A Thesis Entitled

ASPECTS OF SPATIAL DATA ANALYSIS
IN EXPLORATION AND EVALUATION
OF MINERAL RESOURCES

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Faculty of Science, University of Leicester
by
Alfonso E. Monge-Urena

Geology Department
University of Leicester

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ABSTRACT - The data collected in projects of exploration and evaluation of mineral resources is spatially dependent. It is in this context that its analysis should be carried out. The calculation of the Geostatistical semivariance function in as many directions and lags as the data allows permits to identify different degrees of data's internal organization by using iso-semivariance diagrams. Structures within the data are outlined by these diagrams together with their relative position, dimensions and orientations. The technique is arrived at through the development of several computer programs that calculate the semivariance function, transform co-ordinates and produce contourings. Trials on simulated and geochemical exploration data show the close relationship between data's structure and derived functions.

The diagrams are also used to assess goodness-of-fit in polynomial data modelling and to develop a computerized procedure for data filtering as predictive models of spatial variability. This procedure consists of the locally fitting of covariance-weighted regression models. Its applicability as a new method of ore reserve estimation is demonstrated with bore hole data from a porphyry copper and a stratabound base metal deposit. Limitations to the technique are imposed by inadequate sampling patterns or impossibility for defining realistic iso-semivariance diagrams. A practical application of alternative conventional Kriging methods in global evaluation of a Bauxite deposit is also presented.

From these results, the application of probability analysis in financial appraisal of mining ventures clearly defines the deposit as an important and feasible project.
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APPENDIX

Computer programs VAR,
   TRAPOCA, Contouring, LOCMODL,
   PREPROD, RANDEST, ESTDISP, JOLEVAL

Example of the local model program output for copper in Bench 11 West (Gortdrum deposit)

Isotropic variograms of Los Pijiguaos bauxite deposit

Example of the financial analysis program (joleval) output

References
1.1. Introduction

The study of geological environments which may contain economically exploitable mineral resources has traditionally been carried out by measuring variables that are, in one way or another, related to the presence of mineralization. These variables or features can be inherently geological, for example, the thickness of a particular bed, the size and shape of mineral grains, lithological and mineralogical constitutions, or they can be the result of interaction between the parent material and the environmental physico-chemical condition as in the case of secondary metal dispersion.

The behaviour of the variable within an area under study has the form of a defined function, with characteristic variation. The exact knowledge of this function over the entire area will only be known if we measure it at every single locality. This procedure is undoubtedly time consuming and therefore uneconomic. However, certain valid inferences about the total type of variation can be obtained by sampling the parent population of infinite observations at a measurable number of locations which are conveniently placed within the area. The data analysis of this "sub" population will enable us to obtain a better understanding of the variable's function and to objectively assess, during the prospecting stage, how promising the area is and, at the drilling stage its economic potential. Due to the rapid increase in the cost of each successive stage of the exploration programme, a detailed analysis of available data should be carried out in order to decide whether to continue the project to a further step or terminate it and to allocate the funds for exploration in another area. Both decisions are controlled by economics and are made also in the light of the results of data analysis. These results will have a representativity, or in other words
the extent to which they describe the actual natural phenomena, that is largely dependent upon the quality of the data. Hence the need to emphasize the consistency and uniformity when sampling the parental population.

This aspect of the quality of the data is frequently overlooked by the exploration groups and as a consequence, misleading results are more likely to occur from data analysis. Several methods for assessing the quality of the data have been developed in research carried out by Craven (1954), Gy (1968), Garret (1969), James (1970), Thompson and Howarth (1973), and Ingamells (1974). In general, these methods are based on sampling and subsampling, duplicate determinations and the study of theoretical mixtures with different compositions.

However, the purpose of this chapter is to review, though very briefly, the most commonly used univariate methods of analysis of data obtained from programmes of exploration and evaluation of mineral resources.

1.2. Data Distribution

When a large set of data is to be examined, unless the data is highly organized, the human brain is incapable of appreciating the subtleties of its structure. In order to ease this task, the first step is to divide the range of variation (the difference between the highest and the lowest values) into a series of class intervals and count the number of observations per class. The resulting graph (histogram) represents an "empirical frequency distribution". If a smooth curve is fitted to that distribution a new type of distribution is obtained, called the "theoretical frequency distribution" (Fig. 1.1.) which represents an infinite potential number of observations, in contrast to actual observations. The advantage of the theoretical distribution is that a mathematical model can be devised so that certain parameters which
quantitatively describe the empirical population can be calculated. Different types of frequencies can also be calculated. Thus, if the \% of samples from each class in relation to the total number of samples is calculated, a "relative frequency" is obtained. Also, if the number of samples per class are added, either from the highest value to the lowest, or vice versa, a "cumulative frequency" is obtained. The appropriate relative cumulative frequency is worked out in the very same manner as for the relative frequency. An example of these frequencies are illustrated in Table 1.1.

1.3. Parameters of the Distribution

Certain parameters can be calculated from the sampled population which indicate particular characteristics of the distribution. Thus, a measure of the central tendency is obtained by calculating the mean:

\[ m = \frac{\sum x_i}{n} \]  \hspace{1cm} (1.1)

where \( m \) is the population mean, \( \sum x \) is the sum of all observations and \( n \) the number of observations. The amount of spread or dispersion of values is represented by the variance:

\[ s^2 = \frac{\sum (x_i - m)^2}{n-1} \]  \hspace{1cm} (1.2)

and the standard deviation \( S \), which is the square root of the variance is an indicator of the deviation from the central tendency. The ratio of \( S \) to \( m \) gives the coefficient of variation which is a useful measure of relative variability of observations.

The mode is the value which occurs with the greatest frequency, i.e. the value which appears more often, and the median is the value midway in the frequency distribution. Only in a perfectly symmetrical type of distribution will the mean, mode and median have the same value. However, the mean tends to be closer to the parental population mean than either the mode or median as the number of observations increases.
Fig. 1.1. Example of empirical frequency distribution (Histogram) and theoretical frequency distribution (smooth curve fitted to Histogram, after Koch & Link, 1970).

<table>
<thead>
<tr>
<th>Assay interval midpoint, ( % P_2O_5 )</th>
<th>Frequency, ( f )</th>
<th>Relative frequency, ( % )</th>
<th>Cumulative frequency, ( c.f. )</th>
<th>Relative cumulative frequency, ( r.c.f. )</th>
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<tbody>
<tr>
<td>14-16</td>
<td>15</td>
<td>1</td>
<td>0.45</td>
<td>0.45</td>
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<tr>
<td>16-18</td>
<td>17</td>
<td>1</td>
<td>0.45</td>
<td>0.90</td>
</tr>
<tr>
<td>18-20</td>
<td>19</td>
<td>8</td>
<td>3.57</td>
<td>4.47</td>
</tr>
<tr>
<td>20-22</td>
<td>21</td>
<td>21</td>
<td>9.37</td>
<td>13.84</td>
</tr>
<tr>
<td>22-24</td>
<td>23</td>
<td>44</td>
<td>19.64</td>
<td>33.48</td>
</tr>
<tr>
<td>24-26</td>
<td>25</td>
<td>54</td>
<td>24.12</td>
<td>57.60</td>
</tr>
<tr>
<td>26-28</td>
<td>27</td>
<td>56</td>
<td>25.00</td>
<td>82.00</td>
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<td>28-30</td>
<td>29</td>
<td>30</td>
<td>13.39</td>
<td>95.99</td>
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<td>30-32</td>
<td>31</td>
<td>7</td>
<td>3.12</td>
<td>99.11</td>
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<tr>
<td>32-34</td>
<td>33</td>
<td>2</td>
<td>0.89</td>
<td>100.00</td>
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Table 1.1. Example of different types of frequency for 224 phosphate assays (after Koch & Link, 1970).
Hence the mean is the best unbiased estimator of the parental population. If the distribution is either negatively or positively skewed (Fig. 1.2) these three parameters are no longer coincident and the degree of skewness is given by:

\[ w = \frac{m - \text{mode}}{s} \quad (1.3) \]

1.4. Types of Distributions

It is the common practice when sampling a parental population to ensure that the samples are drawn in a random manner. This means that each item has an equal opportunity to be included in the empirical population (also known as the sample population). We aim to obtain a data set which is unbiased. However, if observations with certain characteristics are systematically excluded from the sample population, either deliberately or inadvertently, the data set is said to be biased. The chance of a particular value occurring can be estimated using probability theory, and the different types of theoretical frequency distributions can be studied within this context.

1.4.1. Gaussian or normal Distribution

This has a symmetrical bell-shape and it is mathematically expressed by:

\[ f(x) = \frac{1}{s \sqrt{2\pi}} \exp \left\{ -\frac{(x-m)^2}{2s^2} \right\} \quad (1.4) \]

where \( m \) and \( s \) are the mean and the standard deviation. This distribution can be standardised by expressing \( (x-m)/s \) in \( Z \):

\[ f(Z) = \frac{1}{\sqrt{2\pi}} \exp \left\{ -\frac{Z^2}{2} \right\} \quad (1.5) \]

this standardised normal distribution has a mean of zero and standard deviation of unity (Fig. 1.3)

The cumulative frequency distribution of a normal population has the equation:

\[ F(x) = \frac{1}{s \sqrt{2\pi}} \int_{-\infty}^{x} \exp \left\{ -\frac{(x-m)^2}{2s^2} \right\} \, dx \quad (1.6) \]
Fig. 1.2. Relation between measures of central tendency in an asymmetric (skewed) frequency distribution (after Davis, 1973).

Fig. 1.3. The standardised normal distribution and its properties (after Till, 1974).
for the standardised cumulative frequency distribution.

A most useful property of the standardised normal distribution is that the area under the curve, within any specified range, can be calculated. For example, between \( m + 1S \) we find an area of 0.6827 and 95% of the curve lies between \( m +/- 1.96S \) which correspond to the 95% confidence limit. This means that from the theory of probability there is a chance of 1 in 20 that the observation lies outside that range, provided the observations are taken at random. As the number of such observations increases the distribution tends more and more towards a normal frequency distribution despite the arbitrary shape of the parental populations' theoretical distribution. This last statement is known as the central limit theorem. However, no guidance is given as to how many observations are required for its distribution to be considered nearly normal in shape. It is accepted (Koch and Link, 1970) in geological distributions to be between 50 and 100 observations.

1.4.2 Log normal distribution

There are certain data sets, mainly the content of trace elements in soils, stream sediments and rocks, which present log normal distribution. In this case a strongly skewed frequency distribution is converted to a symmetrical one by plotting the logarithms of the values. Ahrens (1954) was amongst the first to notice this feature. The mathematical equation that describes the log normal distribution is given by:

\[
f(x) = \frac{1}{xSg \sqrt{2\pi}} \exp \left[-\frac{1}{2} \left( \frac{\ln x - mg}{Sg} \right)^2 \right] \tag{1.8}
\]

where \( mg \) = mean value of \( \ln x \)'s (natural)

\( Sg \) = standard deviation of \( \ln x \)'s

The mean and standard deviation of the actual values can be calculated with the following equation:
Several examples of log normal distribution appear in the available literature on the subject:

Ore assays and element concentration in mineral deposits (Krige, 1951), size of oil and gas fields (Kaufman, 1963), Trace element concentration in rocks (Shaw and Bankier, 1954).

When dealing with log normal distributions the tendency is to either use the set of observations \((x)\) and employ the log normal distribution formula or transform to logarithms and use the normal distribution equation. The latter has the advantage of relating probabilities to transformed values and also, many statistical tests based on the normal distribution can then be used.

1.4.3. The Binomial Distribution

This type of distribution is also known with the name of Bernoulli's as it was discovered by James Bernoulli at the end of the 17th Century. It is better understood with the example:

If \(P\) is the probability of one event to happen on one trial (probability of success) and \(Q=1-P\) the probability that the event does not occur (probability of failure), then the probability that such event occurring exactly \(x\) times in \(n\) trials is given by:

\[
P(x) = \frac{n!}{x!(n-x)!} P^x Q^{n-x}
\]

where \(x = 0,1,2,\ldots, n\) and

\[ n! = n(n-1)(n-2)\ldots 1 \quad 0! = 1 \]

The parameters that describe the binomial distribution are as follows:

\[
m = nP
\]

\[
s^2 = nPQ
\]

Coefficient of variation = \[
\frac{\sqrt{Q-P}}{\sqrt{nPQ}}
\]
It is generally agreed that the binomial distribution does not normally occur in geological environments, but other distributions can be derived from it. Indeed, when \( n \) becomes very large, the binomial distribution approximates the normal type. In geological literature there are some examples of binomial distributions in grain to grain contacts in thin section (Khan, 1956), Petrographical Modal Analysis (Chayes, 1956) and Hydrology (Yevjevich, 1971). Agterberg (1974, Page 158), gives a practical example from a vein type mineral deposit.

1.4.4 The Poisson Distribution

Discovered by Poisson, this type of distribution is described by the following probability equation:

\[
P(x) = \frac{\lambda^x e^{-\lambda}}{x!}
\]

where \( x = 1, 2, ..., n \) (number of events)

\( e = 2.71828 \) and \( \lambda \) is a given constant that can be found in specific tables (Spiegel, 1961).

The parameters of the distribution are:

\[
m = \lambda
\]

\[
S^2 = \lambda
\]

Coefficient of variation = \( \frac{1}{\sqrt{\lambda}} \)

Till (1971, page 45) gives an example of Poisson distribution by studying the radioactive element decay when emitting particles (\( \alpha \)). Sichel, (1972) in the study of diamonds deposits from ancient beaches in south west Africa, assumes a Poisson distribution for the densities of occurrences of diamonds.

