted from it, including geological information, e.g. structures. In many cases the influence of geological processes on the mineralisation within an area has not been considered and possibly many mistakes in evaluation could have been avoided if this had been so. This may be the result of the fact that many of the methods used for mineral evaluation have been devised and developed by mathematicians and statisticians who do not fully appreciate many of the geological problems involved. In order to try and emphasise some of the problems some of the possible geological controls are discussed below.

3.2 Geological controls

3.2.1 Segregation of the deposit by rock type

This type of approach is generally favoured in sedimentary deposits, e.g. coal, limestones and placers (Forbes). Sedimentary rocks are typically deposited as a series of horizontal layers and the type of rock produced is dependant upon the type of environmental and geological conditions present at the time of deposition. If a change occurs in the depositional environment, changes in the character of the rock produced would be expected.

Suppose, for example, that the environmental and geological conditions favoured the deposition of an ironstone. As long as the conditions remained constant, ironstone will continue to be deposited at a constant rate. Change the conditions slightly and the rate of deposition will change. Produce a drastic change such as a marine transgression, the ironstone will cease to be deposited and a different type of rock will form on top of the ironstone. Clearly, a change of events has occurred which has completely affected the formation of the ironstone with the result that the overlying rocks bear little or no relationship with those below with respect to their iron content. Thus, in order to value a portion of ground within the ironstone, only samples may be used which are taken from within the ironstone, not from the overlying rocks due to their lack of sample influence. This is the basis for a mineral evaluation based upon the influence of rock types.

A technique using a lithological control alone was initially applied to the Cerro Colorado deposit in an attempt to introduce a geological control to the data. Unfortunately, the results were not as good as at first expected, giving similar predictions to those of previous estimates. This is not surprising since the Cerro Colorado orebody is of an igneous origin, not a sedimentary one, to which this method is better applied.

3.2.2 Segregation of the deposit by grade zone

A new approach to the problem of geological control was developed following the poor success with the segregation by rock type. It involved the development of a concept referred to as a grade zone, which can be best explained as a zone containing samples of similar value. 'Natural' breaks within the copper data were observed in many of the drill hole logs and sections (Fig. 3.1), giving rise to

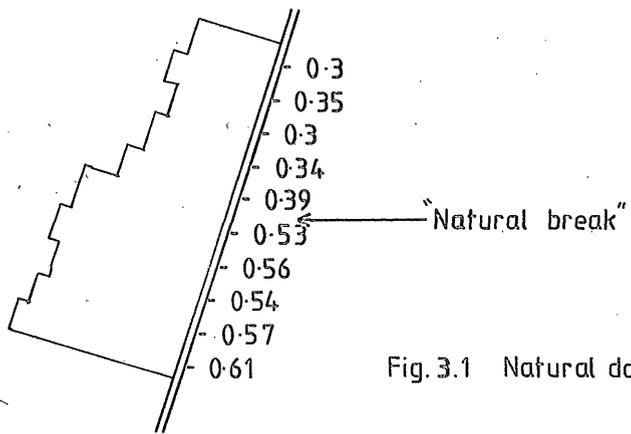


Fig. 3.1 Natural data breaks

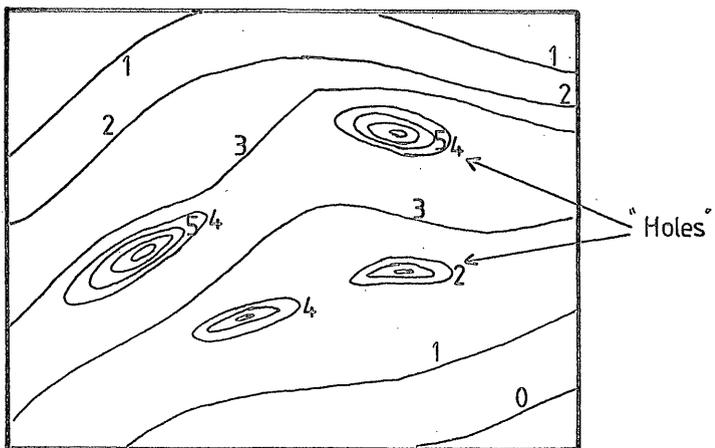


Fig. 3.2 The isograde map

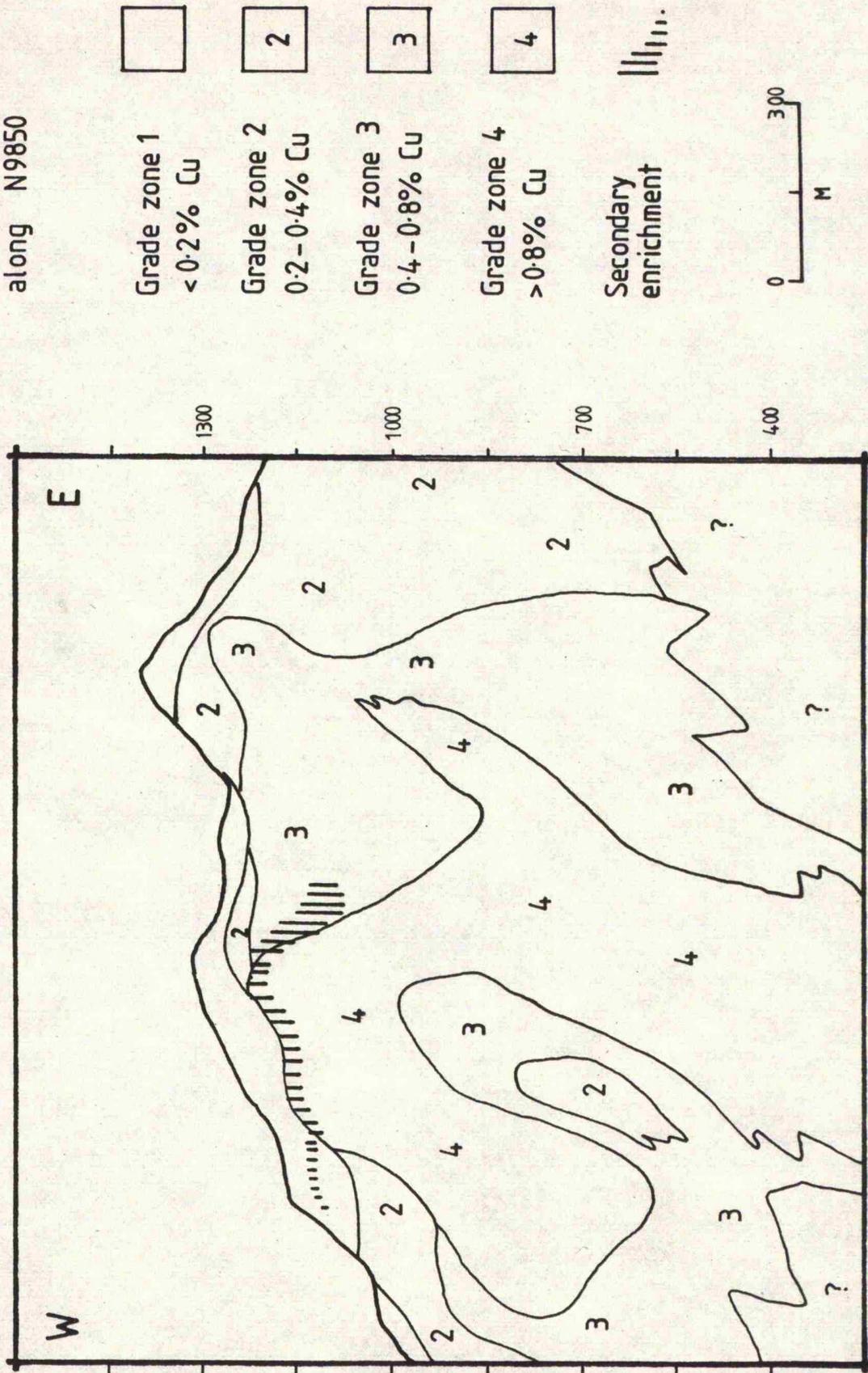
prominent changes in grade values. In the case of Figure 3.1 this occurs between the values 0.53 and 0.39. Examination of every drill hole showed that the natural breaks occurred throughout the whole deposit at approximately 0.25, 0.5 and 0.75 percent copper. Altering the grade value for each zone boundary slightly has very little effect. The natural breaks are so pronounced that if the sections are redrawn using these modified values the shape and boundaries for each grade zone remains virtually unaltered.

It must be emphasised that the grade zones are broad zones and not iso-grades having exact boundaries at 0.25, 0.5 and 0.75 percent copper, meaning that sample values can be found within a grade zone which lies outside its theoretical boundaries. Any misclassifications due to sampling errors are thus minimised and the shapes of the grade zones in section and plan are kept as simple as possible, removing the small 'pockets' and 'holes' highlighted on an isograde map (Fig. 3.2). In some cases, many of these 'holes' may be centred upon one sample value and may represent a 'freak' occurrence and a possible misrepresentation of an area of ground to be valued.

The grade zone concept is often difficult to grasp but helps to make the proposed evaluation method as simple as possible. When the grade zones are combined they show a domed structure in section and a concentric structure in plan (Fig. 3.3). This has been likened to the theoretical porphyry copper model described by Lowell, the grade zones representing regions of alteration brought about by the physical response to the copper solution front migration during mineralisation (Linn). Further work must be done to test this theory.

The grade zones can be used to give a much better local geological control to the process of mineral evaluation, something which the

Fig. 3.3. Grade Zones in section
along N9850



Redrawn from Lin et al

traditional methods and until now kriging has been unable to do. They also help to overcome the problem of using the surrounding low sample values to estimate a much smaller area of high values since areas contained within a certain grade zone will only be valued from sample values taken from within that particular grade zone. It is reasonable to assume that the estimated value will be closer to reality than for the other methods and hence have a lower estimation variance because the sample values used in the estimation will be very similar to that of the actual block or panel value.

3.2.3 'Superblocks' or grade zones

The purpose of many mineral evaluation schemes is to be able to produce a local estimate for the purposes of mine planning. The grade zone technique eventually helped in many ways to overcome some of the problems involved in the Cerro Colorado deposit. In the early stages of valuation, an idea was proposed to divide the deposit as a whole into a series of smaller regions or 'superblocks', each one representing approximately one eighth of the total volume of the deposit. In theory this would be able to produce a local understanding of the data but it must be remembered that a reduction in the area of influence reduces the number of sample values available for use within the local estimation and may lead to less interpretable results. Regions will still remain where areas of predominantly high values contain much smaller areas of low values which may still be difficult to deal with. Sample values from outside each superblock will probably have influences on the sample values within, and unless accounted for will be excluded from the local estimation. A second superblock has to be created to overlap the first two to allow for this with the result that a complicated system may arise.

It could be argued that the sample values within one grade zone can influence sample values within another. This is probably not so since it is believed that the sample values are a result of alterations which are geologically controlled. If this is the case then this is a good argument for segmenting the orebody by grade zone rather than by superblocks for evaluation purposes.

Superblocks may at first appear to be able to impose a local control within the mineral evaluation but the actual siting of the blocks can be very important. It would be very unwise in the case of the Cerro Colorado deposit to place a superblock boundary in the middle of a high grade area thus eliminating any high sample values which are related to the values within the block. Consequently, it would be better to place the boundary between the high and low grade areas, although segregation between the two is difficult because the division between the two is spatially non-linear. The grade zones are able to overcome this division since they are bounded by the samples which are geologically controlled, not by physical constraints.

The superblock approach was not tested for the Cerro Colorado deposit since the grade zones produced good interpretable results. However, it may be worthwhile to produce a comparison between the two methods.

3.2.4 The effects of unfolding and unfauling mineral deposits

This particular aspect of geological control is not required for the evaluation of the Cerro Colorado deposit but is often important in some deposits, especially if structural deformation has occurred. In highly folded deposits, e.g. in North American Precambrian ironstones there is a tendency for the amount of mineralised material to increase in the crest of the fold and decrease in the limbs. Consequently, it

is helpful to know if a drill hole has intersected the crest or limb of the fold, otherwise misleading results will occur. Also, it is possible for the drill hole to run parallel to the limb of a fold and miss an area of high grade mineralisation thus misclassifying the area. It is therefore in the best interest to have a good understanding of the fold structures and to be able to 'unfold' the orebody to its original form during the stages of mineralisation. Evaluation could then be performed on the unfolded orebody which can then be refolded to its present state for mine planning purposes.

Faulting is a geological problem often found in many coal deposits. In most cases the faulting has occurred after the deposition of the coal. When geostatistics or traditional evaluation methods are used the coal seam is valued as a series of panels which would be projected through the fault in the plan of the seam. Because of the action of the fault the influence of the upthrown part of the coal seam may be removed from the valuation of the downthrown part of the coal. The result is that underestimation may occur. Removal of the effects of the fault by placing the coal back to its original position will provide a better valuation of the coal seam and again if the fault is reinstated mine planning procedures can occur.

3.2.5 Other geological influences

Other geological influences include leaching and remobilisation with secondary enrichment. Remobilisation is possibly more difficult to understand since many complicated chemical reactions are involved, often in many stages making the system very complex. At this point very little can be done except for grouping together the areas of remobilisation and evaluating them separately. At Cerro Colorado

secondary enrichment is present in the form of a supergene zone which gave good results when evaluated separately. However, with the birth of the grade zones the supergene could be classified into grade zone four, allowing a much faster method of evaluation to be achieved.

Leaching is a problem at Cerro Colorado, resulting in large tonnages of very low grade material, classified as waste, near the surface of the orebody. Since this is considered to be waste it is pointless to include any sample values from within it in the evaluation of the high grade area underneath. Attempts have not been made to estimate the value of areas of overburden since this will affect the final grade tonnage curve. However, areas of waste and overburden material are considered in the mine planning programme in order to calculate the amount of pre-strip and waste to ore ratios.

3.3 Geological control in the evaluation of the Cerro Colorado deposit

As mentioned earlier, the main source of influence of geological control lies within the grade zones. Sections and plans were drawn up by hand to establish the grade zone boundaries. A matrix was devised consisting of 110 by 120 blocks per bench, each of dimension 25 by 25 metres and of thickness one bench (15 metres), covering the area of the proposed mine. The grade zone number for each block on each bench was digitised into the matrix. For recognition purposes, blocks of unknown material, overburden and air were marked with specific codes as was the drill hole information. This now meant that blocks of overburden and air were not valued and that samples of overburden would not be involved within the evaluation. Blocks contained within a certain grade zone would only be valued from sample values taken from within that grade zone.

A rock matrix was constructed in a similar way, whereby each rock type was assigned a different code. This meant that two possible sources of geological control could be applied to the evaluation of the Cerro Colorado deposit.

CHAPTER 4

4. The geology of the Cerro Colorado deposit

This chapter has been summarised from the works of Lin et al and the Texasgulf feasibility report.

4.1 Regional geology

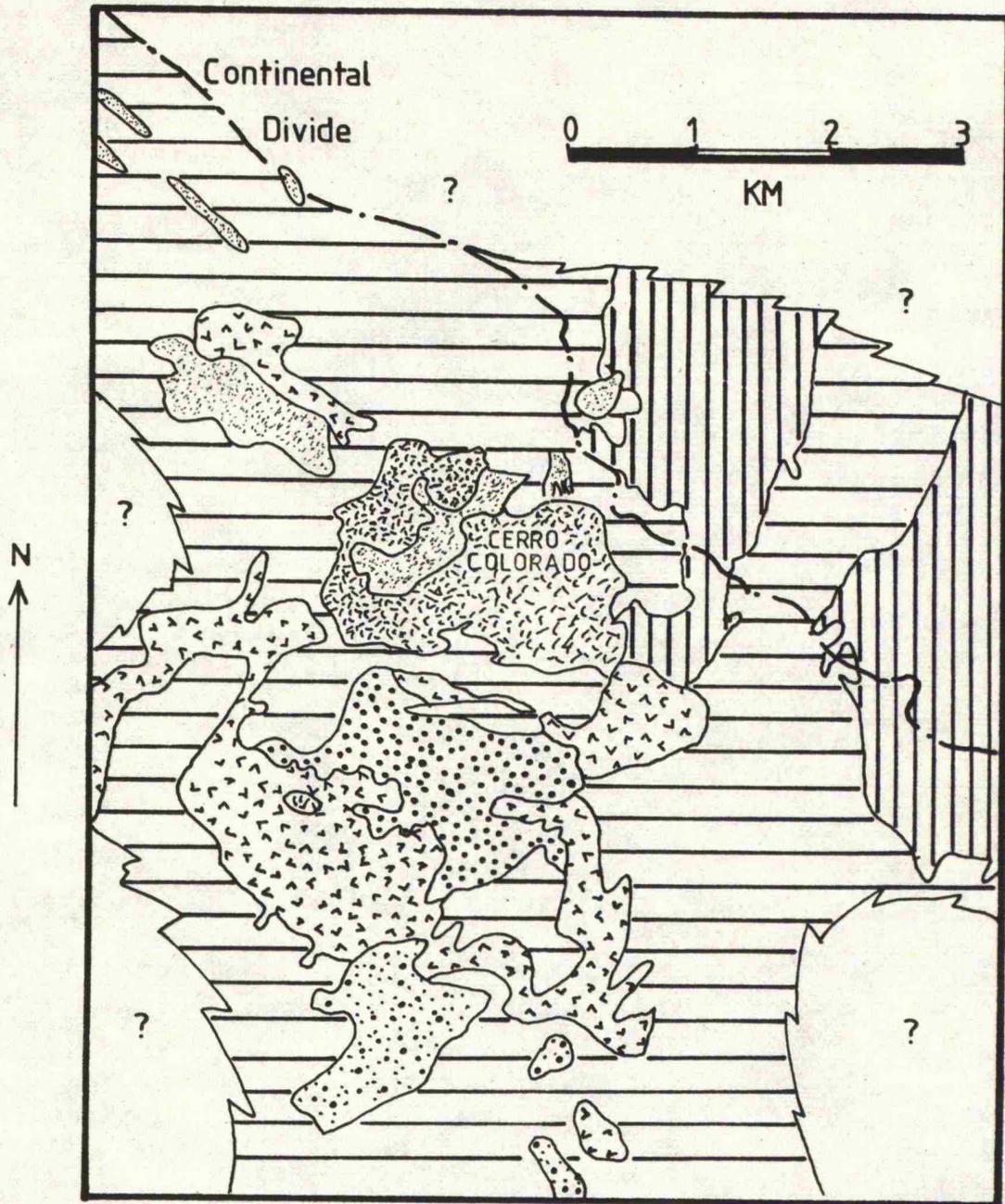
The oldest rock types exposed in western Panama consist of a series of Pre-Eocene basalts, pillar basalts, agglomerates, cherts, andesites and intermediate to silicic intrusions (Fig. 4.1), which are found within the basement complex and have been dated by K-Ar methods at 60-70 million years. Volcanism appears to have been almost continuous during Eocene to Oligocene times, giving rise to numerous andesites and basalts which were later intruded by Miocene diorites and granodiorites. Copper mineralisation is often associated with these intrusions. The andesites and basalts are locally overlain by volcanics interbedded with terrestrial and lake sediments of Upper Miocene and Quaternary age.

A north-west fault pattern is dominant in western Panama, running parallel to the coast line. It is thought to be related to movements of the Pacific Plate, which in turn provided the volcanics by underthrusting the isthmus of western Panama.

4.2 Local geology

The host rocks for the igneous intrusives in the Cerro Colorado area consist of andesite flows and pyroclastics interbedded with sandstones and conglomerates (Fig. 4.2). Copper mineralisation is closely associated with the intrusion of a granodiorite porphyry which was later subject to intense fracturing and hydrothermal alteration. This has been altered to a latite porphyry and then to a feldspar

Fig. 4.1. Regional geology of Cerro Colorado



(after Lin et al)

- | | | | |
|---|-------------------------|---|------------------------------|
|  | Alluvium |  | Granodiorite |
|  | Trachyandesite |  | Andesite |
|  | Post-mineral intrusives |  | Sandstones and conglomerates |
|  | Mineralised porphyries | | |

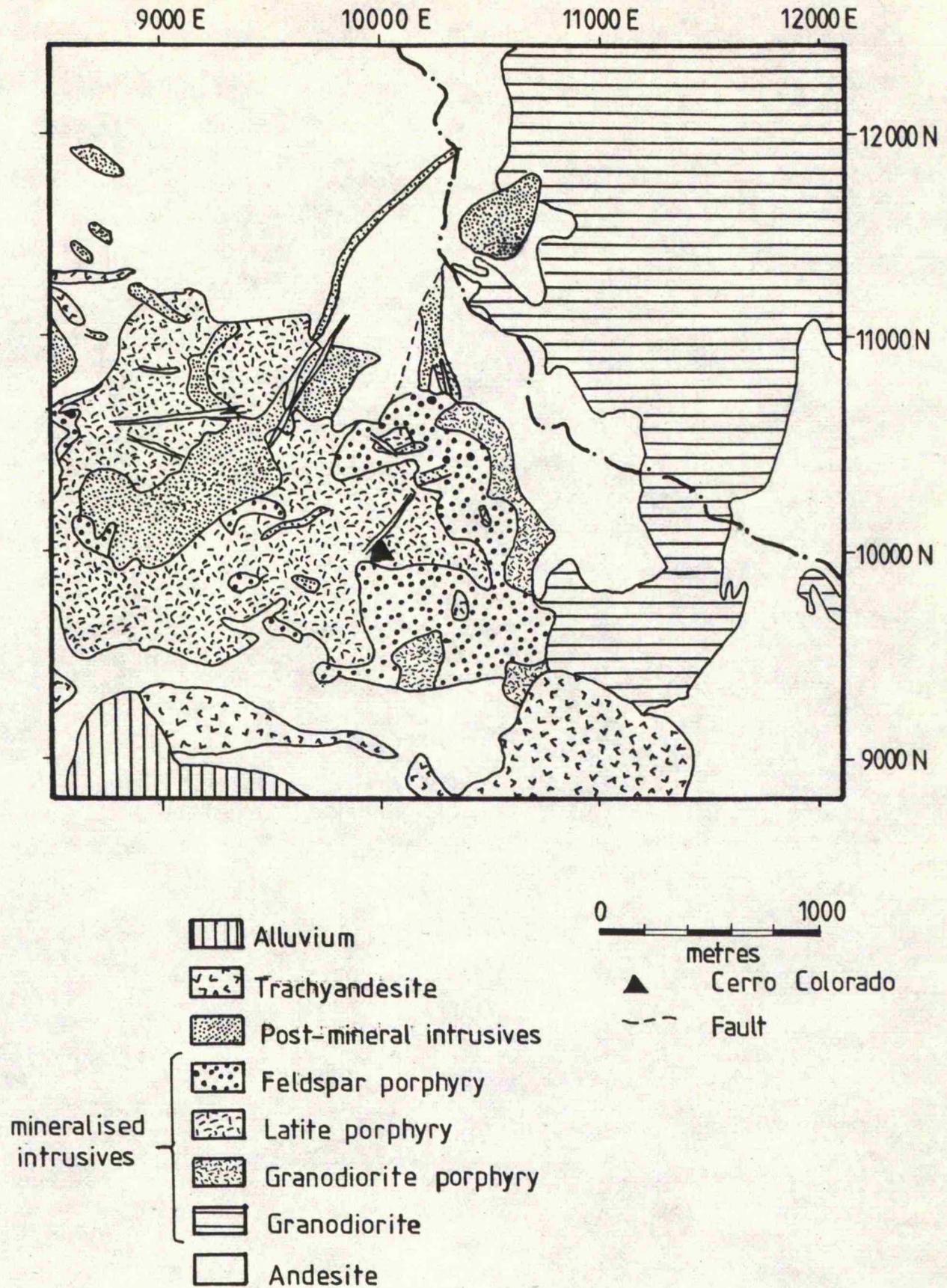


Fig. 4.2. Geological map of Cerro Colorado (after Lin et al)

porphyry which contains the main copper mineralisation. Pliocene post-mineral dykes intrude the mineralised porphyries consisting of either rhyolite porphyry or rhyodacite porphyry. Poorly consolidated trachyandesite tuffs and flows were deposited during the Pliocene but have since been almost totally removed by erosion.

4.3 Rock types

Intense hydrothermal alteration and near surface leaching has occurred within the deposit hindering the recognition of some rock types. For purposes of this evaluation programme the rock types have been divided into three categories, namely pre-ore, ore related and post-ore.

4.3.1 Pre-ore

4.3.1.1 Andesite

The andesite is present as both lava flows and pyroclastics forming the host rocks for the porphyry intrusions. Chemical analyses have shown that their composition approaches that of a basalt. They are usually dark green, grey or black in colour and exhibit either an aphanic or pyroclastic texture with a very fine grained groundmass containing plagioclase, hornblende, pyroxene and magnetite. Strongly zoned phenocrysts of plagioclase are common.

An intensely baked halo of tough dense hornfels surrounds the andesites and often contains visible secondary quartz. Hydrothermal alteration is present in both rock types, though mainly in the andesite. The intensity and style of alteration varies from location to location and has produced a whole series of minerals including epidote, chlorite, sericite and quartz. Sulphide mineralisation may be found in very small amounts in the regions containing visible quartz.

Andesite has been observed as xenoliths in the later mineralised intrusives.

4.3.1.2 Granodiorite

A large granodiorite pluton lies to the north-east of Cerro Colorado and is older than the mineralised porphyries. It is a green-grey, equigranular, tough, occasionally porphyritic, rock, containing medium grained plagioclase, orthoclase, quartz, hornblende and biotite. The green colouration is related to minor chloritic alteration. Disseminated sulphide mineralisation is often present.

4.3.2 Ore related

4.3.2.1 Granodiorite porphyry

This is the least altered rock type within the mineralised porphyry complex. It is very similar to the unaltered granodiorite, having a similar mineralogical composition and a coarse porphyritic texture. The feldspars have been altered to sericite and quartz and the mafic minerals to chlorite, calcite, dolomite, rutile and leucoxene. Sulphide mineralisation is widely disseminated though may occasionally occur in small quartz veins.

4.3.2.2 Latite porphyry

This is the most common of the mineralised porphyries and is centrally located with respect to the copper mineralisation. It is a grey porphyritic rock in which the hydrothermal alteration has produced a series of quartz phenocrysts. Sericite is also present as a result of the alteration of the feldspars, hornblende and biotite. Again, sulphide mineralisation is both disseminated and related to small quartz veins, though the disseminated sulphides tend to occur within some of the altered phenocrysts.

4.3.2.3 Feldspar porphyry

The feldspar porphyry represents the most highly altered variety of the granodiorite porphyry. It is a pale grey rock with few quartz phenocrysts and numerous small altered feldspar phenocrysts. These are often ragged in outline and are possibly the result of the corrosion of larger feldspar grains. As in the case of the latite porphyry the feldspars, hornblende and biotite have been altered to sericite. Alterations and sulphide mineralisation are strongly developed, with copper sulphides being highly disseminated throughout the rock type.

4.3.3 Post-ore

4.3.3.1 Rhyolite porphyry

This is a pale grey rock containing fragments of mineralised porphyries and has a chemical composition similar to that of a dacite. Small phenocrysts of quartz, plagioclase, hornblende and biotite are contained within a groundmass of devitrified glass. Alteration has largely converted the rock into a mass of quartz, sericite, kaolin and calcite. Sulphide mineralisation is sparse and consists mainly of disseminated pyrite cubes with very little chalcopyrite. Most of the mineralisation is contained within the broken fragments of mineralised porphyries. The rhyolite porphyry occurs as one of the two types of post-mineral dykes found in the Cerro Colorado area.

4.3.3.2 Rhyodacite porphyry

This forms the second type of post-mineral dykes and is a brown grey rock containing black shiny flakes of biotite. It is very similar to the rhyolite porphyry having a glassy groundmass containing phenocrysts of plagioclase, hornblende, biotite and quartz. Alteration is less intense than for the rhyolite porphyry with sericite, quartz,

kaolin, chlorite and calcite forming the common alteration products. Mineralisation is again confined to the fragments of mineralised porphyries.

4.3.3.3 Intrusive breccia

A small, pipe-like body occurs in the north-central part of the deposit and is associated with the post-mineral intrusives. It consists of rounded pieces of mineralised porphyry of up to 7 cm across held together in a matrix which can be best described as a rock flour.

4.3.3.4 Trachyandesite

This is the youngest rock type at Cerro Colorado occurring as a light grey tuff. It is very friable, containing small fragments of andesite and giving the appearance of a poorly cemented sandstone. Hydrothermal alterations are not normally present, neither is there any significant sulphide mineralisation.

4.3.3.5 Alluvium

Locally, the alluvium has a thickness of about 200 metres and contains loose fragments of andesite. Layers of volcanic ash and trachyandesite are also present.

4.4 Structures

Obvious signs of faulting are observed along many of the stream beds in the area in the form of fault gouge and fault breccia. The amount of fault movement is uncertain due to the lack of stratigraphic markers. So far the only reliable marker has been that of the displacement of the top of the copper mineralisation where vertical movements of between 50 and 200 metres have been indicated. Most of the faults have been downthrown to the north and west and are often

intersected by cross-faults. Six major fault sets have been detected, the most predominant having a northerly dip. Many faults have been intruded by post-mineral dykes.

4.5 Hydrothermal alteration

Intense alterations have affected most of the rock types within the Cerro Colorado deposit, giving rise to propylitic, argillic, phyllic and potassic alteration features of which the phyllic is dominant.

4.5.1 Propylitic alteration

This is displayed within the granodiorite, granodiorite porphyry and some of the andesites. Dark green or black chloritised mafics are common within the intrusives with epidote occurring within small veinlets. Calcite is often present.

4.5.2 Argillic alteration

This is strongly developed in the latite porphyry and to a lesser extent in the rhyolite and rhyodacite porphyries. It is indicated by the development of kaolin replacing feldspar in a manner analogous to sericite.

4.5.3 Phyllic alteration

Phyllic alteration is found within the granodiorite porphyry, latite porphyry, feldspar porphyry, some andesite and some post-mineral dykes. The latite porphyry and feldspar porphyry are strongly affected and are more obviously affected in thin section.

4.5.4 Potassic alteration

This has been identified only in the deepest portion of the present drilled area and its exact limits are unknown. It is distinguished by

the occasional veinlets of quartz and anhydrite bearing orthoclase which often replaces the host plagioclase. Chlorite occurs as robust flakes and appears to have replaced some of the primary mafic minerals.

4.6 Copper mineralisation

Copper is the most important metal at present within the Cerro Colorado deposit, although gold, silver and molybdenum, as well as zinc and the mineral gypsum, are present. The copper mineralisation has the appearance of an umbrella-shaped array of concentric grade zones (section 4.2.2) and this is made use of in the preparation of a geostatistical orebody model for the deposit. The deposit can be divided into a leached zone near the surface, an underlying secondary or supergene zone and a primary or hypogene zone at depth.

4.6.1 The hypogene zone

The hypogene zone contains chalcopyrite, minor bornite and chalcocite. Small amounts of molybdenite, sphalerite, galena, pyrrhotite and tetrahedrite have been observed, occurring mainly as disseminated grains or small veinlets. Many of the small veinlets occur as fractures which may be completely filled with sulphides, often up to 3 cm across. Disseminated sulphides are generally found in the alteration haloes associated with the quartz veinlets.

The pyrite to chalcopyrite ratio is generally about one to one in the centre of the deposit and increases towards the margins. Usually the two are intergrown, though often occur as separate grains. Primary bornite is significant in the central and western portions of Cerro Colorado.

Quartz is the most abundant gangue mineral occurring as a fine crystalline alteration product with calcite as the second most abundant.

The calcite occurs mainly as an alteration product, though also occurs as small veinlets.

4.6.2 Supergene zone

The supergene zone is present above the hypogene zone occurring as a discontinuous blanket-like zone containing chalcopyrite, digenite, chalcocite, covellite and native copper. Enrichment occurs over the high grade primary sulphide areas and is absent over the low grade primary areas. Occasionally it is extended along fault zones and along the margins of post-mineral dykes. It generally has a thickness of about 60 metres and is considered immature since it has been exposed to weathering comparatively recently, not allowing for the development of a thick and regular supergene zone.

4.6.3 Leached zone

Cuprite is the most common mineral along with minor amounts of malachite and azurite. Usually, the leached zone is confined to near surface, although deep zones of leaching occur along gouge filled fractures and faults. The significant zone of leaching extends between 60 and 250 metres in thickness with an average of 120 metres.

4.7 Molybdenum

The main molybdenum bearing mineral is molybdenite and occurs in some of the non-sulphide copper bearing quartz veins, suggesting that the molybdenum mineralisation phase may be a separate event. Molybdenite usually occurs as minute flakes between 0.01 and 0.05 mm in size. There is no evidence of any zoning until an elevation of 925 metres where there is a zone of structural control, though the nature of the mineralisation is not fully understood.

4.8 Silver

Tetrahedrite is the only silver bearing mineral so far detected, although others probably do exist but have not been found due to their scarcity. Some silver has been dissolved in the leached zone and reprecipitated in the supergene zone, being enriched in a manner similar to that of copper.

4.9 Gold

Gold is present only in trace amounts and it is believed that it has not been remobilised from the leached zone so the supergene zone is not enriched in gold.

4.10 Zinc

Sphalerite is the main source of zinc, but is present on such a small scale to be considered irrelevant to the economics of the orebody. Its presence is noted in as much as its depressant effect on the chalcopyrite.

CHAPTER 5

5. Statistical analysis

5.1 Considerations for the sample values

The actual sample values have the most important role to play within any estimation procedure. These form the basis upon which all calculations, results and conclusions are dependent. It is vitally important to have a good data base and there are four main considerations during sample collection:

1. The purpose to which the data is to be used must be fully understood prior to sampling. Different evaluation methods will require different types of sampling.
2. The manner in which the data is obtained is important. Diamond drilling is expensive, whereas surface rock sampling is cheaper. Obviously, it is unreasonable to sample by means of an expensive and possibly slow technique where a cheaper method will provide the required results. Therefore, select the optimum method of sampling for the purpose required.
3. Sampling is often performed on a large scale to locate any areas of interest and their respective boundaries. In many cases a regular sampling grid is used, though an irregular one may suffice. Exceptional areas of interest can be proven by sampling in more detail, ie. use a more closely spaced sampling grid. The amount of confidence placed in locating a deposit is thus dependant upon the size of the sampling grid and the number of samples obtained.
4. The collected samples must be representative of the area from which they were taken. If a set of samples is representative

of the whole deposit their sample characteristics can be used to infer the characteristics of the deposit.

Once the samples have been collected, the mineral or metal content must be established. The precision is dependant upon the method used. This is important and must be considered in the actual evaluation of an area of interest, i.e. samples valued with a precision of two significant figures cannot be used to produce an estimate to four significant figures.

Numerous tests and calculations may be performed upon the sample values in order to establish the various physical and chemical properties within the data. The mean and variance give an indication to the average and the expected amount of variability respectively within the sample values. If the samples have been valued for two or more variables a correlation between the two will indicate how they are statistically related to each other. Frequency histograms may be used to summarise or categorise the data and give an indication to the statistical distribution of values within the deposit. Many statistical methods assume that the sample values belong to a Gaussian distribution and in many cases this is not so. Many deposits contain sample values which have a Log Gaussian distribution which can be normalised by considering the logarithms of the sample values.

It is important to compare the statistics of the estimated values with the statistics of the sample values used in the estimation. This will help to indicate if the estimated values are representative of the area of interest and thus the validity of the estimate.

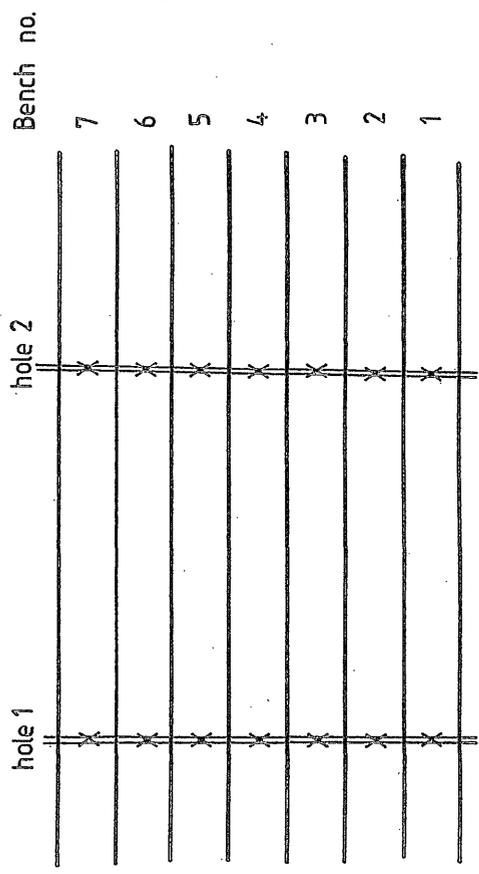
5.2 The nature of the Cerro Colorado data set

Approximately 20,000 samples were obtained from 200 drill holes on an approx. regular grid of 75 metre spacing. Composite samples were collected at

approximately 3 metre intervals down the hole. Each was assayed for its copper, molybdenum, silver, gold, zinc and copper oxide content, although the zinc and copper oxide values have not been considered in this estimate. Previous estimates have only been produced for the copper content based upon the 3 metre composites; the silver and gold content being ignored. An attempt was made to produce an estimate for the molybdenum content, though the actual control upon the mineralisation is not fully understood.

It was decided to attempt an estimation of the copper, silver, gold and molybdenum content of the Cerro Colorado deposit based upon 15 metre sample composites; the advantage being that computer run times would be greatly reduced and it is possible to composite the values on a one sample per bench basis (Fig. 5.1). The data was also segregated into four data files prior to compositing, each one containing the sample values thought to be representative of a particular grade zone. The data thus became available in a form which was easy to handle and not too large to hinder computer turn round.

The main problem with the data set was the presence of the post-mineral dykes. These are very small scale structures of low grade and are unrelated to the mineralisation phases. It was decided to remove any values related to these dykes from the evaluation and then to try and include their presence within the mine planning programme. There is not enough information available at present concerning the understanding of the nature of these dykes, though it is believed that they may constitute a dilution factor of about 5 percent to the actual block values (O'Leary, 1981). All estimates produced within the proposed evaluation and the past feasibility study are done so without considering the presence of the post-mineral dykes and are optimistic.



X: Compositd sample value

Fig. 5.1 Composition of sample values

5.3 Statistics of the Cerro Colorado deposit

Examination of Figure 5.2 suggests that the copper values exhibit a negatively skewed distribution containing two or three small populations. The frequency histograms of the copper values produced a similar appearance, though the degree of skewness was smaller. This is not surprising since this method of normalisation can really only be used for single populations. Figure 5.4 shows the cumulative logarithmic probability plot of the copper values and supports the presence of four sample populations having boundaries at approximately 0.2, 0.4 and 0.8 percent copper. A fifth population may be present, bounded at 1.6 percent copper, though it contains less than 2 percent of the total number of sample values and is considered to be insignificant.

Molybdenum also exhibits a negatively skewed distribution, suggesting that the samples have a log Gaussian distribution (Fig. 5.2). The cumulative logarithmic probability plot (Fig. 5.5) supports this and suggests that only one sample population is present.

Gold and silver also constitute a negatively skewed distribution, though could be considered to be almost Gaussian (Fig. 5.2). Both consist of more than one population, especially in the case of gold, and this is highlighted in the cumulative probability plots (Figs. 5.6 and 5.7).