Three main assumptions are made in deriving the Poisson distribution:

1. The events occur independently

2. The probability of one event occurring is proportional to the length of time since the last event and
1.5. Analysis of Distributions

Many of the empirical frequency distributions that are found within geological environments can be identified with the theoretical distributions briefly described above and the population parameters can be obtained from the appropriate formulae. Several systems of frequency distribution functions have been developed in order to explain empirical distributions which cannot be associated with one of the basic theoretical types. In particular, it is accepted (Agterberg, 1974) that the systems developed by Pearson (1895), Kaptein (1903), Burr (1942) and Johnson (1949) are flexible enough to cover most of the geological empirical distributions. On the other hand, numerical and graphical techniques have been developed to test whether or not an empirical distribution is derived from a parental population of normal character. These tests (f-test, t-test and chi-square) are not described here but can be found in any textbook on elementary statistics. A most widely used test for normality in analysis of geological data is the 95% confidence belt applied by Vistelius (1960) and detailed accounted by Agterberg (1974). The method is based on plotting the cumulative frequency distribution in percentage on a normal probability paper as shown in Fig. 1.4. In that figure, the vertical axis is of an arithmetic scale, and the horizontal scale is devised such that the S-shaped curve of the cumulative frequency curve adopts a straight line. Any truly normal distribution will plot as a straight line on probability paper. The use of this kind of plot in examining geochemical exploration data is widespread and important work of this type has been done by Tennant and White (1959), Williams (1967), Lepeltier (1969), Bolviken (1971), Woodsworth (1972), Sinclair (1974) and Parslow (1974) amongst others.

To construct the 95% confidence belt it is assumed that the calculated mean of the empirical distribution is equal to the mean of the theoretical
one. This corresponds to a 50% cumulative frequency and the calculated standard deviation to 15.9%. The theoretical distribution is then obtained by joining both points. If \( Z_T \) represents the standardised value for a point in the straight line for the theoretical normal distribution and \( Z \) the observed value, the 95% confidence belt satisfies:

\[
Z_T +/- 1.96S(Z)
\]

where \( S(Z) \) is the square root of the following expression:

\[
S^2(Z) = \frac{1}{n} \frac{F(Z_T) \cdot F(-Z_T)}{f(Z_T)}
\]

where \( F(Z_T) \) and \( f(Z_T) \) are functions defined for the standardised cumulative and frequency curves.

The advantage of plotting distributions on a normal or log (if the data is log-normally distributed) probability paper, is that not only can the test of normality be used but also it helps to detect the presence of more than one intermixed distribution as we will discuss below.

1.5.1. Populations

The types of distribution reviewed so far in this chapter are of unimodal character. In other words, they represent, in the general case, a single population. Hence, the value of two standard deviations, in a normal or log normal distribution, suggested by Hawkes and Webb (1962) and used by Lepeltier (1969) and others as the threshold value in defining anomalous values in geochemical exploration data, is valid though arbitrary. However, there are no logical criteria to dismiss the assumption that values above the two standard deviations are anomalous and perhaps related to mineralization.

Nevertheless, the most common situation when dealing with trace element or ore assay data is that the distribution presents more than one population, i.e. it is a polymodal distribution. This appears somewhat obvious since geological environments are in general inhomogeneous.

Physico-chemical conditions, acting at different places and time,
Fig. 1.4. Graphical test of normality for 76 biotite ages from Greenville province, Canadian Shield. Observed frequencies (solid dots) are not all contained within 95% confidence belt on the theoretical normal curve (straight line), indicating departure from normality which is strongest in the upper tail (after Agterberg, 1974).

Fig. 1.5. Frequency histogram and corresponding cumulative frequency plots for polymodal populations (after Hawkes & Webb, 1962).
determine the final characteristics observed to-day. Therefore, the parameters of a single population no longer apply as estimators and several techniques of examining and separating mixtures of various frequency distributions have been devised.

1.5.2. Splitting of Populations

The presence of more than one population can sometimes be easily detected by examining the histogram, but when the degree of overlapping is high such a detection becomes somewhat ambiguous. However, the cumulative probability plot will show a deviation from a straight line as soon as the influence of one population on the other is accentuated. This change of slope (Fig. 1.5) becomes less noticeable as the amount of overlapping increases.

A straightforward procedure of separating the two distributions can be performed by selecting the midway point in the curve section of the least slope. If the breaking point has been selected adequately the two populations will plot on a straight line when recalculated and replotted separately.

Sinclair (1974) accumulates values from the highest to the lowest in order to plot the highest value (Fig. 1.6) and arbitrarily chooses the 99 and 1 cumulative percentiles of A and B populations respectively. However, he claims that in most real cases the 98 and 2 percentiles have proved most useful. In addition, he gives the following relationship to check the partitioning procedure:

\[ PM = FAPA + FBPB \]  
(1.14)

where PM is the probability of the mixture, PA and PB are the cumulative probability of population A and B read from the graph at a specified ordinate level; PA is the proportion of population A and PB = 1-FA.

James (pers.comm) also selects an initial breaking point at the zone of the shallowest slope but the definite partitioning is chosen in such a way that the number of values of one population that are included
Fig. 1.6. Cumulative probability plot of two hypothetical populations A and B combined in the proportion A/B = 20/80. The inflection point is shown by the arrowhead and the thresholds have been arbitrarily chosen at the 1% of B population and the 99% level of A population, which correspond to 78 and 44 ppm respectively (after Sinclair, 1974).
in the other (misclassified samples) are in inverse proportion in
relation to the total number of observations of each population. This
criterion seems to be more logical since the relative importance of each
set of misclassified samples should be based on the size of the population
from which they are derived.

1.6 Relationship between data distribution and spatial distribution

The techniques of examining a data set by looking at the frequency
distribution and its subsequent cumulative probability plot do not give
us information about the spatial interrelationship between the high,
medium and low values of a unimodal distribution, or among successive
populations of polymodal distributions. Theoretically, from the mineral
exploration point of view, observations of a similar range are expected
to appear somewhat spatially correlated. This is usually the case when
dealing with an empirical distribution derived from different litologies
or from areas containing mineralization with suitably developed primary
and/or secondary dispersion.

However, the plotting of a character or colour at the location where
the observation has been taken, which represents a population from a
polymodal distribution, might not depict clearly enough the spatial
interrelation of measurements. This situation might arise due to the
introduction of errors (inadvertedly or not), referred to as bias, during
sampling and or analytical procedures, problems of overlapping
(misclassification of samples) or due to the nature of the mineralization
phenomena itself, which can give rise to very complicated and noisy
functions with a large degree of randomness. Hence the need to perform
sometimes transformations of the raw or empirical function in order to
obtain a simpler underlying function with a rate of variation that can be
more easily recognised and predicted.

1.7 Data transformation
There are basically two types of transformations that can be applied to a data set. The first considers the location of values within the sampled area and the second assumes the data is spatially independent. As examples of the second type of transformation we have logarithmic, square root and addition or subtraction of a particular constant, for example, the gold content in South African Gold Mines (Krige, 1960).

Geological variables become meaningful when they are studied in relation to the location of their measurements, as it will be demonstrated later with a numerical example. Thus the data transformations which consider spatial distribution are more relevant. Hence, the remaining part of this chapter will be allocated to reviewing the most commonly used univariate methods of spatial data analysis.

Moving average techniques are perhaps the most widely used for data filtering due to their simplicity and speed when a computer is available. A cell of a particular size and shape, usually a circle, is moved over the sampled area with fixed displacements. The values enclosed by the cell at any particular time are averaged and the resultant value is allocated to the centre of the cell. If the procedure is done with a computer, a particular character, chosen specifically for a range of values, will be printed at the centre of the cell providing that the average value calculated belongs to that range (grey scale mapping, Howarth, 1971). The way in which the new value is calculated depends upon the degree of smoothing desired. With an increasing severity of smoothing, the average can be (a) ordinary, i.e. arithmetic mean, (b) linear weighted if values are multiplied by a factor of 1 to 0 depending upon the distance to cell centre, (c) exponentially weighted if a particular exponential function is selected.

Lately, trials with a moving mode (Henley, S., pers.comm), calculated from a theoretical distribution generated by the values enclosed in the cell, have proved to be a very satisfactory approach.
There is a fundamental problem when using smoothing techniques which consists of selecting a cell size and type of weighting such that the noise is removed but not the features of the function that might have a direct relationship with the real phenomena. Certain suggestions to overcome this problem are given in Chapter 3.

These generalized forms of moving averages are part of a larger set of gridding techniques. Gridding is a procedure of interpolating randomly spaced two-dimensional data on a regular grid which can then be more easily used for filtering, contouring by hand or automatically and for comparison with another data set. The gridding techniques that have been described in the literature are based upon methods of interpolation at grid locations by using surrounding irregularly spaced data points. Normally, the interpolation procedure is carried out at each grid location using some form of weighted moving average or a linear or quadratic fit to the surrounding data. Weighting is usually selected as the inverse of the distance from the grid point. For instance, Davis (1973, p. 310-322) presents a program for a simple averaging method in which the value at any grid location is given by weighting n points, selected by the user, according to the inverse of the distance to the grid point. This weighting procedure can be combined with different surface fitting to data points in the region of the grid location as it has been done by Palmer (1969) and Walters (1969) who used a linear surface fitting. Also a quadratic fitting within a specified radius of the grid location has been programmed by McIntyre, Polar and Smith (1968) and McLain (1974).

In gridding geophysical data more sophisticated techniques have been employed such as Fourier series (James, 1966), the use of a minimum curvature of the surface as an additional constraint in the fitting procedure (Briggs, 1974) or thereafter, the use of difference equations for minimum curvature (Swain, 1976).

In general, the common problems in gridding methods for geological and geophysical data are: (1) finding an acceptable degree of accuracy
in zones where the data appears sparsely distributed, (2) representing short wavelength variation without including erratic features and (3) avoiding sampled areas' edge effect when interpolating.

An interesting comparison of four random to grid methods is presented by Braile (1978) on an aeromagnetic data set of 200 observations. These methods are graphically shown in Fig. 1.7. In method II the value at the grid point is determined by the weighted average of closest random sample points. At each grid location the random data must be searched to locate the n closest points, where n is any number but is usually chosen to be between 3 and 9. The values of the n closest points are weighted by the inverse of the distance between the grid point and the sample points and then averaged.

Method I is similar to method II in that it estimates grid points values by averaging the closest three points. In the method III, an n\textsuperscript{th} order polynomial surface is fitted to small regions of the sampled area and calculating the surface values at grid locations from the polynomial surface. The gridding area is defined by a small number of grid points in both east and north directions and a surface fitting area is delineated surrounding the gridding area. The sampled data are searched to locate cell points within the surface fitting area and these points are used to calculate the polynomial. The gridding area is then moved successively over the entire grid and the local polynomial fitting is repeated until the gridding is complete. The size of the two rectangular areas is fixed by the user as well as the number of points required to select a particular order of the polynomial, which is usually chosen from 3 to 6.

Method IV requires the user to arrange the sampled data into profiles which need not be straight, nor have even spacing of the sample points. For each grid location the two profiles on each side of the grid point are located and four surrounding points on each profile are found. A cubic (3\textsuperscript{rd} order) polynomial is fitted to each of the four-points sections of the profiles and interpolated values are determined at the four points.
Fig. 1.7. Schematic diagram illustrating four random to grid methods of interpolation. See text for description of each method (after Braile, 1978).

<table>
<thead>
<tr>
<th>Method</th>
<th>Computer time (sec)</th>
<th>RMS Error (gammas)</th>
</tr>
</thead>
<tbody>
<tr>
<td>I. Surrounding triangle</td>
<td>14.0</td>
<td>28.5</td>
</tr>
<tr>
<td>II. Weighted average of closest point</td>
<td>13.8</td>
<td>31.3</td>
</tr>
<tr>
<td>III. Local surface fitting</td>
<td>10.4</td>
<td>21.2</td>
</tr>
<tr>
<td>IV. Intersecting cubic polynomials fit to profiles</td>
<td>2.7</td>
<td>29.3</td>
</tr>
</tbody>
</table>

Table 1.2. Comparison of computer time and relative error in the four gridding methods of figure 1.7 above (after Braile, 1978).
of intersection of a line through the grid point perpendicular to the profile direction and the four profiles. The four intersection points are then fitted by a cubic polynomial and the value at the grid point interpolated. Extrapolation of data is carried out at the edges of the sampled area in order to determine the intersection points.

Accuracy of gridding procedure can be estimated by calculating the differences between the interpolated surface and the original one at the sample points. Davis (1975), using this estimation, demonstrated that a local gridding method such as I is more accurate than procedures employing a larger number of data points. However, Braile, (op.cit.) considers that accuracy should be determined at gridded values for they are more likely to be used in subsequent analysis. Using this criterion he proves that the local surface fitting is more efficient in terms of computer time and gridding accuracy than the other 3 methods considered (Table 1.2). Nevertheless, the performance of random to grid methods is dependent on the complexity of the observed data, the locations of sample points and the characteristic of the gridding technique.

As it has been shown, most gridding procedures use a local strategy in which the value at a grid point is determined from samples restricted to some small distance from a grid location as opposed to global techniques such as polynomial trend surfaces analysis which uses all the data points for the interpolation of the value at each grid location.

A different approach to interpolation appeared in the early 1960's as a consequence of the development of Geostatistics and its Kriging techniques. This procedure together with the global polynomial trend surface will be reviewed separately due to their importance and large controversy they created amongst earth scientists.

1.8. Polynomial data modelling

Deterministic functions such as polynomials have been largely used to deal with the problems of describing "large scale" systematic changes
(trend) and local, apparently non-systematic, fluctuations (residuals) that are superimposed in the large scale patterns. Thus, a geological trend is a systematic change, usually in a geographical direction in the value of a geological variable. These trends are studied using the method of multiple regression, and as observations are areally distributed, usually on a two-dimensional plane (map area): the technique is called trend surface analysis. It was first introduced in Geology by Grant (1957) and extensively developed by Krumbein (1956 and 1959) and programmed by Whitten (1963), Peikert (1963), Esler et al. (1968) and Davis (1973) amongst others.