The correlation between copper and molybdenum, silver and gold, are poor and are summarised in the statistics below:

Fig.5.2 Overall frequency histograms

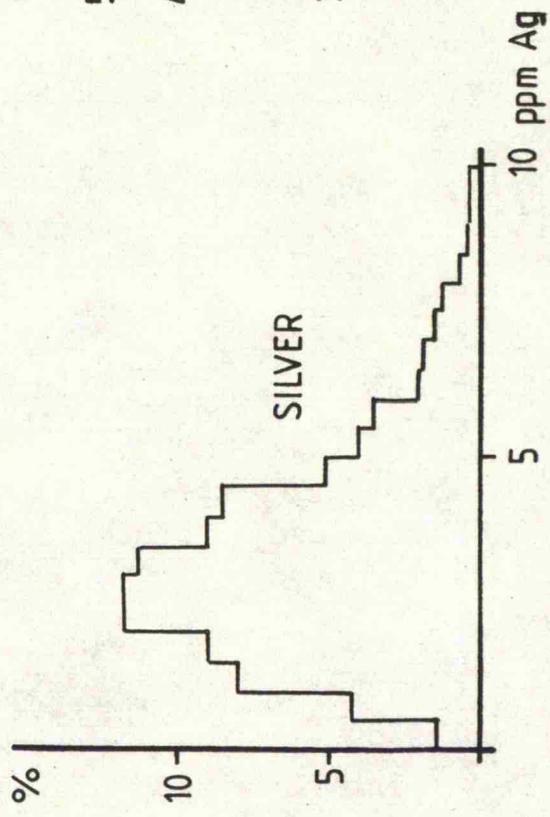
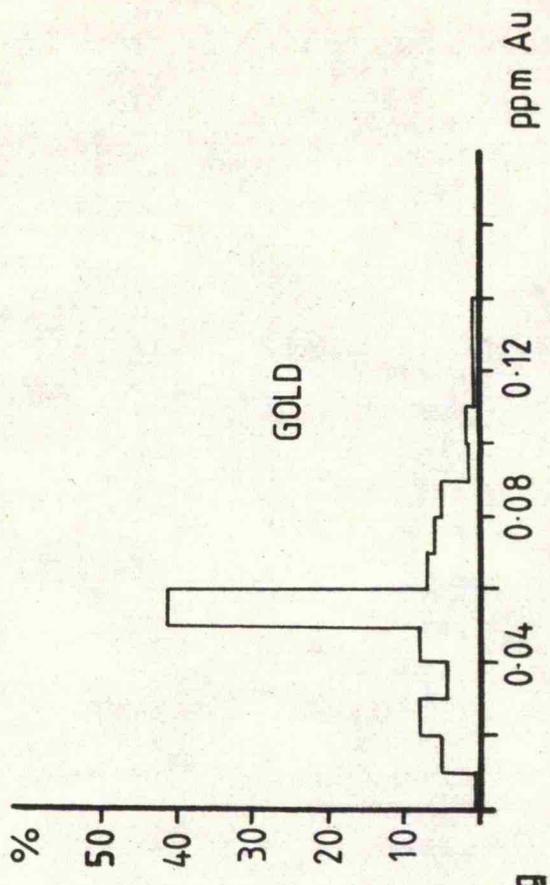
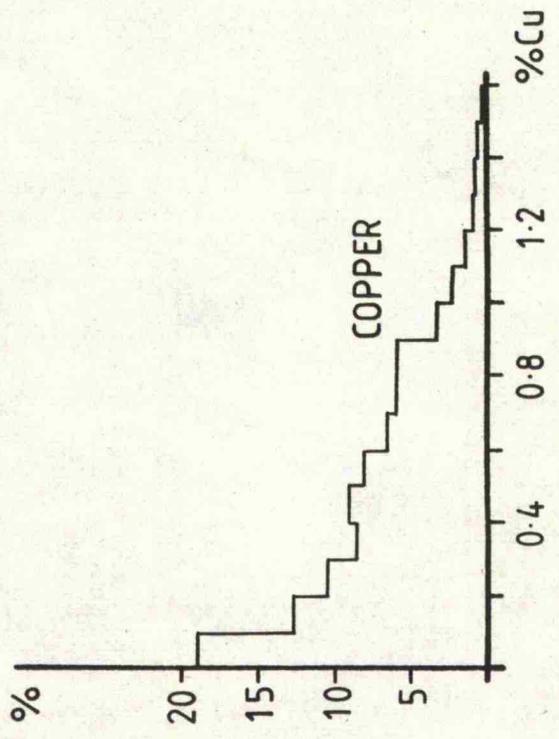
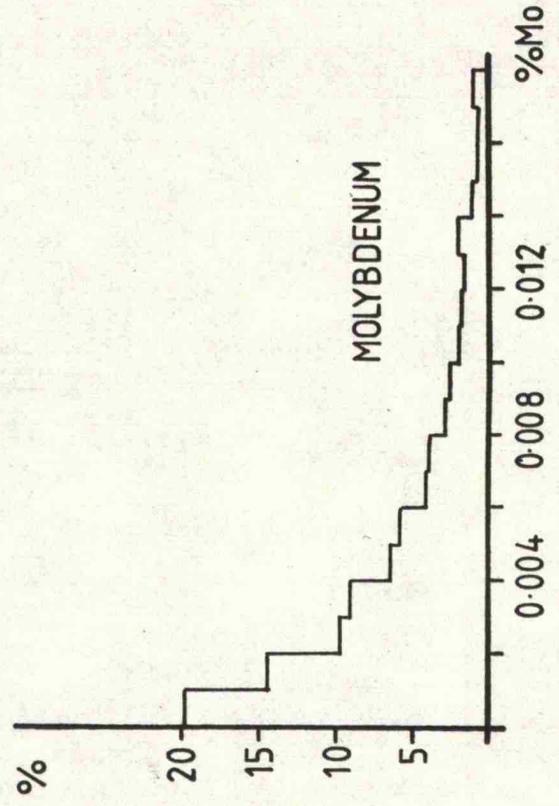
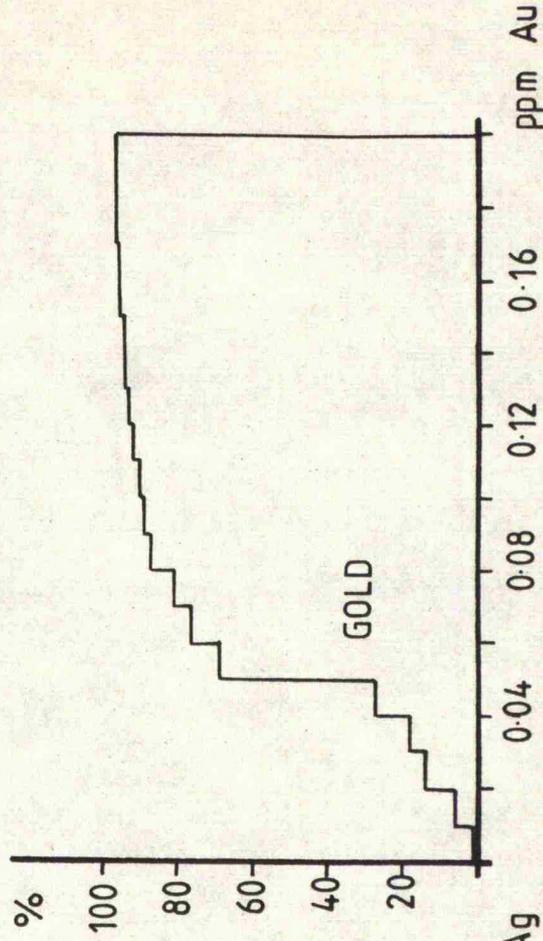
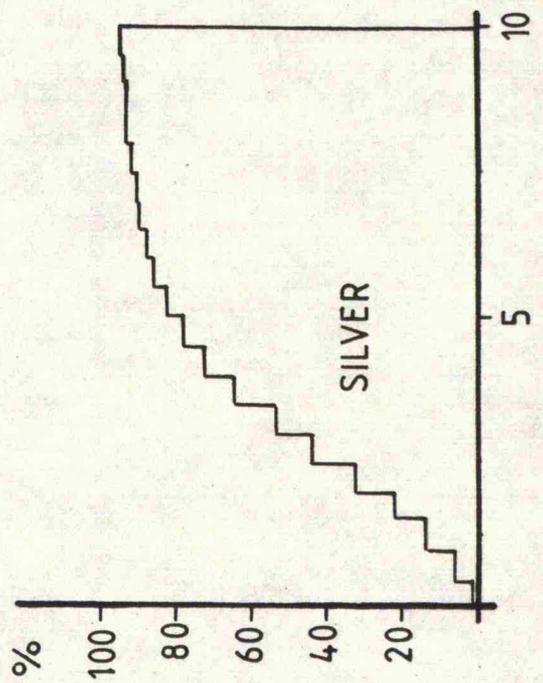
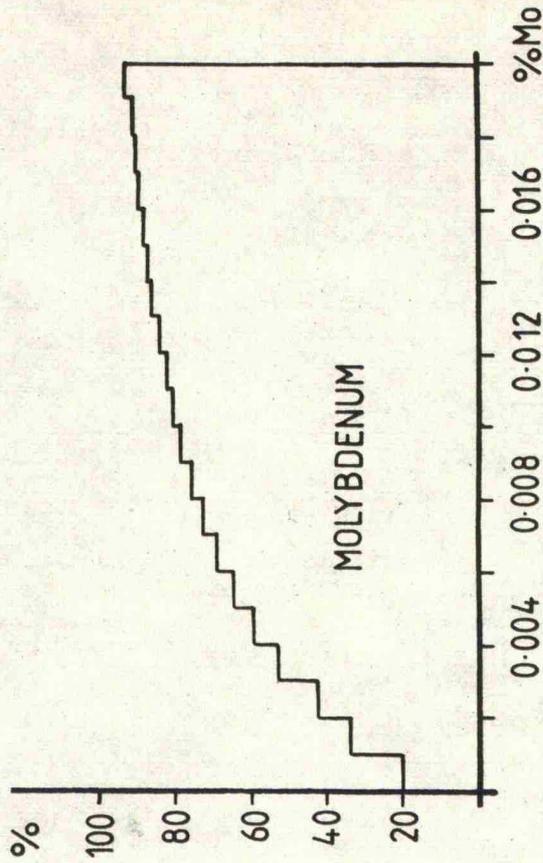
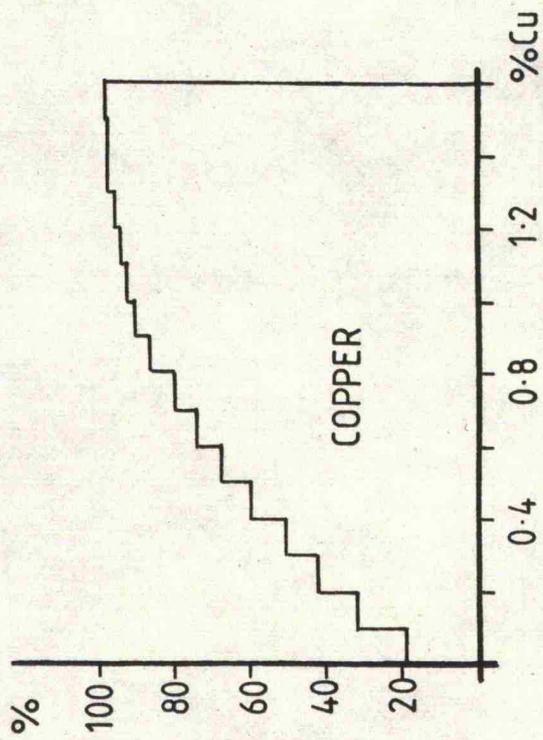


Fig. 5.3 Overall cumulative frequency histograms



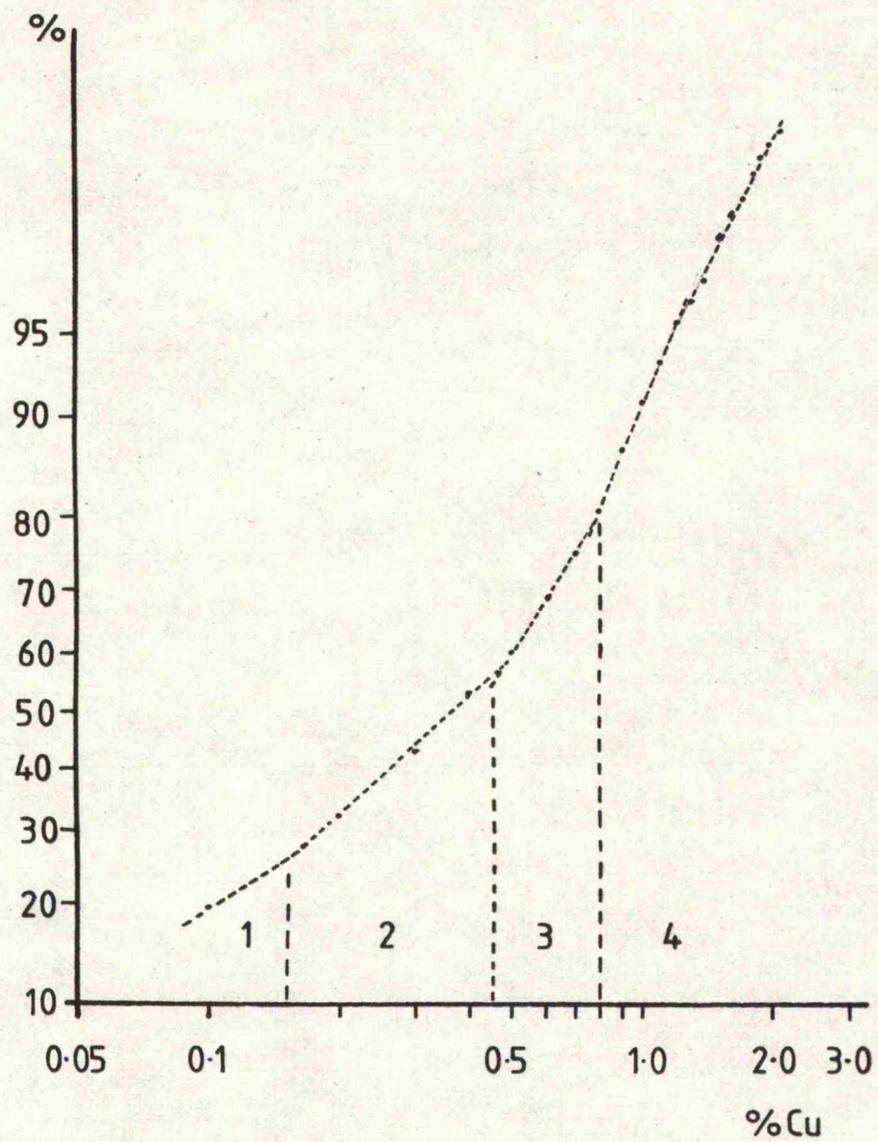


Fig. 5.4 Logarithmic cumulative probability plot for Copper

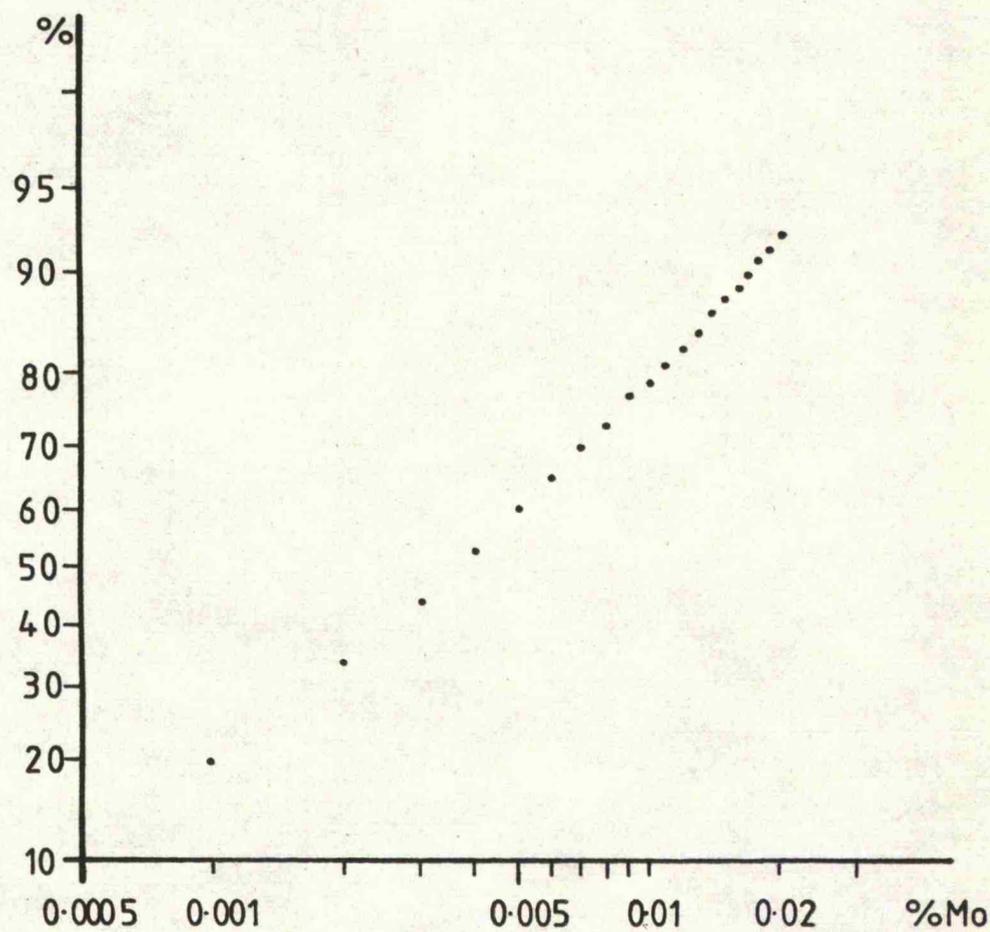


Fig. 5.5 Logarithmic cumulative probability plot for molybdenum

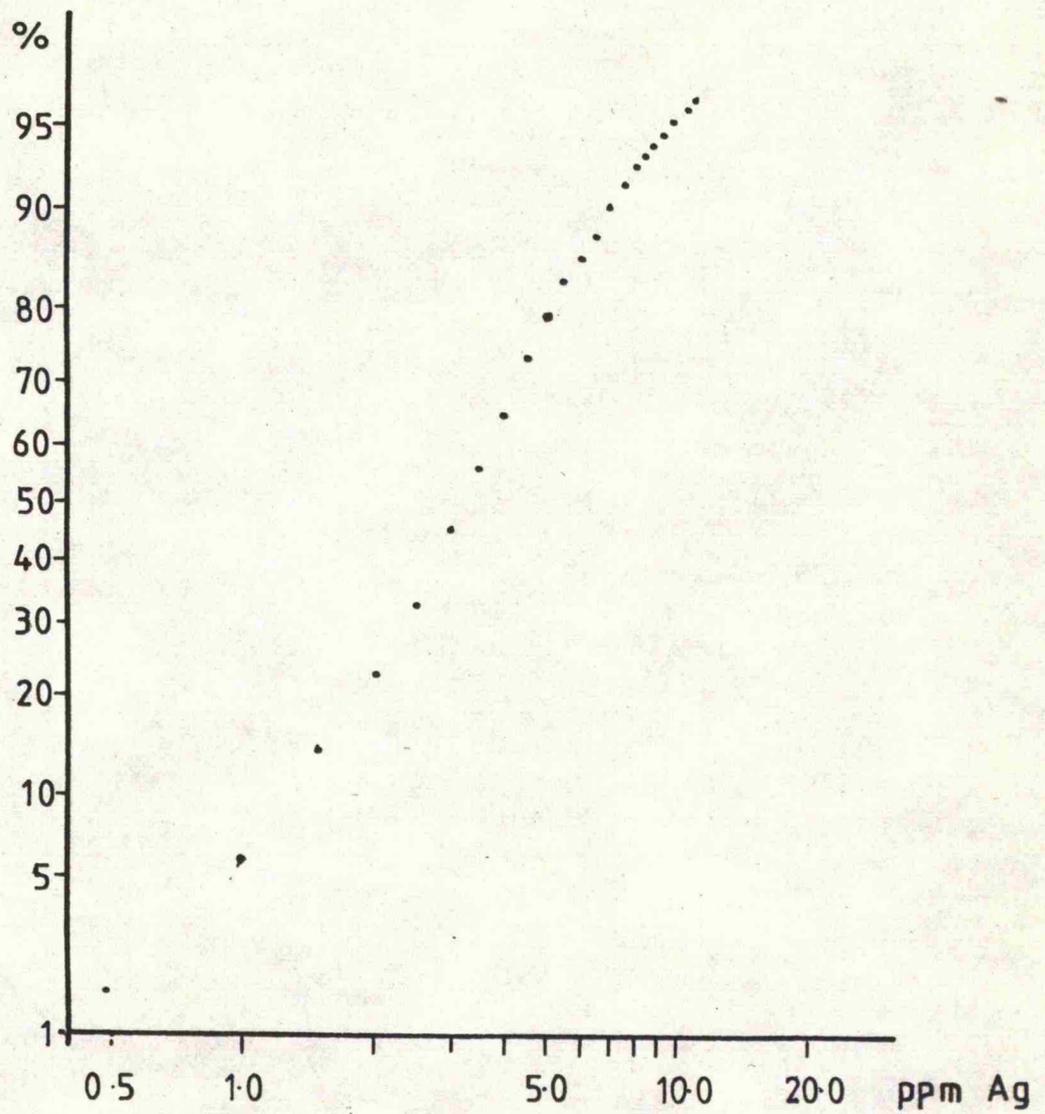


Fig.5.6 Logarithmic cumulative probability plot for Silver

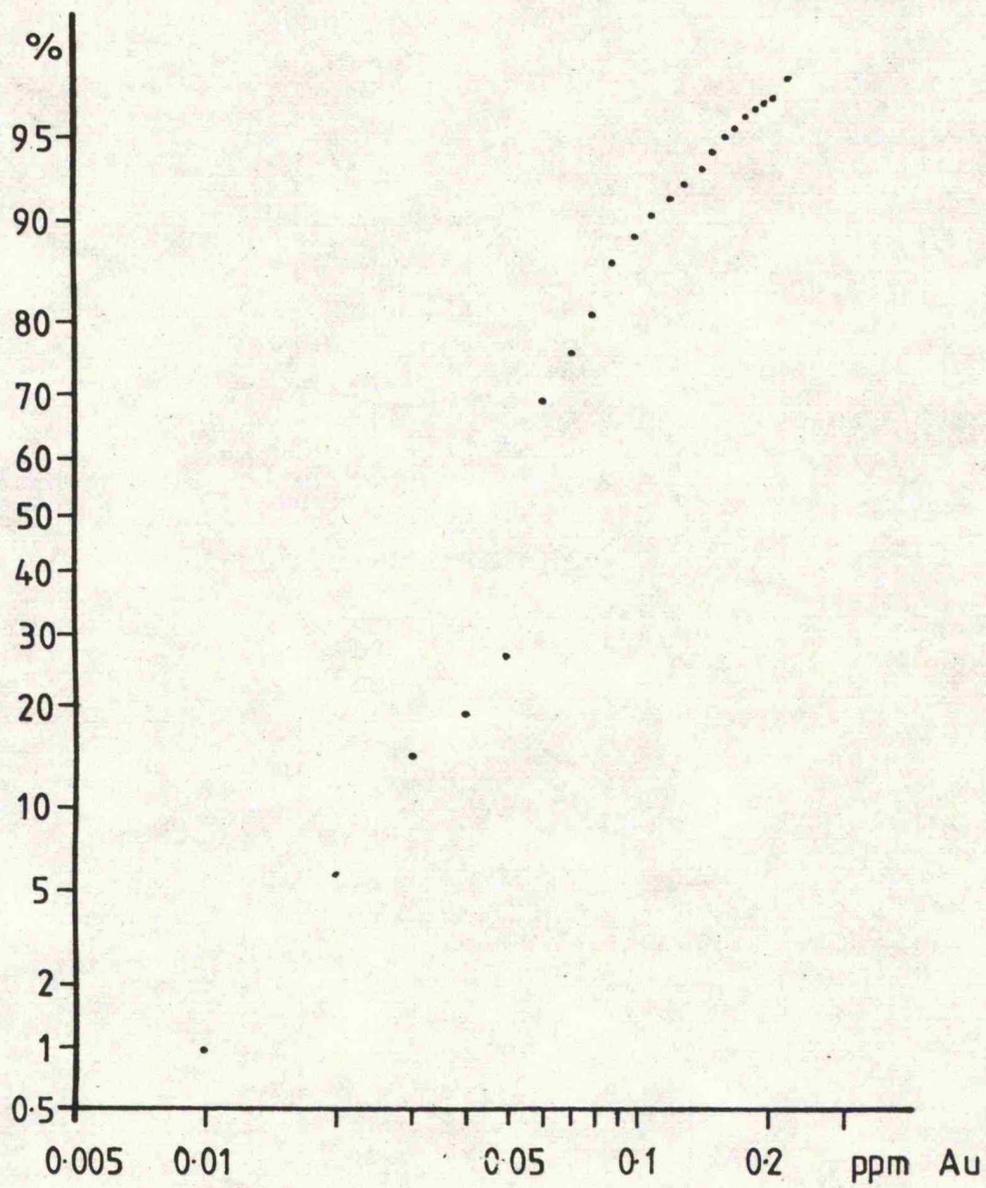


Fig.5.7 Logarithmic cumulative probability plot for Gold

	Arithmetic Mean	Variance	No. of Samples	Correlation Coefficient
Copper	0.466%	0.166	3172	-
Molybdenum	0.0072%	0.0002	2937	0.33
Silver	3.897ppm	9.362	1962	0.40
Gold	0.0712ppm	0.0111	1965	0.16

5.4 Statistics of the grade zones

The data was segregated into four individual data sets, each one containing the sample values thought to be representative of a particular grade zone. It has been demonstrated that the copper values are believed to be contained within four populations whose boundaries approximate to the boundaries of the grade zones. Frequency histograms and cumulative logarithmic probability plots for the copper values suggest that grade zones two, three and four have near Gaussian properties, although grade zone one maintains a log Gaussian distribution. This is not surprising since this particular grade zone contains those very low values close to the precision of the method of measurement and are spread over a small range. Grade zone one is considered to be waste material in the interest of this exercise, and is unlikely to ever be classified as ore. It appears to present no problems within the valuation process. Apart from this, it would seem that the grade zone technique will play a vital role in producing a series of near Gaussian distributed sample values upon which an estimate may be produced. The only problem is that each grade zone is based upon the copper values and thus any silver, gold or molybdenum estimates will be based upon those sample values contained within a particular copper grade zone. Copper is considered to be the only metal of economic importance at this stage and it would seem unnecessary to produce further grade zones based upon the other metals.

Molybdenum still exhibits a negatively skewed distribution for each of the grade zones, whereas silver and gold often show near Gaussian properties. It is anticipated, for the purposes of this study, that the silver and gold values would produce individual grade zones similar to those of the copper and there is sufficient statistical evidence to suggest that this may be the case. Molybdenum is clearly unrelated to the influence of the copper grade zones in so much that it appears as a single population which can be considered to be log Gaussianally distributed. It is debatable whether it is, in fact, log Gaussianally distributed simply because it is present in such small concentrations that it may well present problems similar to those encountered by the copper in grade zone one.

A more precise evaluation of the molybdenum and especially in the case of the silver and gold may be achieved by producing independent grade zones for each metal, although time did not permit this within this study. There is good statistical evidence to suggest that this may be so, i.e. the presence of more than one sample population. The copper content is the main concern of this study, the gold, silver and molybdenum being considered purely out of interest and are not dealt with to the same degree.

The frequency distributions for each metal are given in Figures 5.8 to 5.15 and the statistics for each grade zone are summarised below:

Grade zone one

	Arithmetic Mean	Variance	No. of Samples	Correlation Coefficient
Copper	0.120%	0.013	998	-
Molybdenum	0.0034%	0.00004	912	-0.01
Silver	3.304ppm	14.857	311	0.63
Gold	0.1034ppm	0.052	311	0.39

Grade zone two

	Arithmetic Mean	Variance	No. of Samples	Correlation Coefficient
Copper	0.369%	0.046	799	-
Molybdenum	0.0056%	0.00007	738	0.27
Silver	3.047ppm	5.466	574	0.30
Gold	0.0541ppm	0.0032	574	0.46

Grade zone three

	Arithmetic Mean	Variance	No. of Samples	Correlation Coefficient
Copper	0.559%	0.045	530	-
Molybdenum	0.0084%	0.00014	496	0.23
Silver	3.729ppm	5.169	419	0.47
Gold	0.0561ppm	0.0008	419	0.36

Grade zone four

	Arithmetic Mean	Variance	No. of Samples	Correlation Coefficient
Copper	0.908%	0.184	845	-
Molybdenum	0.0121%	0.0003	791	0.18
Silver	5.030ppm	10.777	658	0.29
Gold	0.080ppm	0.0046	661	0.29