In polynomial data modelling it is assumed that the observed surface is the combination of a trend plus a variation surface. If the trend varies smoothly over the space, its value (height) at any particular point can be expressed in terms of the spatial co-ordinate of that point. Thus, the basic equation of any trend becomes:

$$Z_{obsi} = f(x_i,y_i) + D_i$$  \hspace{1cm} (1.15)

where $Z_{obsi}$ = the observed value of the surface at $i$th point

$x_i$ = east co-ordinate of the $i$th data point

$y_i$ = north co-ordinate of the $i$th point

$D_i$ = deviation (residual) at the $i$th data point

$f(x_i,y_i)$ is a function of the co-ordinates which mean that if the location of any point $i$ is known, as a pair of spatial co-ordinates, then the height of the surface at that point is found by substituting its co-ordinates into the equation or function. The simplest form of this function is the linear type, which is expressed as a plane:

$$Z_{obsi} = (C_0 + C_1 x_i + C_2 y_i) + D_i$$  \hspace{1cm} (1.16)

where $C_0$ = height of the plane at the map origin, $x=y=0$.

$C_1$ = plane's rate of change along east direction

$C_2$ = plane's rate of change along north direction.

Each combination of $C_0$, $C_1$ and $C_2$ will represent a plane of different inclination. These coefficients are usually chosen according to the "least
squares criterion", which states that, in combination, they define a
surface with the best fit to the original data, i.e. the sum of all the
residual values squared is as low as it possibly can be for that surface
shape. So, the previous equation can be rewritten:

\[ S = \sum_{i=1}^{n} (Z_{obsi}-(C_0+C_1x_i+C_2y_i))^2 = \sum_{i=1}^{n} D_i^2 \]  

(1.17)

where \( n \) is the total number of data points available.

Minimization of \( S \), from the theory of calculus in partial differential
equations, is only obtained when:

\[ \frac{\partial S}{\partial C_0} = \frac{\partial S}{\partial C_1} = \frac{\partial S}{\partial C_2} = 0 \]  

(1.18)

where:

\[ \sum_{i=1}^{n} (Z_{obsi}-(C_0+C_1x_i+C_2y_i) \cdot (-1)) = 0 \]

\[ \sum_{i=1}^{n} (Z_{obsi}-(C_0+C_1x_i+C_2y_i) \cdot (-x_i)) = 0 \]  

(1.19)

\[ \sum_{i=1}^{n} (Z_{obsi}-(C_0+C_1x_i+C_2y_i) \cdot (-y_i)) = 0 \]

If each of these equations are divided by 2, the (-1), (-x_i) and (-y_i)
multiplied out and re-arranged, the following system of linear equations
are obtained:

\[ C_0 x_i + C_1 x_i^2 + C_2 x_i y_i = Z_{obsi} \]

\[ C_0 y_i + C_1 x_i y_i + C_2 y_i^2 = Z_{obsi} \cdot x_i \]  

(1.20)

These three normal equations are then solved to obtain the best fit
values for \( C_0, C_1 \) and \( C_2 \), by matrix formulation:

\[
\begin{bmatrix}
\xi_i & \xi y_i \\
\xi_i x_i & \xi x_i y_i \\
\xi y_i & \xi y_i y_i 
\end{bmatrix}
\begin{bmatrix}
C_0 \\
C_1 \\
C_2
\end{bmatrix}
= \begin{bmatrix}
Z_{obsi} \\
Z_{obsi} x_i \\
Z_{obsi} y_i
\end{bmatrix}
\]

(1.21)

There are many mathematical methods of solving this linear system of
equations but perhaps the most widely used in computer trend surface programs is the inversion and simple Gaussian elimination (Esler et al, 1968, Davis, 1973).

The linear equation of a trend surface can be expanded to a quadratic form:

\[ Z_{ti} = C_0 + C_1 x_i + C_2 y_i + C_3 x_i^2 + C_4 x_i y_i + C_5 y_i^2 \]  

(1.22)

or up to a 3rd order by adding the following term to the previous equation:

\[ \ldots + C_6 x_i^3 + C_7 x_i^2 y_i + C_8 x_i y_i^2 + C_9 y_i^3 \]  

(1.23)

The number of coefficients will increase with the order of the polynomial and up to 36 coefficients describe a 7th order surface. However, in Geology it is difficult to imagine a surface of higher order than the 3rd. Some graphical examples of these type of surfaces are given in Fig. 1.8.

There are several parameters that can be calculated and help to assess how good a polynomial models the data. Thus, the percentage of goodness-of-fit is given by:

\[ 100 \% R^2 = \frac{SST}{SSO} \]  

(1.24)

where \( SST \) = sum of squares due to the trend or regression \\
\( SSO \) = sum of the squares of the original data

and the coefficient of multiple correlation is:

\[ R = \sqrt{R^2} \]  

(1.25)

Values of \( R^2 \) and \( R \) above 50\% normally represent acceptable goodness of fit.

The F-ratio can also be calculated as follows:

\[ F = \frac{\% R^2 / df_1}{(100-\% R^2) / df_2} \]  

(1.26)

where \( df_1 \) = degrees of freedom associated with the surface, equal to the number of constants in the trend equation less one for the base term \( C_0 \)

\( df_2 \) = degrees of freedom associated with the residuals, this is given by the total degrees of freedom in the data, less \( df_1 \), those associated with the trend. As usually the
Fig. 1.8. Trend surfaces of one, two and three independent variables for polynomial equations of the first, second and third orders (after Harbaugh, 1964).
total degrees of freedom is \( n \) (Number of observations) less 1, so \( \text{df2} = n-1-\text{df1} \)

Values of the F-ratio for the same degrees of freedom are compared with those of the statistical tables (Davis, 1973, pages 102-104). If the polynomial shows a good fit, the experimental F-ratio should be higher than the theoretical ones. In addition, according to Unwin (1975) a trend surface is representative of the data structure and becomes the best linear unbiased estimator if the residuals have a normal distribution with a mean value of zero, showing no autocorrelation and there are as many data points as coefficients of trend equation plus one.

The application of polynomial data modelling within the mining environment has been well documented especially by Link et al, (1964), the evaluation of the Whalsback Mine in Newfoundland (Agterber, 1968), and the integrated geological-polynomial modelling at Scully Mine, Canada (O'Leary, 1979). The technique has also been successfully applied as an aid in the exploration for Mississippi Valley-type deposits (Trapp et al, 1977) where certain similarities appear to be structurally controlled.

1.9. Geostatistical Data Analysis

The initial steps of the geostatistical approach to the study and estimation of spatial variability started (De Wijs, 1953) when the sample variability was considered by means of the position of the values in relation to each other instead of by means of the standard deviation in the classical manner of comparing each value with the population mean. For example, if we consider five assays of numerical value 1,2,3,4,5 which have been taken at regular intervals in a stope, the mean and standard deviation are:

\[
\begin{align*}
m &= 3.00 \\
S &= 1.58
\end{align*}
\]

We can also compute a type of spatial variation (\( S_4 \)) by comparing each
sample to its neighbour, and measure variation according to sample distribution:

\[ S = \sqrt{\frac{\sum (x_{i-1} - x_i)^2}{2(n-1)}} \]  

(1.27)

If the samples are arranged in a regular order:

1, 2, 3, 4, 5

\[ S = 0.71 \]

But if they are in the conspicuously irregular order:

3-2-5-1-4

\[ S = 2.09 \]

In fact, \( S \) has a range of 0.71 to 2.09 because there are 5! possible arrangements. Therefore, it is obvious that the classical standard deviation over or underestimates (depending on the degree of data organization) the variability of a spatially distributed variable and it is no longer an adequate measure of data variability.

Matheron (1962 and 1971), following the extensive work of Krige concerning the optimal assigning of weights to sample values to be used in estimating the grade of blocks in South African Gold Mines, developed the theory and established the foundations of Geostatistics. He himself defined it as 'the application of the formalism of random functions to the reconnaissance and estimation of natural phenomena'. A natural phenomena is often characterized by the distribution in space of a variable or set of variables which are then called regionalized variables. Some examples of this type of variable are topographic and structural surfaces, water tables and the ore grade within an ore body.

Continuity of the spatial variation is studied in Geostatistics by calculating the semivariance function \( \gamma(h) \) which compares in turn each sample of a data set with the remaining samples at a constantly increasing distance:

\[ \gamma(h) = \frac{1}{2(L-h)} \sum_{0}^{L-h} (f(x+h) - f(x))^2 \]  

(1.28)
where \( f(x) \) and \( f(x+h) \) are assays separated by a distance \( h \), known as the lag, and \( L \) is the length of a linear series of assays. The function \( \xi(h) \) yields therefore a relationship between sample spacing and variability. It is also a symmetric function which means that \( \xi(h) = \xi(-h) \) and non-negative.

The semivariogram, which is the fundamental tool in Geostatistics, is obtained when \( \xi(h) \) is plotted for different values of \( h \). There are five basic types of semivariograms (Fig.1.9) as mathematical models:

1. Continuous or Gaussian. Excellent continuity is observed which is rarely found in geological environments.

\[
\xi(h) = C(1-e^{-a^2})^{h^2} \tag{1.29}
\]

where \( C \) is the sill, i.e. maximum stable value of \( \xi(h) \).

2. Linear. It shows a moderate continuity, observed sometimes on base metal deposits.

\[
\xi(h) = a.h + b \tag{1.30}
\]

where \( a \) and \( b \) are constants.

If the scale in \( h \) is logarithmic the model is called de Wijsian type:

\[
\xi(h) = 3 \alpha \log e h + b \tag{1.31}
\]

\( b \) is a constant, function of the sample size. Also the following variance/volume relationship is observed:

\[
S^2(V/V) = \alpha \log(V/v) \text{ de Wijs formula} \tag{1.32}
\]

In the case of gold mines this formula is reduced to:

\[
S^2(B/A) = \alpha \log(A/B) \tag{1.33}
\]

where \( A \) and \( B \) are areas and \( \alpha \) is called the intrinsic dispersion.

3. Transitive or spherical without nugget effect. Continuity tends to disappear at certain distance, \( a \), called the range, which is interpreted as the zone of internal correlation and the area of influence of sample values. This type of semivariogram is common in sedimentary iron and uranium deposits. The spatial component of the regionalized variable is described by the behaviour of \( \xi(h) \) over the
range.

\[ \gamma(h) = C\left(\frac{3}{2} h - \frac{h^3}{2a}\right) \text{ for } h < a \]  \hspace{1cm} (1.34)

\[ \gamma(h) = C \text{ for } h \geq a \]  \hspace{1cm} (1.35)

4. Transitive or spherical with nugget effect. The presence of certain variance (nugget effect or nugget variance, expressed as CO) at very small distances contain the random component of the regionalized variable and it is related to the scale of observation (Serra, 1967) and measurement errors (sampling and analytical errors). This type of semivariogram is usually present in gold minerlizations and alluvial deposits.

\[ \gamma(h) = CO + C\left(\frac{3}{2} h - \frac{h^3}{2a}\right) \text{ for } h < a \]  \hspace{1cm} (1.36)

\[ \gamma(h) = C \text{ for } h \geq a \]  \hspace{1cm} (1.37)

5. Random. No continuity is observed which means that a high degree of random of the variable distribution exists and \( \gamma(h) \) is then equal to the statistical variance (\( S^2 \)).

The covariance function \( C(h) \) can also be calculated with the following equation:

\[ C(h) = \frac{\sum((f(x)-m)(f(x+h)-m))}{N} \]  \hspace{1cm} (1.38)

but it has the disadvantage in respect to \( \gamma(h) \) in that it requires an estimate of the mean value of some set of samples. However, the covariance can be obtained directly from the experimental semivariogram providing it is of a spherical type:

\[ C(h) = C(\infty) - \gamma(h) \]

In practical terms the behaviour of \( \gamma(h) \) and \( C(h) \) intuitively illustrates that the greater the distance between samples, then, on average, the greater would be the expected differences between their assays values. Therefore, these functions become a valuable tool to estimate the value of the regionalized variable at one particular point by placing a certain degree of importance to the neighbouring data points.
Fig. 1.9: Basic types of semivariograms.
Fig. 1.10. Zones of internal correlation as examples of geometric isotropy and anisotropy.
Experimental semivariograms usually differ markedly from the mathematical models described above and in practice, in the general case of ore reserve estimation, a model of the spherical type is fitted to the experimental semivariograms. These are usually calculated in 4 directions in order to detect the presence of any geometric anisotropy (Fig. 1.10).

If no geometric anisotropy exists, a mean isotropic semivariogram is assumed. The geostatistical procedure of estimating values of the regionalized variable using the information given by the semivariogram is the Kriging systems which have been developed mainly for the case of stationarity, i.e. the spherical scheme semivariogram where the \( \gamma(h) \) function becomes stable after certain lag, in the case of non-stationarity the semivariogram depicts the presence of a trend or drift. The case of stationarity will be considered first. In general terms, the Kriging system finds the estimated value \( Z^* \) of an actual value \( Z \) by using a linear combination of certain coefficients \( \lambda \) and the \( N \) surrounding values, i.e.:

\[
Z^* = \sum_{i=1}^{N} \lambda_i Z_i
\]  

(1.39)

where \( \sum_{i=1}^{N} \lambda_i = 1 \)

The Kriging coefficients are chosen such that:

1. The estimation is unbiased. This means that, on average, the error will be zero for a large number of such estimations, i.e.

\[
(Z - Z^*) = 0
\]

2. The variance of estimation \( S_e^2 = (Z-Z^*)^2 \) is minimal. To minimize this function a Lagrange transformation \( u \) is used and the Kriging system is expressed in matrix form as follows:

Suppose an estimate of the point \( Z_5 \) is required using the \( Z_1, Z_2, Z_3 \) and \( Z_4 \) surrounding points:

\[
\begin{align*}
Z_1 & \quad Z_2 \\
Z_3 & \quad Z_5 \\
Z_4 &
\end{align*}
\]
Then, if the spherical semivariogram has been calculated, the point
or specimen Krigin system is:

\[
\begin{bmatrix}
\gamma_1 & \gamma_2 & \gamma_3 & \gamma_4 & 1 \\
\gamma_5 & \gamma_6 & \gamma_7 & \gamma_8 & 1 \\
\gamma_9 & \gamma_{10} & \gamma_{11} & \gamma_{12} & 1 \\
1 & 1 & 1 & 1 & 0
\end{bmatrix}
\begin{bmatrix}
\lambda_1 \\
\lambda_2 \\
\lambda_3 \\
\lambda_4 \\
-u
\end{bmatrix} =
\begin{bmatrix}
\gamma_15 \\
\gamma_25 \\
\gamma_35 \\
\gamma_45 \\
1
\end{bmatrix}
\] (1.40)

where \( \gamma_{12} \) is the \( \gamma(h) \) value that corresponds to the distance between
points \( Z_1 \) and \( Z_2 \) and so on.

The Kriging variance is defined as:

\[
S^2_K = u + \sum \gamma_i \lambda_i + \sum \gamma_4 \lambda_i + \sum \gamma_3 \lambda_i + \sum \gamma_2 \lambda_i + \sum \gamma_1 \lambda_i
\] (1.41)

If the solution is unbiased:

\[
Z_5 - Z^*_5 = 0
\]

and the estimation variance is equal to the Kriging variance, i.e.

\[
S^2_K = S^2_E
\]

At 95% confidence level the estimated value will have a range of
variation equal to:

\[
Z^*_5 \pm 1.96 S^2_K
\]

Therefore, the Kriging system provides a valuable parameter that determines
how good the estimation is at certain limit of confidence.

As an extension of this approach the mean value of the regionalized
variable over an area of particular dimensions can be estimated in much
the same way but appropriate extension variances of the used values in
relation to the geometric configuration of the area and their position
must be calculated. These extension variances are the \( f \) and \( g \) functions
which are available for consultation on specific tables (Royle, 1978).

Geometric anisotropies are accounted for by transforming to an
isotropic case using coordinates rotation.

When the semivariance function reaches a maximum and then
decreases with increasing lag, the data set is said to exhibit a drift
or a trend. This drift can either be linear or quadratic (Fig. 1.11).

This situation is known in Geostatistics as non-stationarity. The new Kriging systems are called "Universal Kriging".

One Kriging system exists for each type of drift. So for the linear type:

\[
\begin{bmatrix}
\begin{array}{c}
\gamma_1 & \cdots & \gamma_n \\
1 & & \\
\vdots & \ddots & \vdots \\
1 & & 1
\end{array}
\end{bmatrix} \begin{bmatrix}
(X_1-X_C) \\
(Y_1-Y_C)
\end{bmatrix} = \begin{bmatrix}
X_1 \\
Y_1
\end{bmatrix}
\]

where \( \gamma_{nn} \) = mean semivariance between samples

\[
(X_{n-X_C}) \quad (Y_{n-Y_C})
\]

\( X_D \) and the point to be estimated

\( Y_D \) size of search area in X and Y direction

The appropriate system of linear equation for the quadratic is (1.43)

Apart from these basic Kriging systems for cases of stationarity and non-stationarity, there are some other systems developed for particular situations. For example, when the areally distribution of sample values shows certain grouping, a cluster Kriging system is employed. If there are two variables which present a good correlation, a cross semivariogram can be produced to test the spatial correlation. A new Kriging system, known as Cokriging is then formulated.

Sometimes, it is not possible to find an acceptable linear combination of coefficients but instead may be expressed as a particular function. To solve this problem, non-linear geostatistics have been developed with techniques such as log-normal Kriging and disjunctive Kriging.
Fig. 1.11. Example of semivariogram showing the presence of a linear or quadratic drift (trend).
\[
\begin{bmatrix}
    \alpha x y d & 0 \\
    \alpha x y & 0 \\
    \alpha x z & 0 \\
    \alpha x & 0 \\
    \alpha x z & 0 \\
    0 & 0 \\
\end{bmatrix}
\begin{bmatrix}
    0 & 0 & 0 & 0 & 0 & 0 & 0 & 0 & \cdots \\
    0 & 0 & 0 & 0 & 0 & 0 & 0 & 0 & \cdots \\
    0 & 0 & 0 & 0 & 0 & 0 & 0 & 0 & \cdots \\
    0 & 0 & 0 & 0 & 0 & 0 & 0 & 0 & \cdots \\
    0 & 0 & 0 & 0 & 0 & 0 & 0 & 0 & \cdots \\
    0 & 0 & 0 & 0 & 0 & 0 & 0 & 0 & \cdots \\
\end{bmatrix}
\begin{bmatrix}
    0 & 0 & 0 & 0 & 0 & 0 & 0 & 0 & \cdots \\
    0 & 0 & 0 & 0 & 0 & 0 & 0 & 0 & \cdots \\
    0 & 0 & 0 & 0 & 0 & 0 & 0 & 0 & \cdots \\
    0 & 0 & 0 & 0 & 0 & 0 & 0 & 0 & \cdots \\
    0 & 0 & 0 & 0 & 0 & 0 & 0 & 0 & \cdots \\
    0 & 0 & 0 & 0 & 0 & 0 & 0 & 0 & \cdots \\
\end{bmatrix}
\]

\text{(1.43)}
The geostatistical approach to the study and estimation of spatial variation has been extensively used within the mining industry over the last 15 years and examples of applications cover almost all type of deposits. Some of these examples are also described in applied geostatistics text books by David (1977) and Journel and Huijbregts (1978).

1.10. Discussion

The topics covered in this introductory chapter are very wide and therefore many details have been left out. However, certain remarks can be pointed out. Thus, despite the usefulness of the frequency distribution polygons in order to intuitively grasp the range of variation and the degree of spread, it is necessary to produce frequency-probability plots. They help to assess the degree of normality in the distribution and to depict the presence of more than one population. The techniques of splitting populations, though somewhat arbitrary, provide a quick, non computer assisted, procedure for data grouping and pattern recognition. However, when the regionalized variable under study presents sudden fluctuations in the rate of spatial variability, the need for a spatial data analysis becomes apparent.

The use of the original data in creating new regularly-distributed values (Gridding techniques) for either hand or machine contouring facilitates the task of describing the underlying spatial variation of a noisy variable.

The gridding methods that employ some kind of local polynomial modelling of flexible order appear more favourable in comparison with those of distance weighting, triangular or nearest point methods.

In the mining environment, the most widely used predictive models have been trend surface analysis and the geostatistical approach (Kriging systems). Pros and cons for both approaches are abundant. Against regression models it has been said (Matheron, 1967) that a polynomial
is an imposition to a regionalized variable, and that it does not provide a quantification of the error of estimation. On the other hand, global polynomial data modelling may not be flexible enough to explain sudden changes (as in gold and placer deposits) of spatial variation. However, it is obvious that not only the theoretical but the mathematical principles behind the deterministic approach are also easier to be understood by the geologist and mining geologist than the geostatistical approach. In addition it provides the advantage of estimating the mean grade of any point or block without having to carry out the total ore reserve estimation again.

On the other hand, there is no doubt about the semivariogram and the Kriging systems representing a very powerful tool towards the study and precise estimation of variation. However, there are certain aspects which remain somewhat arbitrary. For instance, the selection of a semivariogram model is carried out by trial and error, i.e. the point Kriging program is run several times with different semivariogram parameters. The particular combination of nugget effect, sill and range that produce the smallest Kriging variance is selected as the model. Sometimes this model is notably different from the experimental semivariogram (Forkes, 1979) as is shown in Fig. 1.12. So it is then questionable the need for calculating the semivariance function.

On the other hand, the concept of geometric anisotropy as it is currently defined, only expresses the directional distance over which the values are correlated, yet no indication is given about how good or bad the correlation is. In addition, this type of anisotropy is transferred to a case of isotropy by transformation of the rectangular coordinates. So, estimation is not strictly carried out considering the anisotropy in a proper manner. There is also the common acceptance (Journel, 1978, page 81) that the size of the range determines the rate of change of $\gamma(h)$, i.e. for a short range a faster rate of change, which is not necessarily always true.
Fig. 1.12. Spherical scheme semivariogram models (solid line) fitted to a set of experimental directional semivariograms using the minimum Kriging variance criterion (after Forkes, 1979).
A question that still remains unanswered is when the core size interval, over which a mean grade has been obtained, is considered big enough for the deregularization of the semivariogram to be necessary.

The presence of a trend or drift is assessed in geostatistics by looking at the mean isotropic semivariogram. If it is decided that there is a linear or quadratic drift, such trend will be assumed to exist along all directions at the time the estimation by universal Kriging is carried out. It is difficult to imagine a geological phenomena that has the same trend in all directions.

It should be pointed out, however, that there is an increasing tendency amongst geostatisticians to consider and use more the experimental $\gamma(h)$ values as well as the traditional concept of anomaly (Marechal, personal communication).

It seems therefore that a combined local polynomial-geostatistical modelling approach is more promising. In fact, local polynomials alone have proved (Braile, 1978) to be a very efficient method of Gridding. Another theoretical attempt of combining both approaches has been carried out by O'Hagan (1978) in one dimensional curve fitting, using a polynomial in which the coefficients are allowed to vary in space according to a particular covariance function. Similar approach is also used by Torelli and Tomasi (1977) in interpolating hydrogeological surfaces. In the next chapter a new model is presented in which data structure is the basic criterion for designing the estimation procedure.
CHAPTER 2

2.1. Introduction

The calculation of the geostatistical semivariance function $\gamma(h)$ is without doubt an important procedure towards the understanding and estimation of spatial variability. This function and the derived semivariogram provide three major items of information:

I. It shows up the directional spatial variability. In fact, if there are preferential directions in which the sample values have a tendency to be more alike than in other directions, this phenomenon will be reflected in the semivariogram. So, an intrinsic property of the data is revealed. This property would still remain unknown if any other technique of spatial data analysis is used.

II. The distance beyond which the values are no longer spatially correlated, i.e. it provides a new concept of the area of influence of sample values.

III. The separation of the spatial component (behaviour of $\gamma(h)$ over the range) from the random component (nugget effect) of the regionalized variable. Therefore, predictive models that account for both components can be set up.

The rate of change of the semivariance function represents the spatial variation, i.e. the faster the rate of change the less similar data values are for that particular direction. It is obvious then that in order to describe the spatial variability in a suitable manner, the semivariance function should be calculated in as many directions and lags as the data allows. However, most of the work published on geostatistical analysis of spatial data usually considers only semivariograms in as many as four directions, and in most cases the structural analysis and estimation procedure is based upon the mean isotropic semivariogram.

Nevertheless, there is an important work presented by Krige (1978) in which the semivariograms have been calculated in 8 sectors as is
Fig. 2.1. Sectors and lags in which semivariograms have been calculated for the Hartebeestfontein Mine, South Africa (after Krige, 1978).
shown in Fig. 2.1, with the lag increasing on a logarithmic scale. There is however, no attempt to describe the behaviour of the semivariance function by comparing it from one sector to the next and the main concern of Krige's paper is to obtain a controlled mean semivariogram.

A procedure for studying the semivariance function is presented in this chapter with the development first of a computer program to calculate the semivariance function in up to 18 sectors and as many lags as the data allows. Contouring the \( \gamma(h) \) values leads to the concept of iso-semivariance diagrams and their relationship to the structure of the data.

2.2. The semivariogram computer program VAR

In order to establish a good control over the semivariance function it is necessary to calculate it in as many directions and lags as the data set allows. To this aim the program VAR (O'Leary and Monge, 1980) has been developed and a listing is presented in the appendix.

In two dimensions the assays considered for computing \( \gamma(h) \) are separated by a vector \( R \) which describes the field \( S \). We may derive from the equation for the linear semivariogram (Eqn. 1.28) an equation to describe the variation in two-dimension. In this case is:

\[
\gamma(h) = \frac{1}{2S} \int \left( f(X_{th1}, Y_{th2}) - f(X, Y) \right)^2 \, ds 
\]  

and the three dimensional function may be similarly derived.

The program is devised to calculate \( \gamma(h) \) in three orthogonal planes, so the vector angle component (relative to an orthogonal plane) between a pair of sample points is calculated and this angle is assigned to an integer sector of specific size. Using this method data need not be arranged on a regular grid to obtain all semivariograms in any given direction but instead, semivariograms are obtained for all point pairs falling within an arc of specified size, and the smaller the arc the
Fig. 2.2. Flow diagram of program Var.
better the directional sense of the semivariogram. Semivariograms can be obtained in sectors from each orthogonal plane from 0 to 180 degrees from a major axis and mirror images from 180 to 360 degrees due to the 2-way relationship between each point pair. A generalized flow diagram for the program is shown in Fig. 2.2.

The program is written in FORTRAN IV and is currently set up on a CDC CYBER 73 computer. Up to 5 dependent variables, with no more than 500 data points each, in any particular format, can be processed in one run and a maximum number of lags (radius of search sphere) equal to 100 is allowed. A set of plotting routines named GHOST, developed by CULHAM Laboratories, are used for plotting which can be performed by either graph plotter or line printer.

Various types of variograms can be plotted: one or two dimensional variograms on X-Y, Y-Z and Z-X planes and three dimensional ones. They can be of form either arithmetic, arithmetic-semi log and log-log scale. All three forms can also be plotted in one run. Down bore-hole variograms can be obtained by setting a sector of 90° and selecting either Y-Z or Z-X planes.

The way in which search sectors increase on each of the orthogonal planes is shown in Fig. 2.3.A.

Fig. 2.3.A Directional increase of search sectors.
2.2.1. Control Cards

In order to take full advantage of the versatility of VAR, a set of control cards, indicating processing options, are to be inserted at the front of data deck:

Card 1: Number of dependent variables (up to 5) (Format: I1)

2: Title card for each dependent variable (Format: A8)

3: Variable format card for input variables (Format: 8A10)

4: If data is to be read from file, type 3 (Format: I1)
   If data is to be read from cards, type 5 (Format: I1)

5: Radius of variograms search sphere (Format: F10.0)

6: Size of each search sector in degrees (Format: F10.0)

7: Type of variograms to be plotted:
   (Format: I1)
   For 1 or 2 dimensional variograms in X-Y plane, type 0
   For 1 or 2 dimensional variograms in Y-Z plane, type 1
   For 1 or 2 dimensional variograms in Z-X plane, type 2
   For 3 dimensional variograms, type 3

8: Form of variograms to be plotted (Format: I1)
   For arithmetic scale variograms, type 1
   For arithmetic & semi-log variograms, type 2
   For arithmetic, semi-log & log-log variograms, type 3

9: Title card for data (Format: 8A10)

The data deck, following this set of control cards, should include three independent variables (X, Y, Z) and up to five dependent variables (assay values) with a letter T punched in last column of last data card. Format on which data deck is punched must be specified on control card number 3.