Fig.5.8 Frequency histograms for Copper

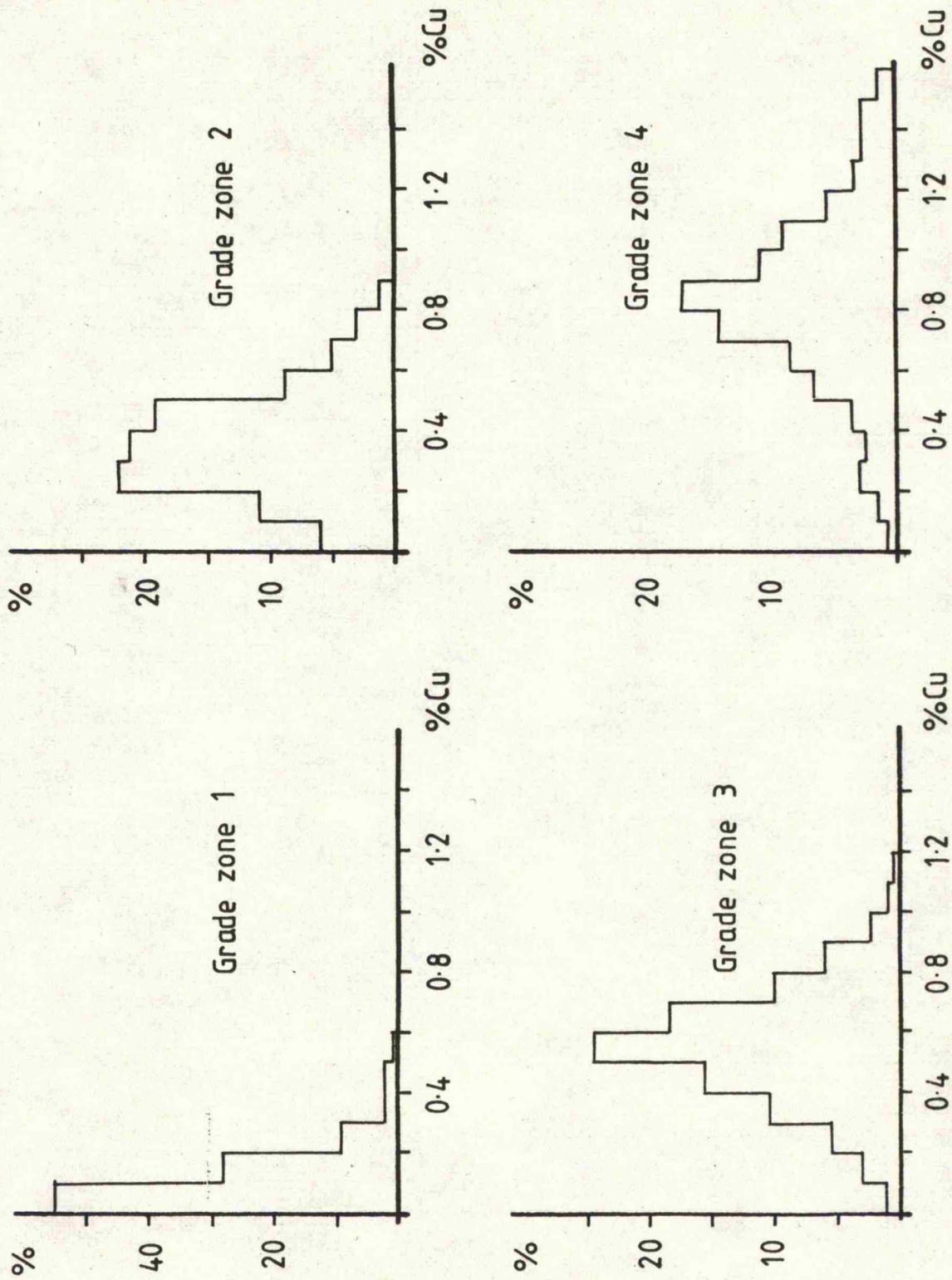


Fig.5.9 Cumulative frequency histograms for Copper

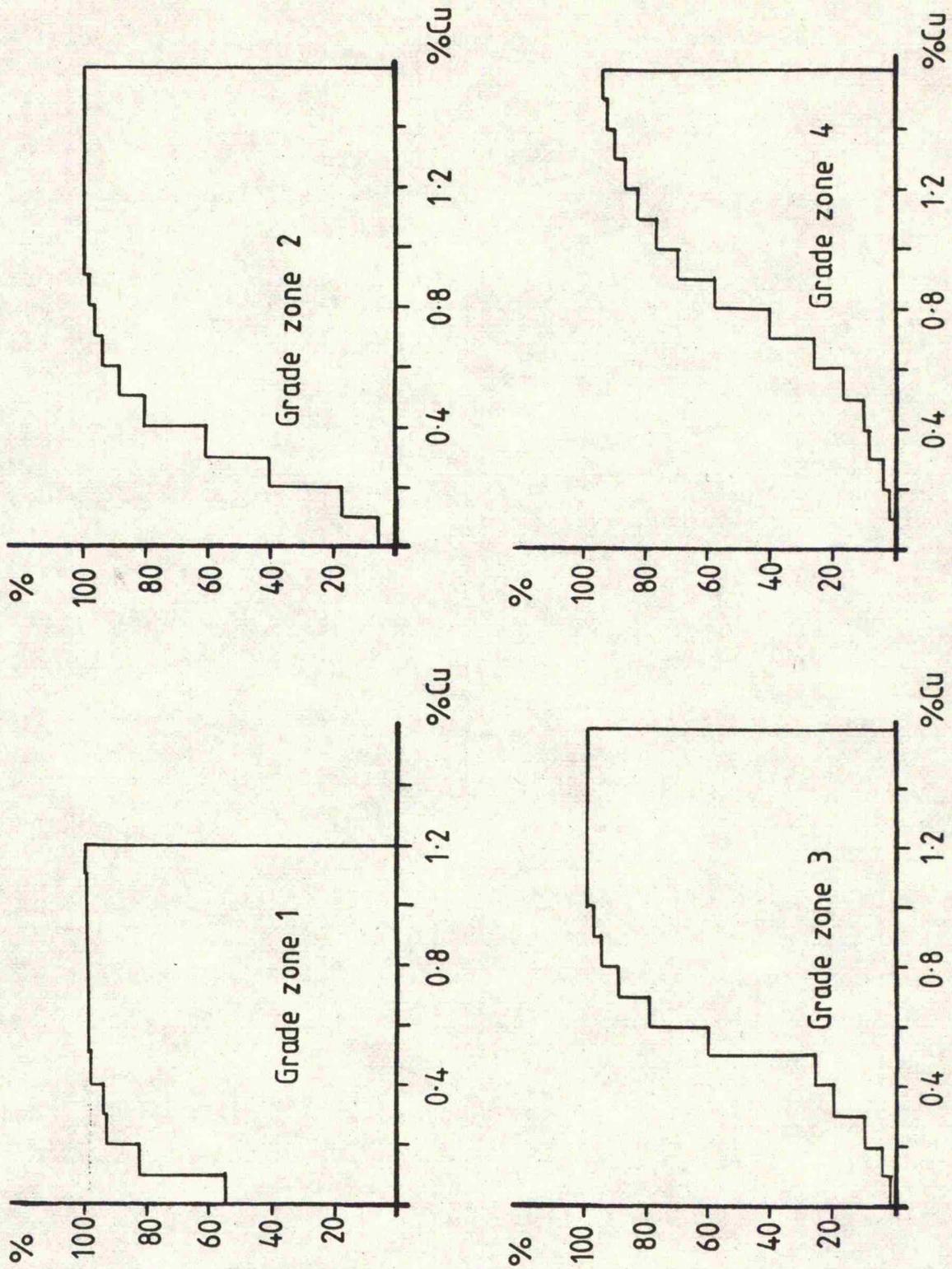


Fig. 5.10 Frequency histograms for molybdenum

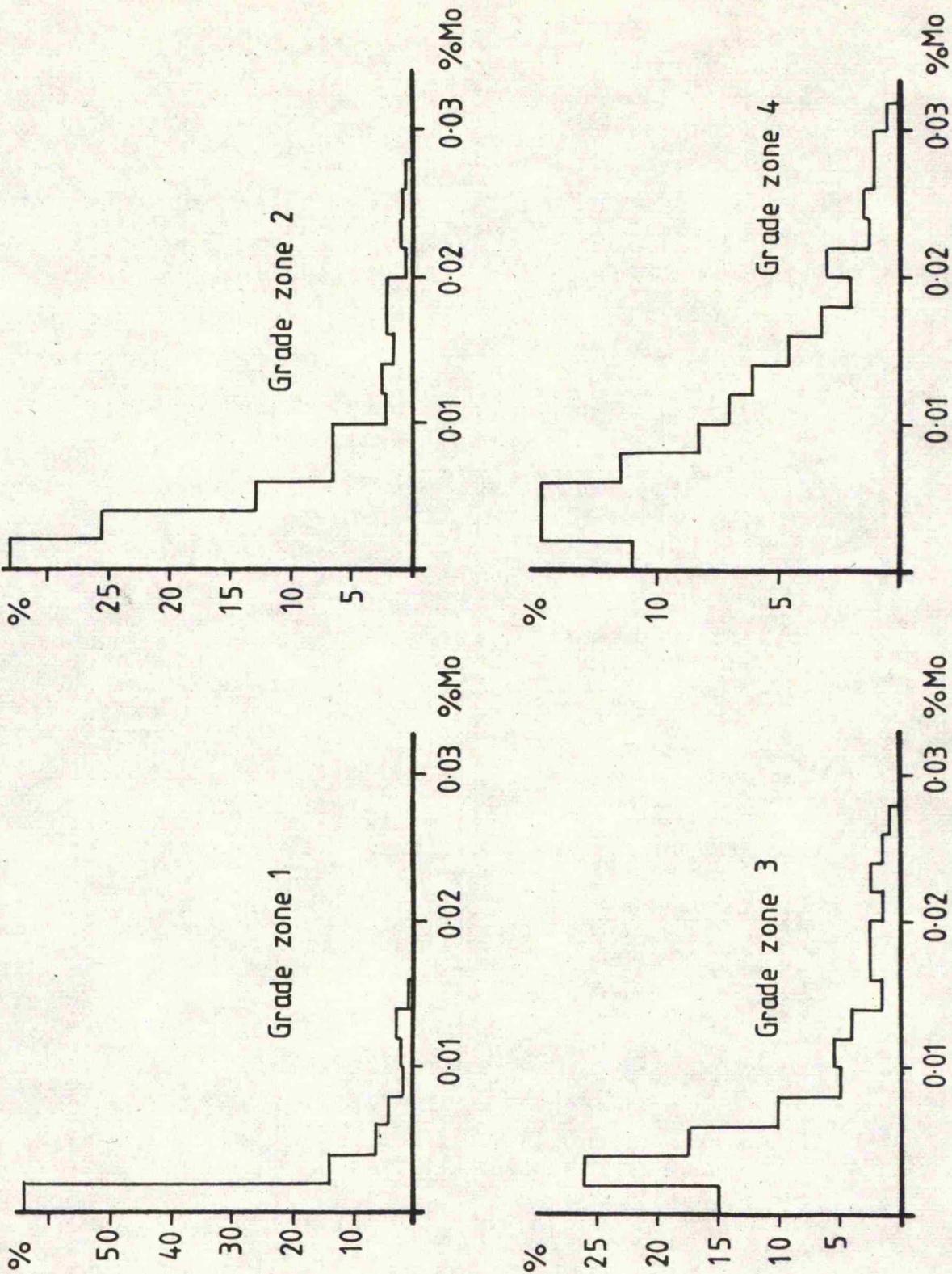


Fig. 5.11 Cumulative histograms for Molybdenum

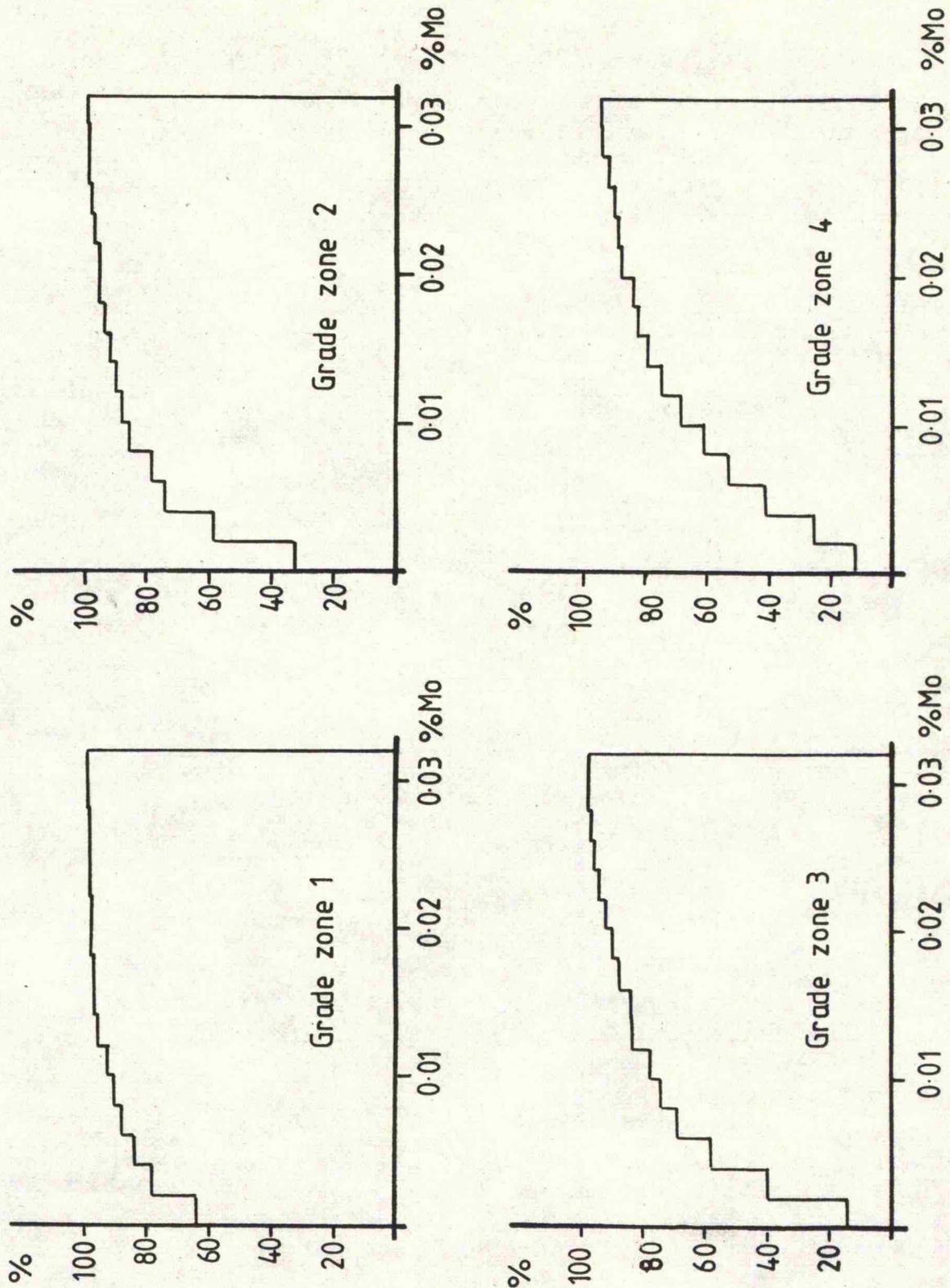


Fig. 5.12 Frequency histograms for Silver

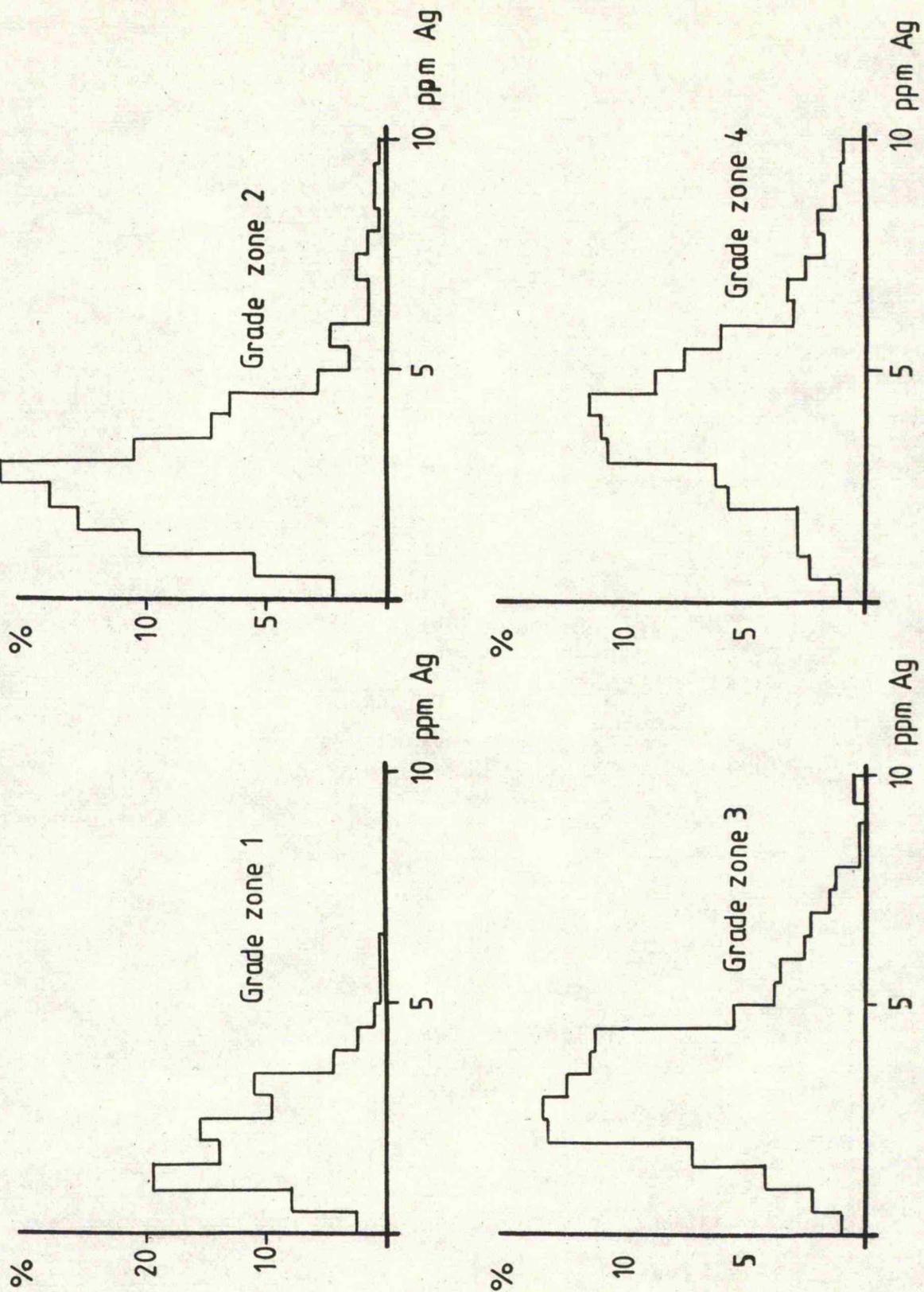


Fig. 5.13 Cumulative frequency histograms for Silver



Fig. 5.14 Frequency histograms for Gold

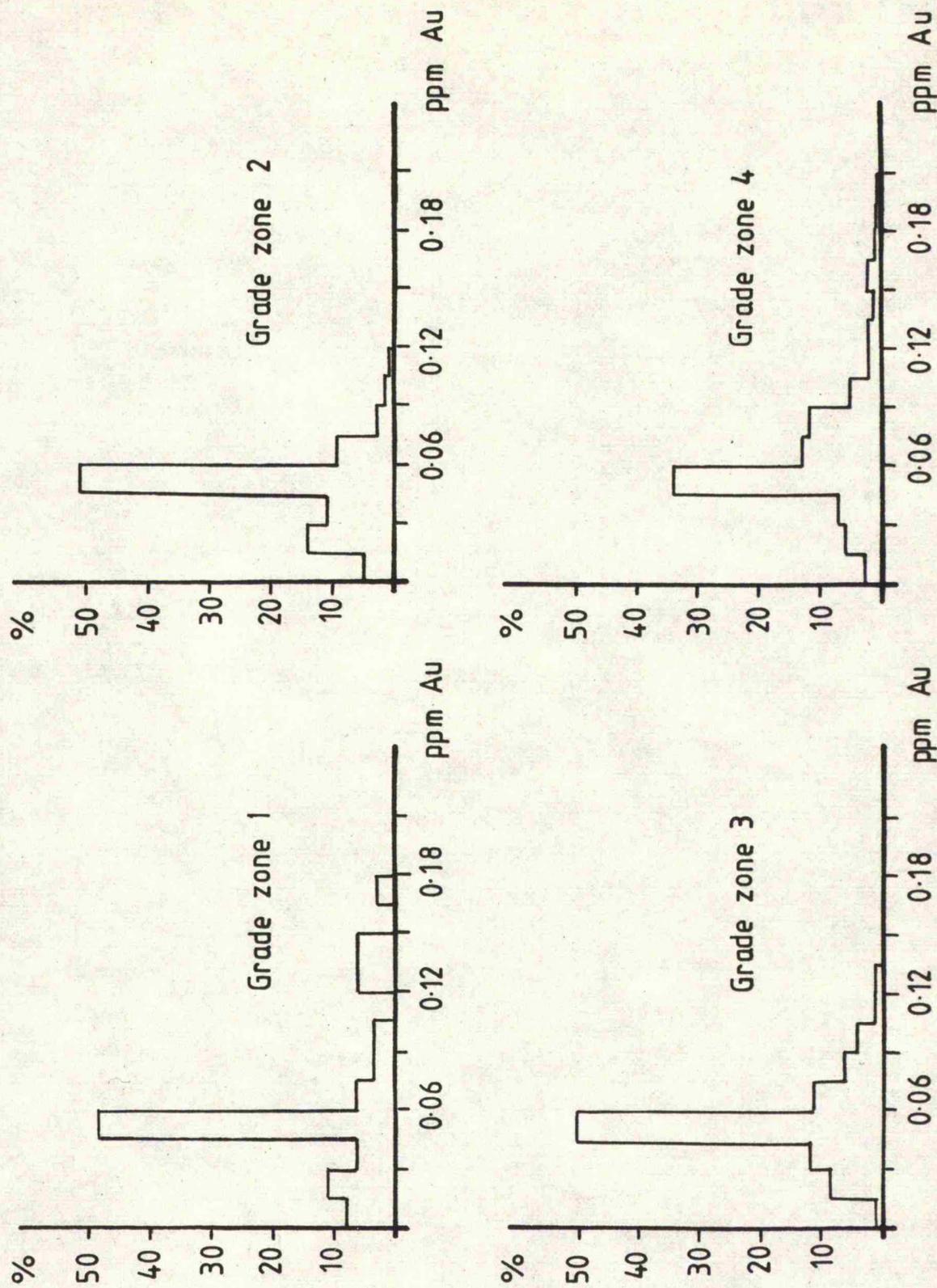
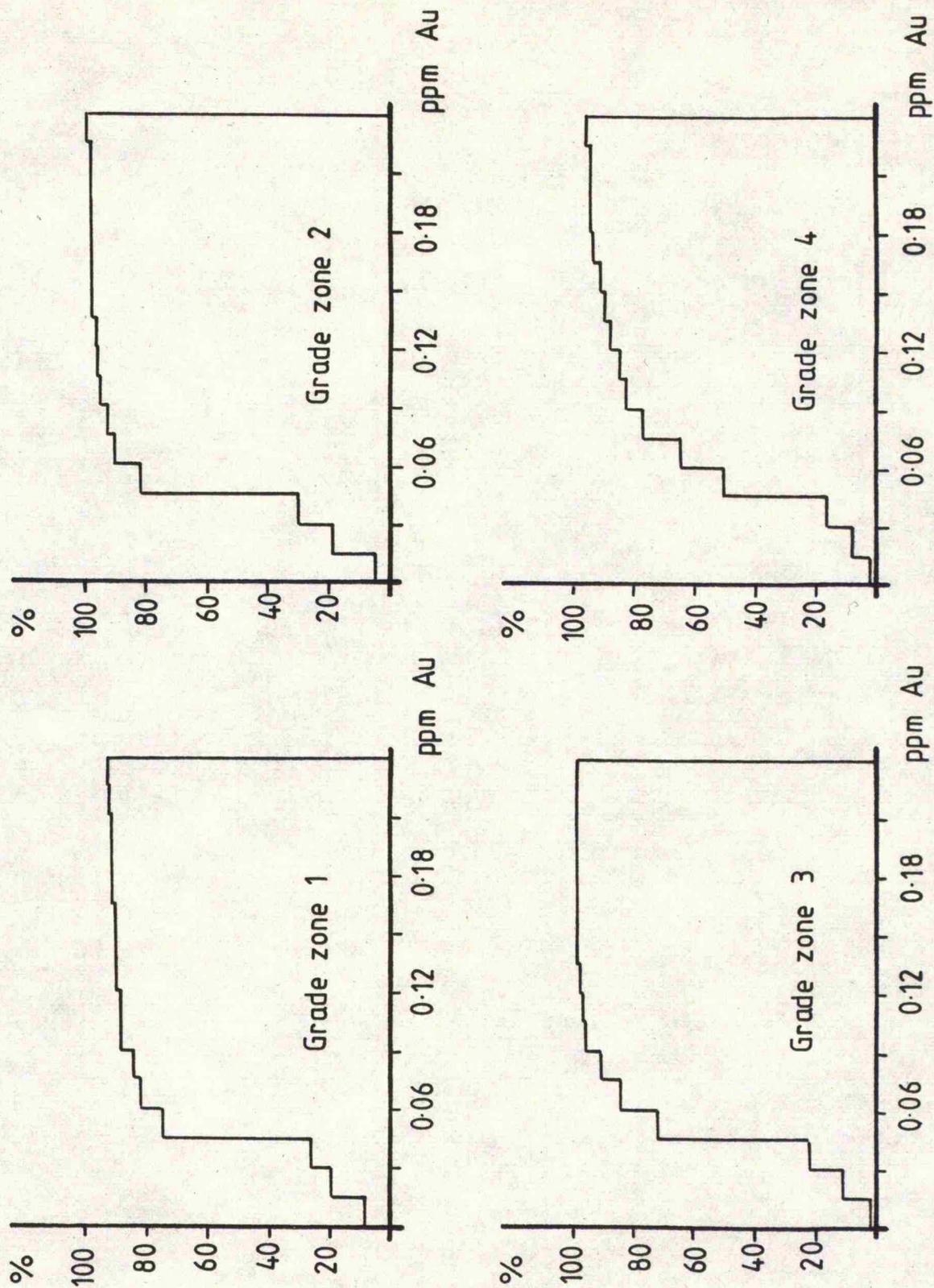


Fig.5.15 Cumulative frequency histograms for Gold



The correlation coefficient between the copper and the other three metals is low for each grade zone, indicating that there is little correlation between the sample values. Also, the mean values of the silver and gold do not tend to increase with the grade zone number as would be expected and thus indicates a further need for the evaluation of each metal by separate grade zones.

5.5 The log Gaussian problem

In many instances it is advisable to 'normalise' a set of data by considering the logarithms of the sample values. All estimates and conclusions are then based upon the logarithmic values. However, this technique tends to be over-emphasised within the field of geostatistics and may not be the best solution in some cases. The problem is that values have to be converted into logarithms, estimates then produced from these which are then converted into a natural number in order to have any significance. Two approximations are made in the conversions to and from the logarithmic values and any results may be affected by this fact, especially when very small sample values are being used. Any inaccuracies such as 'rounding off' errors within a particular method may drastically affect the final result. The problems lie in the actual use and considerations which have to be made to the data, which in the case of geostatistics is the construction of the variogram. Tests within the feasibility study have shown that the molybdenum produced a better variogram of the natural values as opposed to the logarithmic values. Also, the very small molybdenum values could be highly 'influenced' by the conversions to and from the logarithmic values and it is for these two reasons that it was considered unnecessary to convert the molybdenum values into logarithmic values. Again, further tests must be performed to support this, but these are beyond the scope of this study.

CHAPTER 6

6. Structural analysis

6.1 Definition

Journal(1978) defines structural analysis as a procedure for considering the structure of the spatial distribution of a variable and must be considered within the early stages of a geostatistical study. The variogram is able to summarise all of the structural information within a set of data, which can then be interpreted and included within the geostatistical study. Many of the structures which may be found within the variogram have been described in the previous chapter and need not be repeated here. However, it is important to realise that any structures detected, however small, will have a definite cause, and attempts must be made to try and interpret them, unless there are good grounds to suggest otherwise. Usually, the structures are a result of either geological or sampling influences often confined to very small areas. In such cases it may be easier to simply disregard them or to remove and consider them separately. The final judgment must rest with the user as there are no set rules regarding the consideration or rejection of certain variogram structures.

6.2 Structural analysis of the Cerro Colorado deposit

Semi-variograms were produced for each grade zone for the copper, molybdenum, silver and gold sample values. Vertical semi-variograms were relatively easy to obtain since the bulk of the information was contained within vertical drill holes. These were produced for a lag interval of 15 metres, corresponding to the bench composite interval. Few sample pairs were available for many of the holes and so the average downhole semi-variogram was computed.

In order to gain an understanding of the total spatial variability of the samples within the three dimensions of the deposit it was necessary to obtain a series of horizontal semi-variograms. All horizontal semi-variograms were based upon a lag interval of 50 metres, the same as for the previous estimate. A lag interval of 50 metres was chosen simply because it produced interpretable semi-variograms. Most of the sample values were separated by an approximate horizontal distance of 75 metres, though a more closely spaced series of samples were available from the region in and around the adit and from closely spaced and inclined drilling. Sufficient closely spaced samples were available to permit the calculation of the semi-variance over lag intervals smaller than those of the average drill spacing.

Directional horizontal semi-variograms were produced at 45 degree intervals for directions 000, 045, 090 and 135 degrees along the directions of the sample grid in order to try and establish any anisotropies. It was not practical to try and obtain semi-variograms over intervals of less than 45 degrees simply because the number of sample pairs was drastically reduced and the semi-variance was often undefined for the first few lag intervals. In all cases the horizontal directional semi-variograms could be considered to be isotropic.

An isotropic three dimensional semi-variogram was determined for each grade zone in order to establish the average sample variability within the deposit. These were compared with the directional horizontal and downhole semi-variograms. The horizontal semi-variograms were very similar to the three dimensional isotropic semi-variograms. The downhole variogram showed a much shorter range and smaller sill value. Few sample pairs were available within the downhole semi-variogram at distances similar to those of the ranges of the horizontal semi-variograms and so a direct comparison could not be made between the two.

In the previous study, horizontal semi-variograms were obtained for smaller lag distances, i.e. 5 and 15 metres, and could be directly compared with the downhole semi-variograms. Careful analysis of the two suggested that they had similar structures and could be considered to be isotropic over small distances. In view of this fact it was assumed that each three dimensional isotropic semi-variogram was representative of the average sample variability within each grade zone and could be tested and verified by point kriging.

Structures similar to those of a hole effect appeared to be present over distances of about 100 to 200 metres, though these are thought to be related to the presence of the high grade andesite xenoliths. These represent local areas of erratic mineralisation and are often represented by only one sample value. Thus, such occurrences are possible within any one of the four grade zones.

Covariograms were also produced with similar considerations to those of the semi-variograms and were generally stable enough to model. These also showed isotropic tendencies and illustrated the presence of the xenoliths as a series of hollows within the curve. Tests were performed for the presence of drift and it was considered to be virtually non-existent in many cases, except in lags of greater than 1000 metres.

Representative semi-variograms and covariograms are shown in the accompanying figures at the end of this chapter.

6.3 Modelling the variograms and covariograms

The model parameters were tested and refined by point kriging the data values within each grade zone. A three dimensional kriging method was established along the same lines as in Chapter 8, and was used to test the model parameters. The horizontal variogram parameters were firstly established and then the vertical variogram parameters were

adjusted according to the ratio between the kriging variance and estimation variance. In many cases an unbiased model was produced and could be considered to be three dimensionally isotropic. It was also possible to produce an unbiased anisotropic model for the copper variograms with a smaller vertical sill value and range. These generally gave a larger kriging variance than the isotropic models and in view of this fact it was decided to adopt the isotropic model for each grade zone, except for the copper grade zone three. A much lower kriging variance was encountered in the directional approach for this grade zone, although the exact reason for this is unsure. It is possible that there is a strong vertical control of the copper values within this grade zone since this particular grade zone is present as an 'envelope' of limited extent surrounding the higher values within grade zone four.

Numerous input parameters are required to control the point kriging process. Low screen effects were highlighted within almost every semi-variogram, meaning that the more distant sample values could be usefully employed within the kriging system. Large areas of sample search could now be used to govern the maximum and minimum number of samples to define the kriging matrix. Four sample values were considered to be the minimum number necessary to krig a point, though six gave the same overall results but reduced the total number of points kriged. Most of the sample values are contained within the vertical planes of the deposit and care had to be taken to ensure that not all of the points available were confined to the vertical direction. Numerous tests were made to establish the optimum radii of search and the maximum and minimum number of samples to be used. Twelve sample values proved to be adequate for the maximum, though fifteen ensured that 99 percent of the data points were kriged. The horizontal radius of search was

300 metres, which was generally less than or equal to the range of sample values for each metal within each grade zone. A vertical restriction of 150 metres ensured that all of the sample values were not confined to the vertical direction.

Covariograms were also used to test the model parameters, requiring the establishment of the model covariogram. These were generally easier to model than the semi-variograms. Unbiased results were obtained which compared favourably with those produced by the variogram models and computer run times were greatly reduced. The stability and quick computer 'turn rounds' prompted the use of the covariogram for modelling purposes. Models were easily obtained, except for the cases of silver and gold, for grade zone one. In these two cases, high nugget variances were found which were close to the overall gold and silver sample variances for grade zone one. No model parameters could be established and it is believed that pure nugget effects exist in these two cases. The best estimate is now the mean gold and silver value for grade zone one.

The theoretical sill values proved to be stable for most of the semi-variograms, though marginally different from the respective variances for each grade zone. For purposes of this study the variogram sill values can be considered to be the same as those for the respective sample variance, though they are not required during evaluation by means of a covariance model. The model covariogram parameters are listed below:

Copper

Grade zone	C	A	Variance	Sill	Screen effect
1	0.0077	350M.	0.0132	0.013	0.71
2	0.0193	350M.	0.0455	0.045	1.37
3	0.0132	300M.	0.0452	0.044	1.8
Vertical	0.0124	250M.			
4	0.082	350M.	0.184	0.0195	1.28

Molybdenum

Grade zone	C	A	Variance	Sill	Screen effect
1	0.00002	300M.	0.00004	0.00004	1.0
2	0.00003	300M.	0.00007	0.00007	1.25
Vertical	0.00007	250M.			
3	0.00007	250M.	0.00014	0.00015	1.0
4	0.00012	300M.	0.00030	0.00030	1.5

Silver

Grade zone	C	A	Variance	Sill	Screen effect
1	-	-	-	-	-
2	1.02	350M.	5.454	6.50	4.3
Vertical	0.99	300M.			
3	1.62	350M.	5.196	5.2	2.3
Vertical	1.56	300M.			
4	3.46	350M.	10.777	10.8	2.08
Vertical	3.35	300M.			

Gold

Grade zone	C	A	Variance	Sill	Screen effect
1	-	-	-	-	-
2	0.00083	350M.	0.0032	0.004	2.89
3	0.00021	300M.	0.008	0.007	2.33
Vertical	0.00021	250M.			
4	0.0028	350M.	0.0046	0.0047	0.64
Vertical	0.0028	300M.			

Frequency histograms were drawn for both the actual and kriged estimates and were compared with each other. In every case where the optimum model parameters had been obtained, the two distributions were very similar. T tests were also performed and the actual and kriged estimates were statistically compared. These tests highlighted the fact that an unbiased estimate had been produced and the results are listed below:

Copper

Grade zone	1	2	3	4	5	6	7
1	0.119	0.119	0.0005	0.0078	1.00	998	0.12
2	0.370	0.373	-0.0038	0.0344	1.00	796	0.43
3	0.559	0.559	0.0002	0.0406	1.01	530	0.02
4	0.909	0.909	-0.0001	0.1337	1.00	845	0.03

Molybdenum

Grade zone	1	2	3	4	5	6	7
1	0.0034	0.0035	-0.0001	0.000	0.96	912	0.28
2	0.0056	0.0057	-0.0000	0.0001	1.0	735	0.08
3	0.0084	0.0085	-0.0001	0.0001	1.0	496	0.12
4	0.0121	0.0123	-0.0002	0.0002	0.98	791	0.26

Silver

Grade zone	1	2	3	4	5	6	7
2	3.045	3.052	-0.007	5.142	1.0	571	0.07
3	3.735	3.719	1.287	4.420	1.0	416	0.13
4	5.030	5.024	0.0061	8.840	1.0	658	0.04

Gold

Grade zone	1	2	3	4	5	6	7
2	0.054	0.054	0.0001	0.0028	0.99	572	0.02
3	0.056	0.056	-0.0001	0.0007	1.07	419	0.07
4	0.080	0.082	-0.0012	0.0026	1.00	661	0.39

- 1 - Mean of sample values
- 2 - Mean of kriged estimates
- 3 - Average difference between actual and Kriged estimates
- 4 - Average kriging variance
- 5 - Ratio between the kriging variance and estimated variance
- 6 - Number of points kriged
- 7 - T value for comparing the means of the actual and kriged values

Cross-variograms were also produced for each grade zone in order to establish any spatial correlation between the copper and molybdenum, silver and gold values. Molybdenum gave very poor cross-variograms, each one containing drift. Silver provided the best results, exhibiting spherical scheme cross-variograms for grade zones three and four, suggesting that the copper and silver sample values are spatially correlated. The copper and gold cross-variograms were reasonably well defined, suggesting that these two metals were also spatially correlated within grade zones three and four. No further work was done to test these two theories, though they now suggest that it may be possible to employ co-kriging to estimate the gold and silver content of the Cerro Colorado deposit. Similarly, there is sufficient evidence to suggest that the silver and gold grade zone boundaries can be thought to be similar to those of the copper grade zone boundaries. Thus, for the purposes of this study, the gold and silver grade zones may be evaluated upon the basis of their content within the proposed copper grade zones. Molybdenum is obviously independent and must be valued independently, though it is important to consider the possibilities of establishing independent gold, silver and molybdenum grade zones at a later date.

Fig.6.1 Isotropic Copper semi-variograms

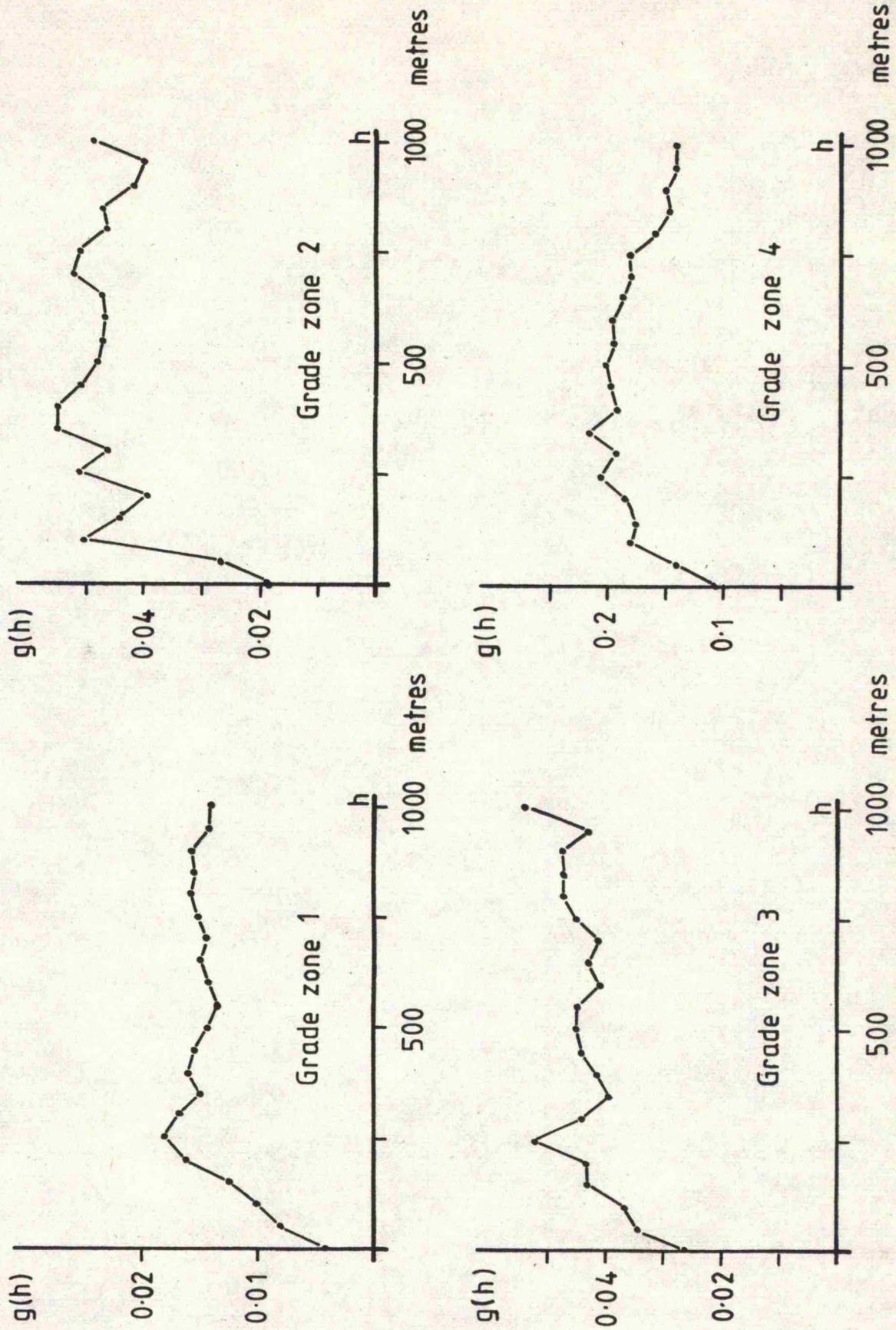


Fig. 6.2 Directional semi-variograms for Copper

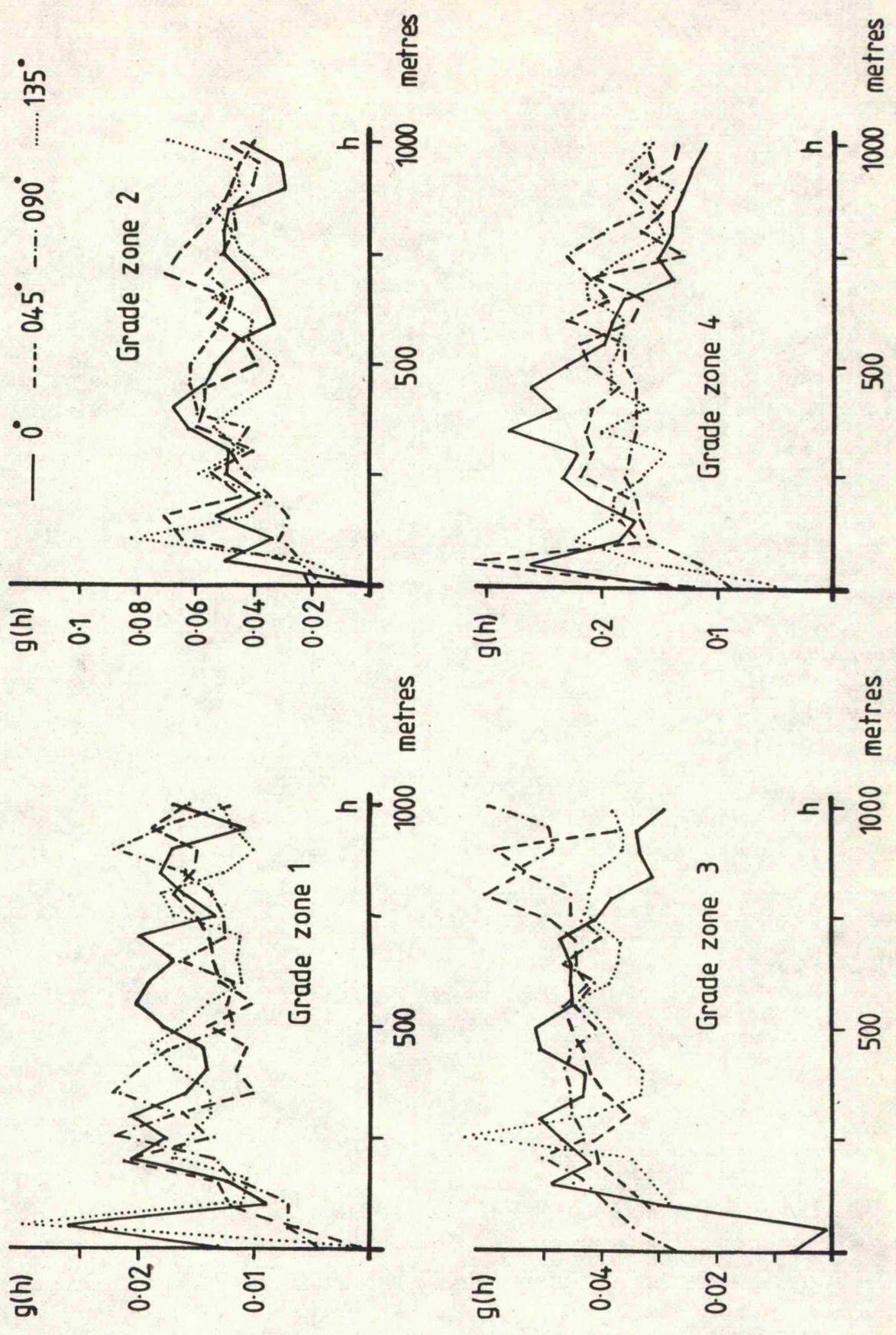


Fig. 6.3 Isotropic Copper covariograms

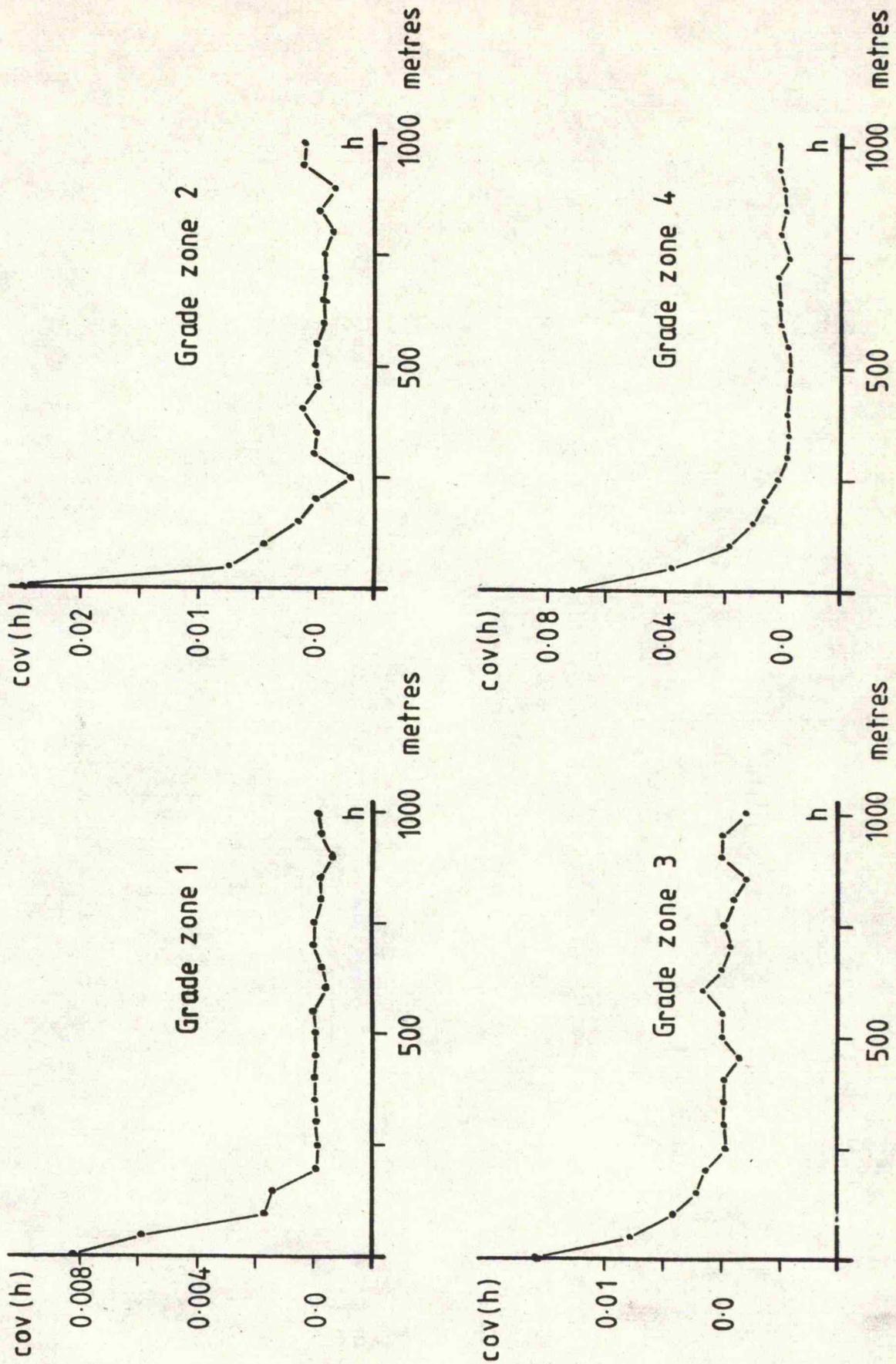


Fig. 6.4 Directional covariograms for Copper

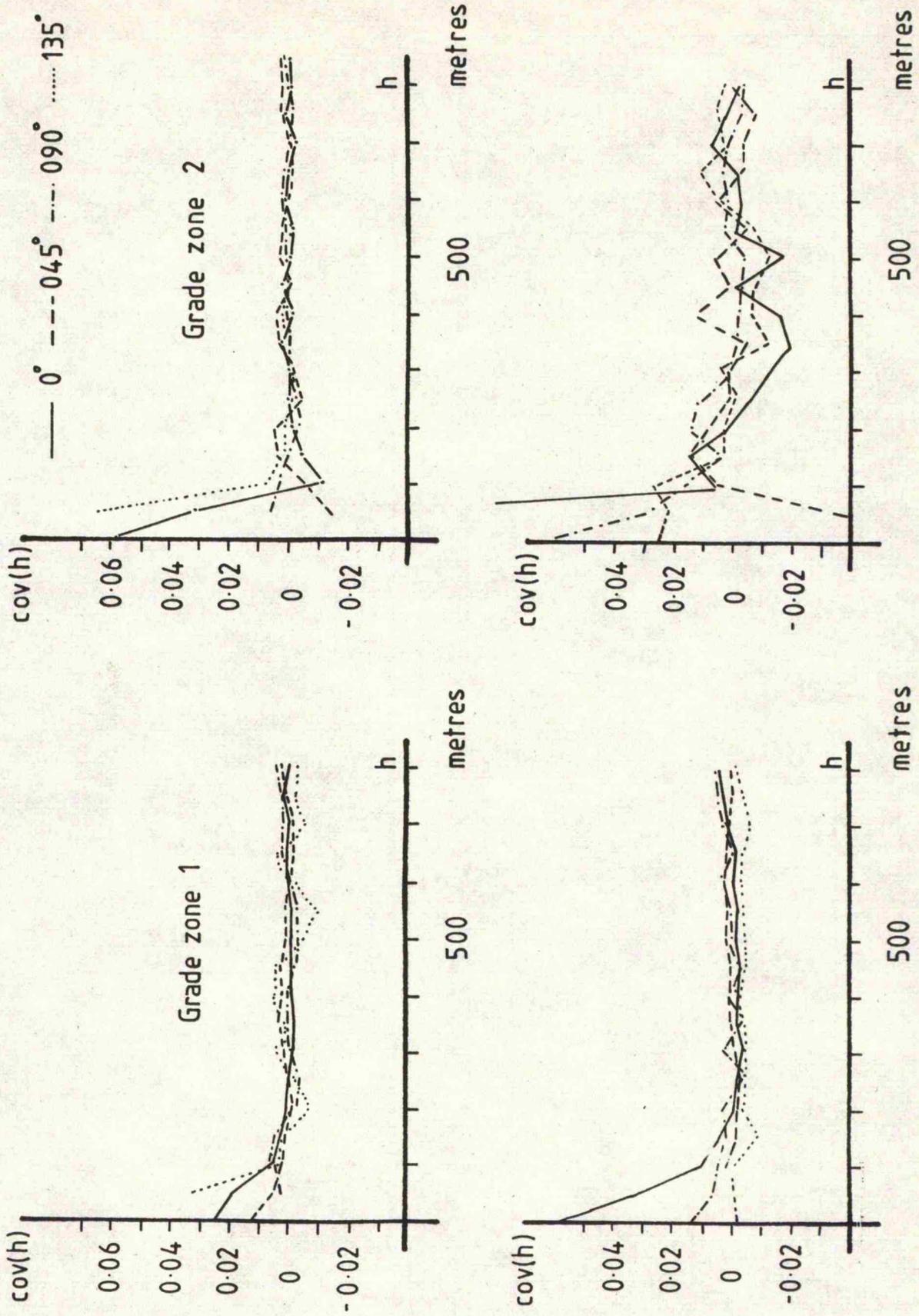


Fig. 6.5 Downhole Copper semi-variograms

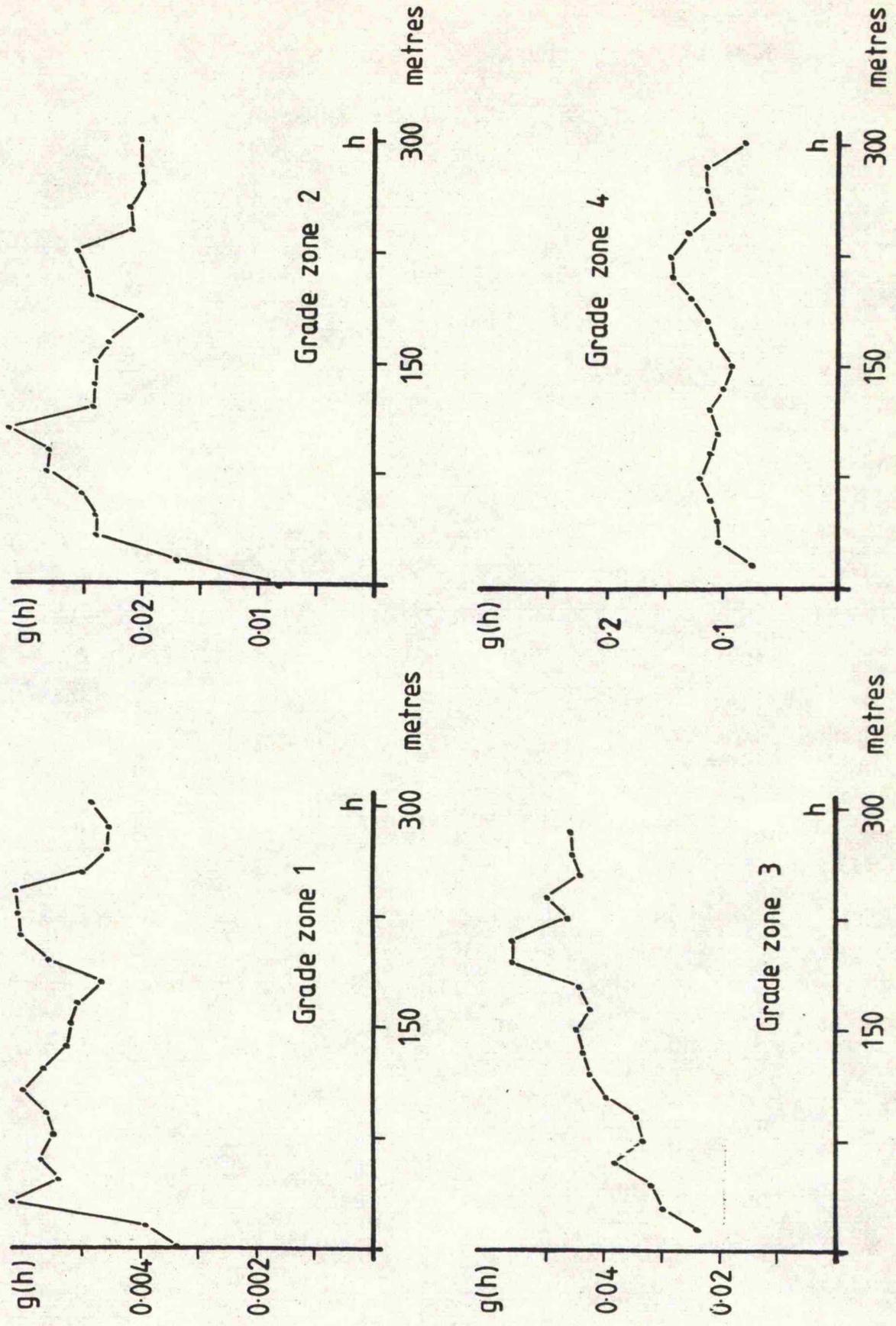


Fig. 6.6 Downhole Copper covariograms

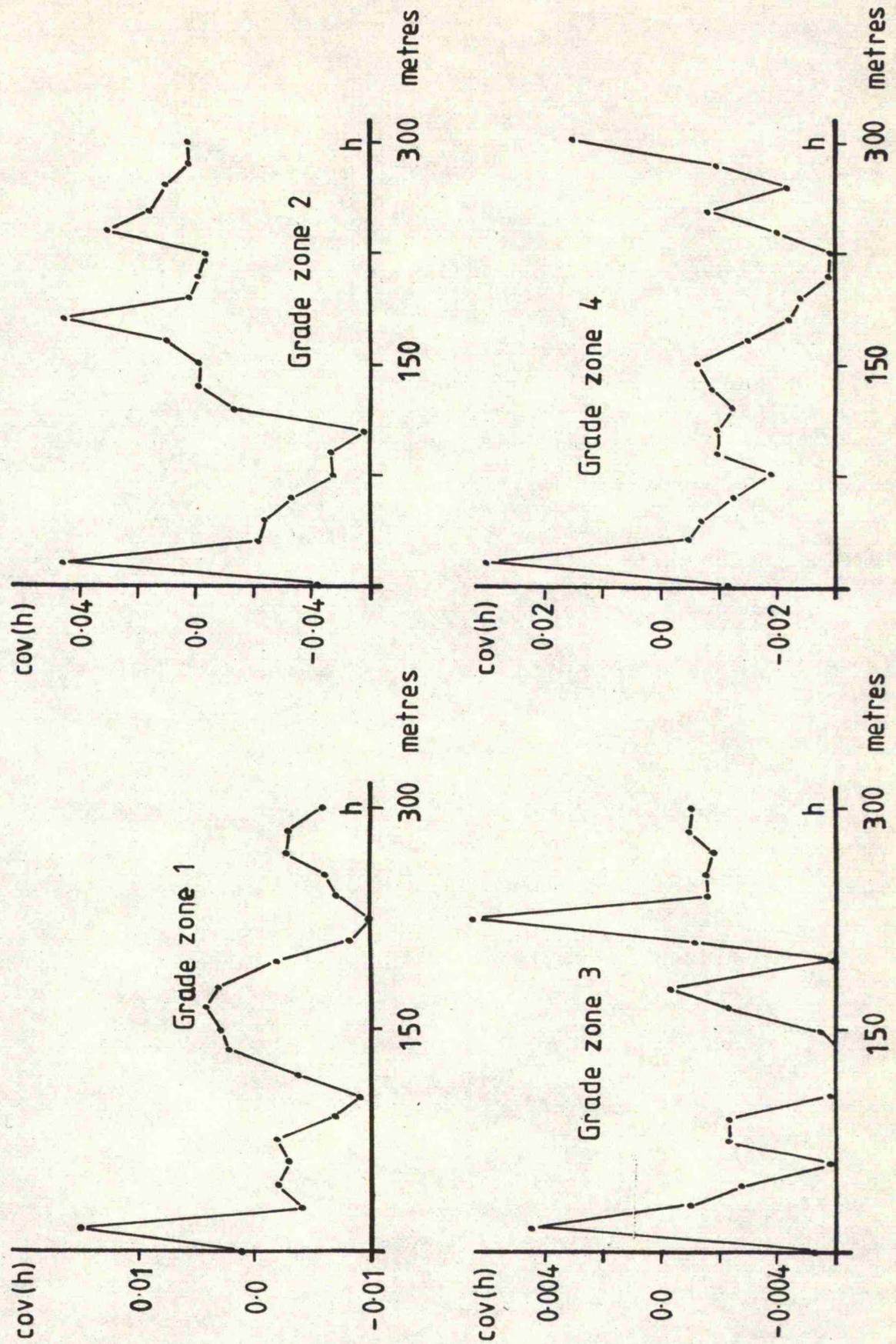


Fig. 6.7 Isotropic drift for Copper

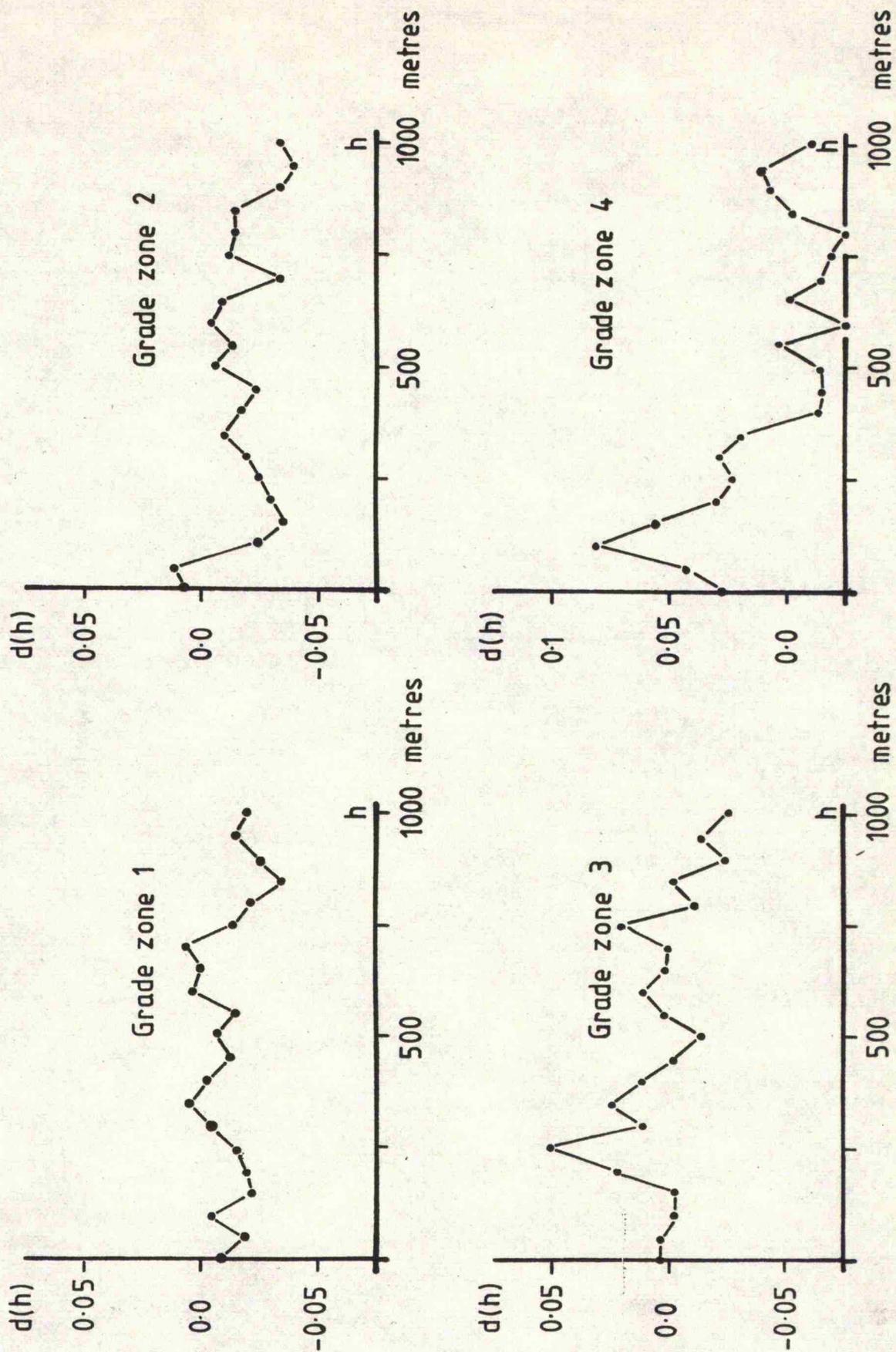


Fig. 6.8 Isotropic sill values for Copper

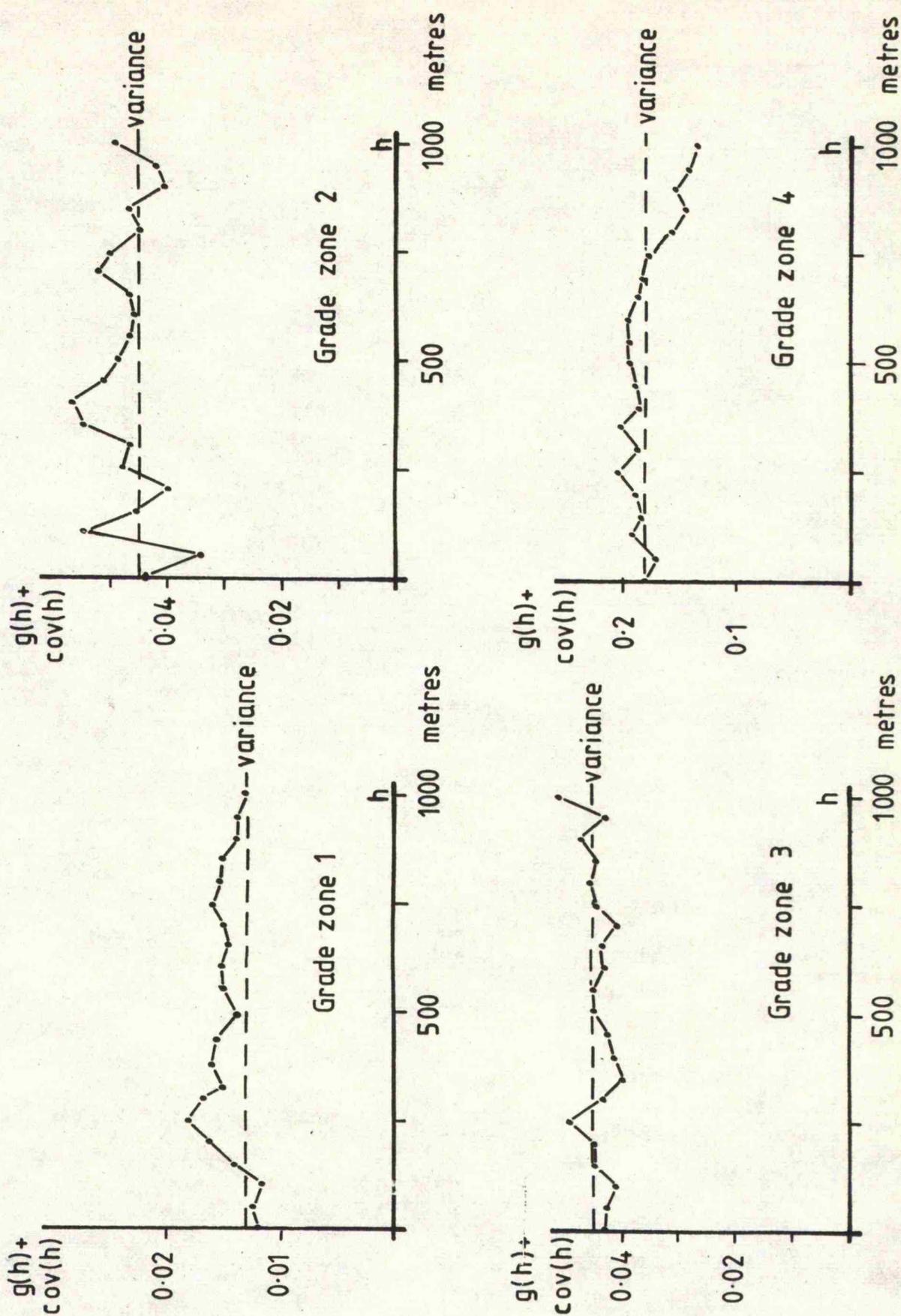


Fig. 6.9 Isotropic semi-variograms for Molybdenum

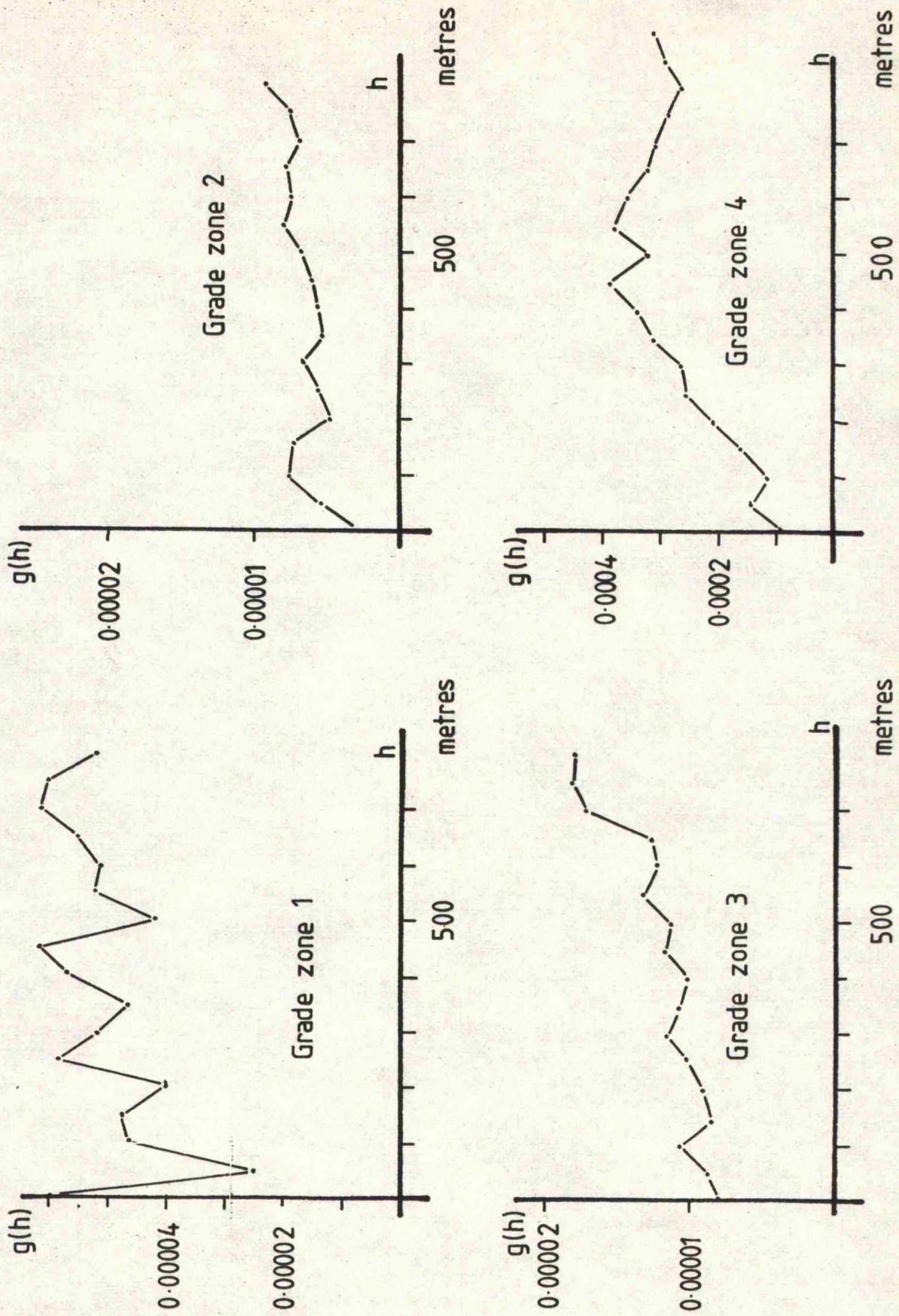


Fig. 6.10 Isotropic covariograms for Molybdenum

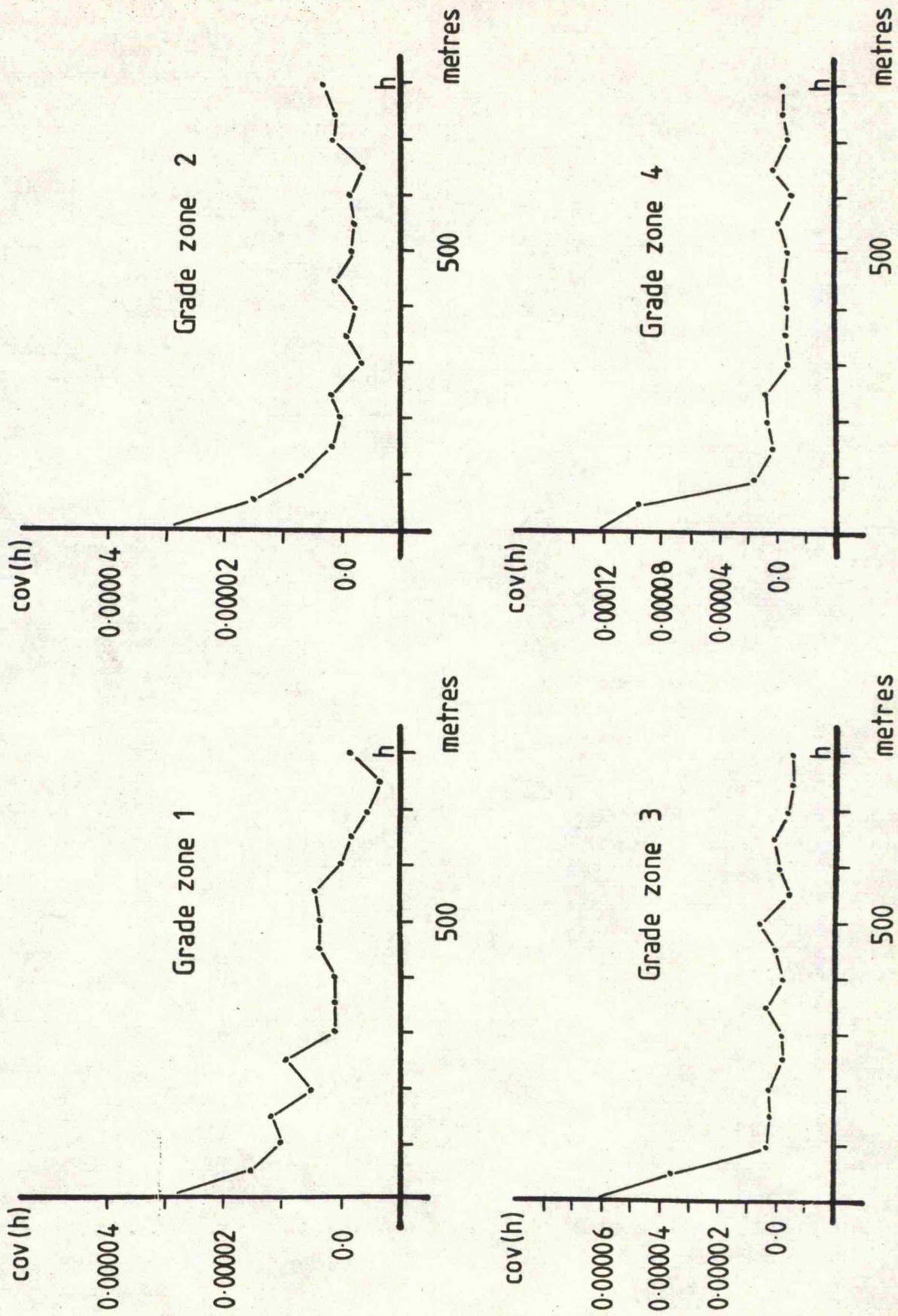


Fig.6.11 Isotropic Silver semi-variograms

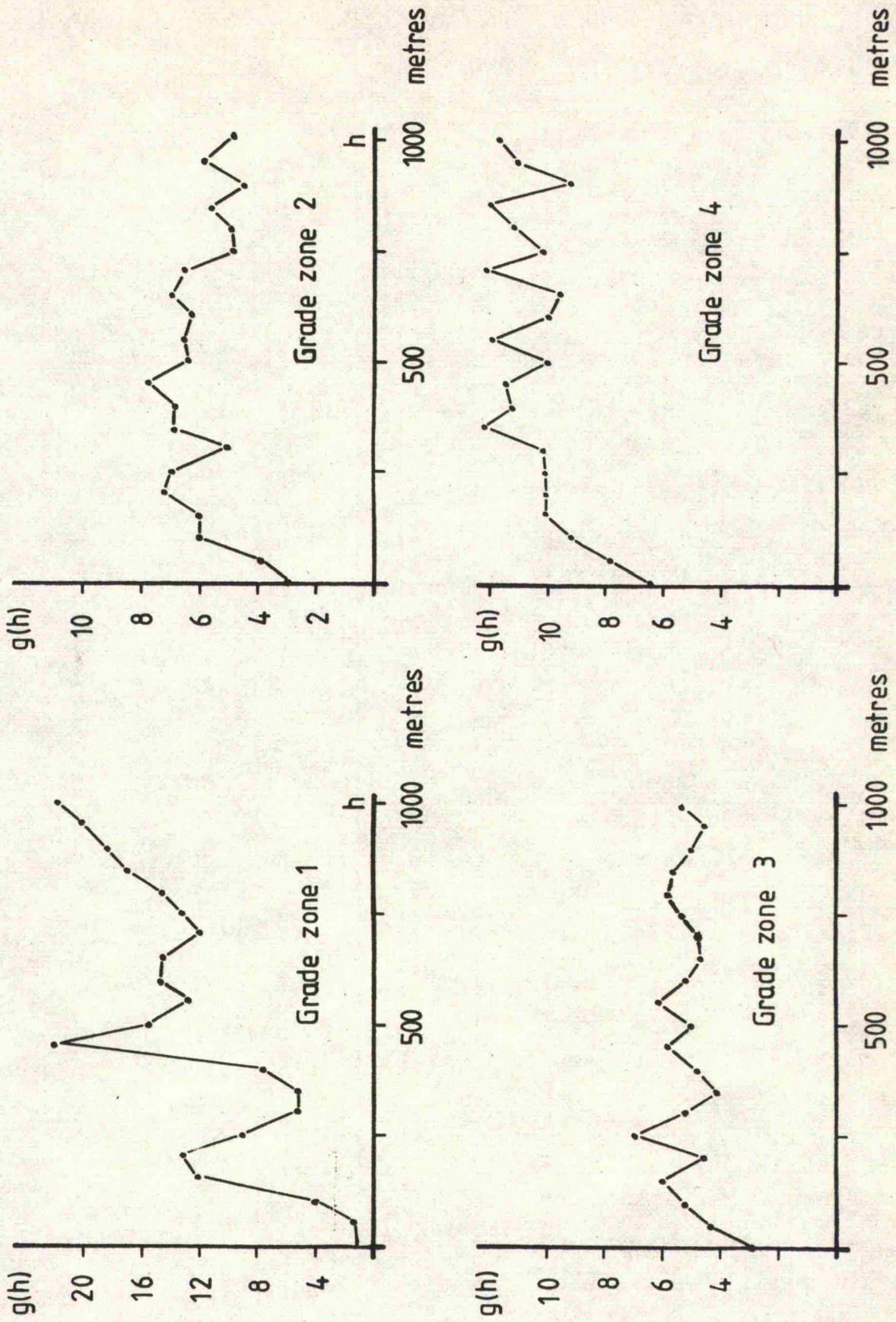


Fig. 6.12 Isotropic Silver covariograms

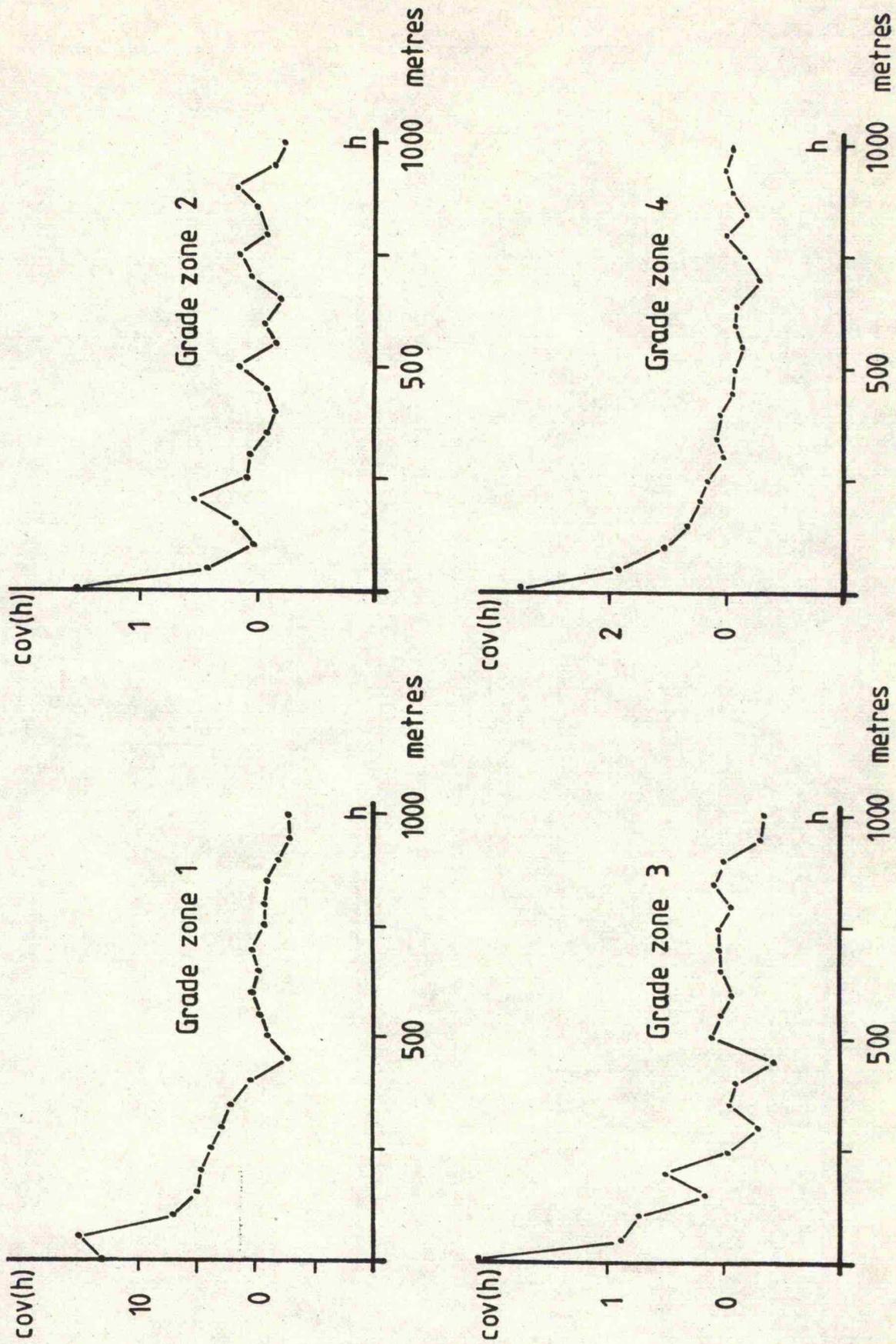


Fig. 6.13 Isotropic Gold semi-variograms

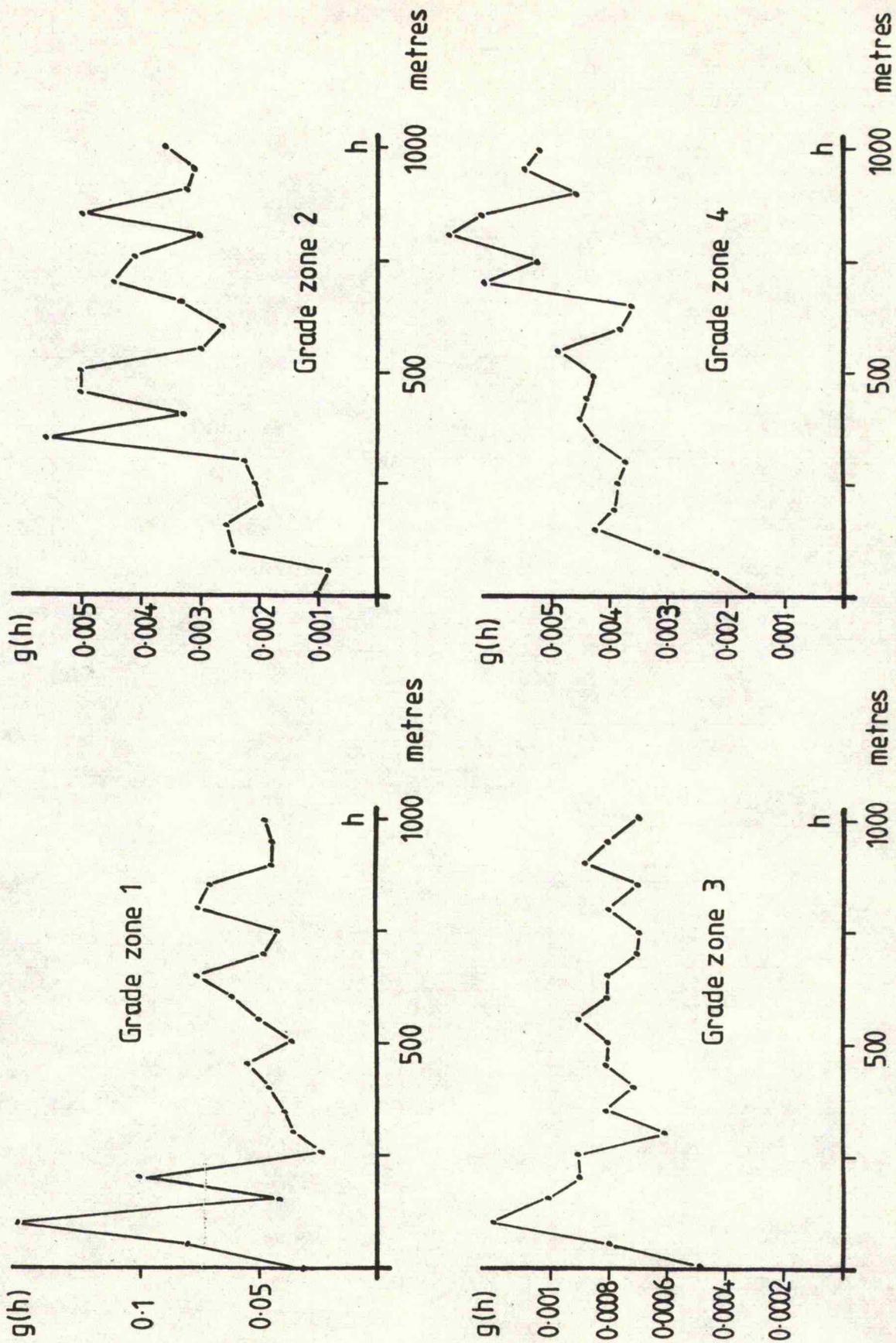
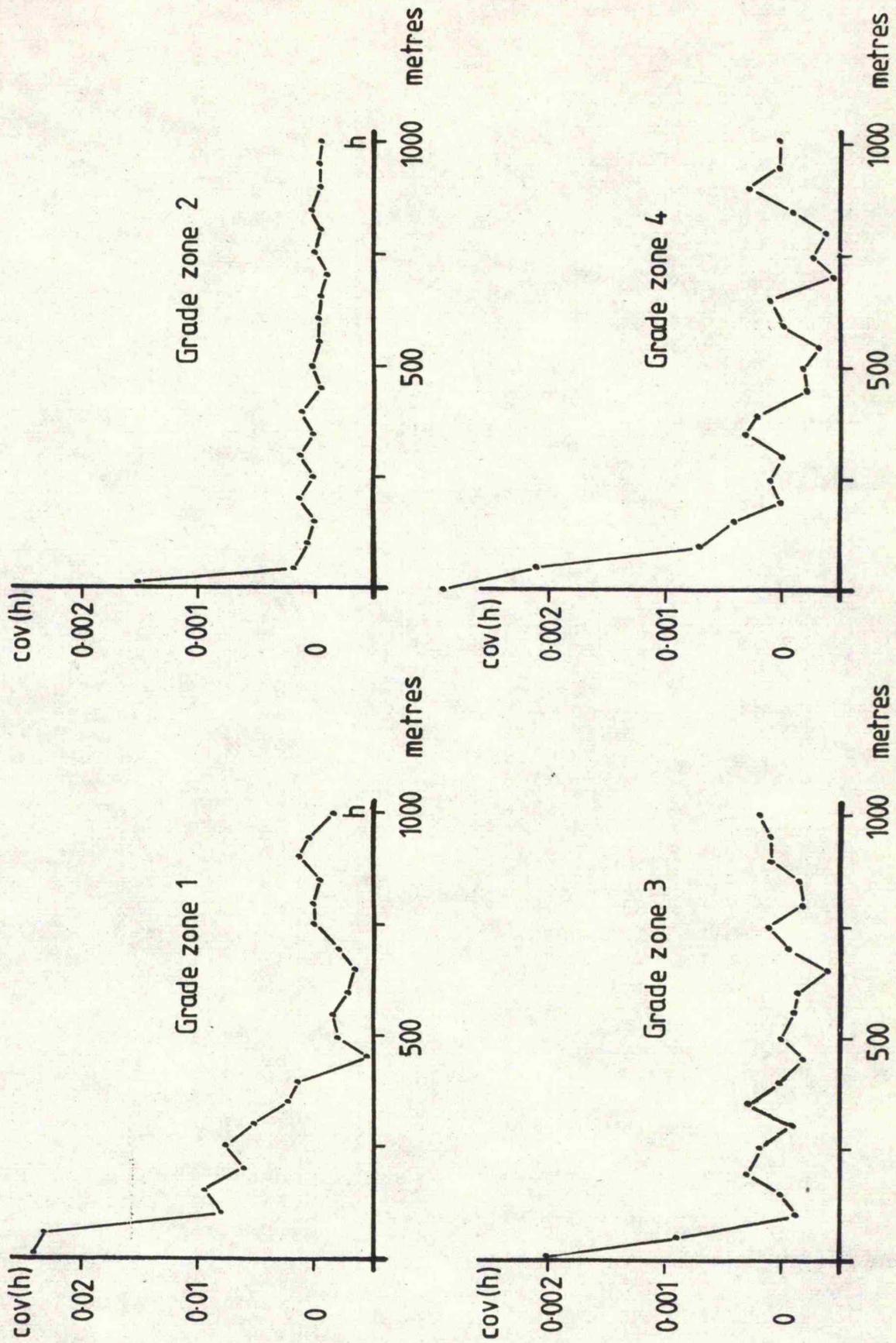


Fig. 6.14 Isotropic Gold covariograms



CHAPTER 7

7.1 Variogram modelling

The first step of a geostatistical estimation procedure is to produce a series of semi-variograms which can be modelled as simply, yet as accurately, as possible. It is important to be able to provide as accurate a model as possible because this will influence all results and conclusions. If poor semi-variograms are available, considerations must be taken as to the validity of geostatistical procedures in the estimation. Further attempts should be made to try and obtain better semi-variograms, e.g. alter the lag distance or to apply some non-geostatistical method. In many cases where good semi-variograms have been obtained the proposed model is considered to be of the spherical scheme type (Fig. 7.1), although other types do exist (Royle, 1977, David, 1977). The expected semi-variance between two sample values, h metres apart, is a combination of the local and spatial variations, i.e.

$$\gamma(h) = C_0 + C(h)$$

where the spatial component $C(h)$ is given by the equation:

$$C(h) = \frac{3}{2} \frac{h}{a} - \frac{(h)^3}{2(a)^3}$$

a is the range of the model variogram in metres.

h is the distance between the two sample values in metres.

A polynomial equation could be established to model the semi-variogram, possibly giving a better fit to the points (Fig. 7.2). This has two drawbacks. Firstly, the resulting equation may be complicated and unstable given certain conditions. Every time a new model is created a new series of integrals and approximations are

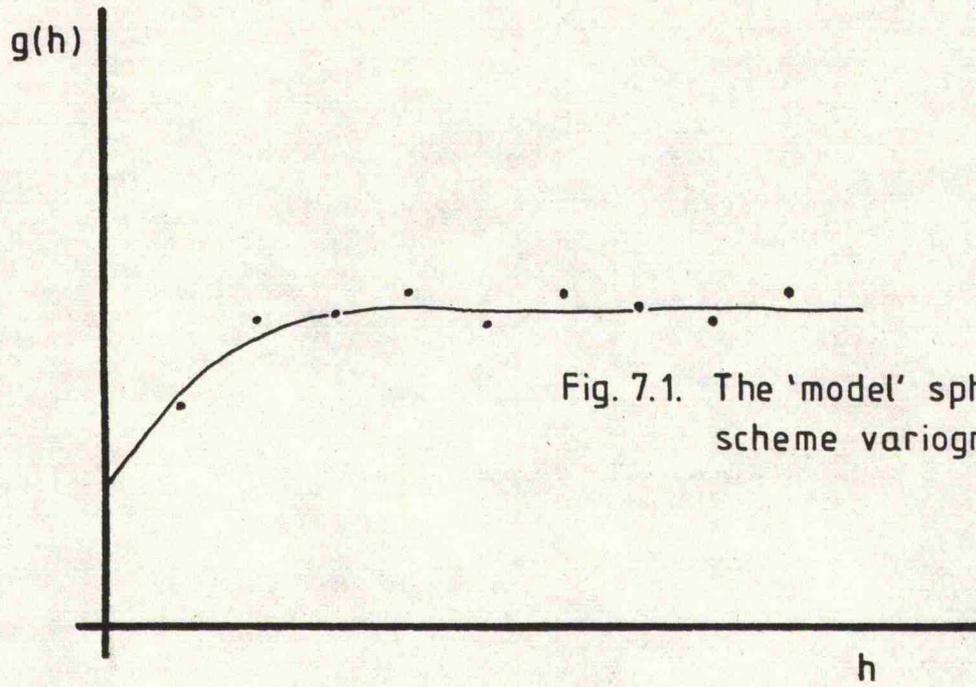


Fig. 7.1. The 'model' spherical scheme variogram

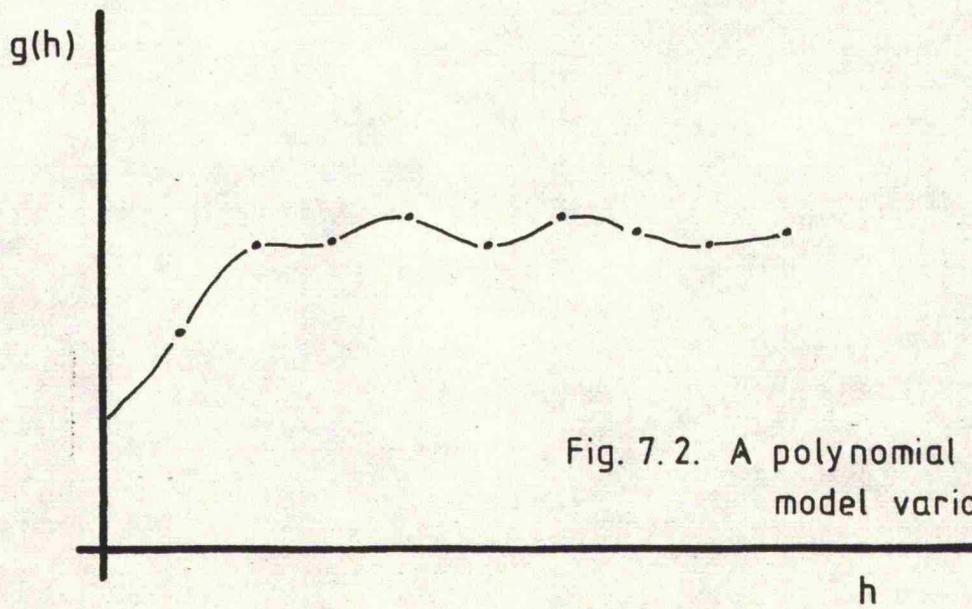


Fig. 7.2. A polynomial 'exact' model variogram

required which may often be time consuming and do not justify any refinements over the spherical scheme model. Secondly, it is not worth trying to fit an exact equation to the points of the semi-variogram since their values are influenced by differing degrees of errors and number of sample values. There are many physical reasons for expecting different degrees of continuity at different scales of investigation (Clarke, 1979) which can be attributed to geological processes.

7.2 Obtaining the optimum variogram model

Numerous cases have arisen where a semi-variogram has been obtained, the model variogram fitted by eye and the 'theoretical' parameters used to kriging the area of interest. No considerations have been taken as to their goodness of fit or if they are indeed the optimum parameters. Kriging gives an indication as to the error of estimation by the size of the kriging variance. It would therefore seem reasonable to try and make use of this fact in order to test the quality of the variogram parameters.

The ideal method for testing any parameters for a particular evaluation model is to take a known sample (z), remove it from the data set and then estimate its value (z^*). Its error of estimation will be $(z - z^*)$ and its estimation variance $(z - z^*)^2$. This can be repeated for every sample value within a test area and the mean error of estimation and estimation variance can be calculated. The optimum parameters will produce an unbiased estimate with the minimum estimation variance, i.e.

$$\bar{z} = \bar{z}^*$$

and σ_{kv}^2 is a minimum.

The model variogram parameters can be tested by point kriging the sample values. Each sample value is removed from the data set and its

value kriged from the nearest surrounding sample values. It is now possible to calculate the overall error of estimation and estimation variance. The method of kriging produces its own estimation variance in the form of the kriging variance and so a direct comparison can be made between the actual and estimated values. The optimum kriging parameters have been obtained when:

$$\sum_{i=1}^n (z_i - z_i^*) = 0 \quad \text{and} \quad \sigma_{kv}^2 = \sigma_{ev}^2$$

7.3 Considerations for variogram modelling

The variogram is based upon the relationship between the actual sample values and provides an indication to the expected variability between two samples. However, irregularities often occur within a set of data and are usually the result of misinterpretations or geological influences. These produce definite patterns of features within the semi-variogram which must be considered when fitting the model variogram.

7.3.1 The hole effect

A hole effect is reflected in the semi-variogram by the presence of one or more 'bumps' on the variogram curve (Fig. 7.3). It is due to the fact that areas exist where high grade values alternate with low grade values over small distances. Occasionally, periodic behaviour (Fig. 7.4) may be detected and is again related to numerous alternations between areas of high and low grade. Usually, the fluctuations tend to decrease with lag distance h . In most cases any bumps or periodicities can be disregarded unless there is a strong, reasonable interpretation for their presence (Delfiner (1979)).