2.2.2. Program description

Mainline

Input-output devices, arrays dimensions and information in COMMON blocks are established in the Mainline. It also reads options selected in the set of control cards together with dependent and independent variables on each data card and initializes number of search sectors in X-Y, Y-Z and Z-X planes. Then, processing subroutines are called and graph-plotter (or line printer) is switched on and off by calling its appropriate routines.
Subroutine Vectpr

Subroutine to calculate vector-distance, vector-angle, and semivariance between all point-pairs.

Vectpr resolves the vector relationship between point-pairs into distance and angle components in 3 orthogonal planes.

Subroutine vectpr calls other subroutines, including:

- **Angle**: calculates vector angle component in three orthogonal planes
- **Sector**: assigns vector angle component to sector of search circle
- **Semivar**: calculates semivariance between all point-pairs
- **Hlist**: lists semivariance values for each sector in 1 plane
- **Vgmplot**: calling subroutine for plotting subroutine (pinist)

**Input values:**
- AS: assay value at each point
- X: X co-ordinate of each point (easting)
- Y: Y co-ordinate of each point (northing)
- Z: Z co-ordinate of each point (vertical depth)
- ANR: radius of search sphere
- ANGL: search angle (degrees)

Subroutine angle

Calculation of a vector between a pair of points.

Commence tree-search to calculate three vector components of all possible point-pair combinations.

- Calculate vector-angle component in X-Y plane relative to positive X-axis (bear)
- Calculate vector-angle component in Y-Z plane relative to positive Y-axis (decly)
- Calculate vector-angle component in X-Z plane relative to positive Z-axis (decly)

Subroutine sector

Subroutine to assign component of vector angle in a plane (vect) to an integer sector of that plane (isect), to increment the distance class-interval counter (KK) for that sector, and to increment the running total (AA) of squared-differences of metal values (assay) between point-pairs for that sector.

**Input values**:
- vect = component of vector angle in a plane
- assay = squared differences in metal value between a pair of points
- int = calculated distance class-interval
- mx = max.no. of class-intervals in each sector
\( (ARRAY) \) aa = list of total squared-differences in metal values between point-pairs in each sector.

\( (ARRAY) \) kk = list of number of point-pair distances in each class-interval of each sector.

Subroutine semivar.
Subroutine to calculate semivariance.

Semivariance \( (H) \) calculated from half the sum of squared differences of assay values \( (A) \), divided by the number of point-pairs \( (K) \) for each sector.

i.e. \( H(I,j) = \frac{A(I,J)}{(K(I,J) - 2)} \)

Input values:  
\( M = \) no. of sectors  
\( Mx = \) max. no. of class intervals in each sector.  
\( (ARRAY) \) KK = list of class intervals in each sector.  
\( (ARRAY) \) AA = list of squared differences in each sector.  
\( (ARRAY) \) HH = list for calculated semivariance values in each sector.  
\( I = \) sector no. (=isect)  
\( J = \) class interval no. (=int)

Subroutine Hlist
Subroutine to list values of semivariance \( (HH) \) for each sector in a plane.

Input values:  
\( M = \) no. of sectors.  
\( Mx = \) max. no. of class-intervals in each sector.  
\( (ARRAY) \) HH = list of calculated semivariance values in each sector.

Subroutine Vgmplot
Calling subroutine for main plotting subroutine \( (pinist) \)

In each sector, values for semivariance \( (HH) \) are assigned to correct class-interval no. \( (J) \) and stored in two arrays \( (din(1)) \) and \( (din(2)) \).

Vgmplot then calls subroutine to plot variogram for each sector \( (pinist) \).

Variograms are plots of semivariance vs. increasing lag (increasing class-interval numbers)

Vgmplot also determines whether arithmetic, semi-log, or log-log variograms are to be plotted.

Any plotting routine may be used to plot semivariance \( (din(2,J)) \) as the Y-component, and lag \( (din(1,J)) \) as the X-component.

Log-transforms of semivariance values, or both semivariance and lag values may be used to give smoother variogram plots.

Input values:  
\( M = \) no. of sectors,  
\( Mx = \) max. no. of class intervals in each sector.  
\( (ARRAY) \) HH = list of calculated semivariance values in each sector.
2.2.3. Output sequence

Printed output contains the following information:
- Number, title and data card format for each dependent variable
- Radius of search sphere (lag)
- Search angle (in degrees)
- Number of search sectors in each plane
- List of dependent and independent variables values
- Number of class intervals in each sector
- List of semivariance values for each lag and sector in each plane
- Ditto for the number of point-pairs (where no pairs exist the semivariance is set to -100.0)
- Variograms for each sector as accordingly selected

The Program also produces warnings when data points present the same X, Y and Z co-ordinates and prints them out. If a too small radius of search sphere has been chosen, the program prints an error message and states a value as a factor by which the radius should be increased.

2.2.4. Possible modifications to the program

Certain alterations can be introduced to the program in order to suit particular data sets and implementations requirements. Thus, if other than GHOST plotting routines are used, the following cards must be deleted from the main line:

```
CALL PAPER (1)
CALL GPSTOP (I page)
CALL GREND
```

Should the number of search sectors, and hence the number of variograms for each plane need to be increased, the dimensions of MIX, MIY, MIZ, MAXX, MAXY, MAXZ, AX, AY, AZ, KX, KY, KZ, HX, HY and HZ must be increased but only in the dimension statement of the main line. The number of sample points to be processed can be enlarged beyond 500 by increasing the dimensions of X, Y, Z, B and C and the limiting value of the second DO-loop.
2.3. The concept of an isosemivariance diagram (ISVD).

The calculation of the semivariance function, carried out in say 18 sectors of 10 degrees each and on as many lags as the data allows, provides a good control for the structural analysis. This analysis can be performed in either the classical geostatistical manner, i.e. fitting models to the experimental semivariograms and obtaining a mean model or, as it is proposed here, by contouring the $\gamma(h)$ values. The surface obtained from this iso-semivariance diagram (I.S.V.D.) will enable us to more accurately describe the behaviour of the semivariance function in relation to the directional variability and degree of organization of the regionalized variable.

2.4. Constructing an I.S.V.D.

The output of the program Var will display $\gamma(h)$ values calculated for each particular sector and lag. Having defined a priori the sector size and the lag, it is possible then to associate any sector number with a bearing or direction. Considering the symmetrical nature of the sectors due to the two-way relationship between each point pair, the centre of the diagram will correspond to the origin of the semivariograms ($\gamma(0)$). The lag number increases outwards from that centre and the sectors in an anticlockwise manner. The semivariance values are plotted along a line that bisects the appropriate sector and the contouring is carried out.

This procedure of setting up the $\gamma(h)$ values to be contoured is extremely time consuming and subject to human errors. In order to ease this task a computer program to transform polar co-ordinates of the $(h,R)$ space to cartesian co-ordinates has been developed. A listing of this program is presented in the Appendix.

The new set of cartesian co-ordinates is amenable for automatic plotting as is shown in Fig. 2.3,B. The usefulness of the computer transformation of co-ordinates is that automatic contouring can then be carried out directly with standard contouring routines. At the present
Fig. 2.3.B. Construction of an iso-semivariance diagram on 18 sectors of 10 degrees in size. Each point has a calculated semivariance value at the appropriate lag.
time, routines for automatically contouring data with associated polar
co-ordinates are not readily available. It should be pointed out that
before any attempt at contouring is made, a check of erratic values of
\( \gamma(h) \) in relation to the number of pair of points that produce such values
must be done. In other words, the importance of a particular \( \gamma(h) \) value
depends upon the number of comparisons performed. The erratic value
should be included in the contouring if the number of pairs is of the
order of 10\% or more in relation to the maximum number of pairs that have
been used to calculate any \( \gamma(h) \) value on that sector. However, if that
percentage is less than 10\%, the erratic \( \gamma(h) \) value can still be considered
provided there are values of similar magnitude on sectors located at
either side and for that particular lag. If this is not the case, the
erratic value should be excluded from the contouring as is shown in
Fig. 2.4. The objective of this preliminary screening is to avoid
meaningless distortions of the semivariance surface.

Hand contouring is time consuming and is prone to human error and
subjectivity. It is then necessary to look for a reliable automatic
contouring routine. To this effect the contouring algorithms devised
by McLain (op.cit.) and implemented on the CDC Cyber 73 computer at
Leicester University under the GHOST plotting package have been selected.
The algorithms produce a regular grid using local polynomial fitting as
the interpolation procedure. The results are quite satisfactory despite
the edge effects. These disturbances only appear beyond a lag that
normally corresponds to a greater distance than 2/3 of the sampled length
in any particular direction. A comparison of I.S.V.D. produced by hand
and automatically is presented in Fig. 2.5 and the program listing in
the Appendix.

It should be noticed that none of the mathematical models of semi-
variograms mentioned in Chapter 1 is fitted to the experimental ones but
the calculated "raw" semivariance values are used instead to produce the
iso-semivariance diagram. Therefore, the problem of subjectivity almost
Fig. 2.4. Example of preliminary screening before contouring of $\gamma(h)$ values is attempted. The semivariance value of sector 4 shown by the arrowhead, must be deleted because it has been obtained only by two pairs and no similar values appear at the same lag on contiguous sectors 3 and 5.
Fig. 2.5.A. I.S.V.D. produced by hand contouring (to be compared to the appropriate automatic contouring of figure 2.5.B.)
Fig. 2.5.B. I.S.V.D. produced by automatic contouring
unavoidable so far in variogram fitting, is surmounted.

2.5. Response of I.S.V.D. to data structure.

It has been said above that the rate of change of the semivariance function represents the spatial variation. Therefore, it should also depict the degree of organization or structure within the data. In order to study the relationship between the I.S.V.D. and the directional variability of real data (O'Leary and Monge, 1980), a data set was firstly simulated and the response of the semivariance contouring to data structure is examined below.

2.5.1. Simulated data and I.S.V.D.

The simulated data set consists of 500 regularly-distributed data points with 4 dependent variables. Each variable represents a differing degree of "structure" and values range from 0 to 4. Variograms in different directions and iso-semivariance diagrams for each variable have been plotted. This has established relationships between the organization within a particular set of data and derived functions.

1. - Random. The appropriate experimental iso-semivariance diagram is shown in Fig. 2.6. The commonest value of \( \gamma(h) \) calculated in all directions and at each possible lag is 0.60 which corresponds to the classical statistics variance. The other \( \gamma(h) \) values, i.e. 0.55, 0.65 and 0.70 are not abundant enough to influence the contouring and the appropriate curve values do not show a parallel arrangement. Therefore, no directional rate of change in \( \gamma(h) \) values can be observed and clearly no structure is defined within this data set.

2. - One structure. In this data set a structure of poor definition trends in a NW-SE direction. The iso-semivariance graph (Fig.2.7) depicts this structure and in the upper right-hand corner a secondary structure may be present. Semivariance curves extend in a general NW direction and the rate of change of the \( \gamma(h) \) function is obtained along the NE direction in
Experimental Iso-Semivariance-Simulated Data (Random)

Fig. 2.6
Experimental Iso-Semivariance - Simulated Data (1 Structure)

Fig. 2.7
Iso-Semivariance Model - Simulated Data (1 Structure)

Fig. 2.8
which, semivariance values gradually increase from the centre of the
diagram. A maximum $\gamma(h)$ value of 0.7 is obtained and corresponds to
the upper part of the main structure. Then curves' intervals get wider,
indicating slow rate of change and in the NE edge a new increase of $\gamma(h)$
is observed, starting the definition of the secondary structure. In an
attempt to find a suitable underlying function, a model (Fig.2.8) has
been fitted to the experimental iso-semivariance in order to attain a
clearer definition of the main structure.

3. - Two structures. A data set with two features present, one trending
NW-SE and the other of circular shape have been simulated. The orientation
between the structures is NE-SW. The experimental iso-semivariance graph
(Fig. 2.9) depicts both structures. The NW structure is defined in the
central part of the diagram and the semivariance curve of 0.65 corresponds
to the centre of that structure. This centre is closer to the origin than
in the previous example of only one structure. This indicates a relative
position of the structure closer to the left lower edge in the data set.
The circular structure has been simulated at the upper right-hand corner
in the data set. Its shape, relative position to the elliptical structure
and their distance apart are observed in Fig. 2.9 and clearly defined in
the fitted model (Fig. 2.10).

4. - Structure-controlled. A highly structured data set containing
a well-defined organization in the NE-SW direction is notably reproduced
in the iso-semivariance graph (Fig.2.11) and there is no need to fit a
model. The range of $\gamma(h)$ values (0.1 to 1.3), larger than in the previous
examples, is due to the directional regularity in the contrast of data
values. The centre of the 1.3 curve corresponds to the upper part of the
highly structured organization.

2.5.2. Testing real data.

As it has been shown in the various simulated data sets, iso-semi-
variance diagrams demonstrate a direct relationship with the degree of
Experimental Iso-Semivariance - Simulated Data (2 Structures)

Fig. 2.9
Iso-Semivariance Model - Simulated Data (2 Structures)

Fig. 2.10
Experimental Iso-Semivariance-Simulated Data

(Structure Controlled)

Fig. 2.11
structure within the data. Therefore, a similar test has been carried out on real data from a conventional geochemical soil survey in an area of 1 x 1.5 km in Devon. A total of 565 soil samples were collected in a regular grid of 25 x 100 m and analysed for copper and lead. Iso-semivariance graphs of each of these elements are shown in Figs. 2.12 and 2.13. For copper, a more and less continuous structure, trending EES-WWN and covering almost the whole sampled length along this direction, can be identified. For lead, also a general E-W trend is recognised with a possible extension to the north in the central part, and a sub-structure trending N-S is located at the right edge of the sampled area.