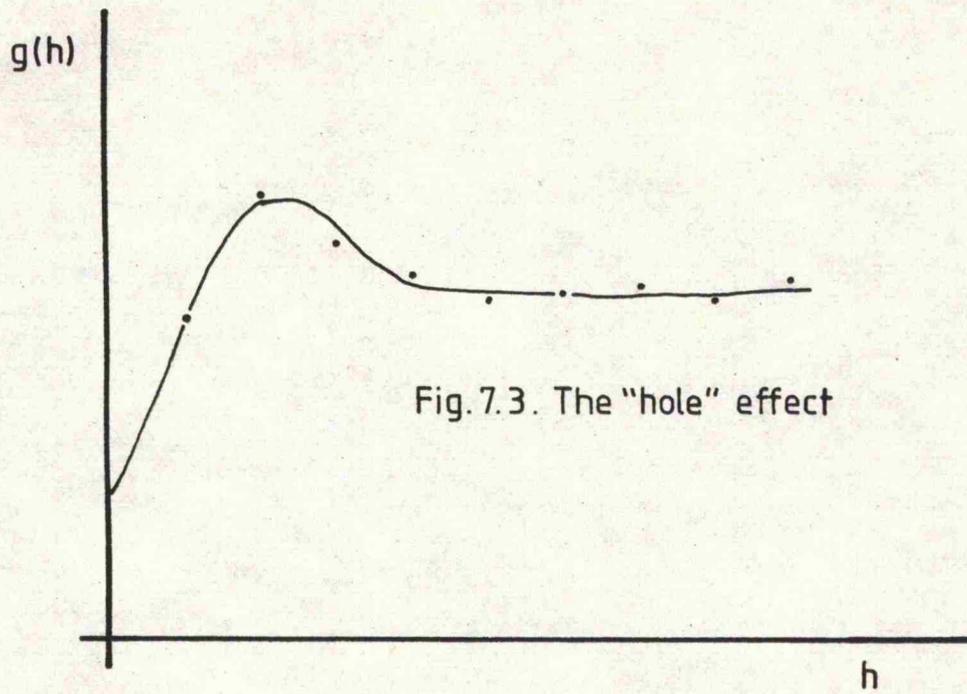


Fig. 7.3. The "hole" effect

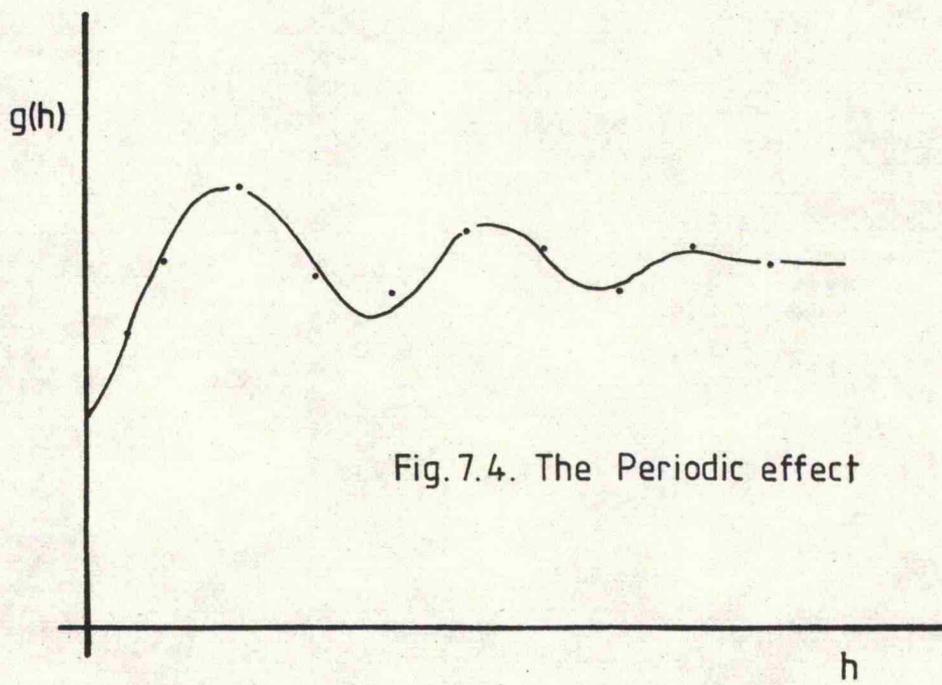


Fig. 7.4. The Periodic effect

7.3.2 Compound structures

More than one structure may be present within the variogram and is highlighted by the presence of two or more sills (Fig. 7.5). This suggests that there is a superposition of sample variation at different scales within the deposit. These are often related to the presence of small scale structures within larger ones, e.g. the location of small mineralised lenses within the main mineralised province. All of the sources of variability may act simultaneously to produce a series of nested structures.

7.3.3 Anisotropies

Any anisotropy within the mineralisation can be detected by producing semi-variograms of sample values taken in different directions (Section 6.2). If the degree of continuity, the nugget effect and range are the same for each direction, the semi-variograms are said to be isotropic and are indicative of similar sample variation in all directions within the deposit. In many cases this is not so, often with the variogram having different ranges in different directions (Fig. 7.6). A geometrical anisotropy is said to exist when a simple geometrical transformation will transform the variogram for one direction onto the variogram for another direction. This type of variogram is typical of lens shaped deposits and indicates that a particular sample value has a greater influence upon the samples lying in one direction than in another.

A second type of anisotropy exists where the semi-variance is greater in one direction than in another, though the range of sample influence usually remains constant (Fig. 7.7). This is often referred to as zonal anisotropy, which is typically related to large zones of high and low grade values within a deposit. Proportional effects have

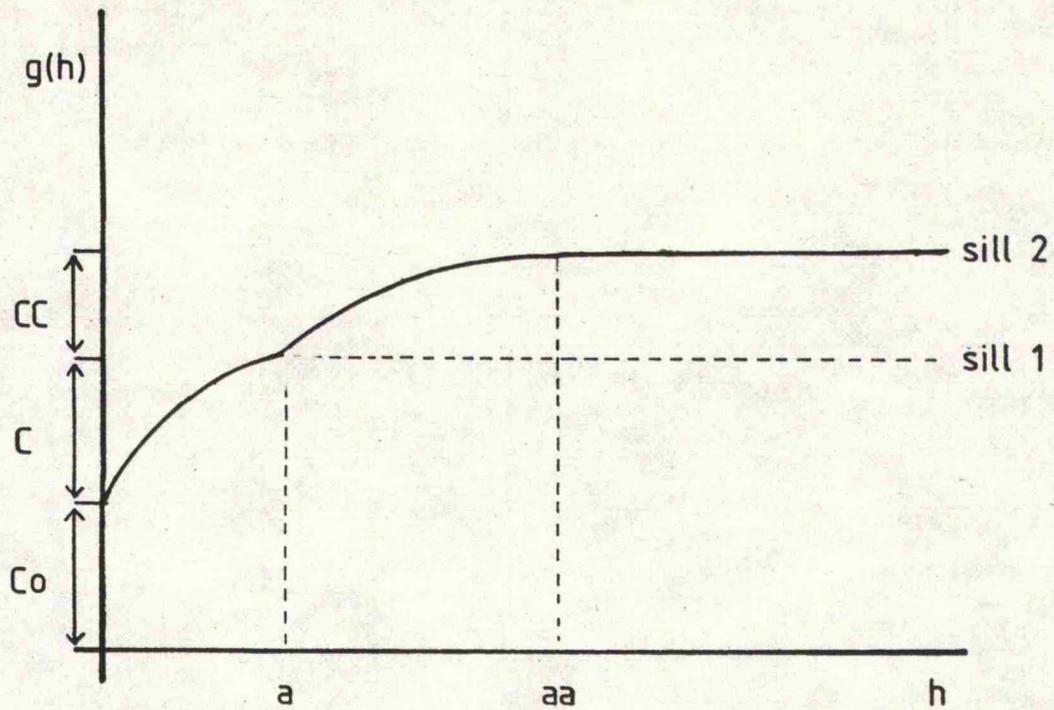
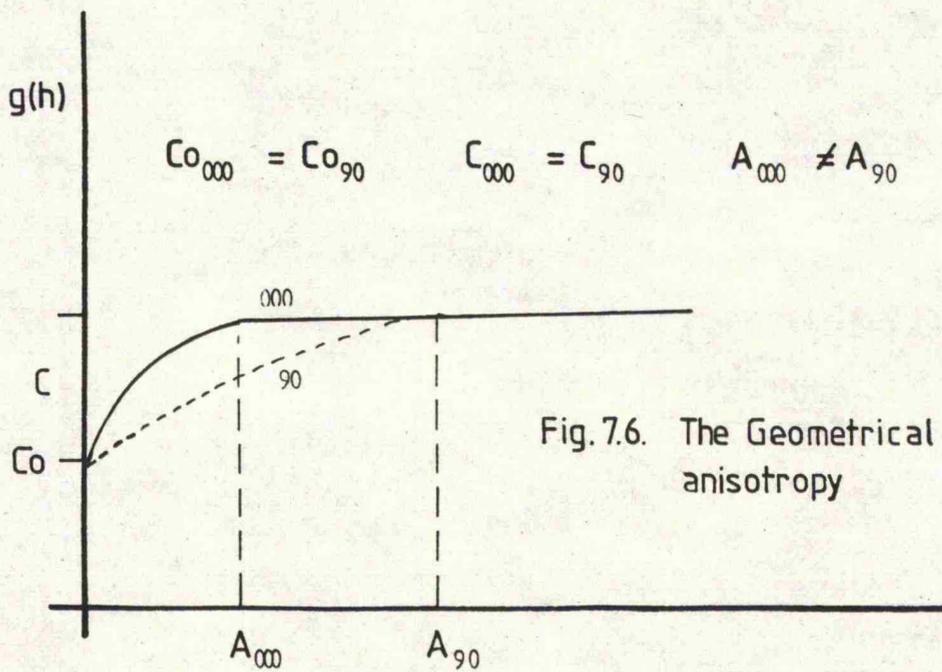
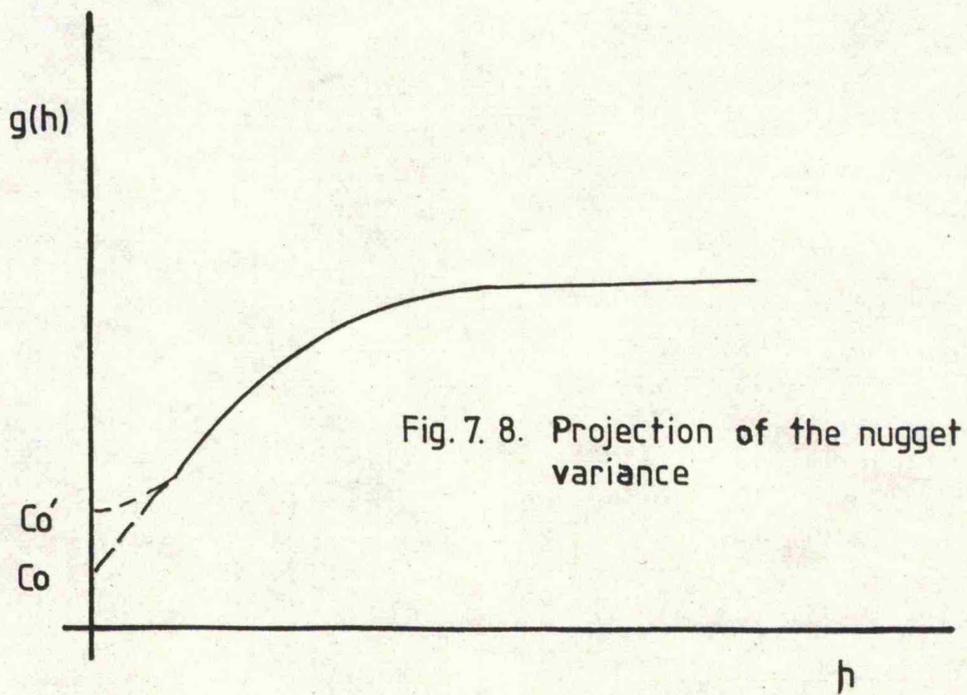
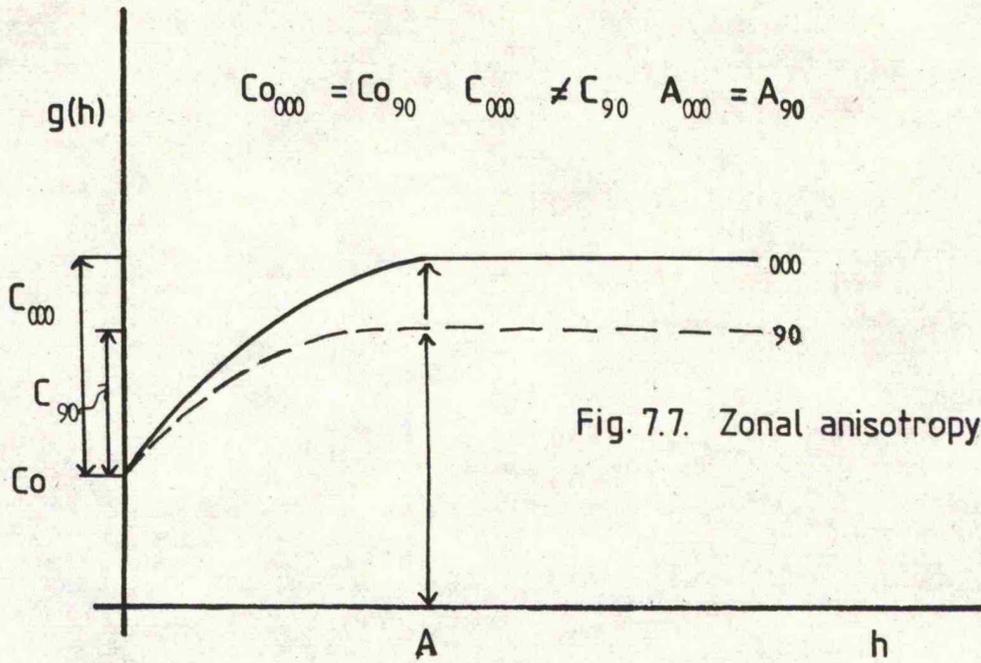


Fig. 7.5 The Compound structure





a similar appearance representing a greater discrimination between the high and low grade values in a particular direction. These may be removed by taking the relative semi-variogram, i.e. dividing the semi-variance for each lag interval by the square of the mean of the sample values used in its calculation.

7.3.4 The nugget effect

The semi-variogram is based upon the variability between the sample values at various distances apart and so there must be a minimum distance below which there are no sample pairs available for use. As a result, the model variogram has to be extrapolated over the distance between the first known point of the semi-variogram and the Y-axis. If the variogram curve intersects the Y-axis at some positive value, a nugget effect is present. If $\gamma(h) = 0$ when $h = 0$, no nugget effect is present. The interpolation to lag zero is subjective and is often a major source of error (Fig. 7.8). In most cases a nugget effect will be found although a detailed examination of the variogram shows that it has a value of zero at lag zero and a positive value at a very small lag interval which gives the appearance of a nugget effect (Royle, 1978A).

7.3.5 The screen effect and the number of samples to be used in kriging

The screen effect gives an indication to the amount of influence produced by the more distant sample values and is represented by the ratio of the nugget effect to the continuity, i.e.

$$E = \frac{C_0}{C} \quad (\text{Royle})$$

In kriging, the greatest weighting coefficients will be assigned to the nearest sample values, the smallest to the furthest from the block to be kriged. As the distance between the block and sample

increases so the weighting coefficients tend to zero. A screen is established in this manner by the near sample values which removes the influence of the furthest sample values. A high screen effect is indicated when the value of E tends to zero and as the nugget effect decreases so does the effect of the screen. In some cases the nugget effect is so large that the variogram is considered to be a pure nugget effect and the best estimate for any block or panel is that of the mean value of the whole deposit (Royle, 1977).

The screen effect can be used to determine the maximum size of sample search for the kriging program. Obviously, too large an area would provide too many sample values, many of which will have very little influence in the evaluation. Too many unnecessary sample values will increase the computer run times adding nothing to the accuracy of the estimate. Too small an area may not provide enough sample values, or assign them too large coefficients, thus emphasising any errors. The optimum size of sample search is best found by examination of the sampling pattern, though it should not exceed the range of influence of the sample values unless the sample data is sparse.

It is unwise to krig with too few sample values and pointless with too many. Six sample values are often considered to be the minimum (Royle, 1977A), although four may often suffice. The maximum number is usually governed by the screen effect and the area of sample search, though usually between twelve and fifteen are adequate. Care must be taken to ensure that the sample values are well distributed in and around the block or panel and not confined to a small cluster or linear arrangement, though it may not be possible to avoid this situation.

7.3.6 The continuity

The kriging variance is directly proportional to the continuity (Royle) and this fact can be made use of in the refinement of the variogram parameters. A variogram model can be tested by point kriging the data and noting the ratio between the estimation and kriging variances. Assuming that the nugget effect and range of sample influence remain the same, the continuity can be adjusted according to this ratio. The point kriging procedure can be repeated with the new parameters until an unbiased result can be obtained. If the results remain biased after two or three refinement attempts there is the possibility that either the nugget variance or range of influence is wrong and completely new parameters are required.

7.3.7 The range of sample influence

The greater the range of the variogram, the greater the distance over which the sample values are correlated. Thus the magnitude of the weighting coefficient assigned to a sample value during kriging will depend upon the range. The shorter the range the quicker the rate at which the sample influence decreases and a distant sample value will be assigned a small weighting coefficient regardless of any screen effect.

7.4 Checking the variogram model

Point kriging can be used to give an overall indication of the amount of bias involved in the estimate. Listing each sample value and its kriged estimate will give an indication of the bias involved in each estimation. However, this has no practical value since it is not possible to provide zero bias for each local point. A more meaningful result may be achieved by considering the average local bias. This would involve the use of the cross-variogram.

The average semi-variance for a series of sample values, h metres apart, is given by the equation:

$$\gamma(h) = \sum_{i=1}^n \frac{[z_i(x) - z_i(x+h)]^2}{n}$$

and the cross-variance:

$$\gamma_{cv}(h) = \sum_{i=1}^n \frac{[(z_{1i}(x) - z_{1i}(x+h)) (z_{2i}(x) - z_{2i}(x+h))]}{n}$$

where n is the number of sample pairs.

In this case the cross-variance will be:

$$\gamma_{cv}(h) = \sum_{i=1}^n \frac{[(z_i(x) - z_i(x+h)) (z_i^*(x) - z_i^*(x+h))]}{n}$$

In the unbiased case the average of the estimated values in the cross-variogram of actual and estimated values will be very similar to the original semi-variogram. This approach was tested on a set of data from a Chilean porphyry copper deposit with success.

7.5 Calculation of the theoretical sill value

In the case of the spherical scheme variogram, the sill of the variogram is considered to be that of the variance of the sample values within the deposit (David, 1977). This fact is often used in variogram modelling and in establishing the covariance kriging matrix and assumes that only one population of sample values is present. In reality, slight irregularities can occur within a set of data, either due to errors or some direct geological control. In the first case, filtering the values may have an effect but should not really be used in the second case as crucial samples may be removed. In order to check for

this it may be worthwhile to compare the theoretical sill directly with the variance.

Consider the variograms and covariograms for a theoretical set of standardised data which contains only one population of values. Since the variance is 1.0 it can be considered that the variogram will have a sill value of 1.0. It is assumed that there is a nugget effect of zero (Fig. 7.9). The covariogram will have a C value of 1.0 at lag zero which will decrease to zero at the range of the variogram. Thus, the covariogram is a direct mirror image of the variogram about the value $\gamma(h) = 0.5$. Therefore, if the covariance and semi-variance are added together for each particular lag interval, a value of 1.0 is obtained which is equal to the sill value. Introduce a nugget effect and repeat the procedure, a similar result will occur.

Thus, for any given lag interval the standard equation will apply:

$$\text{Cov}(h) = \text{Variance} - \gamma(h)$$

or, in this case:

$$\text{Cov}(h) = \text{Sill} - \gamma(h)$$

The covariance and semi-variance may be calculated directly from a set of sample values and so it may be possible to calculate the theoretical sill value for each lag interval. This was tested on numerous data sets and gave very stable results, even in cases where the semi-variogram or covariogram appeared to be unstable. In many cases the theoretical sill value approximated to the variance, although in some cases the theoretical sill value was less than that of the variance, especially in cases where more than one sample population was present.

This technique was applied to the Cerro Colorado data set, giving relatively good results. These are illustrated and discussed in the section concerned with the structural analysis (section 6.3).

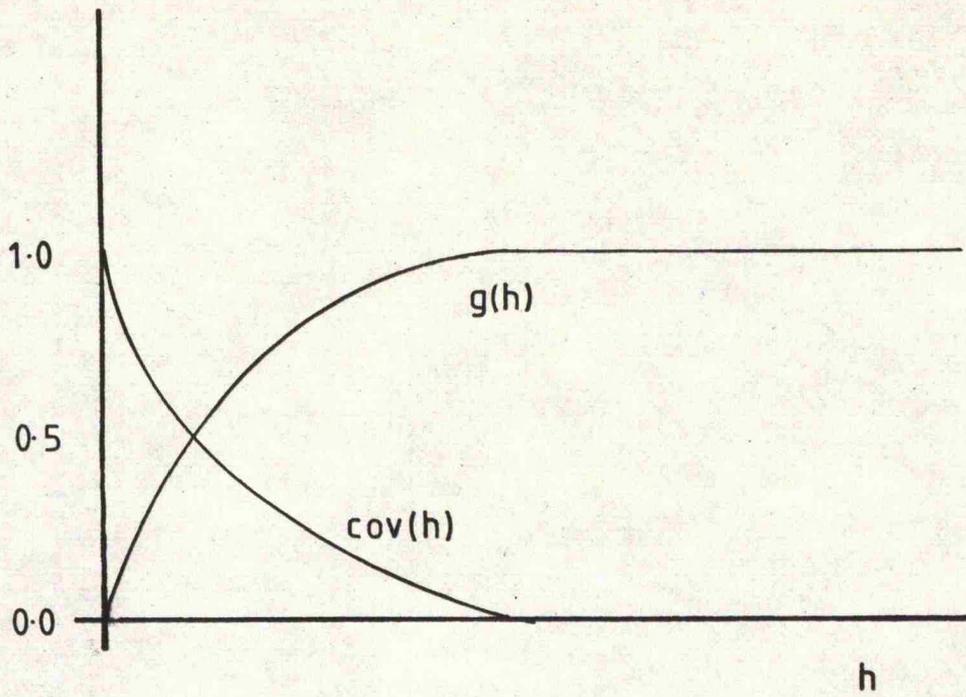


Fig. 7.9. The theoretical relationship between the variogram and covariogram

Obviously, a great deal more work has to be done on this topic to see how stable the concept is for any type of data. It is interesting to note that encouraging results are produced from unstable semi-variograms and covariograms. It is possible that by using the theoretical sill value in a covariance, kriging matrices will produce a better result than when using the variance.

7.6 Use of the covariogram as opposed to the semi-variogram

In many cases the covariogram is unstable and difficult to model and usually any required variances and covariances can be obtained from the semi-variogram which is generally easier to model. However, it is much quicker to krig from a computational point of view by using a covariance matrix as opposed to a semi-variance matrix. A less complicated matrix inversion algorithm is required since it is not necessary to solve a matrix with a diagonal filled with zeros. Comparisons of run times show that the covariance matrix is up to 30 percent quicker and hence slightly cheaper to use.

Two methods are available in order to establish a covariance matrix. The first is to model the covariogram and calculate the covariance directly. In most cases a spherical scheme covariogram can be used, although, as in the case of the variogram, other models do exist. For two sample values h metres apart the covariance will be:

$$\text{Cov}(h) = C - C(h)$$

where $C(h)$ is the same as in the semi-variance equation.

The advantage of this method is that no estimation is required of the nugget variance. Instead, it is assumed that the variance of the samples is the same as that of the deposit and as a result the leading diagonal of the covariance matrix is set to the sample variance. Point kriging can be used to test the model covariogram parameters and can be

refined in a manner similar to that of the variogram. Because no nugget effect is present, the only refinements required are to the range and continuity.

The second method converts from a semi-variance into a covariance by the equation:

$$\text{Cov}(h) = \text{Variance} - \gamma(h)$$

or, in most cases

$$\text{Cov}(h) = \text{Sill} - \gamma(h) \quad (\text{for the spherical scheme case})$$

It must be remembered that the kriging variance is now calculated differently, although the calculation of the estimated value remains the same. The kriging variance is now calculated by the equation:

$$\sigma_{kv}^2 = \text{Variance} - \sum_{i=1}^n a_i \gamma_{Ai} - \mu$$

or in more practical terms:

$$\sigma_{kv}^2 = \text{Sill} - \sum_{i=1}^n a_i \gamma_{Ai} - \mu$$

CHAPTER 8

8. Kriging in three dimensions

8.1 When to use a three dimensional approach

The type of kriging to be employed in a mineral evaluation programme is dependent upon the type of deposit and the quality of the data available. Usually, the data is gained from a series of boreholes drilled through the area of interest. Most boreholes are vertical, though may be inclined in order to obtain a better understanding of certain structures. The information can be considered to represent a series of point locations whose positions can be described by a series of X, Y and Z co-ordinates relative to a given origin. In some cases the vertical co-ordinates of the sample values are unnecessary in the evaluation calculations. Certain types of sedimentary deposits may be valued using this assumption. Sedimentary deposits consist of a series of initially near horizontal layers of sediments whose sample values may not be related to one another (section 3.2.1). Any sample values located outside the layers of interest will have no influence on those within and the relevant sample values may now be thought to represent a series of point values on a two dimensional plane (Clarke, 1977). Panel kriging may be suitably applied to this type of evaluation.

Many non-sedimentary type deposits, e.g. porphyry coppers, contain sample values whose influences are not restricted by geological boundaries in the same manner as for sediments. Mineralisation has resulted from the diffusion of mineral bearing fluids through a given volume of rock and is dependent upon the structure and the mobility of the fluids. Thus, there is a greater chance of a given sample being influenced by a series of samples located within constraints in the three dimensional space of the deposit. Any valuation for this type of

deposit must try and include this three dimensional influence of the sample values.

8.2 Problems involved in the three dimensional valuation of a deposit

Consider the two sample points A and B located at positions (X_1A, X_2A, X_3A) and (X_1B, X_2B, X_3B) respectively (Fig. 8.1). A can be related to B and B to A in terms of a vector displacement which is dependent upon the direction and distance apart of the two sample values. Their statistics are independent of direction and distance but may be selectively measured to do so. Semi-variograms can be produced for any sample directions in order to establish any directional trends and the amount of sample variability for a given direction. This may be repeated for numerous directions and orientations providing that there are enough sample pairs for use. The nature of the sampling ensures that most of the sample pairs lie within either the horizontal or vertical planes of the deposit. Therefore, it is customary to consider only the vertical and horizontal semi-variograms unless many of the drill holes are inclined. The main problem is that the boreholes are separated over a much greater horizontal distance than the vertical sampling interval within the boreholes. This means that the vertical semi-variograms will be based upon a much smaller sample support than the horizontal semi-variograms and will be better defined over small lag distances. It is impractical to produce vertical semi-variograms based upon the same lag interval as the horizontal semi-variograms simply because the vertical extent of the data is not as great as in the horizontal directions. Consequently, only two or three lag intervals would result and would give a meaningless interpretation of the vertical sample variability.

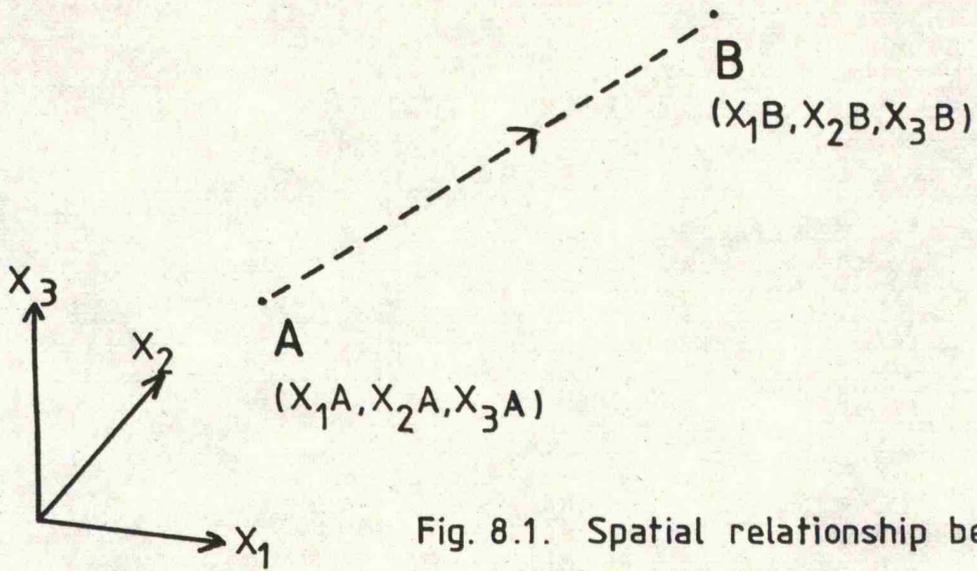
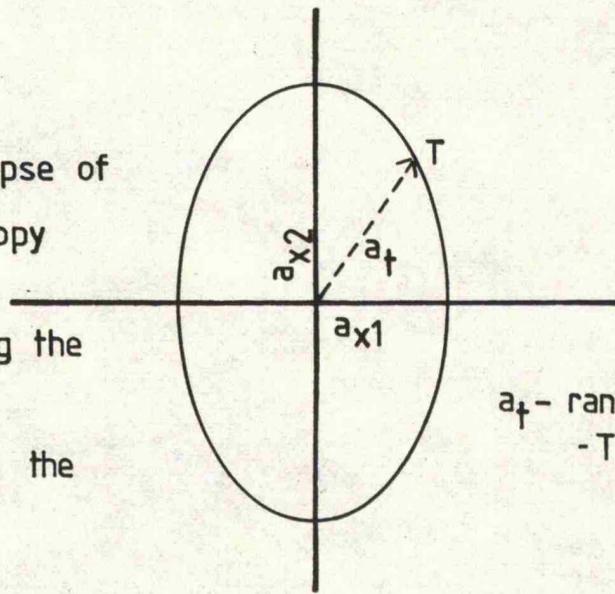


Fig. 8.1. Spatial relationship between two sample points

Fig. 8.2. The ellipse of anisotropy

a_{x2} - range along the x_2 axis

a_{x1} - range along the x_1 axis



a_t - range along direction $-T$

This may not be so large a problem as many consider it to be. Many vertical semi-variograms exhibit much shorter ranges, and/or smaller sill values than those of the horizontal variograms and can be explained as a result of the differences in sampling intervals. However, some types of deposits have a good deal of vertical control within the geology and this would also account for the differences in the variogram structures. The problem lies in the fact that there is very rarely any closely spaced sampling within the horizontal plane which can be used to determine the structure of the horizontal variogram over small lag intervals. This may be remedied by further sampling on a closely spaced grid over a small area. Here again, sample values measured over different supports will be used though, providing that there are enough samples, this should have little effect on the construction of the semi-variogram at small lag intervals.

It must also be remembered that it is often unnecessary to provide a model which describes the whole of the semi-variogram, i.e. only fit a model to the portion of the semi-variogram which will be needed for evaluation purposes. Suppose the range of the model variogram was 500 metres and the screen effect showed that samples over 200 metres away had very little influence in kriging. It seems reasonable to suggest that the only relevant part of the variogram curve is that between 0 and 200 metres and only this portion should be modelled. In reality, anisotropic variograms may be adequately modelled as an isotropic one over the distances of effective sample influences for the estimation procedure.

8.3 Present methods of overcoming the three dimensional problem

Numerous attempts have been made by many authors to try and overcome the three dimensional problem. One approach is to simply

forget about the problem and consider the deposit to consist of a series of horizontal blocks whose values can be estimated on a bench by bench basis. All data, both below and above the bench concerned is ignored and only the values within the bench are used in the estimation. Others involve the use of the three dimensional auxiliary functions in order to understand the block-block, block-core and core-core relationships with the variograms. Clarke (1977) suggests the use of the standard auxiliary function with a series of vertical 'cores' parallel to the block in question. This is probably a quicker technique involving fewer approximations and calculations than the previous method. Journel (1978) uses a series of geometrical projections of the three dimensional samples onto a two dimensional plane, treating the estimation as a two dimensional problem. Most of these methods use complicated mathematical functions and approximations and become difficult to understand by the non-mathematician. It must be possible to devise some method which limits the amount of mathematical approximations and is simple to understand, but is still able to account for any geological controls. A new type of kriging system was devised with this in mind.

8.4 Directional kriging

Semi-variograms may be produced for any given direction and orientation of sample values in an attempt to detect any trends or preferred directions within the data. O'Leary (1980) has shown that the semi-variogram may be contoured to obtain the size and orientation of any structures by means of the iso-semivariance diagram. This would suggest that the variability between two sample values is not only a function of their distance apart but their direction and orientation from one another.

Returning to Figure 8.1, suppose the value of sample A is Z_a and for B, Z_b . Their difference in value is $(Z_a - Z_b)$ and their variability $(Z_a - Z_b)^2$. Similarly, their expected semi-variance can be established from the variogram constant for any direction and orientation of sample values, the expected semi-variance between samples A and B can be considered to be:

$$\gamma_{\phi}(h) = C_0 + C_{\phi}(h)$$

It may be possible to produce an equation to describe the surface of the iso-semivariance diagram, but this is thought to be unnecessary as the same problems would be encountered as for fitting an exact model to the semi-variogram (Chapter 7). A much easier approach would be to describe the iso-semivariance diagram in terms of a series of ellipses based upon a two dimensional spherical scheme variogram model. In three dimensions this would have the appearance of an ellipsoid.

It is possible to describe the position of B relative to A in terms of their distance apart with respect to the three principal axes, i.e. x_1 axis, x_2 axis and x_3 axis. When combined these may be used to describe the distance and direction of B relative to A. Likewise, it may be possible to describe the expected semi-variance between the two samples in a similar way. Semi-variograms may be produced for each of the principal axes and a model variogram produced for each one. If the data is isotropic the nugget variance, continuity and range of sample influence will remain constant for each axis. Figure 8.2 demonstrates the ellipse of anisotropy for a two dimensional case which is elongated along the x_2 axis. Sample values lying along direction X will have a smaller range of influence than those lying in direction Y. Similarly, samples lying along direction T will have a range of influence somewhere between the two. If the zone of anisotropy is considered to be

elliptical the range of influence of the samples lying along direction T will be:

$$a_T = \left[a_{x1}^2 + a_{x2}^2 \right]^{\frac{1}{2}}$$

In the three dimensional case when the range in the x_3 direction differs from both the x_1 and x_2 directions, the range of sample influence for direction T will be:

$$a_T = \left[a_{x1}^2 + a_{x2}^2 + a_{x3}^2 \right]^{\frac{1}{2}}$$

This may be repeated for the nugget variance and the continuity when they differ along any of the three principal axes. Thus, the expected semi-variance between two samples can be calculated with respect to a directional influence and can consider any geometrical anisotropies and proportional effects.

The kriging matrix is established in exactly the same way as for simple kriging. Each sample pair is taken in turn and their direction, orientation and distance apart are used to calculate their directional variogram parameters. Their expected directional semi-variance is then established from these. Apart from these minor alterations, the kriging procedure remains exactly the same as before.

Numerous successful comparisons were made between the three dimensional approach and the two dimensional methods for point values. In many cases the estimated values remained similar (overall they were the same), although the kriging variances were slightly higher in the three dimensional case and computer run times were marginally increased.

A similar approach could be established for the covariance type of kriging. At present it is assumed that the variance of the samples is set to the leading diagonal of the covariance matrix. The covariance

between a sample and itself is directionless and so the overall variance may be used since this is not directionally controlled.

Following the success with the three dimensional point kriging it was decided to try and produce a three dimensional block kriging method along similar lines. This would basically involve slight alteration to the panel kriging technique. Panels would be replaced by blocks and the average semi-variance calculated by considering the distances and directions of a series of hypothetical points within the block (section 4.7). Very few alterations are required and again good success was achieved with this method.

8.5 Advantages and disadvantages of the directional kriging system

The advantage of a directional approach to kriging is that it attempts to try and consider the directional relationships between sample values in more detail. It could be argued that this is basically an extension of the use of the ratio of anisotropy in the considerations of proportional effects and removal of anisotropies. Other methods of kriging require that the data is made isotropic beforehand and must be readjusted upon completion of the evaluation. Directional kriging has the advantage that the actual variogram parameters are used and do not require adjustments to provide the isotropic state. Comparisons between directional kriging and conventional kriging show that the directional kriging is able to cope with the anisotropies without adjustment.

The disadvantage of directional kriging is that it uses sample values which are based upon different supports and does not correct the vertical variograms to compensate for this. This is not considered to be a major problem and has so far been overcome with relative ease with the aid of point kriging. Models are firstly established for the

horizontal variograms and then the vertical parameters are adjusted according to the ratio of the kriging variance to estimation variance for the estimation. Numerous tests have shown that the vertical model approximates very closely to the vertical semi-variogram. The three dimensional parameters now produce an unbiased estimate which can be used in the evaluation of blocks of ground. Thus, the kriging process is kept as simple as possible, undertaking more considerations than before and still producing an unbiased result. On average three kriging runs are required to establish the three dimensional directional kriging parameters.

CHAPTER 9

9. The valuation of the Cerro Colorado deposit

9.1 The grade-tonnage curve

The grade-tonnage curve is a graphical representation of the tonnage and grade of ore above a certain cut off grade within a deposit or contained within a mine plan. It is obtained by firstly dividing the orebody into a series of mining units (see below) and assigning a grade value to each one. A frequency histogram can be produced to summarise the total number of blocks whose grade value is greater than that of a given cut off grade. The total tonnage of ore is then calculated and can be plotted against the cut off grade to give the grade-tonnage curve (Fig. 9.1). It must be remembered that the resulting curve is based upon a series of estimates produced from actual sample values. No mathematical technique, however complex, can be used to improve the quality of the data (David, 1972) and so the resulting grade-tonnage curve will be no more representative of the orebody than the actual sample values.

A second type of curve may be constructed from the global data (David, 1972). The orebody is not divided into a series of blocks but is treated as a single unit. Many authors believe that this type of curve may be more representative of the orebody and provide a more accurate tonnage of ore for a given cut off grade. However, it is not possible to locate the areas of high and low grade material and hence the boundaries between ore and waste are undefined. Thus, the overall grade of the orebody is known although the location of the ore is undetermined and will be of little use to the mine planning engineer.

Tonnage curves may also be produced for the waste material (Fig. 9.2). These provide an indication to the amount of waste material which can be expected for the given cut off grade.

Tonnes
of
Ore

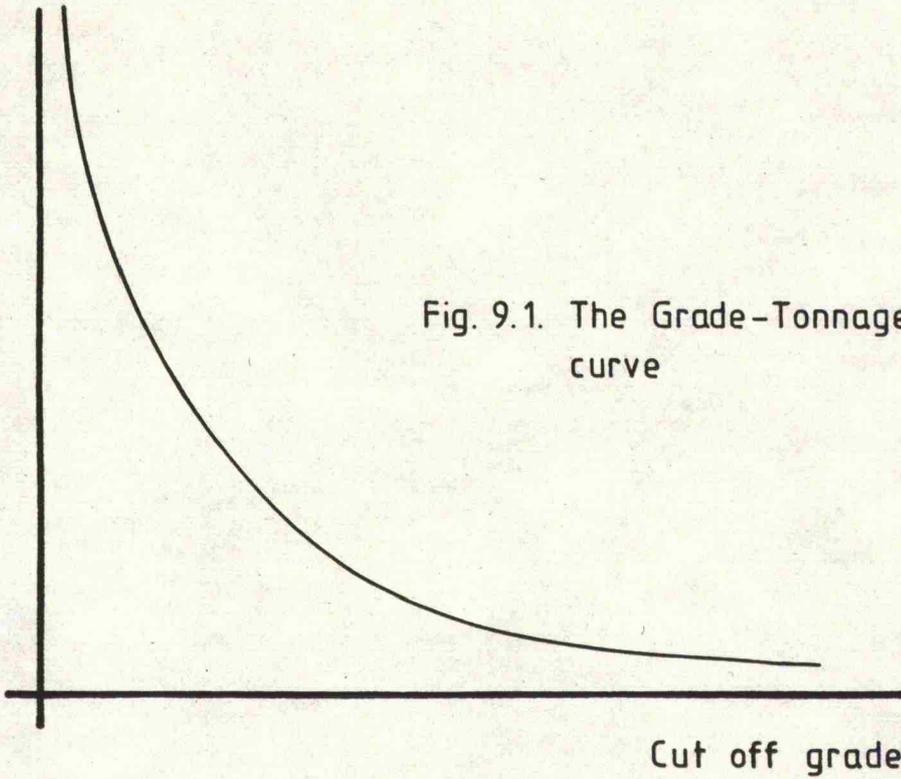


Fig. 9.1. The Grade-Tonnage
curve

Tonnes
of
waste

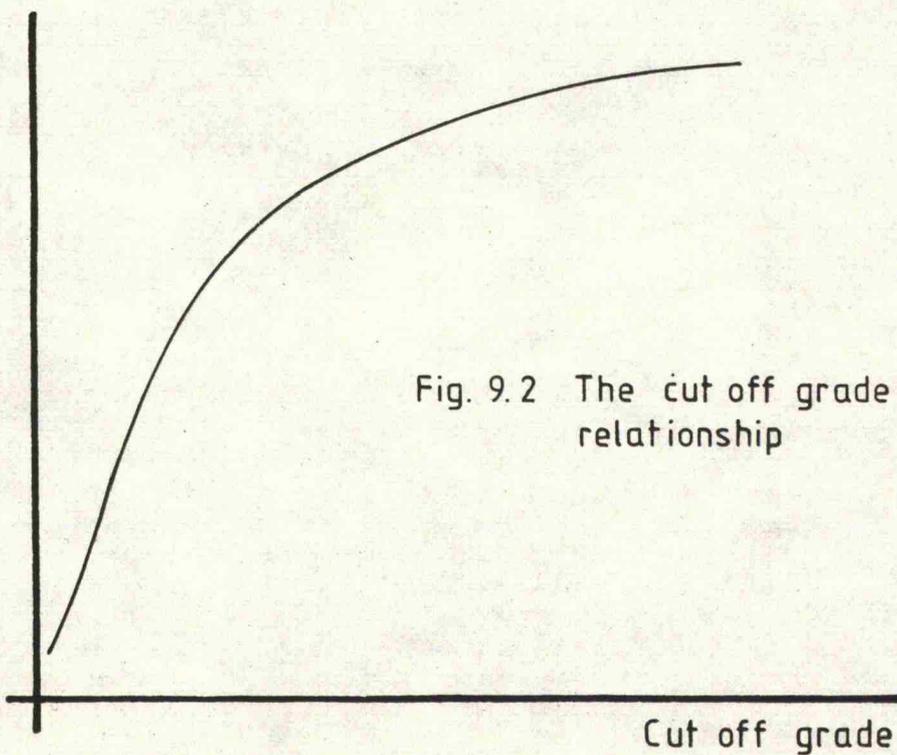


Fig. 9.2 The cut off grade - waste
relationship

The selection of the mineable blocks is based upon their classification as either ore or waste. Blocks whose grade values are less than that of the cut off grade are considered to be waste, those whose values are above that of the cut off are ore. Errors may occur within the methods of estimation (i.e. 'rounding off' errors) or classification of sample values, meaning that some blocks may become misclassified due to being either over- or underestimated (Fig. 9.3). Consequently, blocks may be sent to the wrong destination once they have been mined. It is therefore vitally important to be able to classify the blocks as accurately as possible.