In order to compare results with well known automatic mapping techniques, ordinary moving average (Monge, 1977) and trend surfaces maps (Esler et al., 1968) have been obtained for both elements. These maps, Figs. 2.14 to 2.17 show a clear congruence with the iso-semivariance diagrams.

2.6. The relation between I.S.V.D. and polynomial modelling.

According to Unwin (op.cit.) a trend surface is representative of the data structure and becomes the best linear estimator only if the residuals have a Gaussian distribution with a mean value of zero, showing no autocorrelation and if there are as many data points as coefficients of trend equation plus one. Although other statistical parameters such as the F-ratio coefficient of determination and coefficient of multiple correlation can be calculated and also assess the goodness-of-fit of a trend, the assumption of no autocorrelation in the residuals is the most difficult to obtain.

However, as it has been shown above, iso-semivariance diagrams reveal the degree of internal organization within a data set. If the polynomial adequately models the raw data, then the iso-semivariance diagram generated in the regression will be similar to that generated on the raw data. Furthermore, the iso-semivariance on the residuals from the model will
Experimental Iso-Semivariance ($10^3$) - Copper in Soils (DEVON)

Fig. 2.12
Experimental Iso-Semivariance ($\times 10^2$) - Lead in Soils (DEVON)

Fig. 2.13
Ordinary Moving Average Grey Scale Map
Copper in Soils (DEVON)
Fig. 2.14
Fig. 2.15. 2nd Order trend surface of copper in soils (Devon). Values in ppm (X100).
Ordinary Moving Average Grey Scale Map
Lead in Soils (DEVON)

Fig. 2.16
Fig. 2.17. 2nd Order trend surface of lead in soils (Devon). Values in ppm (X100).
be random, showing no autocorrelation.

To illustrate this procedure, data from the Ptolemais Lignite deposit of Northern Greece (Hatzigiannis, 1978) is being used. Iso-semivariance diagrams for the variable "thickness of recoverable lignite" (in metres) show a main E-W structure (Fig. 2.18) and most of the variation is explained by a quadratic trend (Fig. 2.19) because no autocorrelation is observed in the residuals (Fig. 2.20). However, a second order trend (Fig. 2.22) of the variable ash (%) is not representative of the total variation (Fig. 2.21) since an internal correlation is still present in the residuals (Fig. 2.23).

2.7. Conclusions

When the geostatistical semivariance function is calculated in all possible directions and lags as the data allows, and iso-semivariance diagrams are produced data structure can be identified. This type of graph also permits the detection of more than one structure and if present, it establishes their mutual spatial relationship together with their dimensions and orientation within the data set. As a direct application of this approach, iso-semivariance graphs of geochemical soil survey data may influence the design of a sampling grid for further detailed studies. In addition, the assumption of non-autocorrelated residuals, as a condition of significance in trend surface fitting, can be tested with iso-semivariance diagrams. Having established the relationship between iso-semivariance diagrams and the degree of internal organization within a data set, a new technique is presented in the next Chapter in which a weighting function for data filtering as predictive model is based upon the appropriate iso-semivariance diagram.
Figure 2.18 - Experimental Iso-semivariance of thickness of recoverable Lignite raw data. Ptolemais Lignite deposit (N GREECE) after HATZIGIANNIS (1978)
Figure 2.19 - Experimental Iso-semivariance on quadratic order regressed values. Thickness of recoverable Lignite (m)
Figure 2.20 Experimental iso-semivariance on residuals from a quadratic order Polynomials. Thickness of recoverable Lignite (m)
Figure 2.21 - Experimental Iso-semivariance of ASH raw data. (%)
Figure 2.22 - Experimental Iso-semivariance of a second order Polynomial fitted values of ASH. (%)
Figure 2.23: Experimental Iso-semivariance of residuals from a second order Polynomials fitting to ASH values (%)
CHAPTER 3

3.1. Introduction

The advantages and disadvantages of the most commonly used predictive models of spatial variability, namely polynomial data modelling and the Geostatistical approach were discussed in the first chapter. It was said that overall regression models may not explain the sudden changes in the variability pattern in a satisfactory manner and that conventional Kriging systems either assume that the data does not exhibit a trend (specimen Kriging) or if there is actually a trend it is assumed to exist in all directions (Universal Kriging). In an attempt to obtain a better model for predicting spatial variability, Braile (1978) performs local regressions over an area of arbitrary size and shape and O'Hara (1978) allows the polynomial coefficients to vary with space. However, no geological or data structure criteria has been defined to control local fitting.

An alternative approach is developed in this Chapter in which it is demonstrated how the iso-semivariance diagrams can be used to select the number of points to be included in the locally fitted models. The method is devised for the $\gamma(h)$ function behaving according to the transitive scheme. A weighting procedure is then also developed using the covariance function $C(h)$ obtained from the component transitive model (Fig.3.1) where $C(h) = C - \gamma(h)$, i.e. $C(h) = C(0) - \gamma(h)$, $C$ being the sill of the iso-semivariance diagram. Practical applications of the technique in the mining environment are presented in this Chapter together with a description of computer programs and an analysis of the methods' limitations.

3.2. Description of a locally-fitted covariance weighted regression model

The method can be described, for the purpose of ore reserve estimation, as a covariance weighted moving average trend surface. The cell size and shape over which a surface is calculated is defined by the area of the internal correlation obtained from I.S.V.D. An example of the $\gamma(h)$
Fig. 3.1 Relationship between the semivariance and the covariance function on a transitive scheme.

Fig. 3.2 Hypothetical example of a transitive scheme in two dimensions, showing a stronger spatial correlation along E-W than in N-S direction.
function in two dimensions is given in Fig. 3.2.

This diagram shows a strong correlation along the E-W direction defining an elongated zone of internal correlation. However, in the N-S direction the function exhibits a very rapid change which indicates that assay values are weakly correlated along this direction. The iso-semivariance contour beyond which contouring loses its consistency is taken as the outer limit of the zone of internal correlation and therefore it defines the size and shape of the cell over which sample selection can be made for use in a smoothing function. The appropriate \( \gamma(h) \) value of this iso-semivariance line (5 in Fig. 3.2) will have the value of \( C \) the sill.

The centre of this cell is placed at any one time on the centre of the block for which a mean grade is to be estimated. A series of data points will be enclosed within the cell. The influence of each individual data point value on the block estimated grade is accounted for by weighting according to the covariance function \( C(h) \) graphically described on Fig. 3.1. This function decreases directionally with distance from the centre of the cell in a similar way to the increase of the \( \gamma(h) \) function. Therefore, from the cell of Fig. 3.2. a larger weight will be given to data points located along E-W direction than those on the N-S direction, distance remaining constant. The weighting procedure should also take into account the nugget effect. This nugget variance can be then more easily obtained from the I.S.V.D. in any particular direction by extrapolating towards \( \gamma(0) \) the rate of change of the semivariance function in cross sections along the appropriate direction.

In order to obtain a local model of equal mean the weighted data sub-set is standardised by multiplying each point value by the ratio of raw data sub-set's mean to the mean of the weighted sub-set. A regressed model of first, second or third order is locally fitted to the standardised data points. This procedure is repeated for all the blocks to which an estimated mean grade is to be assigned. An example is
illustrated in Fig. 3.3.

The order of the polynomial is selected according to the requirements that at least n+1 data points must be included in a regressed model defined by an equation of n coefficients. In the examples cited below the order of polynomial is selected as follows:

If 4 points or less are included, no model is fitted. The First order will be fitted to a subset of 5 - 8 data points and a second order to a cell containing from 9 to 15 points. If the number of points is greater than 15, a 3rd order is then selected. In order to place a certain degree of confidence on the amount of variation explained by the model, the fitting parameters (goodness-of-fit and coefficient of multiple correlation) are calculated. Also, the standard error of the model is obtained using the relationship:

$$\text{ste} = \frac{1 - R^2}{\sqrt{n}}$$

where R is the coefficient of multiple correlation and n the number of samples included in the local model.

The estimated value of any point within a particular local model can be worked out by substituting its x and y co-ordinates in the model equation. Also, a mean value of the model within an area of dx by dy dimensions can be estimated by integrating the volume beneath the surface and dividing by that area, provided the limits of such area do not fall outside the zone of internal correlation, i.e. outside the area over which the surface has been generated. When the area of a block to be estimated is considerably smaller in relation to the size of the I.S.V.D. as is the general case in a mining operation, the surface is relatively constant over that block area and the mean grade can also be estimated with the average of a limited number of points conveniently placed on the block (corners and the centre). This latter procedure is preferred when appropriated, due to the optimisation of computing time to perform the total ore reserve estimation. In addition, the coefficients of individual local models can be saved in a file together with the x and y co-ordinates.
Fig. 3.3. Elliptical zone of internal correlation defining the cell over which a surface is generated. (Graphical example of locally-fitted regression models).
of the appropriate model centres and be used for instance, to estimate mean grades of blocks of any dimension located at random within the sampled ore body, given the above restrictions.

All these possibilities are available with the combination of the Fortran IV computer programs LOCMODL, PREPROD and RANDEST that are described below and their listings are presented in the Appendix.

3.2.1. The program LOCMODL

The program LOCMODL fits models to data sub-sets in two dimensions. Sub-sets are defined by data points falling within the zone of internal correlation obtained from the iso-semivariance diagram. Data points are weighted according to the covariance function in relation to each local model centre. For ore reserve estimation purposes the centre of each local model should preferably coincide with the centre of the block for which a mean grade is to be estimated.

A polynomial of selected order is fitted to the smoothed sub-set and the calculated coefficients are used for estimating mean grade of blocks. If major blocks are divided into sub-blocks and their estimated mean grades are required, additional information should be provided and to run this program in conjunction with the program PREPROD.

The maximum number of data points to be included in any particular local model is 1000, and the maximum input data points, number of models and sub-blocks is also 1000.

The program has been devised in a flexible manner for input data and processing options such that the following control cards are to be included in front of data deck:

Card 1 : Title card for project Format (8A10)

Next : Sub-block estimation option: Format (II)
  if estimation required type 1
  if estimation is not required type 0 (skip next control card)
Next: Sub-Block control card: Format (I3,2X,I3,4F10.0)
No. of subblocks (per block) in x-direction
No. of subblocks (per block) in y-direction
Distance between sub-block centres (x-direction)
Distance between sub-block centres (y-direction)
x and y co-ordinates of first sub-block centre (bottom left)

Next: Local models control card including: Format (2I3,3X,5F10.2)
Number of models in x and y direction
Distance between model centres in x and y direction
x and y co-ordinates of first model (centre of bottom left hand corner block)
Size of zone of internal correlation major axe

Next: Variograms parameters card including: Format (I1,F5.0,4X,7A10)
Input device No. for variograms parameters (type 3 for data on file or type 5 for data on cards)
Size of search sectors in which variograms parameters are available.
Variable format for variograms parameters (FMT1)

Next: Sample data card including: Format (I1,I4,5X,7A10)
Input device No. for data set co-ordinates & grade (type 3 for data on file or 5 for data on cards)
Number of sample points (no more than 1000)
Variable format card for data set (FMT2)

Next: Variograms parameters (from I.S.V.D.): Format (FMT1)
Isect: sector No. counted anticlockwise from x-axis
Range: cut-off lag
Sill: semi-variance cut-off
Anugget: nugget effect

Next: x, y Co-ordinates and dependent variable (assay values) Format (FMT2)

The main line reads all necessary input data and initializes the main DO loop to find the centre of each block in turn. It writes out, at the end of a successful run, the following information for each local model:

- x and y co-ordinates of the centre
- number of points included
- goodness-of-fit, coefficient of multiple correlation and standard error
- coefficients of trend equation and mean estimated grade

Subroutine Search:

It sorts out the data points that fall within the zone of internal correlation and selects the order of the polynomial to be locally fitted.
If no more than 4 points have been found for a particular block no model is fitted and an error message is printed out. Standardisation of smoothed data sub-set and calculation of the standard error is also carried out in this subroutine.

Subroutine Angle:

It calculates the vector angle component, anticlockwise, between the model centre and a sample point.

Subroutine Dtest:

It tests whether sample points fall within zone of internal correlation of the appropriate sector. Then it calculates the covariance function between sample point and model centre and weight the sample value accordingly.

Subroutine Polynom:

It computes a polynomial of a particular order selected within the subroutine search. It also calculates deviations for each observation, goodness-of-fit and coefficient of multiple correlation.

Subroutine Linequa:

It solves a system of linear equations using the pivoting method (Davis, 1973).

Subroutine Global:

It calculates a mean grade for major block enclosed by the isosemivariance diagram using the coefficients of trend equation with the four corners and the centre of the block.

3.2.2. The program PREPROD

PREPROD enables a more detailed estimation of grades. It is assumed that we want to estimate grades of sub-blocks within major blocks over which local models have been fitted. Therefore, the input data is the file
containing the results from the program LOCMODL. The appropriate coefficients of trend equation are used with the four corners and the centre of each sub-block in order to estimate its mean grade. If no model has been previously fitted to a particular block no estimation is carried out for the sub-blocks.

3.2.3. The program RANDEST

Ranest computes a random estimation. The program has been developed to estimate the mean grade of a sub-block or set of sub-blocks located at random within the area in which local models have been fitted to major blocks. Again, the input data (coefficients and local model co-ordinates) is the output file resulting from program LOCMODL. Also the user should provide the number of sub-blocks to be estimated together with the co-ordinates of sub-block centre and dimensions in east and north directions. The program search for the nearest local model to the particular sub-block in terms of distance between their centres and uses the coefficients of nearest model trend equation to estimate the mean grade as in program PREPROD.

The program can be very useful for production scheduling and when a particular blend of ore is required.