The kriging variance is a measure of the expected amount of variability within the estimated value. A small kriging variance is an indication of a good estimate, a large kriging variance of a poorer estimate. It may be worthwhile to construct a grade-tonnage curve based upon the kriging variances of the estimates, since this will be able to relate the tonnage and grade to the accuracy of the estimation of the block values. However, this will have no significance for the mine planning since the kriging variance is seldom considered during this stage.

9.2 The importance of block size

Parker refers to the mining block as the selective mining unit (SMU) or the smallest practical volume which can be classified as either ore or waste. On a small scale this may be an individual truck load or on a larger scale an entire shift's production. Usually, the size is governed by the scale of operations. Each block can be considered to contain an infinite number of sample values which can be used to calculate the mean and variance of the block. Obviously, the actual mean and variance will never be known until it has been

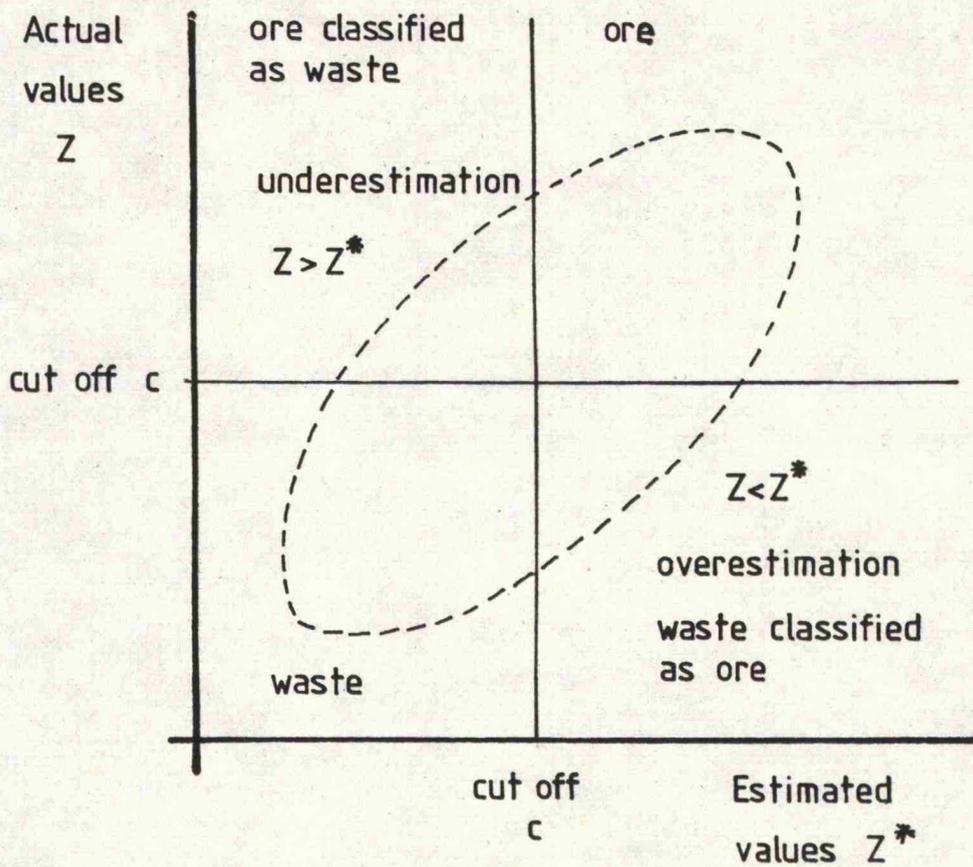


Fig. 9.3. Scattergram of actual against estimated values, summarised to demonstrate the misclassification of blocks of material

AFTER KRIGE 1951

completely mined out. The problem is to try to predict these values for each block from a series of surrounding known sample values. It may be assumed that each block consists of a series of smaller blocks, each one having the same average grade as that of the mining block. Shurtz has shown that the variance of the sample values within the block increases as the size of block decreases. Thus, if a small mining block is to be considered it would be expected to have a high sample variability and the accuracy of estimation will decrease. For any block value the expected contained sample variance will be the variance of estimation minus the expected block variance. Kriging assumes that the sample values exhibit a near normal or Gaussian frequency distribution, as do the kriged estimates. Consequently, all kriged estimates may be assigned an error margin with the use of the standard deviation. It is now believed that the minimum block size permissible can be defined with respect to the kriging variance.

For a Gaussian distribution, 95 percent of the sample values will be expected to lie within the interval:

$$\bar{z} - 1.96sd < z < \bar{z} + 1.96sd$$

A kriged estimate can be significant if it is standardised and the following expression is true:

$$(\bar{z}_s^* + 0.5) - 1.96 (\sigma_{kv}^2)^{\frac{1}{2}} < z^* < (\bar{z}_s^* + 0.5) + 1.96 (\sigma_{kv}^2)^{\frac{1}{2}}$$

The minimum block size possible will occur when no more than 2.5 percent of the standardised kriged estimates have a negative value once two 'standard deviations' have been added. Sometimes the practical sized mining unit is too small to warrant reasonably accurate estimates. In other cases the minimum unit possible is impractical from a production point of view. In all cases, a decision must be made as to the requirement of either accuracy or a practical mining unit.

One of the problems with kriging is that it is a very expensive procedure to employ and is very inflexible with regard to altering the block size. A complete rerun is required when estimates are required when based upon a different block size. Monge has tried to overcome some of the problems with the establishment of a local modelling system based upon polynomial surfaces. In many cases a surface is fitted with a high accuracy. A simple integration of the surface equation within boundaries of a given block will provide the mean and variance of the block. Changes in block size can easily be accounted for since only the limits on the boundaries of the integration need be altered.

9.3 Kriging the Cerro Colorado orebody

It was decided to divide the deposit into a matrix containing 120 by 110 by 91 blocks (Fig. 9.4), each one of dimensions 25 by 25 by 15 metres, approximating to 25,000 tonnes of rock. This was considered to be the minimum mining unit possible from a practical point of view and is equivalent to a day's production of a P & H 2800 shovel. Approximately 250,000 of the 1.2 million blocks contained within the matrix need to be kriged, the rest represent the air, overburden or unknown material and were temporarily removed from the evaluation procedure. The remaining blocks were categorised, dependent upon the grade zone they were within, i.e. category one contained all of the blocks thought to be contained within grade zone one.

One method of kriging is to take each bench and krig each contained block in turn, commencing at one corner of the grid (Fig. 9.5). This method was considered to be impractical in this case in view of the vast amounts of data involved and the necessity of a complicated computer program to input the necessary geological control. Similarly,

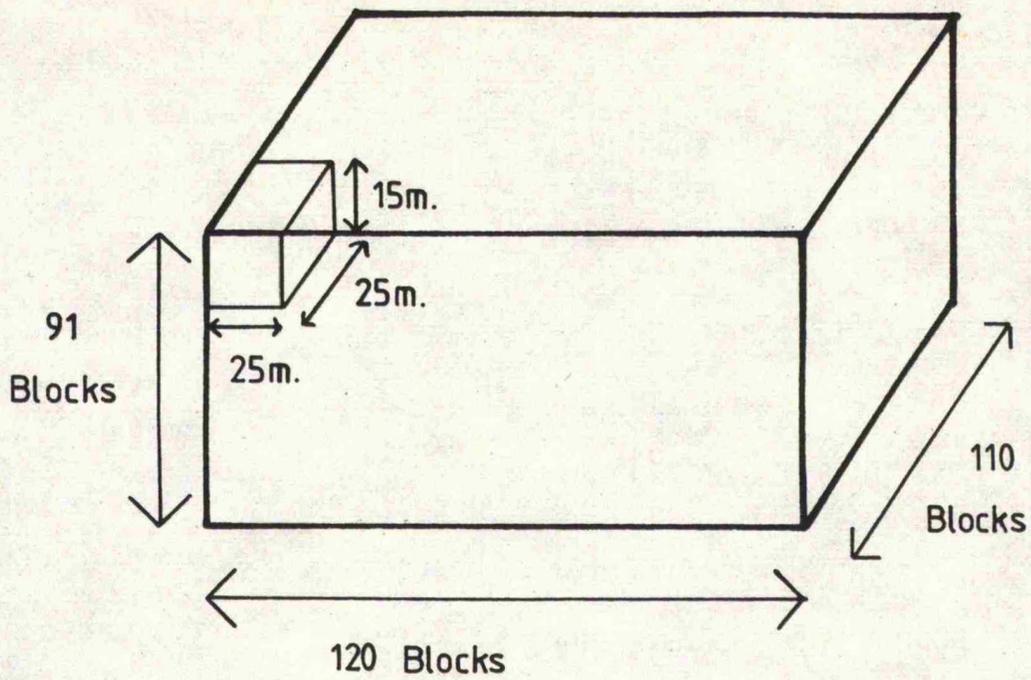


Fig. 9.4. Dimensions of the Cerro Colorado block matrix

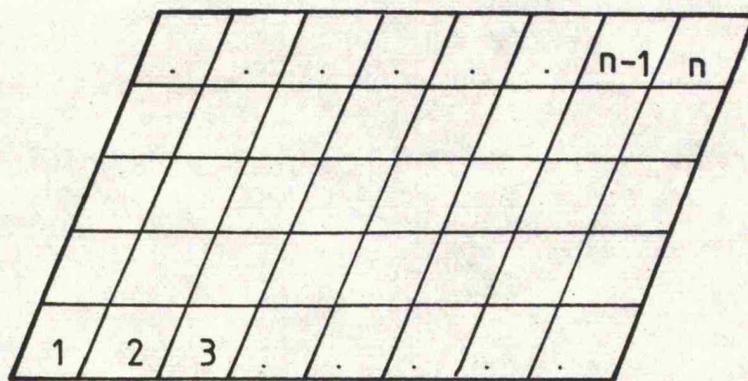


Fig. 9.5 Order of kriging the n blocks on a bench by bench basis

many benches contain more than half of the blocks which are not able to be kriged since they contain either overburden, unknown material or air. The kriging process was speeded up by the inputting of the required block co-ordinates separately. Each grade zone was individually kriged by taking each required block in turn and locating the nearest surrounding representative grade zone samples in a manner similar to that of the point kriging (Chapter 8). A three dimensional approach was taken with the same radii of search and covariogram parameters as were used in the point kriging process. In all cases the block co-ordinates, bench number, kriged value and kriging variance were written to a computer file for later use. Blocks which were unable to be kriged due to a lack of surrounding sample values were assigned the mean value of the representative grade zone and a kriging variance of -1.0. This was done to aid in the mine planning programme. Once all of the blocks had been kriged they were combined together to form a matrix identical to the grade zone and rock type matrices. Matrices were also produced for the copper, molybdenum, silver and gold estimates.

9.4 Mineral inventory

The mineral inventory is used to describe the grade and tonnage of ore which has been estimated during the mineral evaluation programme. It is not possible to tabulate the mineral inventory for the Cerro Colorado deposit as obtained in this study or for any previous studies due to reasons of confidentiality. However, it is possible to compare and contrast this estimate with the previous ones.

To date, a total of six estimates (excluding this one) have been carried out on the Cerro Colorado deposit. Of these six, four were produced by using traditional methods, i.e. sections and plans; the

remainder by geostatistical methods. It is not possible to provide a direct comparison between all of the methods since they are based upon different concepts and ideas. However, it is worthwhile to note the recent RTZ and Texasgulf estimates are in agreement, especially since the recent RTZ estimate has been based upon a very strong geological control.

The estimate produced within this study differs slightly from the most recent RTZ estimate in so much that the grade is slightly lower (approximately 0.05 percent) and the tonnage is increased by approximately 20 percent. This is believed to be a result of the influence of the composition of the data values, with the effect that more blocks are unable to be kriged due to a lack of surrounding data values. Consequently, more blocks are assigned a mean value, which would account for the overall drop in grade and increase in tonnage. Comparisons between each of the frequency histograms of the actual (data values) and the kriged estimates for each grade zone suggests that the two have a similar frequency distribution. In all four cases the mean of the actual and kriged values are the same, although the variance of the kriged estimates was lower than that of the actual values. This was to be expected since the same phenomenon was encountered during the point kriging procedure.

Statistical tests, using the method described previously, indicated that the block sizes were not too small as was at first feared. As the grade zone number increased so did the percentage of negative standardised values. Again, this would be expected since the variance of the actual values also increased with grade zone number. The results are tabulated below:

<u>Grade zone</u>	<u>Percentage standardised blocks < 0</u>
1	0.37
2	1.08
3	1.22
4	1.70

It is now believed that it may be possible to evaluate the Cerro Colorado deposit on a smaller block size, possibly on the scale of 15 by 15 by 15 metre blocks.

CHAPTER 10

10. Conclusions

It is now possible to suggest that geological considerations can greatly aid the mineral evaluation process. After all, it must be remembered that geological factors were responsible for the formation and structure of the deposit. The grade zone concept is an initial step in the direction for geological involvement, as was shown in the evaluation of the Cerro Colorado deposit. It not only simplifies some of the problems encountered within the geostatistical modelling of the orebody but is also able to easily describe the nature and structure of the mineralisation. To date few mineral evaluation procedures have actually considered any geological aspects within their methods of evaluation, with the result that poor results have often been obtained. It must seem obvious to suggest that any geostatistical calculations would be expected to be more realistic if they are based upon sample values which are directly related to each other or, as in this case, have been influenced by similar geological factors.

Subdividing the deposit into zones or areas containing sample values which are believed to be geologically related means that it is now possible to provide a 'local' estimation. Previously, areas of ground were evaluated using the nearest sample values with the result that anomalous areas of suspected high and low values were smoothed out as a result of the estimation process, e.g. kriging. Results have shown that it is now possible to value a piece of ground by using standard kriging techniques and still maintaining the areas of anomalous values without smoothing them out. Consequently, a better definition can be obtained between the areas of ore and waste and also the regions of highest mineralisation, which can be a great asset to the mine planning engineer.

The present trend regarding the use of computers is to employ larger and faster machines with the result that new geostatistical procedures are becoming more complex and take longer to solve. As a result, methods are devised which produce and solve large algorithms which practically attempt to fit an exact surface to a set of data. In all fairness, this may seem a logical approach to some of the problems but it is not considered to be the correct approach. It has been stated by many authors that it is impractical to fit exact surfaces to a set of data due to the presence of irregularities and errors which may be contained therein. Again, the grade zone concept is believed to overcome many of these problems. No vast development is required to the present kriging system, simple kriging is sufficient. The only development required is that of the adaptation to a three dimensional directional kriging system which attempts to overcome some of the problems of anisotropy. Again, this is only a slight alteration to the simple kriging system. When combined with the grade zone concept, the two provide a very powerful technique for the evaluation of the Cerro Colorado deposit. In test cases computer run times were greatly reduced since only specific areas of ground were kriged using only the relevant surrounding sample data.

Many authors tend to use the variogram as the basis for all of their geostatistical calculations. One of the problems is that it requires a complicated matrix inversion algorithm for the kriging procedure. The covariogram does not require such a complicated algorithm and comparisons in computer run times show that calculations involving the covariogram are much quicker and hence are cheaper to use. In many instances the covariograms are unstable and difficult to model and it is for this reason that they are not used. In the case of the Cerro

Colorado deposit very good covariograms were obtained and provided a good justification for their use in place of the semi-variogram. One small point to consider with the use of the covariogram is that it requires only two parameters, i.e. the continuity and range, whereas the variogram requires three, i.e. nugget effect, continuity and range. Hence, there is a greater chance of introducing errors if the semi-variogram is used. The only problem with the covariogram is that it requires an estimation of the variance of the whole deposit which is assumed to be the same as that of the sample values. This is no longer thought to be a problem since the purest of geostatisticians argue that the sill of the variogram is equal to the sample variance. This being the case, no error is introduced.

From an economical point of view it is cheaper to use the covariogram to provide the kriged estimates, especially in this case since there is so much data and over 250,000 blocks of ground have to be kriged. Tests on the Leicester University Cyber 73 have shown that the variogram type of kriging is approximately 1.25 times more expensive than the covariance type kriging. Since the two give the same results it is economically viable to use the covariance method of kriging.

One gratifying fact to note is that the evaluation of the Cerro Colorado deposit by using a strong geological control compares very favourably with the recent standard geostatistical methods used in the previous estimate by Texasgulf. The only difference is that the more recent (RTZ) studies give a much better definition of the boundaries between ore and waste. The present study compares very well with the previous two despite the fact that the tonnage and grade differ slightly. It is still believed that the differences are a result of the compositing effect brought about by using 15 metre samples as

opposed to 3 metre samples. The advantage of using these 15 metre composites is that it greatly reduces computer run times without which this study may have been impossible on the Leicester University system. This estimate is possibly a more conservative estimate and it would be interesting to compare the economics of the present proposed pit with the economics of the same pit based upon this estimate.

To summarise, geological influences have a major role to play in future mineral evaluation programmes. It is now believed that there is no need at present for any major developments in geostatistical concepts, all developments must be produced within the interpretation of the data available. A good understanding of this will probably lead to a much better mineral evaluation.

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THE MINPAK USER MANUAL

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Chapter 1

1. Introduction

MINPAK is a completely new computer package of programs and subroutines designed for use within the mining industry. It has been written by J. E. FORKES of Leicester University from further developments of the highly successful GEOSTAT, JOLPACK and OPENPIT packages. These three packages have been extensively modified and partially rewritten to form one package which is capable of a complete study of a set of data. This ranges from a statistical and geostatistical analysis to the design of an open pit mine.

This package has been written to allow it to be as user friendly as possible, thus giving the user as much freedom as possible in the running of his program. The user still has to write a small mainline program to read in the data and to call the relevant subroutines. Most of the arrays are undeclared and are dimensioned through the users' own program. This is a great asset since the total amount of core required is governed by the user, not by the program. Similarly the routines have been written as efficiently as possible to ensure that they run quickly and are cheap to use. It was felt necessary to include a series of mainline programs to enable the user to readily use some of the more complex routines ie. the open pit routines. Again these have been made as efficient as possible, though unfortunately the user is not able to govern the size of the program since the array dimensions are already declared.

One of the problems with a package of this nature is that of portability ie. the ability to transport the package from one machine to another. All of the routines have been written in standard FORTRAN and will compile with either the FORTRAN IV or FORTRAN V compilers. One or two of the routines call functions which are specific to CDC computers and their use has been unavoidable. These include routines TIME(K) and DATE(K) (time and date routines) though it is anticipated that a minor alteration to the relevant routines will overcome this problem when used on a non-CDC machine.

Another unfortunate problem concerns that of the matrix inversion routines. Such a routine is required by the kriging subroutines and the difficulty lies with the inversion of a matrix whose leading diagonal contains 0's. To date the optimum routine found to cope with this is contained within the NAG library - notably subroutine F04AAf. MINPAK has been set up to call this particular routine. At present, work is being pursued to replace this routine with one which is to be self contained within this package.

Graphical routines are also contained and these too require the access to routines which are external to this package. MINPAK is set up to access the GINO-F graphical package which is not present on all machines. Again work is in progress to try and overcome this problem.

It must be emphasised that this package has a useful purpose to serve and is not a toy. A meaningful result will be only be returned from meaningful data. The user must first decide what to

do with the data before it is input to the relevant routines. It is also assumed that the user has some knowledge of FORTRAN programming and is capable of writing a small program to call the required MINPAK routines. He must also have some knowledge of geostatistics and mine planning and it is for this reason that detailed routine descriptions and equations are omitted from this manual. All of the equations and methods used may be found in any good statistical or geostatistical text book.

Chapter 2

2.1 Creating a relocatable library

The MINPAK library is written in such a way as to allow the user to easily transform it into a relocatable library. It has been written in an UPDATE format ie. each program or subroutine is identified by a *DECK. To date MINPAK and its predecessors GEOSTAT and OPENPIT have been implemented mainly on CYBER machines and examples will be given as to their methods of library creation. However library creation on some other types of machines may cause problems since more than one main program is contained within MINPAK. This may be rectified by using the SYSEDIT parameter on the fortran compiler.

In the following examples the source version will be referred to by the filename SOURCE and the relocatable library filename as MINPAK.

2.2 UPDATE

In this example the source deck is converted into an update library before being passed via the COMPILE file to the compiler.

```
job card
USER card
CHARGE card
UPDATE,N.
SAVE,NEWPL=MINPAK.
FTNS,I=COMPILE.
eor
*DECK .....
.
.
.
end
<eoi>
```

2.3 MAKELIB

MAKELIB is a procedure at Leicester University for creating a relocatable library. The relocatable library is written onto a local file called NEWLIB which includes all of the modules on the local file LGO. For example:

```
job card
USER card
CHARGE card
GET,SOURCE.
FTNS,I=SOURCE.
EXEC,MAKELIB(X=1).
SAVE,NEWLIB=MINPAK.
<eor>
<eof>
```

Since there is more than one main program present, the X=1 parameter must be added to the EXEC,MAKELIB command.

For other methods for creating a relocatable library it is

suggested that the user consults either the relevant FTN manual or seeks advice from a systems analyst.

2.4 Using the MINPAK library

The stages in using a library in a typical job are:

1. compile the program and its subroutines (if in source form).
2. make the library local to the job.
3. tell the loader to search for the modules which are required from the library to make the job run.
4. execute the program.

The vital command in this job is the LDSET command since this begins the load command and instructs the loader to search the library for the required routines.

In order to use the MINPAK library with a source program called ABC, the job control for a CYBER would be as follows:

```
job card
USER card
CHARGE card
GET,ABC.
FTN5,I=ABC.
GET,MINPAK.
LDSET,LIB=MINPAK.
LGO.
<eor>
      data
<eof>
```

where MINPAK is the MINPAK library.

In the second case where the user wishes to use one of the contained MINPAK main programs he must tell the loader which program to access and load. This may be done in one of two ways:

1. job card
USER card
CHARGE card
GET,MINPAK.
LDSET,LIB=MINPAK.
XYZ.
<eor>
data
<eof>

where XYZ is the required contained program.

2. job card
USER card
CHARGE card
GET,MINPAK.
LDSET,LIB=MINPAK,USE=XYZ.
EXECUTE.
<eor>
data
<eof>

where XYZ is the required contained program.

Chapter 3

3.1 Borehole datafiles

Data files may be presented for mineral evaluation purposes in one of two forms:

a). On a down the hole basis where each assay value is measured over a given distance down each hole.

b). On a point basis where each assay value is assigned a co-ordinate location within the dimensions of the deposit.

The second case is of more use when persuing an ore reserve since this is usually dependent upon the location of the sample values within the deposit. However the first case also has its' uses since it is able to indicate the distances over which each sample has been measured. This will be important if it is required to produce an ore reserve based upon sample values measured over a different distance ie. using composites.

It is also useful to provide a graphical log of the assay values contained within each hole. This may be done with subroutine:

```
HOLEPT(NOCC, FROM, TO, FTGRAD, GAXIS, GINC, ITL, NHOLE, LPR)
```

which plots the grade values in the form of a bar chart.

Subroutine:

```
READAT(NROW, NDATA, DATA, WKSP, NCOL, FMT, INP, ICHAR, IX)
```

reads in a set of drill hole information measured on a from-to basis down the hole. A typical from-to file would have a format similar to:

```
1000.0 1000.0 100.0 0.0 0.0 1
1 0.0 10.0 6.0
1 10.0 20.0 5.0
1 20.0 30.0 2.0
1 30.0 40.0 3.0
1 40.0 50.0 1.0
9
1050.0 1000.0 100.0 0.0 0.0 2
:
:
9
end-of-file
```

The first record states the X, Y and Z co-ordinates of the drill hole collar, the dip and azimuth of the hole and the hole number. This record is not read in by subroutine READAT, though the information may be required later by other subroutines. The following records specify a character identifier, the starting and finishing distance down the hole over which the assay(s) has been measured and finally the assay value(s). These are read in and stored in array DATA (except for the character identifier). The character identifier is used to indicate the bottom of the hole

and terminates the read-in sequence. The subroutine is returned to allow further processing and/or to read in the next set of drill hole information. In the case of the above example, the read in sequence will be terminated if ICHAR is assigned a value of 9. 5 rows of data are to be read in and stored in array DATA. Array DATA would then be used by further subroutines to establish a series of point data files.

An end-of-file marker terminates the read-in sequence and is marked by the return of IX = 100. A value of IX = 101 indicates that no data values have been read in.

3.2 Creating a point data file

Once array DATA has been initialised by subroutine READAT subroutine:

```
ROTDDH(DATA,NCOL,NOCC,XCOL,YCOL,ZCOL,NHOLE,DIP,AZM,LPR,ISWO,
        FMTO,BASE,BGHT,NBEN)
```

may be called to convert the assay values measured on a from-to basis into a series of point values with known co-ordinates. The output may be unformatted (ISWO > 0) or formatted (ISWO < 1) when written to output device number LPR under a format of FMTO. FMTO must accomodate for 2 integer values and NCOL+1 real values. NOCC records will be written per drill hole and each record will contain:

- a). The hole number (integer)
- b). The bench number on which the sample lies (integer)
- c). The x co-ordinate of the assay (real)
- d). The y co-ordinate of the assay (real)
- e). The z co-ordinate of the assay (real)
- f). (NCOL-2) assay values (real)

3.3 Sample composites

Once array DATA has been initialised by subroutine READAT, subroutine:

```
CMPDDH(DATA,NCOL,NOCC,XCOL,YCOL,ZCOL,NHOLE,DIP,AZM,LPR,ISWO,
        FMTO,BASE,BGHT,NBEN,WKSP)
```

may be called to composite the assay values over a distance equal to that of the bench height (BGHT). This routine works in the same way as subroutine ROTDDH, requiring a similar form of input data and providing output in exactly the same format.

3.4 Splitting up the data according to rock type

In some cases an ore reserve may be required which is based upon a certain portion of the data set ie. a reserve which is based upon the separate evaluation of individual rock units. In such an example a volume of ground containing a given rock type would need to be valued using only the samples which are contained within that particular rock type. If the rock types present in each hole have been measured in a similar way to those of the assay values ie. measure the from-to distance of each rock type intersection down the hole, subroutine:

```
RKSPLT(NROW,NDATA,DATA,WKSP,NCOL,FMT,INP,ICHAR,FROK,TROK,  
        ICODE,ICD,NV2)
```

may be used to remove and process only the assay values belonging to rock type ICD. This routine works in a similar manner to subroutine READAT except that only a certain portion of the data is stored in array DATA.

The from-to data file for the rock type intersections for one hole may look like this:

0.0	25.0	3
25.0	28.0	2
28.0	29.0	1
29.0	30.0	6
30.0	40.0	3
40.0	41.0	2
41.0	48.0	3

If ICD has been assigned a value of 3, only rock type 3 sample values, lying within the required from-to distances will be stored in array DATA.

3.5 Program example

```
PROGRAM C1 (INPUT, OUTPUT, TAPE3, TAPE5=INPUT, TAPE6=OUTPUT,
1TAPE7)
C
C A PROGRAM TO READ IN DRILL HOLE INFORMATION AND TO COMPOSITE
C THE DATA. THE DATA IS READ IN ON TAPE 3 AND OUTPUT TO FILE
C ON TAPE 7
C
C DIMENSION DATA(9,500),WKSP(9),FMT(8),FMTO(8)
C
C READ IN INPUT DATA FORMAT
C
C READ(5,1000)FMT
C
C READ IN OUTPUT FORMAT
C
C READ(5,1000)FMTO
C
C READ IN BENCH PARAMETERS
C
C READ(5,1001)BASE,BGHT,NBEN
C
C INITIALISE DIMENSION VARIABLES
C
C NCOL=9
C NROW=500
C
C TERMINATE READ IN ON DETECTION OF A 9
C
C ICHAR=9
C
C INPUT ON TAPE 3
C
C INP=3
C
C OUTPUT TO TAPE 7
C
C LPR=7
C
C OUTPUT IS TO BE FORMATTED
C
C ISWO=0
C
C READ IN 10 DRILL HOLES
C
C DO 500 I=1,10
C
C READ IN HEADER
C
C READ(INP,1002)XCOL,YCOL,ZCOL,AZM,DIP,NHOLE
C
C READ IN DATAFILE FOR HOLE
C
C CALL READAT(NROW,NOCC,DATA,WKSP,NCOL,FMT,INP,ICHAR,IX)
C
C CHECK FOR SUCCESSFUL CALL
C
C IF(IX.EQ.0)GO TO 100
C
```

```

C   GIVE MESSAGE AND READ NEXT HOLE
C
      WRITE(6,2000)NHOLE
      GO TO 499
100  CONTINUE
C
C   COMPOSITE DATA
C
      CALL CMPDDH(DATA,NCOL,NOCC,XCOL,YCOL,ZCOL,NHOLE,DIP,AZM,
      LLPR,ISWO,FMTO,BASE,BGHT,NBEN,WKSP)
499  CONTINUE
500  CONTINUE
      STOP
1000 FORMAT(8A10)
1001 FORMAT(2F10.0,I5)
1002 FORMAT(5F10.0,I5)
2000 FORMAT(/,5x,22HNO VALUES IN HOLE NO: ,I3,/)
      END

```

Chapter 4

4.1 Statistical analysis

The statistical analysis of a set of data is the general procedure early in a mineral evaluation study. This will help to provide an understanding of the distribution of the sample values and their relationships with one another. The use of simple statistics such as frequency distributions, means and variances and correlations can reveal a great deal about the properties of the data. Such statistical routines are contained within this package and use standard equations which may be found in any statistical text book. The equations used are the same as those described by Speigel and in view of this fact it is felt unnecessary to explain how and when these techniques should be used.

Sufficient routines are available to provide an adequate analysis of a set of data.

4.2 The routines

CHI7DF(NVAL,A,CHI2)

calculates the chi-square value for seven degrees of freedom for a set of sample values.

CORREL(NVAL,A,B,R,COVR,AØ,GRAD,SEST)

establishes the correlation coefficient, standard error of estimate and average covariance between two sets of sample values. It also calculates the equation of the line of regression.

COVAR1(NVAL,A,B,COVR)

calculates the average covariance between two sets of sample values.

FRATIO(F,AVAR,BVAR,NVALA,NVALB,IDFA,IDFB)

compares the variances of two sets of data by means of an F-test.

HISTGM(NVAL,A,NCLASS,XO,DX,NOBS,PERC,ICUM,CPERC,ITL,LPR)

calculates and lists the frequency distribution of a set of data.

HISTPT(NCLASS,FREQ,DX,ITL,LPR)

plots a histogram of values or a cumulative histogram if the maximum value contained within array (FREQ(I),I=1,NCLASS) is greater than 99.Ø.

ONEPOP(NVAL,A,ITL,LPR)

calculates the overall statistics of a single set of data.

RANGE1(NVAL,A,AMAX,AMIN)

calculates the maximum and minimum values contained within a set of data.

SCATTR(NVAL,A,B,ITLA,ITLB,ISET,JSET,LPR)

plots a scatter graph of variable A against variable B.

SKWKUR(NVAL,A,SKEW,AKURT)

calculates the degrees of skewness and kurtosis for a set of data values.

STANDD(NVAL,A,STAND)

standardises a set of data values so that the resulting mean is 0.0 and the variance is 1.0.

TTEST1(TVAL,NVALA,NVALB,AMEAN,BMEAN,AVAR,BVAR,IDF)

compares the means of two sets of data by means of a t-test.

TWOPOP(NVALA,NVALB,A,B,ITLA,ITLB,LPR)

compares and contrasts the statistics of two separate sets of sample values.

VARIAN(NVAL,A,XMEAN,XVAR)

calculates the mean and variance of a set of data.

4.3 Program example

Example 4.1

```
PROGRAM S1(INPUT,OUTPUT,TAPE5=INPUT,TAPE6=OUTPUT)
C
C A PROGRAM TO CALCULATE AND PLOT A HISTOGRAM
C
C DIMENSION A(100),PERC(30),CPERC(30),NV(30),ICUM(30)
C
C READ IN 100 DATA VALUES
C
C NVAL=100
C DO 100 I=1,NVAL
C READ(5,1000)A(I)
100 CONTINUE
C
C READ IN NUMBER OF CLASS INTERVALS
C
C READ(5,1001)NCLASS
C
C READ IN HISTOGRAM PARAMETERS
C
C READ(5,1002)XO,DX
C
C READ IN NAME OF HISTOGRAM OF UP TO 10 CHARECTERS
C
C READ(5,1003)ITL
C
C WRITE OUTPUT TO TAPE 6
C
C LPR=6
C
C CALCULATE FREQUENCY DISTRIBUTION AND WRITE TO NEW
C PAGE
C
C WRITE(6,1004)
C CALL HISTGM(NVAL,A,NCLASS,XO,DX,NV,PERC,ICUM,CPERC,
1 ITL,LPR)
C
C PLOT HISTOGRAM ON NEW PAGE
C
C WRITE(6,1004)
C CALL HISTPT(NCLASS,PERC,DX,ITL,LPR)
C
C PLOT CUMULATIVE HISTOGRAM ON NEW PAGE
C
C WRITE(6,1004)
C CALL HISTPT(NCLASS,CPERC,DX,ITL,LPR)
C STOP
1000 FORMAT(F10.2)
1001 FORMAT(I5)
1002 FORMAT(2F10.2)
1003 FORMAT(A10)
1004 FORMAT(1H1)
END
```

Example 4.2

```
PROGRAM S2(INPUT,OUTPUT,TAPE5=INPUT,TAPE6=OUTPUT)
C
C A PROGRAM TO CALCULATE THE MEAN AND VARIANCE OF A SET
C OF DATA
C
C DIMENSION A(10)
C
C READ IN 10 VALUES
C
C NVAL=10
C DO 100 I=1,NVAL
C READ(5,1000)A(I)
100 CONTINUE
C
C CALCULATE MEAN AND VARIANCE
C
C CALL VARIAN(NVAL,A,AMEAN,AVAR)
C
C TABULATE RESULTS
C
C WRITE(6,1001)AMEAN,AVAR
C STOP
1000 FORMAT(F10.2)
1001 FORMAT(//,5X,7HMEAN = ,F10.2,5X,11HVARIANCE = ,F10.2)
END
```

Example 4.3

```
PROGRAM S3(INPUT,OUTPUT,TAPE5=INPUT,TAPE6=OUTPUT)
C
C A PROGRAM TO CALCULATE THE STATISTICS OF A SINGLE SET
C OF DATA
C
C DIMENSION A(100)
C
C READ IN 100 DATA VALUES
C
C NVAL=100
C DO 10 I=1,NVAL
C READ(5,1000)A(I)
10 CONTINUE
C
C READ IN NAME OF DATA OF UP TO 10 CHARECTERS
C
C READ(5,1001)ITL
C
C WRITE OUTPUT TO TAPE 6
C
C LPR=6
C
C CALCULATE STATISTICS
C
C CALL ONEPOP(NVAL,A,ITL,LPR)
C END
1000 FORMAT(F10.2)
1001 FORMAT(A10)
C END
```

Chapter 5

5.1 Structural analysis

The structural analysis of a set of data is performed with the use of the variogram. This measures the average variability between a set of sample values with respect to their distance apart. The most common type of variogram is the spherical scheme variogram and has three main properties:

a). Nugget variance represents the average variability between the sample values lying zero distance apart ie. a sample is compared with itself. In theory this value is zero though in practice this is never the case since it has some positive value. This value is related to the effects of sampling errors and the presence of small scale structures within the data.

b). The continuity measures the rate at which the variability between the sample values increases with distance. It also represents the rate at which the influence of one sample on another decreases with distance of separation.

c). The range of the variogram indicates the distance beyond which the samples are no longer related to each other. This occurs when the variogram reaches a plateau ie. the continuity is zero.

5.2 The semi-variogram

The semi-variogram is established by calculating the average variability between pairs of sample values and categorising this according to their distance apart. In order to ensure that there are sufficient sample pairs for each distance of separation the samples are categorised over a distance referred to as a lag interval. If the distance of sample separation is DIST and the lag interval is H, then the lag number LAG under which the two samples are categorised is:

$$\text{LAG} = (\text{DIST}/\text{H}) + 0.5$$

The resulting semi-variogram is referred to as an isotropic semi-variogram.

Similarly the covariogram may be produced by comparing each sample pair with the mean of the sample population to which they both belong. Also the drift may be established by calculating the difference in value between each sample pair and categorising in the same way as for the semi-variance.

5.3 The cross-variogram

The cross-variogram may be used to establish the possibilities of a spatial correlation between two sets of sample values. Each sample locality must be sampled for two commodities A and B and the cross-variance between two samples located at positions 1 and 2 is given by the equation:

$$\text{cross-variance} = (A_1 - A_2)(B_1 - B_2) / 2.0$$

and may be categorised in a similar manner to that of the

semi-variance to form the cross-variogram. If the two variables are spatially correlated, the resulting cross-variogram will have an appearance similar to that of a variogram.

5.4 Directional semi-variograms

A directional semi-variogram is one in which the average semi-variance is calculated from a series of samples lying along a specific direction eg. samples taken along a vein. In order to ensure that there are sufficient sample pairs per lag interval, an angle of regularisation is used. If the direction of the semi-variogram is to be ALPHA and the angle of regularisation is BETA, the resulting semi-variogram will only consider sample pairs lying in a direction:

$$(\text{ALPHA}-\text{BETA}) < x < (\text{ALPHA}+\text{BETA})$$

Directional cross-variograms and covariograms may also be produced.

5.5 Sector semi-variograms

Sector semi-variograms may be produced from a series of directional semi-variograms measured in numerous directions, spaced over regular intervals. If NSEC sector semi-variograms are required the resulting semi-variograms will be produced for every

180/NSEC degrees starting at 000 degrees.

The resulting angle of regularisation will be:

180/(2*NSEC) degrees.

Likewise sector covariograms and cross-variograms may also be produced.

5.6 linear semi-variograms

In some cases data is collected at regular intervals along a line of section eg. down a drill hole or along a drift. If semi-variograms, covariograms and cross-variograms are produced from such data they are termed "linear", since they are based upon data which has been obtained from linear sampling. This term must not be confused with that used with the technique of Universal Kriging.

5.7 The routines

CXVDIR(NVAL,NLAG,H,X,Y,Z,A,B,CXVAR,DIST,NPAIRS,ALPHA,BETA)

calculates the directional cross-variogram.

CXVISO(NVAL,NLAG,H,X,Y,Z,A,B,CXVAR,DIST,NPAIRS)

calculates the isotropic cross-variogram.

CXVLIN(NVAL,NLAG,A,B,CXVAR,NPAIRS,FILT)

calculates the linear cross-variogram.

CXVOPT(NLAG,H,CXVAR1,DIS,NPAIR,ALPHA,BETA,ITLA,ITLB,LPR)

tabulates the cross-variance for each lag distance.

CXVSEC(NVAL,NLAG,H,X,Y,Z,A,B,CXV2D,DIS2D,NP2D,NSEC)

calculates the sector cross-variogram.

DISLAG(XC1,YC1,ZC1,XC2,YC2,ZC2,LAGNO,DISTL,H)

establishes the vector distance between two sample values and calculates the corresponding lag number underwhich the sample pair is to be categorised.

SECTNO(ANGLE,NOSEC,SECT,NSEC)

establishes the sector number in which a sample pair lies and is used in the calculation of the sector semi-variograms and cross-variograms.

SECTOR(SECTB,NSEC)

establishes the upper and lower boundaries of each sector used by the sector semi-variogram and cross-variogram routines.

VARCXV(AA1,AA2,BB1,BB2,CXV)

calculates the cross-variance between two sample values.

VARDIR(NVAL,NLAG,H,X,Y,Z,A,GAMMA,COV,DIST,NPAIRS,ALPHA,BETA,DRIFT)

calculates the directional semi-variogram.

VARISO(NVAL,NLAG,H,X,Y,Z,A,GAMMA,COV,DIST,NPAIRS,DRIFT)

calculates the isotropic semi-variogram.

VARLIN(NVAL,NLAG,A,GAMMA,COV,DRIFT,NPAIRS,FILT)

calculates the linear semi-variogram.

VAROPT(NLAG,H,GAMM,COVA,DIS,NPAIR,DRIF,ALPHA,BETA,ITL,LPR)

lists the semi-variance for each lag distance.

VARSEC(NVAL,NLAG,H,X,Y,Z,A,GAM2D,COV2D,DIS2D,NP2D,DRF2D,NSEC)

calculates the sector semi-variogram.

VARVAL(AA1,AA2,XMEAN,GAMA,COVR,DRFT)

calculates the semi-variance, covariance and drift between two sample values.

5.8 Program examples

Example 5.1

```
PROGRAM V1(INPUT,OUTPUT,TAPE5=INPUT,TAPE6=OUTPUT)
C
C A PROGRAM TO PRODUCE AN ISOTROPIC SEMI-VARIOGRAM
C
DIMENSION X(100),Y(100),Z(100),A(100)
DIMENSION GAMMA(30),COV(30),DRIFT(30),NPAIRS(30)
DIMENSION DIST(30)
C
C READ IN NUMBER OF LAGS REQUIRED AND LAG INTERVAL
C A MAXIMUM OF 30 LAGS IS PERMITTED
C
READ(5,1000)NLAG,H
C
C READ IN 100 DATA POINTS
C
NVAL=100
DO 10 I=1,NVAL
READ(5,1001)X(I),Y(I),Z(I),A(I)
10 CONTINUE
C
C CALCULATE SEMI-VARIOGRAM
C
CALL VARISO(NVAL,NLAG,H,X,Y,Z,A,GAMMA,COV,DIST,NPAIRS,
1 DRIFT)
C
C OUTPUT ON TAPE 6
C
LPR=6
C
C READ IN TITLE OF SEMI-VARIOGRAM
C
READ(5,1002)ITL
C
C SET DIRECTION TO 0.0 AND ANGLE OF REG. TO 180.0
C
ALPHA=0.0
BETA=180.0
C
C WRITE OUTPUT TO NEW PAGE
C
WRITE(6,1003)
CALL VAROPT(NLAG,H,GAMMA,COV,DIST,NPAIRS,DRIFT,ALPHA,
1 BETA,ITL,LPR)
STOP
1000 FORMAT(I2,F10.2)
1001 FORMAT(4F10.2)
1002 FORMAT(A10)
1003 FORMAT(1H1)
END
```

Example 5.2

```
PROGRAM V2(INPUT,OUTPUT,TAPE5=INPUT,TAPE6=OUTPUT)
C
C A PROGRAM TO CALCULATE A SECTOR CROSS-VARIOGRAM
C
DIMENSION X(100),Y(100),Z(100),A(100),B(100)
DIMENSION CXV2D(4,30),DIS2D(4,30),NP2D(4,30)
DIMENSION C1(30),D1(30),N1(30)
C
C SET UP FOR 4 SECTORS AND 30 LAGS
C
NSEC=4
NLAG=30
C
C READ IN LAG INTERVAL
C
READ(5,1000)H
C
C READ IN NAME OF EACH COMMODITY
C
READ(5,1001)ITLA,ITLB
C
C READ IN 100 DATA VALUES
C
NVAL=100
DO 10 I=1,NVAL
READ(5,1002)X(I),Y(I),Z(I),A(I),B(I)
10 CONTINUE
C
C CALCULATE CROSS-VARIOGRAM
C
CALL CXVSEC(NVAL,NLAG,H,X,Y,Z,A,B,CXV2D,DIS2D,NP2D,
1 NSEC)
C
C CALCULATE SECTOR ANGLES
C
SINT=180./FLOAT(NSEC)
BETA=SINT*.5
C
C SET INITIAL ANGLE TO 0.0
C
ALPHA=0.0
C
C WRITE TO UNIT 6
C
LPR=6
C
C LIST EACH SECTOR
C
DO 100 I=1,NSEC
DO 90 J=1,NLAG
C
C CONVERT TO A 1-D ARRAY
C
D1(J)=DIS2D(I,J)
C1(J)=CXV2D(I,J)
N1(J)=NP2D(I,J)
90 CONTINUE
C
```

```
C      WRITE OUTPUT TO NEW PAGE
C
      WRITE(6,1003)
      CALL CXVOPT(NLAG,H,C1,D1,N1,ALPHA,BETA,ITLA,ITLB,LPR)
C
C      RESET
C
      ALPHA=ALPHA+SINT
100  CONTINUE
      STOP
1000 FORMAT(F10.2)
1001 FORMAT(2A10)
1002 FORMAT(5F10.2)
1003 FORMAT(1H1)
      END
```

Chapter 6

6.1 Kriging

Kriging is a method which may be used to evaluate areas or volumes of ground by considering the expected variability between the contained sample values. The variogram has an important role to play since this is able to summarise all of the information about the expected sample variability required for the kriging process. Many people consider this to be a difficult technique to understand, though in practice it is quite simple.

The basics of a kriging program are:

1. State the input requirements -
 - a). the maximum and minimum number of sample values to be used to krige a block of ground. In practise these may be set at 12 and 4 respectively.
 - b). the radius of sample search surrounding the block in which the sample values must lie to be incorporated within the calculations.
 - c). the variogram parameters.
 - d). the dimensions of the block to be kriged.
2. Calculate the average variability of the samples contained within the dimensions of the mining block - the extension variance.
3. Locate the nearest sample values to the block centre.
4. If insufficient samples are located, move to the next block and step 3.
5. Establish the kriging matrices -
 - a). set up the S-matrix which considers the expected variability between each of the nearest surrounding values and themselves.
 - b). set up the T-matrix which considers the expected variability between each of the nearest surrounding values and the block centre.
6. Calculate the kriging coefficients by inverting the S-matrix and multiplying by the T-matrix.
7. Calculate the kriged estimate and its kriging variance.
8. Locate the next block and go to step 3.

It is also possible to krige a set of data by considering the expected covariance between the sample values. In such cases the covariogram is used as opposed to the variogram. This produces an S-matrix which is able to be inverted more easily by the computer and is quicker and more cheaper to use.

6.2 Calculating the expected variability between two sample values

The variogram is used to calculate the expected variability between two sample values separated by a distance of h metres. If the variogram has a range of A metres, a nugget variance of C_0 and a continuity of C , the expected variability $g(h)$ will be:

$$g(h) = C_0 + C(h)$$

This equation forms the basis for the formation of the kriging matrices.

6.3 Calculation of the kriged estimate and kriging variance

Once the kriging coefficients have been established, these are multiplied by their representative sample values to give the kriged estimate. The kriging variance is calculated from the sum of the products of the weighting coefficients and their respective sample-block variances. An extra constant - the Lagrange multiplier is added to ensure that the kriging variance is minimised.

6.4 Checking the variogram model

A technique referred to as Point kriging is used to test the validity of the variogram parameters. A point value is now estimated, not a block value. The kriging method remains the same except that each sample value is taken in turn, removed from the data set and its value kriged from the nearest surrounding sample values. In order for the model parameters to be valid, two criteria must be adhered to:

- a). The mean of the kriged estimates must equal the mean of the actual values.
- b). The average kriging variance must equal the average variance of estimation.

6.5 Panel kriging

In some cases panels of ground need to be valued, not blocks of ground. It is now assumed that all of the calculations will be performed upon data values which lie on a horizontal plane. This technique is often applied to sedimentary type deposits.

6.6 Universal kriging

This is a method of kriging to be used when the variogram shows the presence of drift. The difference between this type of kriging and simple kriging is that the kriging matrices are expanded to consider not only the expected sample variability but the distances between the samples. There is now a tendency for the kriging matrix to become unstable when inverted and consequently is rarely used. Instead it is better to fit a simple surface to the data values and then perform a geostatistical study upon the residuals.

6.7 Directional kriging

Directional kriging considers the expected variability between the sample values with respect to their direction from one another. If the variogram parameters are known for the directions lying along those of the three principal axes, it is possible to describe the sample variability for any sample pair lying within the confines of the three dimensional space. The kriging matrices remain in exactly the same format as before.

6.8 Covariance kriging

In some cases, the covariogram may produce a better model than that of the semi-variogram and may be used to kriging a set of sample data. The method of kriging remains exactly the same as before, except that the covariogram is now used. This can speed up the kriging procedure since the kriging matrix no longer contains a diagonal of 0's, permitting the use of a much simpler matrix inversion algorithm.

6.9 The routines

6.9.1 The extension variance

```
EXTVAR(CO,C,RAN,CC,RAN2,RAT,XINC,YINC,EXTV2D)
```

calculates the extension variance of a panel with the use of the semi-variance.

```
EXTVR1(XINC,YINC,CO,C,EXTVUK)
```

calculates the extension variance of a panel for use with Universal Kriging.

```
EXTV3D2(CO3D,C3D,RAN3D,CC3D,RAN23D,XINC,YINC,ZINC,EXTV3D)
```

calculates the directional three dimensional extension variance of a block.

```
EXTV3D3(C3D,RAN3D,CC3D,RAN23D,XINC,YINC,ZINC,EXTC3D,AVAR)
```

calculates the directional three dimensional extension covariance of a block.

6.9.2 Selecting the nearest available points for kriging

```
SLEC3D(NVAL,NMIN,NMAX,NPR,X,Y,Z,A,XR,YR,ZR,AR,WKS,XPK,YPK,  
ZPK,XMAX,YMAX,ZMAX,IY)
```

selects the nearest NMAX points to the point or block centre located at position (XPK,YPK,ZPK).

```
SELECT(NVAL,NMIN,NMAX,NPR,X,Y,A,XR,YR,AR,WKS,XPK,YPK,XMAX,  
YMAX,IY)
```

selects the nearest NMAX points to the point or panel centre located at position (XPK,YPK).

6.9.3 The point kriging matrix

PTKMAT(NP,NV,NU,X1,Y1,XPK,YPK,CO,C,RAN,CC,RAN2,RAT,SR,TR)

establishes a two dimensional point kriging matrix. This is a covariance matrix, converted from a semi-variance matrix.

PTUKMT(NP,NV,X1,Y1,SR,TR,CO,C,XMAX,YMAX,XPK,YPK,NU)

establishes a two dimensional point kriging matrix for use with Universal kriging.

PTKM3D(NP,NV,NU,X1,Y1,Z1,XPK,YPK,ZPK,CO3D,C3D,RAN3D,CC3D,
RAN23D,SR,TR)

establishes a three dimensional point kriging matrix.

PTKM3D1(NP,NV,NU,X1,Y1,Z1,XPK,YPK,ZPK,C3D,RAN3D,CC3D,RAN23D,
SR,TR,AVAR)

establishes a three dimensional point kriging matrix for use with the covariogram.

6.9.4 The panel and block kriging matrices

PNKMAT(NP,NV,NU,X1,Y1,XPK,YPK,CO,C,RAN,CC,RAN2,RAT,SR,TR,
XINC,YINC)

establishes a panel kriging matrix. This is a covariance matrix, converted from a semi-variance matrix.

PNUKMT(NP,NV,X1,Y1,SR,TR,CO,C,XMAX,YMAX,XPK,YPK,NU,XINC,YINC)

establishes a panel kriging matrix for use with Universal kriging.

PNKM3D1(NP,NV,NU,X1,Y1,Z1,XPK,YPK,ZPK,CO3D,C3D,RAN3D,CC3D,
RAN23D,SR,TR,XINC,YINC,ZINC)

establishes a three dimensional block kriging matrix.

PNKM3D2(NP,NV,NU,X1,Y1,Z1,XPK,YPK,ZPK,C3D,RAN3D,CC3D,RAN23D,
SR,TR,AVAR,XINC,YINC,ZINC)

establishes a three dimensional block kriging matrix for use with the covariogram.

6.9.5 Inversion of the kriging matrix

SKRMAT(S,T,FLAMR,NU,NV,IFAIL)

Inverts the kriging matrix and calculates the weighting coefficients.

NOTE: This routine cannot cope with a leading diagonal of zeros and can only be used to invert a covariance type matrix. A semi-variance type matrix may be inverted by the external NAG library routine:

F04AAF(S,NV,T,NV,NU,IORD,FLAMR,NV,WKS,IFAIL)

which is fully described in the NAG library manual.

6.9.6 Calculation of the kriged estimate

SOLVE1(NP,NU,FLAM,T,A1,EXTC,ZEST,VU,SILL)

calculates the kriged estimate and kriging variance produced from a covariance type kriging matrix.

SOLVE2(NP,NU,FLAM,T,A1,EXTV,ZEST,VU)

calculates the kriged estimate and kriging variance produced from a semi-variance type matrix.

6.9.7 Calculation of the directional semi-variance

GAMA3D(XC1,YC1,ZC1,XC2,YC2,ZC2,CO3D,C3D,RAN3D,CC3D,
RAN23D)

calculates the expected directional semi-variance between two points.

6.9.8 Point kriging

PTKRIG(NVAL,NMIN,NMAX,CO,C,RAN,CC,RAN2,RAT,X,Y,A,XMAX,YMAX,
LPR,ITL)

point krige a set of two dimensional data. A semi-variance matrix is established for each point and the converted to a covariance type matrix.

PTUKRG(NVAL,NMIN,NMAX,CO,C,X,Y,A,XMAX,YMAX,ITKG,LPR,ITL)

point krige a set of two dimensional data with the use of Universal kriging.

PTKG3D(NVAL,NMIN,NMAX,CO3D,C3D,RAN3D,CC3D,RAN23D,X,Y,Z,A,
XMAX,YMAX,ZMAX,LPR,ITL)

point krige a set of data with the use of directional kriging.

PTKG3D1(NVAL,NMIN,NMAX,C3D,RAN3D,CC3D,RAN23D,X,Y,Z,A,XMAX,
YMAX,ZMAX,LPR,ITL)

point krige a set of data with the use of directional covariance type kriging.

6.9.9 Panel kriging

PNKRIG(NVAL,NMIN,NMAX,CO,C,RAN,CC,RAN2,RAT,X,Y,A,XMAX,YMAX,
XINC,YINC,ANROT,XLHP,YLHP,NX,NY,ICHAN,LPR,ITL)

panel krige an area of ground commencing at point (XLHP,YLHP) - the bottom left hand panel contained within the grid.

PNUKRG(NVAL,NMIN,NMAX,CO,C,X,Y,A,XMAX,YMAX,XINC,YINC,ANROT,
XLHP,YLHP,NX,NY,ICHAN,ITKG,LPR,ITL)

panel kriges an area of ground using Universal kriging.

6.9.10 The model variogram

GMODEL(CO,C,RAN,CC,RAN2,NLAG,H,ITL,LPR)

lists the value of the model variogram for each lag distance.

6.10 Program example

```
PROGRAM K1(INPUT,OUTPUT,TAPE3,TAPE5=INPUT,
1          TAPE6=OUTPUT)
C
C   A PROGRAM TO POINT KRIGE A SET OF DATA
C
C   DIMENSION X(1000),Y(1000),Z(1000),A(1000)
C   DIMENSION CO3D(4),C3D(4),RAN3D(4),CC3D(4),RAN23D(4)
C
C   READ IN 1000 DATA POINTS FROM TAPE 3
C
C   NVAL=1000
C   DO 100 I=1,NVAL
C     READ(3,1000)X(I),Y(I),Z(I),A(I)
100    CONTINUE
C
C   READ IN VARIOGRAM PARAMETERS
C
C   DO 110 I=1,3
C     READ(5,1001)CO3D(I),C3D(I),RAN3D(I),CC3D(I),
1    RAN23D(I)
110    CONTINUE
C
C   READ IN RADII OF SAMPLE SEARCH
C
C   READ(5,1002)XMAX,YMAX,ZMAX
C
C   READ IN MINIMUM AND MAXIMUM NUMBER OF VALUES
C   REQUIRED TO KRIGE A POINT
C
C   READ(5,1003)NMIN,NMAX
C
C   READ IN TITLE OF DATA
C
C   READ(5,1004)ITL
C
C   WRITE OUTPUT TO TAPE 6
C
C   LPR=6
C
C   CALL MAIN SUBROUTINE
C
C   CALL PTKG3D(NVAL,NMIN,NMAX,CO3D,C3D,RAN3D,CC3D,
1    RAN23D,X,Y,Z,A,XMAX,YMAX,ZMAX,LPR,ITL)
C   STOP
1000  FORMAT(4F10.0)
1001  FORMAT(5F10.0)
1002  FORMAT(3F10.0)
1003  FORMAT(2I5)
1004  FORMAT(A10)
END
```

Chapter 7

7.1 The grade tonnage curve

A grade tonnage curve can be used to indicate the tonnage and grade contained within a volume of ground for a series of given cut off grades. It is essentially a reversed cumulative frequency histogram of the estimated grade values produced from the ore reserve. Routine:

```
GTCURV (NINT,GDX,GXO,NBLKS,GRADE,TONS,INP,FMT,TFAC,CONS,ITL,  
LPR,VNEG)
```

may be used to provide a histogram of grade distribution and the grade tonnage curve.

It is assumed that the grade values produced by the ore reserve are stored on a computer file as a series of individual records. These are read into the subroutine on input device number INP under format FMT, wher FMT accounts for a single real value. For example if the grade values have been output in a format of (4F6.1) where columns 1, 2 and 3 represent the x, y and z co-ordinate of the block and column 4 the estimated grade value, FMT must be input into the subroutine in the form of:

```
(18x,F6.1)
```

Any negative grades are noted and are assigned a value of VNEG.

Occasionally grades are estimated in ppm. as opposed to a percentage and in such instances need to be converted into a percentage. The value of CONS indicates the constant value by which each grade value must be multiplied to convert it into a percentage. For example, if the grade values are in percentages CONS must be assigned a value of 1.0. If the grades are measured in ppm,CONS must be assigned a value of 1000.0.

7.2 Program example

```
PROGRAM G1(INPUT,OUTPUT,TAPE3,TAPE5=INPUT,
1 TAPE6=OUTPUT)
C
C A PROGRAM TO ESTABLISH A GRADE TONNAGE CURVE
C
DIMENSION NBLKS(50),GRADE(50),TONS(50)
DIMENSION FMT(10)
C
C SET UP FOR 50 INTERVALS STARTING AT A GRADE VALUE
C OF 0.0
C
GXO=0.0
NINT=50
C
C READ IN THE GRADE VALUES ON TAPE 3
C
INP=3
C
C READ IN DATA FORMAT
C
READ(5,1000)FMT
C
C READ IN THE TONNAGE OF ROCK PER BLOCK, CONVERSION
C CONSTANT AND GRADE INTERVAL
C
READ(5,1001)TFAC,CONS,GDX
C
C READ IN VALUE TO BE ASSIGNED TO NEGATIVE VALUES
C
READ(5,1002)VNEG
C
C READ IN NAME OF METAL UPON WHICH THE CURVE IS BASED
C
READ(5,1003)ITL
C
C OUTPUT TO TAPE 6
C
LPR=6
C
C CALCULATE TONNAGE CURVE
C
CALL GTCURV(NINT,GDX,GXO,NBLKS,GRADE,TONS,INP,FMT,
1 TFAC,CONS,ITL,LPR,VNEG)
STOP
1000 FORMAT(10A10)
1001 FORMAT(F10.2,F10.3,F10.3)
1002 FORMAT(F10.2)
1003 FORMAT(A10)
END
```

Chapter 8

8.1 Miscellaneous routines

Numerous miscellaneous routines are required to fulfill simple yet essential needs. These include conversion routines which convert a value measured in one unit system into another, calculating bearings and the rotation of co-ordinate systems.

8.2 The routines

ANGRAD(ANG,RADR)

converts an angle measured in degrees into an angle measured in radians.

BEARNG(XC1,XC2,YC1,YC2,RADR,DISTL)

calculates the distance and bearing in radians from point (XC1,YC1) to point (XC2,YC2).

RADANG(RAD,ANGLE)

converts an angle measured in radians into an angle measured in degrees.

ROTATE(NVAL,X,Y,XROT,YROT,XAXOR,YAXOR,ANG)

rotates a series of co-ordinates through angle ANG degrees about point (XAXOR,YAXOR).

HEADER(LPR,NAME,IPAGE)

writes a title and page number to the top of a new page of output. CDC routines TIME(K) and DATE(K) are called to record the time and date of the output.

8.3 Program example

```
PROGRAM E1(INPUT,OUTPUT,TAPE5=INPUT,TAPE6=OUTPUT)
C
C A PROGRAM TO CONVERT AN ANGLE MEASURED IN DEGREES
C INTO AN ANGLE MEASURED IN RADIANS
C
C READ IN ANGLE IN DEGREES
C
C READ(5,1000)ANGLE
C
C CONVERT
C
C CALL ANGRAD(ANGLE,RAD)
C
C OUTPUT RESULTS
C
C WRITE(6,1001)RAD
C STOP
1000 FORMAT(F10.2)
1001 FORMAT(//,13HTHE ANGLE IS ,F10.2,8H RADIANS)
END
```

Chapter 9

9.1 The rock matrix

The deposit is divided into a series of slices (usually horizontal) known as a bench. Each one of these benches is subdivided into a series of smaller units referred to as mining blocks. It is now possible by studying the geological plans for each bench to establish the principal rock type within each block and store this information in a computer file to form the rock matrix. Each block location is assigned an integer value which represents a particular rock type eg. all of the blocks whose rock type is rock type 1 may be assigned the value of 1.

9.2 Structure of the matrix

The resulting matrix consists of NBEN benches, each one containing NX blocks in the east-west direction by NY blocks in the north-south direction. Each bench is separated by a header which states the bench number written out five times. The benches are sequently stored, the first bench being the bottom bench, the last one, the bench nearest the surface. The file is unformatted.

The first record contained in the file is the bench header written out five times. The next NY records contain the bench information. Each record contains NX pieces of information and are read into the array (MAT(I,J),I=1,NX,J=1,NY).

The first record of the bench contains the individual block information for row 1 ie. blocks lying at locations ((I,1),I=1,NX). The last record refers to row NY. Consequently array location MAT(1,1) gives the rock type found in block (1,1) (the bottom left hand edge) and MAT(NX,NY) the rock type in block (NX,NY) (the top right hand edge).

A rock matrix containing NBEN benches, each of NX by NY blocks may be read in by the following program segment:

```
DO 100 K=1,NBEN
C
C   READ IN BENCH HEADER
C
C   READ(3)NO,NO,NO,NO,NO
C
C   READ IN BENCH INFORMATION
C
C   DO 50 J=1,NY
C     READ(3)(MAT(I,J),I=1,NX
50   CONTINUE
100  CONTINUE
```

9.3 The grade matrix

The grade matrix is of exactly the same format as the rock matrix except that the estimated grade of each mining block is stored, not the rock type. Different companies have different methods of creating their own grade matrix and so it is not

proposed to include a suite of programs to create such a matrix.

Chapter 10

10.1 Open pit design

A series of programs are contained within this package which are capable of aiding the mining engineer in the design of an open pit mine. These consist of four mainline programs, each one having a specific purpose. The four are:

a). Program PITMAT displays a series of given geological sections and plans on the lineprinter and will plot the corresponding pit limits on the sections if required.

b). Program PITDES displays a series of given sections and plans of grade values on the lineprinter and will plot the corresponding pit limits on the sections if required.

c). Program PITVAL calculates the tonnage and grade contained within a given pit shell for up to 6 different cut off grades. It is possible to display the bench plans once they have been mined. Sequence mining is also possible.

d). Program PITMET calculates the tonnage and grade for up to 6 metals and 6 cut off grades based upon the cut off grades of the first metal. It is also possible to provide sequence mining.

All four programs require a matrix or matrices of a format similar to those described in the previous chapter. The maximum dimensions of the matrices are 120 blocks in the east-west direction and 110 blocks in the north-south direction. The number of benches permitted is unlimited. Single characters are used to display the rock types and ranges of grade values on the sections and plans. However certain characters may not be used in certain routines:

a). Programs PITDES and PITMAT

Characters '\', '=' and '/' are used to display the pit walls on the sections. PITDES also uses characters 'O' and 'U' to represent the blocks of overburden and unknown material.

b). Programs PITVAL and PITMET

Characters ' ' (blank), 'O' and 'U' are used to represent blocks of air, overburden and unknown material. '.' is used to plot the outline of the pit base on each bench and is used as a marker for the grade and tonnage calculations. Character '@' is not displayed though used to define the pit base in the air and is used as a marker in the grade and tonnage calculations.

A maximum of 50 characters may be used, excluding those representing air, overburden and unknown material.

It is not possible to calculate the tonnage and grade of re-entrant type pits.

10.2 Displaying sections and plans

Program PITMAT is used to display a series of geological bench plans and sections on the lineprinter taken from the rock matrix. The type of section ie. north-south section, east-west section or bench plan is selected and the required geological

section stored in a two dimensional array. This is then converted into a two dimensional array of character values, each character value being representative of a given rock type. The pit walls may be inserted if required and the section plotted. This is repeated for each relevant section.

The input parameters are:

- Card 1: The name of the project of up to 30 characters
format(3A10)
- Card 2: Relevant comments of up to 80 characters
format(8A10)
- Card 3: The dimensions of the rock matrix
format(4I5)
 - NX - number of blocks in the east-west direction
(max 120)
 - NY - number of blocks in the north-south direction
(max 110)
 - NBEN - number of benches (no limit)
 - NC - number of plotting symbols to be used (max 50)
- Card 4: Decision card regarding possible pit slopes
format(I5,F10.0)
 - ISW1 - If ISW1 is set to 0, pit slopes will be plotted on the north-south or east-west sections. This is suppressed by any non-zero value
 - SLOP - The average slope angle of the pit walls measured in degrees from the horizontal. If the value of ISW1 is non-zero SLOP should be set to 0.0 degrees.
- Card 5: Plotting requirements format(3I5)
 - IBASE - The bench number upon which the pit base is to be situated. If ISW1 is non-zero IBASE should be set to 0
 - NSEC - The total number of sections and plans to be plotted.
 - NBOT - The bottom bench number contained within the matrix.
- Card 6: The plotting characters and corresponding rock type names. format(A1,2A10)
 - IC - The single character to represent rock type IROK.
 - IROK - The name of the rock type represented by character IC of up to 20 characters.

Card 6 has to be read in NC times.
- Card 7: Plotting requirements format(4I5)
 - ITYPE - The type of section to be plotted
 - ITYPE < 0 - an east-west section
 - ITYPE = 0 - a bench plan
 - ITYPE > 0 - a north-south section
 - NO - The number of the section to be plotted
 - IL - The block number of the western edge of the

pit base on section number NO. If ISW1 has been set to a non-zero value IL should be set to 0
IR - The block number of the the eastern edge of the pit base on section number NO. If ISW1 has been set to a non-zero value IR should be set to 0

Card 7 should be read in NSEC times.

The rock matrix is read in on input device number 3.

Example 10.1

It is required to plot 6 north-south sections of the Tinpot deposit with their respective pit slopes of 45 degrees. The matrix consists of 50 by 50 blocks by 10 benches. The bottom bench number is 10 and the pit base is situated on bench number 5. The pit walls are located at positions:

section no	left hand edge	right hand edge
20	20	25
25	18	28
30	12	30
35	15	30
40	20	28
45	20	25

6 rock types are present

number	code name
1	air
2	A rock type 1
3	B rock type 2
4	C rock type 3
5	D rock type 4
6	U unknown

The input data for program PITMAT would be:

```
TINPOT DEPOSIT
DEMONSTRATION FOR EXAMPLE 6.1
  50  50  10  6
   0  45.0
   5   6  10
AIR
A ROCK TYPE 1
B ROCK TYPE 2
C ROCK TYPE 3
D ROCK TYPE 4
U UNKNOWN
  1  20  20  25
  1  25  18  28
  1  30  12  30
  1  35  15  30
  1  40  20  28
  1  45  20  25
<end-of-file>
```

10.3 Displaying sections and plans of grade values

Program PITDES can be used to display a series of plans and sections of grade values on the lineprinter. Bench plans, north-south sections or east-west sections may be plotted and once the type and section number has been selected, the relevant part of the grade matrix is read in and stored in a two dimensional array. This is then converted into an array of character values where each character represents a particular range of grade values. Pit walls may also be plotted on each section if required. This is repeated for ever required section.

The input parameters for program PITDES are as follows:

- Card 1: The name of the project of up to 30 characters
format(3A10)
- Card 2: Relevant comments of up to 80 characters
format(8A10)
- Card 3: The name of the metal whose grade values are to be plotted of up to 10 characters
format(A10)
- Card 4: The dimensions of the grade matrix
format(4I5)
NX - The number of blocks in the east-west direction
(max 120)
NY - The number of blocks in the north-south
direction (max 110)
NBEN - The number of benches required (no limit)
NC - The number of plotting symbols required
(max 50)
- Card 5: The plotting symbols format(50A1)
IC - The plotting symbols read in on one record.
Remember to omit ' ', 'O' and 'U'.

Card 6: The plotting boundary and interval
format(2F10.0)
XO - The minimum grade to be plotted. Grades having
a value of < XO will be plotted as character
IC(1).
DX - The grade interval.

Card 7: Decision card regarding pit slopes
format(I5,F10.0)
ISW1 - If ISW1 is set to 0, pit slopes will be plotted
on the north-south and/or east-west sections.
This is suppressed by any other value of ISW1.
SLOP - The average pit slope measured in degrees from
the horizontal. If ISW1 has a non-zero value
SLOP should be set to 0.0 degrees.

Card 8: Plotting requirements format(3I5)
IBASE - The bench number upon which the pit base is to
be placed. If ISW1 is non-zero set IBASE to 0
NSEC - The total number of sections and plans to be
plotted.
NBOT - The bottom bench number of the grade matrix.

Card 9: Plotting requirements format(4I5)
ITYPE - The type of section to be plotted
ITYPE < 0 - an east-west section
ITYPE = 0 - a bench plan
ITYPE > 0 - a north-south section
NO - The number of the section to be plotted
IL - The block number of the western edge of the pit
base on section number NO. If ISW1 has a
non-zero value set IL to 0
IR - The block number of the eastern edge of the pit
base on section number NO. If ISW1 has a
non-zero value set IR to 0

Card 9 should be read NSEC times.

The grade matrix is read into the program on input device
number 3.

Example 10.2

It is required to plot the copper grade values of the
north-south sections given in example 10.1. The minimum grade
value to be plotted is 0.0 with a grade interval of 0.1. 11 grade
ranges are to be plotted.

The input for program PITDES is:

```
TINPOT DEPOSIT
DEMONSTRATION FOR EXAMPLE 6.2
COPPER
  50  50  10  11
0123456789*
   0.0      0.1
  0  45.0
  5  6  10
  1  20  20  25
  1  25  18  28
  1  30  12  30
  1  35  15  30
  1  40  20  28
  1  45  20  25
<end-of-file>
```

10.4 The calculation of the grade and tonnage contained within a pit shell

Program PITVAL calculates the tonnage and grade contained within a given pit shell whose base is defined by a maximum of 100 points measured in a clockwise direction. Each point is read in as a series of integer values, each one defining the centre of a block, resident on the bench on which the pit base lies. It is assumed that a block will be either mined or left in place, portions of blocks are not dealt with.

The grade values contained on that bench on which the pit base lies are located and stored in a two dimensional array. These grade values are then converted into a series of characters representing the grade intervals in which they belong and are also stored in a two dimensional array. The points defining the pit base are then inserted into the character matrix and joined by a series of dots. If part of the pit base lies in air, the base in this portion is defined by the character '@'. The grade and tonnage contained on this bench is then calculated by noting the blocks contained within the pit perimeter.

The next bench upwards is then selected and its grade values are stored in the two matrices as before. Since this bench lies above the bench on which the pit base was defined, it is necessary to project this pit base upwards. This involves the use of geometrical methods.

The bearing is noted between the base points

(X_i, Y_i) and (X_{i-1}, Y_{i-1}) and

(X_i, Y_i) and (X_{i+1}, Y_{i+1})

The mid bearing between the two point pairs is noted. 180 degrees is either added or subtracted to this bearing to ensure that the point is moved outwards from the pit centre and the new position of the base point is then calculated for the bench.

This procedure is repeated and each bench valued in a similar manner until either the top of the matrix is encountered or all of the blocks contain air.

The input parameters for program PITVAL are:

- Card 1: Name of the project of up to 30 characters
format(3A10)
- Card 2: Relevant comments of up to 80 characters
format(8A10)
- Card 3: Name of the metal for which the pit is to be valued of
up to 10 characters format(A10)
- Card 4: The dimensions of the grade matrix
format(4I5)
NX - The number of blocks in the east-west direction
(max 120)
NY - The number of blocks in the north-south
direction (max 110)
NBEN - The number of benches (no limit)
NC - The number of plotting characters required
(max 50)
- Card 5: The plotting symbols format(50A1)
IC - The plotting symbols read in on one record
(max 50). Omit characters ' ', 'U', 'O', '.',
and '@'.
- Card 6: The plotting boundary intervals
format(2F10.0)
XO - The minimum grade to be plotted. Grades having
a value of < XO will be plotted by character
IC(I).
XINC - The grade interval.
- Card 7: The tonnage and grade conversion factor
format(2F10.0)
TON - The tonnage of material contained within a
block.
TFAC - The conversion factor to convert grade values
into %.
- Card 8: Output type card format(2I3)
IOUT - Any non-zero value allows the mined out bench
plans to be plotted on the lineprinter.
NPIT - The number of pits to be sequenced. ie. a
second or further pits may be mined from the
previous mined out matrix.
- Card 9: Grade of overburden and unknown material
format(2F10.0)
GOVB - The grade assigned to the overburden
GUNK - The grade assigned to the unknown material

Note: It is important that these values are input as
precisely as possible since these values are
compared with the values contained in the grade
matrix and may give round off errors if
precautions are not taken.
- Card 10: The pit base format(2I5)

NPTS - The number of points to be used to define the
pit base (max 100).
NBASE - The bench number upon which the pit base is
situated.

Card 11: The pit base co-ordinates format(2I5)
IPT - The I co-ordinate of each point
JPT - The J co-ordinate of each point

Repeat card 11 NPTS times

Card 12: The cut off grade, slope angle and cut off grade
interval format(3F10.0)

COFF - The initial cut off grade to be used
CINC - The cut off grade interval
SLOP _ The average pit slope angle in degrees

The grade matrix is read in on input device number 3.

10.5 The calculation of the grade and tonnage for a series of metals contained within a given pit shell

Program PITMET may be used to calculate the grade and
tonnages of up to 6 metals contained within a given pit shell,
based upon the cut off grades of one particular metal. The
program works in exactly the same way as program PITVAL except
that more than one metal is used. The input data is exactly the
same except for cards:

Card 3: The names of the 6 metals each of up to 10 characters.
The first name is that of the metal of primary
concern. format(6A10)

Card 4: The dimension of the grade matrix
format(5I5)
NX - as before
NY - as before
NBEN - as before
NC - as before
NMET - The number of metals required (max 6) which
includes the main metal concerned

The primary metal is read in on input device number 3, the
extra metals on tapes 10, 11, 12, 13 and 14 (in that order).

Chapter 11

11.1 Graphical output

Three subroutines are contained within this package which are capable of providing graphical output. These employ a series of external subroutines contained within the GINO-F graphical library which unfortunately is not available on every computer system. All output is drawn to A4 paper size and presents plots which are titled and display the time and date on which the routine was run. An example of output is given at the end of this chapter.

Before any one of these routines are called it is necessary to switch on the plotter by means of the GINO-F subroutine:

```
CC936N
```

Likewise it is necessary to switch off the plotter once plotting has ceased with GINO-F subroutine:

```
DEVPAP
```

If more than one plot is required subroutine PICCLE must be called after each plot has been done.

11.2 The routines

```
PLOT1(X,Y,NVAL,SCALE,NAME,ISWP)
```

plots a map of the drill hole collars to a scale of SCALE. The drill hole locations may be numbered if ISWP is assigned a non-zero value on entry.

```
PLOT2(FREQ,NCLASS,NAME,METAL,DX,XO)
```

plots a frequency distribution in the form of a histogram.

```
PLOT3(A,B,NVAL,NAME,METAL,JTLA,JTLB)
```

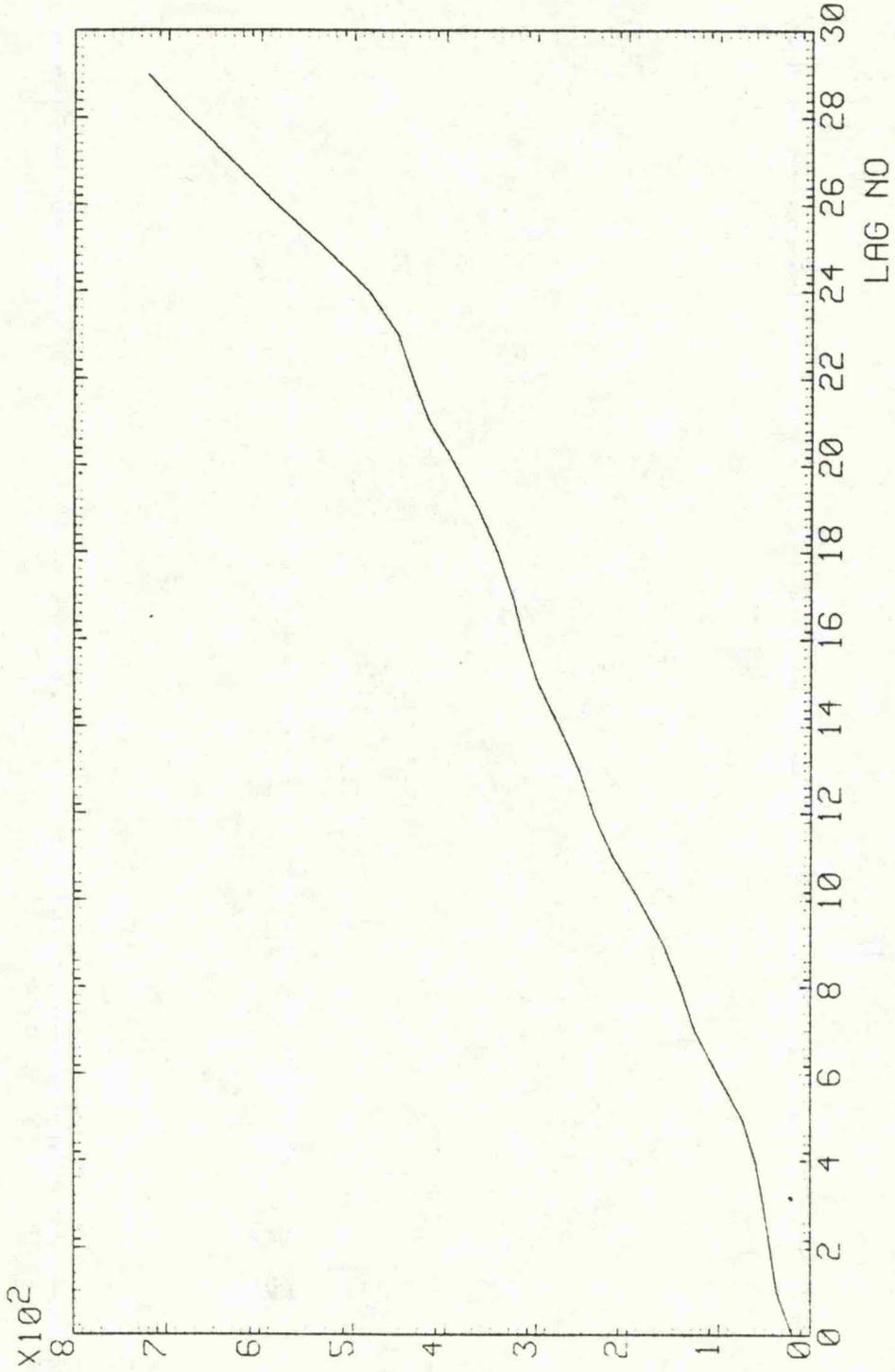
plots a graph of variable A against variable B.

SEMI-VARIOGRAMS FOR YEARLY PRICE

ELEMENT: COPPER

DATE: 82/11/04.

, GAMMA(H)



11.3 Program example

```
PROGRAM P1(INPUT,OUTPUT,TAPE5=INPUT,TAPE6=OUTPUT)
C
C A PROGRAM TO PLOT A GRAPH OF GRADE AGAINST DISTANCE
C DOWN THE HOLE
C
C DIMENSION A(10),B(10),ITLS(6)
C
C READ IN 10 DATA VALUES
C
C NVAL=10
C DO 10 I=1,NVAL
C READ(5,1000)A(I),B(I)
10 CONTINUE
C
C READ IN TITLES OF AXES
C
C READ(5,1001)JTTLA,JTLB
C
C READ IN TITLE OF PLOT OF UP TO 60 CHARECTERS
C
C READ(5,1002)ITLS
C
C READ IN NAME OF REQUIRED METAL OF UP TO
C 10 CHARECTERS
C
C READ(5,1003)METAL
C
C SWITCH ON PLOTTER
C
C CALL CC936N
C
C PLOT GRAPH
C
C CALL PLOT3(A,B,NVAL,ITLS,METAL,JTTLA,JTLB)
C
C SWITCH OFF PLOTTER
C
C CALL DEVEND
C STOP
1000 FORMAT(2F10.2)
1001 FORMAT(2A10)
1002 FORMAT(6A10)
1003 FORMAT(A10)
END
```

Chapter 12

12.1 Error descriptions

Once a subroutine is called, checks are immediately performed to ensure that the supplied data will not cause the subroutine to fail. This mainly consists of checking for zero values. When an error is detected routine:

ERRORS(NM1,IFAIL,LPR)

is called which lists the MINPAK error number and the name of the routine in which the error was detected. Execution is then terminated.

The MINPAK error numbers and their descriptions are as follows:

Error number	description
1	The number of data values (NVAL) is less than 1
2	The number of data values (NVALA) contained within the first population is less than 1
3	The number of data values (NVALB) contained within the second population is less than 1
4	The sum of the values (SUMA) contained within the first population is zero
5	The sum of the values (SUMB) contained within the second population is zero
6	The sum of the values squared (SUMA2) contained within the first population is zero
7	The sum of the values squared (SUMB2) contained within the second population is zero
8	The standard deviation of the first population (SDEVA) is zero
9	The standard deviation of the second population (SDEVB) is zero
10	The variance of the first population (AVAR) is zero
11	The variance of the second population (BVAR) is zero
12	The intercept of the regression line (A0) with the Y-axis is infinite
13	The gradient of the regression line (GRAD) is infinite
14	The lag interval (H) is zero

15 The number of lags required (NLAG) is zero

16 The number of sector semi-variograms required (NSEC) is zero

17 The number of sector semi-variograms required (NSEC) is greater than 18

18 The range of the variogram (RAN) is zero

19 The range of one or more of the directional semi-variograms (RAN3D) is zero

20 The ratio of anisotropy (RAT) is zero

21 The dimension of the panel (XINC or YINC) is zero

22 The dimension of the block (XINC, YINC or ZINC) is zero

23 The radius of sample search (XMAX, YMAX or ZMAX) is zero

24 The number of points (NP) to be used to set up the kriging matrix is less than 1

25 The dimension of the S-matrix (NV) is zero

26 The number of kriging coefficients (NU) is less than 1

27 The number of kriging coefficients (NU) required exceeds the dimension of the S-matrix (NV)

28 The minimum number of points (NMIN) required to set up a kriging matrix is less than 1

29 The maximum number of points (NMAX) required to set up a kriging matrix is less than 1

30 The minimum number of points (NMIN) required to set up a kriging matrix exceeds the maximum (NMAX)

31 The kriging matrix is singular

32 The number of points (NP) to be used to set up the kriging matrix exceeds the number of kriging coefficients (NU)

33 Array DATA has less than 1 column (NCOL) of data

34 Array DATA has less than 1 row (NROW) of data

35 Too many data points are to be stored in array DATA

36 The data is to be composited over a distance of zero

37 End-of-file detected

38 The number of columns (NCOL) is less than or equal
to 2

Chapter 13

13.1 Parameter descriptions

The parameter descriptions for the MINPAK subroutines are listed below. Accompanying diagrams are also included at the end of this chapter to illustrate and further describe some of the parameters.

Unless specified otherwise all parameters are returned unaltered.