3.3. Grade and standard error-tonnage relationships: the program ESTDISP

The evaluation of a mineral deposit is basically concerned with assigning an estimated grade to blocks of ore. Having defined the three-dimensional size of the block and measured the specific gravity of the material, a calculation of the tonnage is directly obtained. This amount of ore becomes meaningful when it is considered alongside with its estimated grade and with the confidence of this estimation. The ore reserves can then be classified not only in groups of a particular grade interval, but also in terms of the degree of certainty in which the estimation has been carried out. This procedure is by far less ambiguous than the traditional proved, possible and probable categories.
In the geostatistical manner, the Kriging variance, once accepted as an estimation variance, provides the 95% confidence level of the estimation. In the local models method presented here, the standard error is a measure of the amount of variation not explained by each local model and therefore can be used as an indicator of the degree of certainty for the estimated grade.

The most commonly used procedure of displaying the ore reserve estimation results is by calculating grade-tonnage and error of estimation-tonnage tables and plotting the appropriate curves. To this effect, the program ESTDISP (estimation display) has been written. The program displays the ore reserve estimation results from the local models method using the GHOST plotting package.

Two tonnage tables are produced: one considers accumulation of tonnes in decreasing order of grade and the other in an increasing order of standard errors. Also, if it is desired, a graph for each of these tables and a map showing the estimated grade and standard error for each block can be produced by selecting the appropriate options in the control cards and providing the plotting routines with their required information.

The input data is:
- tonnage factor (LxLxspec.Grav.)
- block thickness
- lowest and highest grade to be considered
- grade class interval
- minimum and maximum standard error
- standard error class interval
- ghost plotting routines requirements
- local models output file results

A detailed account of the grade and standard error-tonnage relationship is obtained by running the program several times with different limits of grade and standard error together with their class intervals. When either the estimated grade or standard error of a particular block fall outside the
specified limits the graph plotter reproduces those figures in red on the block map.

The program also calculates the tonnes of metals and mean grade of each group or class of blocks as well as the overall mean and variance of grade and standard error of the blocks brought into the ore reserves estimation. Examples of the program output will be shown later in practical applications of the local models method and the listing is presented in the Appendix.

3.4. Applications of the local models procedure.

The locally-fitted covariance weighted regressed model presented above has been tested with real data from drilling projects in three different geological environments. A brief description of these settings together with the nature of available data, discussion of the results and limitations of the technique is presented below.

3.4.1. The Quellaveco Porphyry Copper

The diamond drill data from this deposit has been kindly made available by Candiotti (1979) and description of the geology and mineralization have been taken entirely from his M.Sc. thesis.

The deposit is located in the south western sector of the Peruvian Andes and within the major South American porphyry copper belt (Fig.3.4). The ore body appears partially exposed in a small narrow valley and it shows a lens-like shape in plan with the major axis trending NW-SE. It is situated within a calc-alkaline volcanic and intrusive rocks assemblage, between 3500 and 3800 metres above sea level and about 7 km to the west of the main Andean volcanic chain.

A sequence of Late Cretaceous-Lower Tertiary volcanic rocks (andesites and rhyolites) were intruded by the Quellaveco stock (granodiorite and monzonite porphyry) during a Middle Tertiary phase of active major faulting and erosion. A subsequent volcanic cycle during the Early Pliocene
Fig. 3.4. Geographical location of the Quellaveco porphyry copper ore body, together with similar deposits in the South American porphyry copper belt. (After Candiotti, 1979)
originated the rhyollitic and trachytic tuffs of the Huaylillas formation which covers the previous calc-alkaline volcanics and intrusives. After this volcanic cycle and as a result of the uplift of the Andes the area has been subjected to intense erosion leaving the ore body partially exposed at the present time.

Three distinctive alteration zones (Fig. 3.5) are identified: potassic, sericitic and propylitic. Their relationship with the intrusive stock is shown in Fig. 3.6. The mineralization is distributed in different areas and the abundance of metallic sulphides exhibits a close relationship with the various wall rock alteration zones. The inner area, with low sulphides content and composed mainly of chalcopyrite, pyrite and bornite, is associated with the potassic zone. The intermediate area showing abundant sulphide minerals (chalcopyrite and pyrite) is coincident with the sericitic zone. The outer area contains little pyrite and appears related to the propylitic zone.

The primary mineralization is present in veins and disseminations. In the veins, the sulphide minerals are deposited along with quartz and other gangue minerals in continuous small fractures and the disseminated mineralization - more abundant than the vein type, appears replacing silicate minerals. Secondary enrichment is not properly developed and only occurs as a thin blanket containing chalcocite digenite and minor covellite and cuprite. A set of mineralized dacitic and rhyodacitic dykes crosscut the intrusive stock and have been emplaced along small NE-SW trending faults.

The bore hole data consists of total copper assay values obtained in 188 intersections with the 3500 metres level in primary mineralization. The drill pattern is shown in Fig.3.7 and the Fig. 3.8 corresponds to the iso-semivariance diagram from variograms calculated with a lag of 50 metres. This diagram clearly depicts the NW-SE spatial distribution of the mineralization and also a secondary correlation along NE-SW is observed and
ALTERATION PATTERN AT
QUELLAVECO PORPHYRY COPPER DEPOSIT
(After Candiotti, 1979)
Fig. 3.6 NE cross section that shows the relationship between Lithologies and alteration zones in the Quellaveco ore body (After Candiotti, 1979)
Fig. 3.7. Sampling density (DDH) map on the Quellaveco copper deposit (Level 3500 m)
Fig. 3.8. I.S.V.D. of Copper values from the Quellaveco deposit.
it might well be related to the fractures and dykes that trend in that direction. The contouring loses consistency at the contour line 4 which is the \((h) = 0.04\) and this defines the value of the sill.

The nugget effect and the range have been obtained in cross sections (Fig. 3.9) that bisect sectors of 20 degrees in size. It should be pointed out that once a controlled \(\gamma(h)\) surface has been calculated there is no need to carry out estimation of spatial variability using sectors of smaller size than 20 degrees.

The sampled ore body was divided into blocks of 200 x 200 m (Fig. 3.7) with a thickness of 15 metres, and the local models were fitted to them using the variograms parameters mentioned above. The results of the estimation are presented in figures 3.10 and 3.11 and Tables 3.1 and 3.2. The upper number on the estimation block map (Fig. 3.10) corresponds to grade and the lower to standard error. The first digit of the grade defines the centre of the block. When either the grade and/or the standard error fall outside the specified limit, the figures are printed in red by the Graph Plotter, but for the purpose of reproduction an asterisk has been drawn instead.

The estimation of reserves is a simulation because no specific gravity of the ore has been made available. It can be observed on Fig. 3.10 that if a maximum standard error of 15\% is allowed the technique is sensitive to edge effects and estimation of blocks situated on the outline of the sampled area shows a relatively high standard error. Nevertheless, the mean error of estimation within the specified limits is only 7\%.

3.4.2. The Gortdrum deposit

The Cu-Ag-Hg stratabound gortdrum deposit is situated in the Republic of Ireland (Fig. 3.12 and Table 3.3) and it appears, like many other Irish bases metal deposits, within the Lower Carboniferous rocks of various sedimentary facies adjacent to a major normal fault trending approximately east-northeast (Morrisey et al, 1971). This fault throws beds of the
Fig. 3.9 Graphical estimation of the nugget effect (Co) and range (a) in cross sections on the appropriate I.S.V.D. along lines that bisect sectors of 20 degrees in size. (Quellaveco copper deposit).
** INPUT PARAMETERS **

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** RESULTS **

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** GRADE-TONNAGE TABLE **

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** STANDARD ERROR-TONNAGE TABLE **

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## PORPHYRY COPPER DEPOSIT

**BLOCKS INCLUDED IN THE RESERVES**

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Table 3.2
MINIMUM GRADE = 0.100
MAXIMUM GRADE = 1.000
MAXIMUM STD.ERROR = 0.150

Fig. 3.10. Block grade estimation of copper in the Quellaveco deposit. (X = Values outside specified range).
Carboniferous lower limestone and lower limestone shales to the north, against sandstones, shales and silts of the Devonian to the south.

Carboniferous-age beds lie conformably upon the Devonian north of the fault zone, while to the south the sandstones and shales dip gently to the south west (Gortdrum Mines Ltd, technical report, 1971).

Carboniferous beds are markedly altered with dolomitization, local marble development introduced carbonate veining, barite and rarer silica and recrystallization of younger limestone. These beds are extremely disturbed and fractured, mainly in the vicinity of the main fault and elsewhere can exhibit complex folding and thrusting.

The ore occurs in the Carboniferous limestone and shales, without preference, with the axes of mineralization parallel to the main fault and also to other hanging wall faults. In the upper part of the mine, mineralization consists predominantly of chalcopyrite and tennantite and bornite-chalcopyrite. The deeper section of the east end of the ore body contains bornite-chalcocite as the copper ore with lesser tennantite and sparse chalcopyrite. Silver is largely native in origin and mercury is associated with tennantite and cinnabar.

The ore body was discovered after a geochemical stream sediment sampling in 1962 followed up by a detailed soil survey that delineated the anomaly (up to 1200 ppm Cu). The IP survey in 1964 showed a definite anomaly and gave improved drilling target definition (Schultz et al., 1971). A later trial on soil sampling for mercury showed a marked anomaly.

The development for open cast mining and plant construction began in early 1966 and a production rate of 15000 short ton/day started in late 1967. By the end of 1970 the output was 1,578,000 short tonnes of ore, 14,639 tonnes of copper and 1,221,341 oz. of silver in concentrates. Also 1,340 flasks of mercury was produced in 1970 from the concentrated roasting plant. At the end of that year the calculated reserves were 1,500,000 tonnes of 1.4 per cent copper and 0.98 oz. of silver/ton.
Fig. 3.12 Geological map of Ireland showing the location of Gotdrum and other base metals deposits. See Table 3.3 for key to map. (After Schultz, 1971).
1. Abbeytown Zn-Pb Deposit
2. Aberlow Cu-Ag Deposit
3. Allihies (Mountain Mine) Cu Deposit
4. Amos Cu-Pyrite Deposit
5. Ballinacork Zn-Pb Deposit
6. Ballinslea Pb Deposit
7. Ballvihergin Cu Deposit
8. Bannow Cu Deposit
9. Civicinella Zn-Pb Prospect
10. Castleblaney Area Pb-Zn-Sb-Au
11. Glendalough Pb Deposits
12. Gantown Cu-Ag-Hg Deposit
13. Keel Zn-Cd Deposit
14. Meecameena Mo Prospect
15. Meighera Cross Cu Prospect
16. Mante Zn-Pb Prospect
17. Muirney Ma Prospect
18. Navan Zn-Pb Deposit
19. Silvermines Zn-Pb Deposits
20. Tynagh Pb-Zn-Ag-Cu Deposits
21. West Carbery District Cu + 8o

Key to Fig. 3.12

Table 3.3
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<th>Orebody</th>
<th>Ore tonnage and grade</th>
<th>Structural setting</th>
<th>Host rocks</th>
<th>Styles of mineralization</th>
<th>Wallrock alteration</th>
<th>Epigenetic mineralogy</th>
<th>Apparent paragenesis</th>
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<td>Gortdrum</td>
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<td>On the down-faulted northern flank of a ENE–WSW trending Devonian inlier. In fractured limestones in the hanging-wall zone of a large normal fault. This down-throws to the north a max. of about 400 ft</td>
<td>Bioclastic lower limestones and shales of Tournaisian age. These are cut by a series of highly altered basic dykes of presumed Carboniferous age</td>
<td>As disseminations within the more shaly rock members and as ramifying veinlets, infillings and replacements in the more massive limestones</td>
<td>Localized dolomitization and silicification</td>
<td>Economic sulphides</td>
<td>(1) Iron sulphides and tennantite</td>
<td>Chalcocite and tennantite (hence, also, Hg) concentrated towards the top of the orebody. Bornite and chalcocite intimately intergrown and concentrated in the lower parts of the orebody</td>
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*Short tons.

Table 3.4 Summarized characteristics of Gortdrum deposit (after Morrisey, 1971).
Mining operations ended in 1975 (?) once reserves were exhausted.

A list of the main characteristics of the Gortdrum deposit are summarized in Table 3.4. Several blast hole data sets from benches in the east and west parts of the ore body were kindly made available by Dr. P. Leyshon. Variograms and I.S.V.D. were produced and are discussed below.

The I.S.V.D. of bench 12 east (Fig. 3.13) shows that the calculation of 8 variograms in sectors of 22.5 degrees each has been good enough in this case, to define the directional rate of change of the semivariance function in a satisfactory manner. The diagram clearly depicts the general E-W distribution of the mineralization and it is also noticeable close to the centre, the slightly E-EN arrangement of the semivariance contour lines 4, 5 and 6. This could perhaps be related to small scale structural features of the deposit such as a system of "en-echelon" faults or axes of folds originated as a side-effect of the major E-W faulting?

The sill of the zone of internal correlation is marked by the contour 8, i.e. the value of $\gamma(H) = 2.4$. The local model method was applied however to bench 11 West due to certain peculiarities of the appropriate I.S.V.D. (Fig. 3.14). In this diagram the general E-W distribution of the mineralization can still be identified up to the curve 5 ($\gamma(H) = 0.5$) but in the NW direction the section between line 7 and 8 is noticeably wider which may indicate either the presence of two levels of internal correlation or a drift.

Overall regressed polynomial models of 1st, 2nd, 3rd, 4th and 5th order were fitted to raw data and the regression of 4th order showed the best fit (coefficient of multiple correlation = 0.7). However, despite the residuals showing a gaussian distribution about 0 (Fig. 3.15) they maintain certain amount of spatial correlation (Fig. 3.16) in the NW direction. Therefore, the 4th order polynomial does not adequately explain the variation along that direction despite the relative similarity of the
Lag = 10ft. Base contour (1) = 0.30
Contour interval = 0.30

Fig. 3.13. I.S.V.D. of copper in Bench 12 E (Gortdrum deposit)
Fig. 3.14 I.S.V.D. of copper in Bench 11 W (Gortdrum deposit)
Fig. 3.16 I.S.V.D. In residuals from a 4th order regression for copper in Bench 11 W - Gortdrum deposit.