A	Real array of dimension A(NVAL) On entry specifies the grade of each value located at positions ((X(I),Y(I),Z(I)) for I=1,NVAL)
A0	Real On return specifies the value of the intercept of the regression line with the y-axis
A1	Real array of dimension A1(NMAX) On entry specifies the values of the nearest NP samples to the point or block centre located at (XPK,YPK,ZPK)
AA1	Real On entry specifies the value of a given sample
AA2	Real On entry specifies the value of a second sample
AKURT	Real On return specifies the degree of kurtosis
ALPHA	Real On entry specifies the bearing measured in degrees along which the directional semi-variogram is to be calculated
AMAX	Real On return specifies the maximum value
AMEAN	Real On entry specifies the mean value
AMIN	Real On return specifies the minimum value
ANG	Real On input specifies the angle measured in degrees
ANGLE	Real On return specifies the bearing measured in degrees
ANGR	Real On return specifies the angle measured in radians
ANROT	Real On entry specifies the angle of rotation measured in degrees in a clockwise direction to transform the

sample co-ordinates onto the mine grid

AR Real array of dimension AR(NMAX)
On return specifies the values of the nearest NP samples to the point or block centre located at (XPK,YPK,ZPK)

AVAR Real
On entry specifies the variance

AZM Real
On entry specifies the direction of the dip of the drill hole measured in degrees ie. the azimuth

B Real array of dimension B(NVAL)
On entry specifies the value of a second variable located at positions ((X(I),Y(I),Z(I)) for I=1,NVAL)

BASE Real
On entry specifies the height O.D. of the bottom bench toe

BB1 Real
On entry specifies the value of a second variable of a sample pair

BB2 Real
On entry specifies the value of a second variable of a second sample pair

BETA Real
On entry specifies the angle of regularisation measured in degrees

BGHT Real
On entry specifies the height of each bench

BMEAN Real
On entry specifies the mean value of a second set of sample values

BVAR Real
On entry specifies the variance of a second set of sample values

C Real
On entry specifies the continuity of the isotropic variogram or covariogram ie. the sill value less the nugget variance

CC Real
On entry specifies the continuity of the second structure of the variogram or covariogram ie second sill less the first sill. If only one structure is present set CC to 0.0

C3D Real array of dimension C3D(4)
On entry specifies the continuity of the variogram or covariogram along the x-axis (C3D(1)), y-axis (C3D(2)) and z-axis (C3D(3)). On return C3D(4)

specifies the directional continuity

- CC3D Real array of dimension CC3D(4)
On entry specifies the continuity of the second structures of the variogram or covariogram for the x-axis (CC3D(1)), y-axis (CC3D(2)) and z-axis (CC3D(3)). On return CC3D(4) specifies the directional continuity.
If only one structure is present for any axis (CC3D(I), I=1,4) must be set to 0.0
- CHI2 Real
On return specifies the Chi square value for seven degrees of freedom
- CO Real
On entry specifies the nugget variance of the isotropic variogram
- CO3D Real array of dimension CO3D(4)
On entry CO3D(1) specifies the nugget variance for the x-axis variogram, CO3D(2), the nugget variance for the y-axis variogram and CO3D(3), the nugget variance for the z-axis variogram. On return CO3D(4) specifies the calculated directional nugget variance
- CONS Real
On entry specifies the conversion factor by which grade values may be converted into percentages ie. grades measured in ppm. require a CONS value of 1000.0
- COV Real array of dimension COV(NLAG)
On return specifies the average covariance for each lag distance
- COVA Real array of dimension COVA(NLAG)
On entry specifies the average covariance for each lag distance
- COVR Real
On return specifies the average covariance
- COV2D Real array of dimension COV2D(NLAG,NSEC)
On return specifies the average covariance per lag interval for each sector semi-variogram
- CPERC Real array of dimension CPERC(NCLASS)
On return specifies the cumulative frequency percentage per class interval
- CXVAR Real array of dimension CXVAR(NLAG)
On return specifies the average cross-variance per lag distance
- CXVAR1 Real array of dimension CXVAR1(NLAG)
On return specifies the average cross-variance per lag distance

CXV Real
On return specifies the average cross-variance

CXV2D Real array of dimension CXV2D(NLAG,NSEC)
On return specifies the average cross-variance per lag distance for each sector cross-variogram

DATA Real array of dimension DATA(NCOL,NROWS)
On entry specifies the from-to information for one drill hole

DATAR Real array of dimension DATAR(NCOL,NROWS)
On return specifies the from-to information for one drill hole

DIP Real
On entry specifies the dip of the drill hole measured in degrees

DIS - Real array of dimension DIS(NLAG)
On entry specifies the average of sample separation per lag distance

DIST Real array of dimension DIST(NLAG)
On return specifies the average distance of sample separation per lag distance

DISTL Real
On return specifies the vector distance between the two sample values

DIS2D Real array of dimension DIS2D(NLAG,NSEC)
On return specifies the average distance of sample separation per lag interval for each sector semi-variogram or cross-variogram

DRF2D Real array of dimension DRF2D(NLAG,NSEC)
On return specifies the average drift per lag distance for each sector semi-variogram

DRIF Real array of dimension DRIF(NLAG)
On entry specifies the average drift per lag distance

DRIFT Real array of dimension DRIFT(NLAG)
On return specifies the average drift per lag interval

DX Real
On entry specifies the histogram class interval

EXTC Real
On entry specifies the extension covariance of a panel or block. If point kriging is to be used EXTC must be set to the variance on entry

EXTC3D Real
On return specifies the three dimensional extension covariance of a block

EXTV Real
On entry specifies the extension variance of a panel or block. If point kriging is to be used EXTV must be set to 0.0 on entry

EXTV2D Real
on return specifies the extension variance of a panel

EXTVUK Real
On return specifies the extension variance of a panel for use with Universal Kriging

EXTV3D Real
On return specifies the three dimensional extension variance of a block

F Real
On return specifies the F-value

FILT Real
On entry acts as a filter. Sample values having a value of equal to FILT will be removed from further calculations

FLAM Real array of dimension FLAM(NMAX+6)
On entry specifies the NU kriging coefficients

FLAMR Real array of dimension FLAMR(NMAX+6)
On return specifies the NU kriging coefficients

FMT Alphanumeric array of dimension FMT(8)
On entry specifies the format of the input data of up to 80 characters

FMTO Alphanumeric array of dimension FMTO(8)
On entry specifies the format of the output of up to 80 characters

FREQ Real array of dimension FREQ(NCLASS)
On entry specifies either the frequency percentage or cumulative frequency percentage of values per histogram class interval

FROM Real array of dimension FROM(NOCC)
On entry specifies the distance down the hole from which each value (FTGRAD(I), I=1, NOCC) was taken

FROK Real array of dimension FROK(NV2)
On entry specifies the distance down the hole from which a particular rock type (ICODE(I), I=1, NV2) commences

FTGRAD Real array of dimension FTGRAD(NOCC)
On entry specifies the sample value observed between (FROM(I), TO(I), I=1, NOCC)

GAMA Real
On return specifies the average semi-variance

GAMM Real array of dimension GAMM(NLAG)
On entry specifies the average semi-variance per lag distance

GAMMA Real array of dimension GAMMA(NLAG)
On return specifies the average semi-variance per lag distance

GAM2D Real array of dimension GAM2D(NLAG,NSEC)
On return specifies the average semi-variance per lag distance for each sector semi-variogram

GAXIS Real
On entry specifies the minimum grade value to be plotted

GDX Real
On entry specifies the grade interval of the grade tonnage curve

GINC Real
On entry specifies the grade interval for plotting purposes

GRAD Real
On return specifies the gradient of the regression line

GRADE Real array of dimension (NINT)
On return specifies the average grade of material above each cut of grade

GXO Real
On entry specifies the initial cut off grade of the grade tonnage curve

H Real
On entry specifies the lag interval

ICD Integer
On entry specifies the rock type number

ICHAN Integer
On entry specifies the output device number to which the kriged estimates are to be written for storage purposes. A value of ICHAN=0 suppresses the output

ICHAR Integer or alphanumeric
On entry specifies the number or character(s) to be used to terminate a read in sequence

ICODE Integer array of dimension ICODE(NV2)
On entry specifies the rock type number found between distances (FROK(I), TROK(I), I=1,NV2) down the hole

ICUM Integer array of dimension ICUM(NCLASS)
On return specifies the cumulative number of observations per class interval

IDF Integer
On return specifies the number of degrees of freedom for the T-test

IDFA Integer
On return specifies the number of degrees of freedom for the F-test for the first set of sample values

IDFB Integer
On return specifies the number of degrees of freedom for the F-test for the second set of sample values

IFAIL Integer
A returned value of 1 indicates that the kriging matrix is singular

INP Integer
On entry specifies the input device number upon which a datafile is to be read into a subroutine - default value 3

IORD Integer
On entry specifies the order of the kriging matrix. In this case it is 1

IPAGE Integer
On entry specifies the page number and is increased by 1 on output and subsequent return

ISET Integer
On entry is used to define the origin of the x-axis for plotting purposes. A value of ISET=0 sets the origin to 0.0. Any other value sets the origin to the minimum value

ISWO Integer
On entry acts as a switch to govern the type of output required.
ISW0 > 0 gives unformatted output
ISW1 < 1 gives output under format FMTO

ISWP Integer
A value of 0 on entry enables the drill hole collars to be numbered when plotted

ITKG Integer
On entry specifies the type of drift required for Universal Kriging.
ITKG=1 indicates that quadratic drift is required.
Any other value permits linear drift to be used

ITL Alphanumeric
On entry specifies the name or title of the sample values of up to 10 characters

ITLA Alphanumeric
On entry specifies the name or title of the first set of sample values of up to 10 characters

ITLB Alphanumeric
On entry specifies the name or title of the second set of sample values of up to 10 characters

IX Integer
A returned value of 0 indicates that one or more records of information have been read in.

IY Integer
On returned value of 0 indicates that sufficient sufficient sample values are available to establish a kriging matrix

JSET Integer
On entry is used to define the origin of the y-axis for plotting purposes. A value of JSET=0.0 sets the origin to 0.0. Any other value sets the origin to the minimum value

JTLA Alphanumeric
On entry specifies the title of the x-axis of up to 10 characters

JTLB Alphanumeric
On entry specifies the title of the y-axis of up to 10 characters

LAGNO Integer
On return specifies the lag number corresponding to the lag distance between two sample values

LPR Integer
On entry specifies the output device number to which output is to be written - default value 6

METAL Alphanumeric array of dimension METAL(2)
On entry specifies the name of the commodity to be plotted of up to 20 characters

NAME Alphanumeric array of dimension NAME(6)
On entry specifies the title to be output of up to 60 characters

NBEN Integer
On entry specifies the bottom bench number

NBLKS Integer array of dimension NBLKS(NINT)
On entry specifies the number of blocks whose grade is greater than that of the cut off grade, for each cut off grade interval

NCLASS Integer
On entry specifies the number of class intervals required to produce a histogram - max. 30.
Note: Class interval number 1 contains the number of samples having a value of less than XO. Class interval number NCLASS contains the number of samples having a value of greater than ((NCLASS-2)*DX)+XO

NCOL Integer
 On entry specifies the number of columns of from-to data to be read into the DATA array as specified in the main program

NDATA Integer
 On return specifies the number of observations read in for the hole

NHOLE Integer
 On entry specifies the hole number

NINT Integer
 On entry specifies the number of cut off grade intervals required by the grade tonnage curve

NLAG Integer
 On entry specifies the number of lag intervals required

NMAX Integer
 On entry specifies the maximum number of sample values to be used to kriging a point or block value

NMIN Integer
 On entry specifies the minimum number of sample values to be used to kriging a point or block value

NM1 Alphanumeric
 On entry specifies the name of the subroutine of up to 8 characters in which the error occurred

NOBS Integer array of dimension NOBS(NCLASS)
 On return specifies the number of observations per class interval

NOCC Integer
 On entry specifies the number of observations within the hole

NOSEC integer
 On return specifies the sector number in which two sample values lie

NP Integer
 On entry specifies the number of sample values to be used to kriging a point or block value

NPAIR Integer array of dimension NPAIR(NLAG)
 On entry specifies the number of sample pairs per lag distance

NPAIRS Integer array of dimension NPAIRS(NLAG)
 On return specifies the number of sample pairs per lag distance

NPR Integer
 On return specifies the number of sample values to be used to kriging a point or block value

NP2D Integer array of dimension NP2D(NLAG,NSEC)
On return specifies the number of sample pairs per lag interval for each sector semi-variogram or cross-variogram

NROW Integer
On entry specifies the maximum number of rows to be read into array DATA as specified in the main program

NSEC Integer
On entry specifies the number of sector semi-variograms or cross-variograms required - max 18

NU Integer
On entry specifies the number of kriging coefficients. For simple kriging ie. non Universal Kriging NU should equal NP+1. In the case when Universal kriging is required, NU should be set to NP+3 for a linear drift and NP+6 for a quadratic drift

NV Integer
On entry specifies the number of rows contained within the S-matrix as defined in the main program

NV2 Integer
On entry specifies the number of changes of rock type boundary within the hole

NVAL Integer
On entry specifies the number of data values

NVALA Integer
On entry specifies the number of data values contained within the first set of sample values

NVALB Integer
On entry specifies the number of data values contained within the second set of sample values

NX Integer
On entry specifies the number of blocks or panels to be kriged along the x-axis

NY Integer
On entry specifies the number of blocks or panels to be kriged along the y-axis

PERC Real array of dimension PERC(NCLASS)
On return specifies the frequency percentage of observations per histogram class interval

R Real
On return specifies the correlation coefficient

RAD Real
On entry specifies the angle measured in degrees

RADR Real
On return specifies the angle measured in radians

RAN Real
On entry specifies the range of the isotropic variogram

RAN2 Real
On entry specifies the range of the second structure of the isotropic variogram. If only one structure is present, set RAN2 to any non zero value.

RAN23D Real array of dimension RAN23D(4)
On entry specifies the range of the second structure of the variogram along the x-axis (RAN23D(1)), y-axis (RAN23D(2)) and z-axis (RAN23D(3)). On return RAN23D(4) specifies the directional range of the second structure. If only one structure is present along any one of the three axes (RAN23D(I), I=1,4) must be set to any non-zero value

RAN3D Real array of dimension RAN3D(4)
On entry specifies the range of the variogram along the x-axis (RAN3D(1)), y-axis (RAN3D(2)) and z-axis (RAN3D(3)). On return RAN3D(4) specifies the directional range

RAT Real
On entry specifies the ratio of anisotropy ie. range of the variogram along the x-axis divided by the range along the y-axis

S Real array of dimension S(NV,NV)
On entry specifies the S-matrix and is reduced to the identity matrix on return

SCALE Real
On entry specifies the scale to which the map is to be plotted

SECT Real array of dimension SECT(19)
On entry specifies the bearings of the sector semi-variogram boundaries measured in radians

SECTB Real array of dimension SECTB(19)
On return specifies the bearings of the sector semi-variogram boundaries measured in radians

SEST Real
On return specifies the standard error of estimation

SILL Real
On entry specifies the sill value of the variogram

SKEW Real
On return specifies the degree of skewness

SR Real array of dimension SR(NV,NV)
On return specifies the S-matrix

STAND Real array of dimension STAND(NVAL)
On return specifies the set of standardised values

T Real array of dimension T(NMAX+6)
On entry specifies the T-matrix

TFAC Real
On entry specifies the tonnage of material contained
within a block

TO Real array of dimension TO(NOCC)
On entry specifies the distance down the hole to
which each sample value is measured

TONS Real array of dimension TONS(NINT)
On return specifies the tonnage of ore for each
cut off grade

TR Real array of dimension TR(NMAX+6)
On return specifies the T-matrix

TROK Real array of dimension TROK(NV2)
On entry specifies the distance down the hole to
which each rock type boundary is measured

TVAL Real
On return specifies the T-value

VNEG Real
On entry specifies the grade to be assigned to a
block with a negatively kriged estimate

VU Real
On return specifies the kriging variance

WKS Real array of dimension WKS(NMAX)
Used as workspace

WKSP Real array of dimension WKSP(NCOL)
Used as workspace

X Real array of dimension X(NVAL)
On entry specifies the east-west co-ordinate of each
sample value (A(I), I=1, NVAL)

X1 Real array of dimension X1(NP)
On entry specifies the x co-ordinates of the nearest
NP samples to the point or block centre located at
(XPK, YPK, ZPK)

XAXOR Real
On entry specifies the x co-ordinate of the point
about which a series of points are to be rotated

XC1 Real
On entry specifies the x co-ordinate of a sample
value

XC2 Real
On entry specifies the y co-ordinate of a second sample value

XCOL Real
On entry specifies the x co-ordinate of the drill hole collar

XINC Real
On entry specifies the size of the block or panel along the x-axis

XLHP Real
On entry specifies the x co-ordinate of the centre of bottom left hand panel in the mining grid

XMAX Real
On entry specifies the radius of sample search along the x-axis

XMEAN Real
On return specifies the mean of the set of sample values

XO Real
On entry specifies the starting point of the histogram

XPk Real
On entry specifies the x co-ordinate of the point or block centre to be kriged.

XR Real array of dimension XR(NMAX)
On return specifies the x co-ordinates of the nearest NP samples to the point or block centre located at (XPk,YPk,ZPk)

XROT Real array of dimension XROT(NVAL)
On return specifies the new x co-ordinates following a rotation

XVAR Real
On return specifies the variance

Y Real array of dimension Y(NVAL)
On entry specifies the north-south co-ordinate of each sample value (A(I),I=1,NVAL)

Y1 Real array of dimension Y1(NP)
On entry specifies the y co-ordinates of the nearest NP samples to the point or block centre located at (XPk,YPk,ZPk)

YAXOR Real
On entry specifies the y co-ordinate of the point about which a series of points are to be rotated

YCOL Real
On entry specifies the y co-ordinate of the drill

hole collar

YC1 Real
On entry specifies the y co-ordinate of a sample value

YC2 Real
On entry specifies the y co-ordinate of a second sample value

YINC Real
On entry specifies the size of the block or panel along the y-axis

YLHP Real
On entry specifies the y co-ordinate of the centre of the bottom left hand panel in the mining grid

YMAX Real
On entry specifies the radius of sample search along the y-axis

YPK Real
On entry specifies the y co-ordinate of the point or block centre to be kriged

YR Real array of dimension YR(NMAX)
On return specifies the y co-ordinates of the nearest NP values to the point or block centre located at (XPK,YPK,ZPK)

YROT Real array of dimension YROT(NVAL)
On return specifies the new y co-ordinates following a rotation

Z Real array of dimension Z(NVAL)
On entry specifies the vertical co-ordinates of each sample value (A(I),I=1,NVAL)

Z1 Real array of dimension Z1(NP)
On entry specifies the z co-ordinates of the nearest NP values to the point or block centre located at (XPK,YPK,ZPK)

ZCOL Real
On entry specifies the z co-ordinate of the drill hole collar

ZC1 Real
On entry specifies the z co-ordinate of a sample value

ZC2 Real
On entry specifies the z co-ordinate of a second sample value

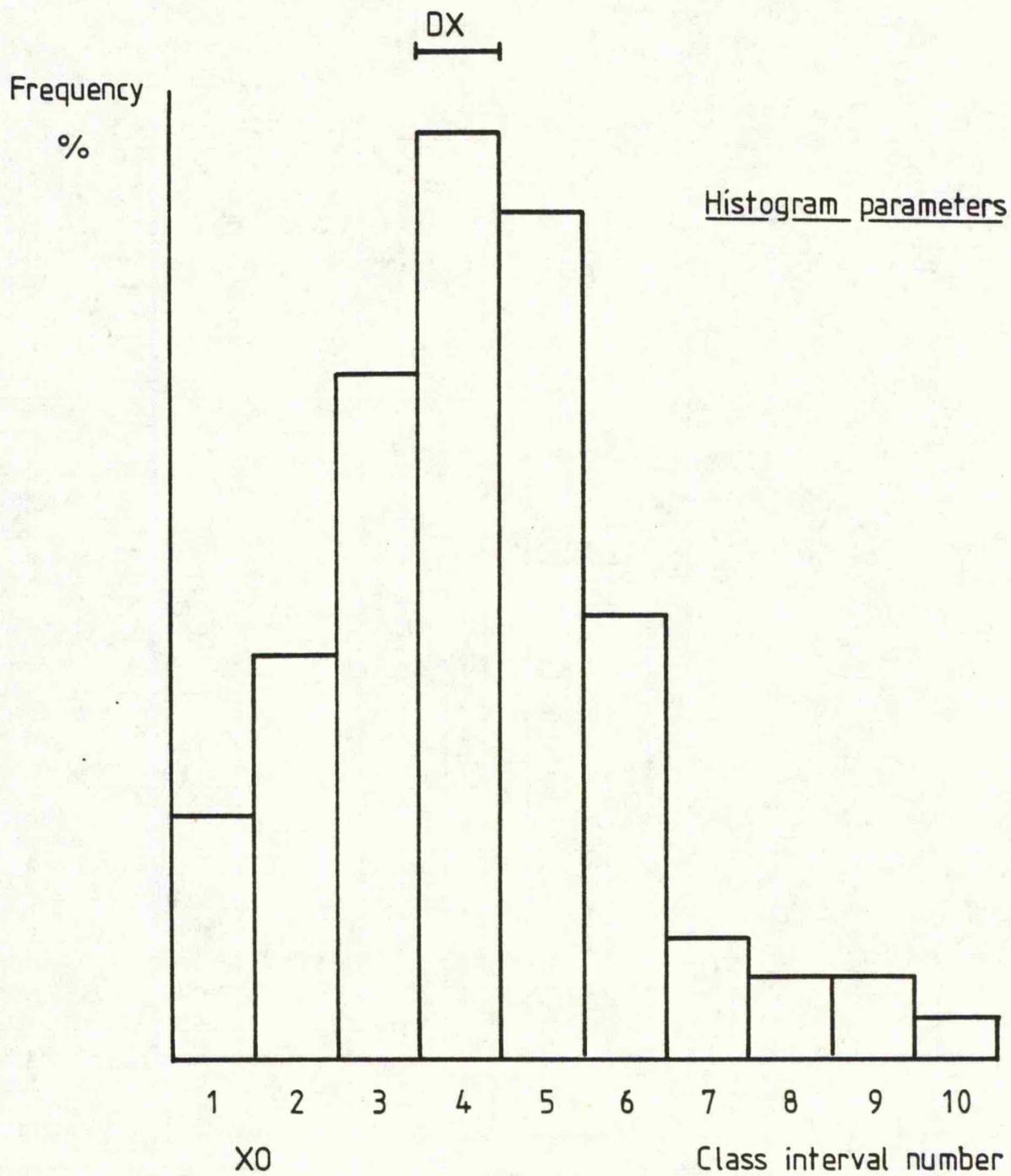
ZEST Real
On return specifies the kriged estimate

ZINC Real
On entry specifies the size of the block along the
z axis

ZMAX Real
On entry specifies the radius of sample search along
the z axis

ZPK Real
On entry specifies the z co-ordinate of the point
or block centre to be kriged

ZR Real array of dimension ZR(NMAX)
On return specifies the z co-ordinates of the nearest
NP values to the point or block centre located at
(XPK, YPK, ZPK)

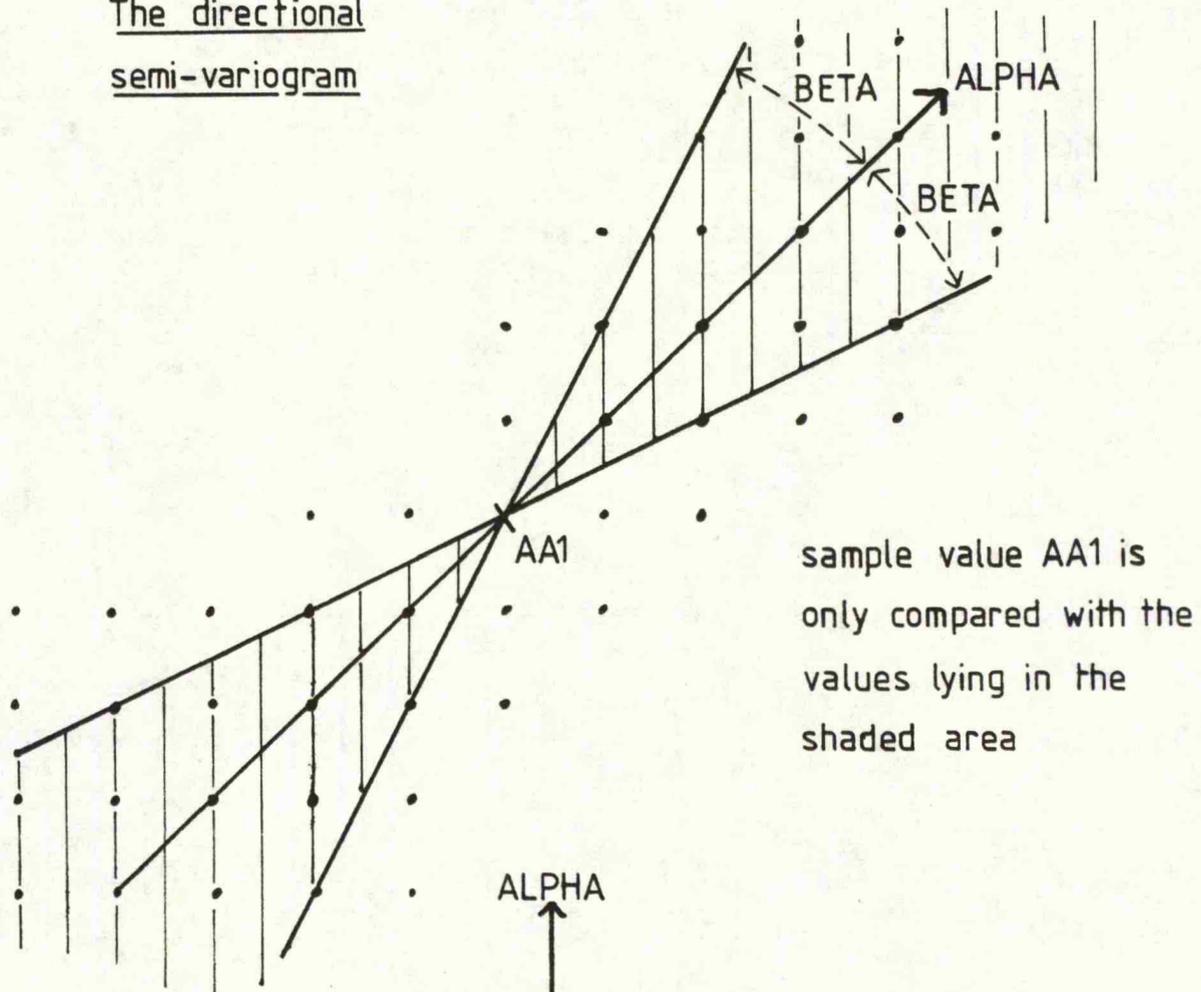


NCLASS = 10

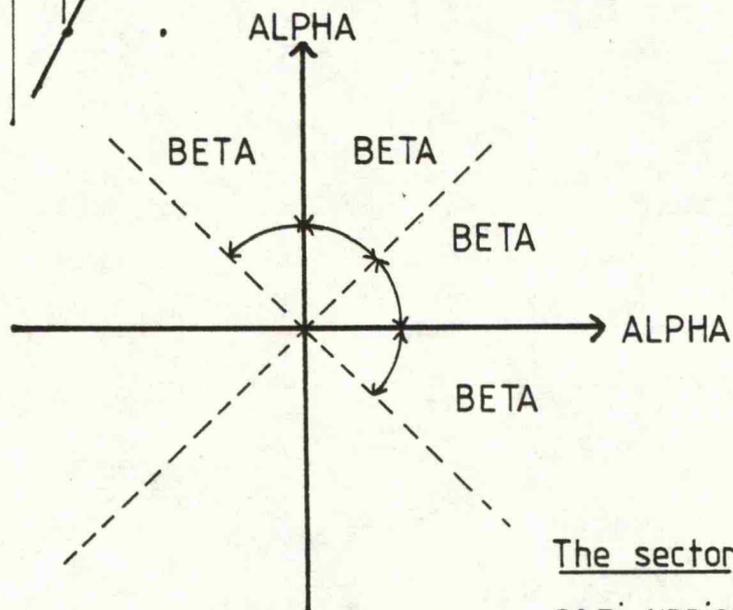
Values \leq X0 categorised in cell 1 ie. FREQ(1)

Values $>$ (8x DX) + X0 categorised in cell 10
ie. FREQ(10)

The directional
semi-variogram



sample value AA1 is only compared with the values lying in the shaded area



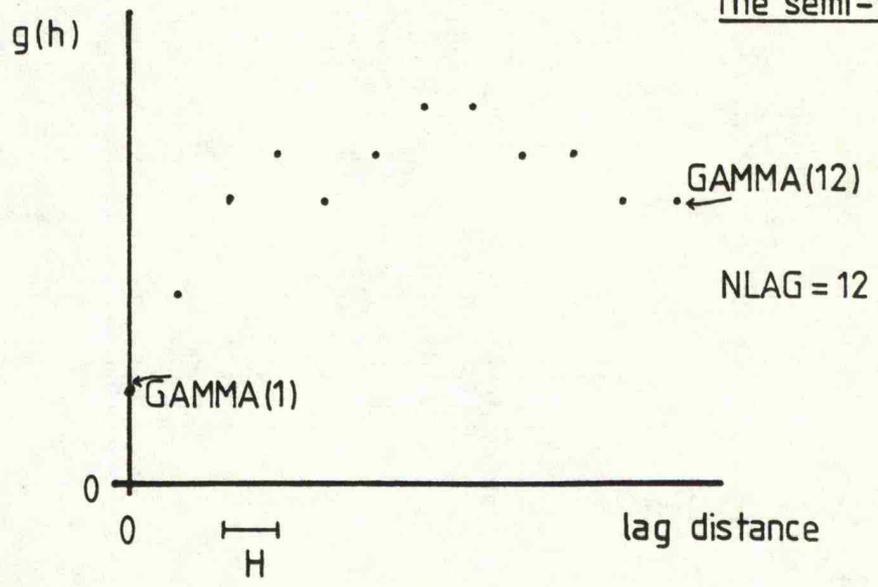
The sector
semi-variogram

$NSEC = 2$

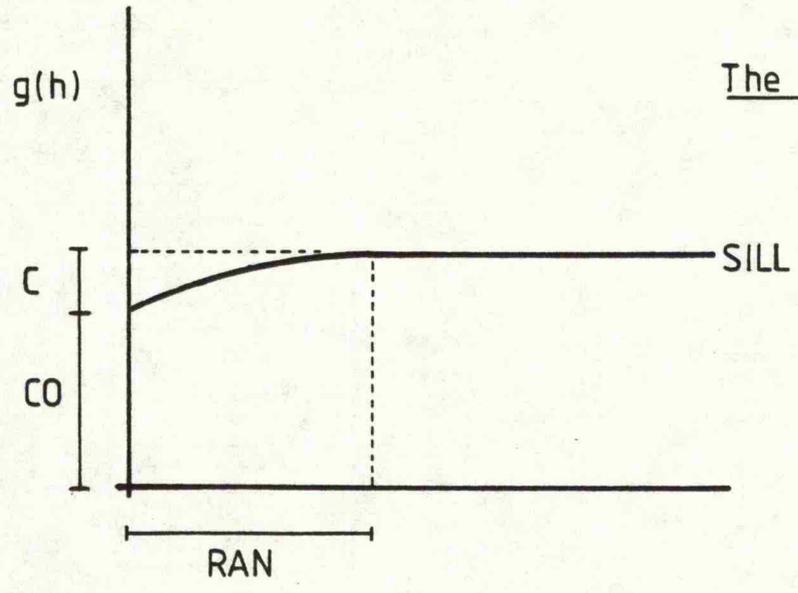
$\therefore ALPHA = 000 \text{ and } 090 \text{ degrees}$

$BETA = 45 \text{ degrees}$

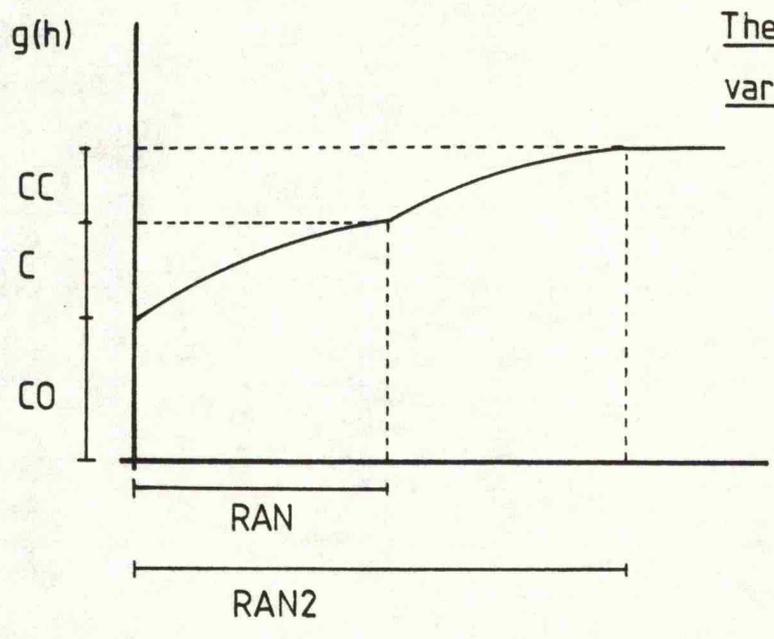
The semi-variogram

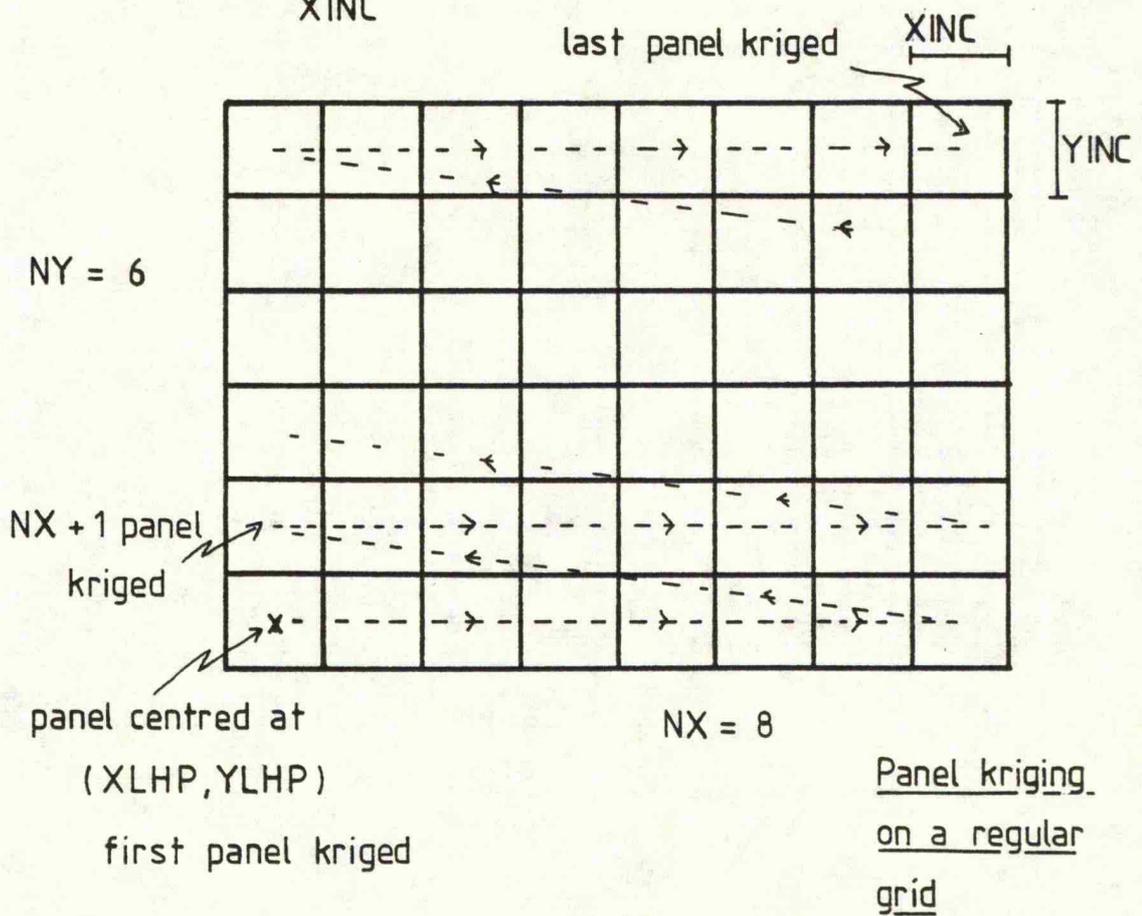
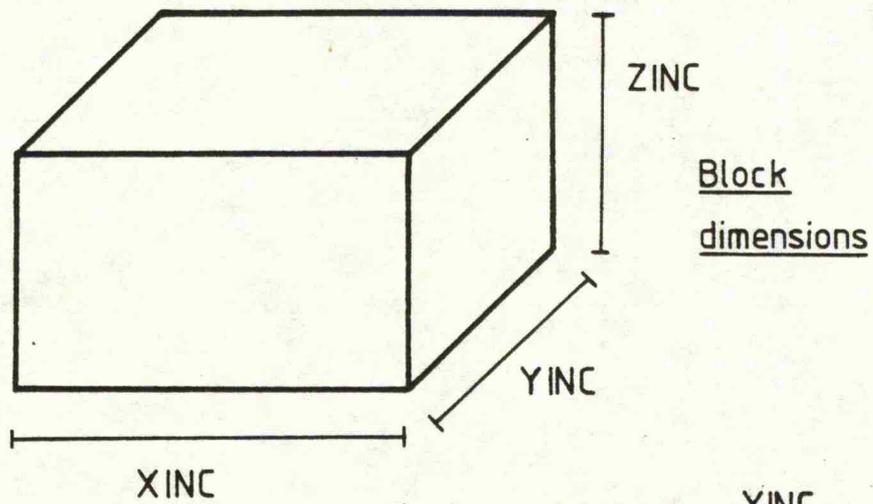
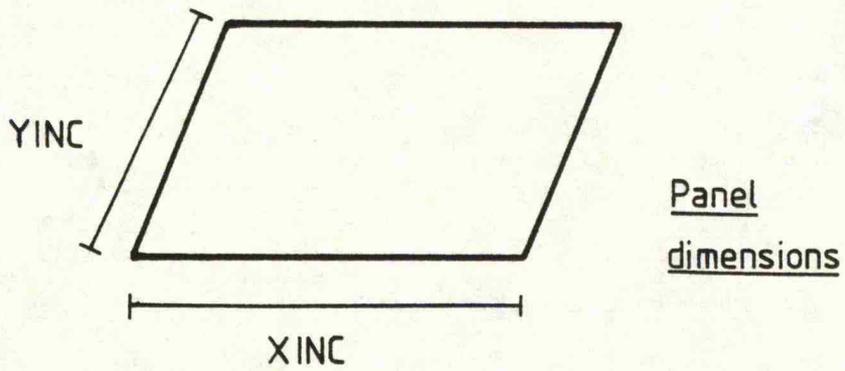


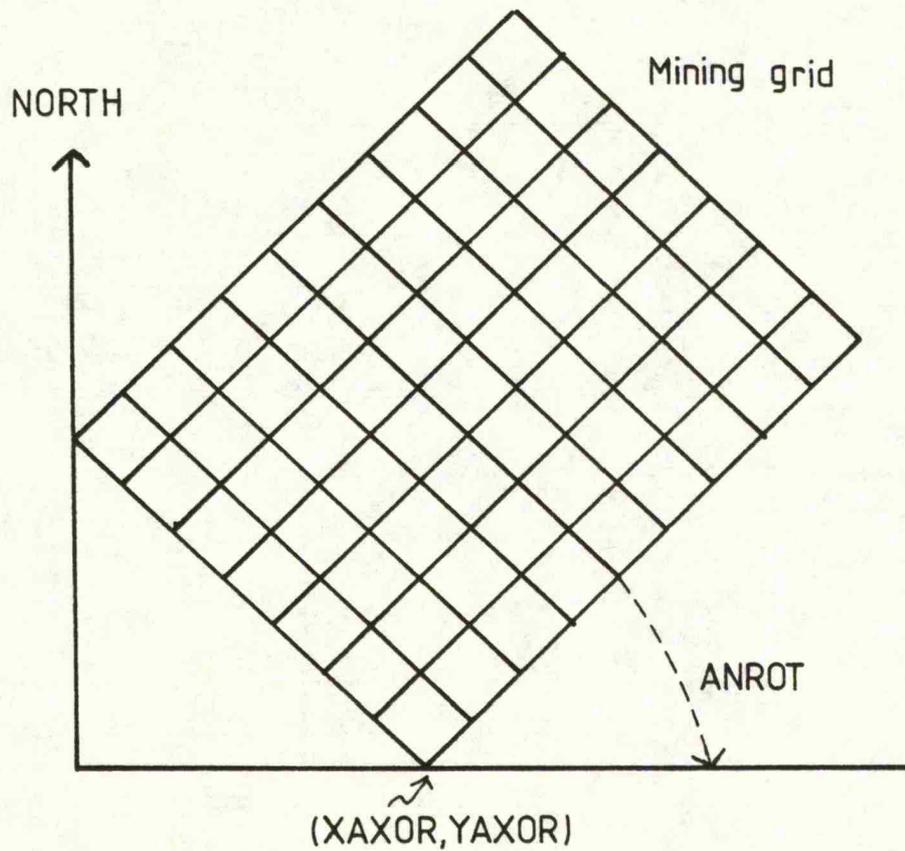
The variogram



The double sill variogram

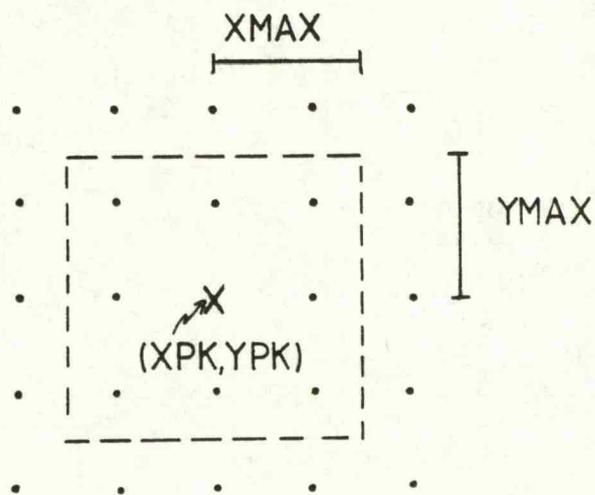






Rotation of a
co-ordinate system

Radii of sample search



point values to be used
lie within square

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