Lag = 10 ft  No. of sectors = 18
Base contour (1) = 0.05
Contour interval = 0.05
LAG = 10  NO. OF SECTORS = 18
BASE CONTOUR = 0.10  CONTOUR INTERVAL = 0.10

FIG. 3.17.-ISO-SEMIVARIANCE MAP OF
4TH ORDER REGRESSED COPPER VALUES
(BENCH 11 WEST)
appropriate I.S.V.D. (Fig. 3.17) with the raw datas diagram (Fig. 3.14).

If the structural analysis is carried out on a Geostatistical fashion the need for Universal Kriging would appear obvious. However, it has been shown that the drift is only present in the NW direction.

The application of the local models procedure to this bench has been done using a sill value of 0.5 (curve 5 in fig. 3.14). The ranges and nugget effects were calculated in the same manner as it was explained above for the porphyry copper example. The data set consists of 412 copper values from samples located in the bench as is shown in Fig. 3.18. The values in each blast hole were obtained by analysing cores of 25 feet long. So, the sampled area was divided into blocks of 20 x 25 x 25 ft. and an arbitrary specific gravity of 1 was used for testing the method and producing an ore reserve estimation. The results are displayed in figures 3.19 and 3.20 and tables 3.5 and 3.6.

It can be observed from fig. 3.19 that, in this case, standard errors above 10% are mainly related to edge effects. The very high (greater than the largest sample value) and negative estimated grades appear also related to edge effects or to blocks for which the considered data points sub-set largely lie on one side of the block. Therefore, the technique is more reliable when spatial distribution of sampling is adequate, i.e. surrounding the block to be estimated. This is however the most common situation at the mining stage.

3.4.3. Los Pijiguaos Bauxite deposit.

A data set and appropriate information from the Los Pijiguaos deposit has been kindly made available, under strict confidentiality, by A. Sarmentero, Geologist of the Corporation Venezolana de Guayana (Bauxiven). This bauxite deposit is located in the Bolivar State of Venezuela, at about 450 km due south from Caracas and 500km SW of Ciudad Guayana (fig. 3.21).
** INPUT PARAMETERS **

TONNAGE FACTOR = 1.1617x10^3  
BLOCK THICKNESS = 1.76235x10^1  
LOWEST GRADE = 1.00009x10^2  
HIGHEST GRADE = 4.00009x10^1  
MINIMUM STANDARD ERROR = 0  
MAXIMUM STANDARD ERROR = 1.10000x10^0  
SPAN CLASS INTERVAL = 1.0000E+00  
STANDARD ERROR CLASS INTERVAL = 1.0000E-01  

** RESULTS **

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<td>0.100</td>
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<td>2.100</td>
<td>2.100</td>
</tr>
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<td>1.040</td>
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<td>0.850*</td>
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<td>0.840</td>
<td>0.560</td>
<td>0.440*</td>
<td>0.040</td>
</tr>
</tbody>
</table>

MINIMUM GRADE = 0.010
MAXIMUM GRADE = 4.000
MAXIMUM STD. ERROR = 0.100

Fig. 3.19  BLOCK GRADE ESTIMATION OF COPPER IN BENCH 11 W (GORTORUM DEPOSIT)
Fig. 3.20
The deposit is geologically located within a set of intrusives of the Guayana shield (fig. 3.22). Two main lithological groups are defined in the area:

- the Cuchivero Group, at the base, consists of quartz-latitic and rhyollitic tuffaceous metalavas of the Caicara formation and intrusive granites of San Pedro and Santa Rosalia.

- the "upper" group consists of the Parguaza and Los Pijiguaos granites.

Rb/Sr age determinations dates the Cuchivero group with ages varying from 1500 to 2000 million years and the upper group between 151 ± 39 million years.

The Parguaza batholith is considered the parent rock (Besozzi, 1978) of the bauxite deposit. The granite shows a rapakivi texture and presents the following mineralogical composition: large potasic felspar (pertite-oligoclase) crystals, quartz, biotite, muscovite and hornblende. Apatite, magnetite, ilmenite and sphene are the most important accessory minerals. The average (in %) chemical composition is:

\[
\begin{align*}
\text{SiO}_2 & \quad 69.1 & \text{Fe}_2\text{O}_3 & \quad 2.9 & \text{Na}_2\text{O} & \quad 3.4 \\
\text{Al}_2\text{O}_3 & \quad 14.4 & \text{MnO} & \quad 0.1 & \text{K}_2\text{O} & \quad 5.1 \\
\text{TiO}_2 & \quad 0.6 & \text{MgO} & \quad 0.1 & \text{P}_2\text{O}_5 & \quad 0.2 \\
\text{P}_2\text{O}_5 & \quad 1.4 & \text{CaO} & \quad 2.0 \\
\end{align*}
\]

The bauxite ore has been originated as a result of in situ weathering of the granite in heavy rain and high temperature conditions, together with a good drainage that allowed the leaching of K, Na, Mg, Ca and reactive silica. The ore consists of: gibbsite, quartz, kaolinite, hematite and anatase.

The bauxitization process has been developed mainly at the top of plateaux with an average altitude of 600 metres above sea level. Four zones were vertically distinguished at the exploration stage:

- top soil: an almost continuous layer with a mean thickness of 0.5 metres.
- bauxite crust: a discontinuous hard layer, more than 8 metres thick in
PHOTOGEOLOGY OF THE RIO SUAPURE AREA
Realised by Codesur 1971

Legend:
Lithological units
- Q Quarternary
- Cr Cinaruco Formation
- Rfm Roraima Formation
- SPg Parguaza Granite
- Csg Santa Rosalia Granite
- Ccfm Caicara Formation

Structural symbols
- Geological contact
- Fault
- Discordance
- Syncline
- Anticline

Fig. 3.22. Geological map of the vicinity of Los Pijiguaos Bauxite deposit.
certain areas. It consists of pisolitic gibbsite and goethite with some quartz relics.

- earthy bauxite: a soft, red coloured layer of fine grain gibbsite and iron oxides.

- micaceous clay: a yellow-reddish sericitic and kaolinitic layer that gradually passes to a highly altered zone with preserved original granitic texture.

In general, the bauxite layer has over ten metres thickness and a slope that rarely exceeds 10 degrees but the boundaries of plateau are very abrupt. It has been classified as a blanket type bauxite.

The available data set consists of average% of AL203 and SI02 in 10 metres drill core from 123 bore holes. The position of these holes appears in fig. 3.23. The drilling pattern obeys to restrictions imposed by the topography. The areas in between bore hole lines correspond to sections of eroded plateau. This pattern creates obvious difficulties in spatial data analysis that takes into account directional rates of change and therefore the I.S.V.D. for AL203 (fig. 3.24) and SI02 (fig. 3.25) do not show a well defined zone of internal correlation, indicating a high randomness of data's variability and its variance is almost entirely a nugget variance.

Different types of I.S.V.D. models were fitted to the experimental ones but no reliable local models results were obtained. This was also due to the random but strongly linear distribution of bore holes and high standard errors are obtained when fitting a surface of any particular order to a linear series of data points that lies outside or cross-cut a block.

Conventional kriging techniques were then applied using the trial and error approach to variogram modelling. The mean isotropic variograms that produced the smallest kriging variances for AL203 and SI02 are shown in figures 3.26 and 3.27.
Fig. 3.23. Drilling pattern and outline of Los Pijiguaos Bauxitic plateau.
Fig. 3.24. I.S.V.D. of Alumina in Los Pijiguaos Bauxite deposit.

Lag = 500m  No. of Sectors = 18
Base Contour (1) = 5 contour interval = 5
Fig. 3.25. I.S.V.D. of Silica in Los Pijiguaos Bauxite deposit.

Lag = 500m  No. of sectors = 18
Base contour (1) = 5  Contour interval = 5
The trial and error procedure consists on fitting initially a spherical scheme and to run the point Kriging program with the obtained variogram parameters. A second run is carried out with a new value of the sill equal to the first one multiplied by the ratio of the mean estimated variance to the mean Kriging variance obtained in the first run.

The aim is to lower the mean Kriging variance and to bring the mean algebraic and absolute differences between actual and estimated values as close to zero as possible. The ratio mentioned above is also adjusted to 1.

If the second run does not improve these optimal conditions of estimation, extra runs can then be attempted using values of the nugget effect slightly different from the value selected at the beginning of the procedure. In the case of $\text{Al}_2\text{O}_3$ and $\text{SiO}_2$, the selected models gave a minimum mean Kriging variance of 8 and 14 respectively. The ratio of the mean estimated variance to the mean Kriging variance was 1.2 in both estimations.

Once the variogram parameters have been chosen as the optimal ones, the block Kriging estimation is carried out. Usually, if the parameters have been selected properly the mean Kriging variances for the panel estimation is lower than in point Kriging.

The Bauxite deposit was divided into blocks of 250 X 250 m with a mean thickness of 10 m. The mean specific gravity of the material (as bulk density) has been reported as 1.5 gr/cm$^3$. The area of search for data points to be used in the block estimation was selected as a 1000 x 1000 m quadrangle and a minimum of 3 points was established as necessary to carry out the estimation of a block. This minimum is usually chosen as 6 in conventional Geostatistics but due to the very peculiar drilling pattern and being a global ore reserve estimation, the selected minimum was considered adequate in order to optimize the coverage of the drilled area.
Al₂O₃ - Avg. 10m

1 lag = 500m

Fig.3.26. Mean Isotropic variogram of Alumina. Los Pijijuaos Bauxite deposit.

SiO₂ - Avg. 10m

1 Lag = 500m

Fig.3.27. Mean Isotropic variogram of Silica. Los Pijijuaos Bauxite deposit.

N.B. The variogram models (solid line) have been obtained in both cases with the minimum Kriging variance criterion.
The shape of this area is extremely irregular and therefore its calculation by any integration or manual method would be a rather difficult and time consuming task. The selection of estimated blocks to be included in the ore reserve estimation was carried out with the following criterion: only those blocks with half or more of its area falling within the outlined Bauxite surface were included in the mineral inventory file. It is hoped that on average the amount of ore included in certain blocks that had part of their area outside the Bauxite boundary is balanced out with the blocks that were rejected for having less than half of their area within the ore zone. It should be emphasized that no local or pre-production evaluation is attempted. Therefore, this problem of misclassified reserves would be rectified with updated information to give local estimation as the deposit is mined.

The global estimation results are shown in Tables 3.7 and 3.8 and Figs. 3.28 and 3.29. It is noticeable from these results the severe smoothing effect of the Kriging technique in as much as the variance of the estimated grades is 3.67 compared to 10.5 which is the variance of the raw data. Their respective mean value is, as suspected, very similar i.e. 46.94 and 47.07 and the global tonnage is slightly greater than 200 million tonnes.

Due to the restrictions imposed by the selected area of search and minimum number of points to estimate a block, certain part of the drilled ore body have not been included in the ore reserve estimation. Therefore, it is clear that no over valuation has been introduced and the estimated tonnage is likely to increase once a more detailed drilling is carried out.

However, the 'quality' of a Bauxitic ore not only depends on the Al₂O₃ content but also on the SiO₂ concentration (Marechal and Roullier, 1970). The lower the % of silica the better the Bauxite is regarding processing techniques: when the Bayer method is used to separate alumina, the two simplified reactions are:

\[
\text{Al}_2\text{O}_3 + 2\text{NaOH} \rightarrow 2\text{AlO}_2 + \text{NaOH} + \text{H}_2\text{O}
\]
\[
5\text{SiO}_2 + 3\text{Al}_2\text{O}_3 + 6\text{NaOH} \rightarrow 5\text{SiO}_2\cdot 6\text{AlO}_2\cdot \text{Na}, 2\text{H}_2\text{O} + \text{H}_2\text{O}
\]  
(2)

In reaction (1) only a small part of the NaOH is consumed but in reaction (2), an insoluble silico-aluminate of NaOH is formed and it consumes most of the NaOH and Al\textsubscript{2}O\textsubscript{3}.

Block Kriging was then carried out for SiO\textsubscript{2} in the very same manner than for Al\textsubscript{2}O\textsubscript{3}. The obtained values for the mean Kriging variance and mean of estimated grades are 6.5 and 11.5 respectively. In order to examine in detail the spatial distribution of the blocks of different Bauxite quality, a new calculation of tonnages and maps were produced (Tables 3.9 and 3.10 and Figs. 3.30 and 3.31) for various ranges of silica content. These calculations were made for Bauxite containing less than 15 and less than 10\% of silica.

The behaviour of the alumina and silica content in metre intervals from the surface down to metre 10 is summarized in the statistical and geostatistical Table 3.11. It can be observed in this table that the quality of the Bauxite gradually decreases in depth and from the mean content in Al\textsubscript{2}O\textsubscript{3} several zones can be distinguished. The top 5 metres present a similar level of alumina concentration, with an average of 49.3\%. The 6th and 7th metres are also similar, showing a mean value of about 47\%, the 8th metre seems to mark also a particular unit with an average of 45.2\%. The metres 9 and 10 are characterized by the lowest mean content of alumina (average of 43\%).

With regard to the SiO\textsubscript{2} content the same zones can be distinguished and perhaps the top one could be divided into two: (A) the top 3 metres, with an average of 5.5\% and (B) the 4th and 5th metres which contain about 8\%. Again, the metres 6 and 7 are similar with an average of 12.5\%. This concentration notably rises to 16.3 in the 8th metre and to 19 in the metres 9 and 10.
Fig. 3.28. Centres of 250x250 m Kriged blocks for Alumina.
** INPUT PARAMETERS **

NAGE FACTOR = .9375J+05  BLOCK THICKNESS = .10000E+02
EST GRADE = .40000E+02  HIGHEST GRADE = .52000E+02
INUM KRIG. VARIANCE = .23000E+01  MAXIMUM KRIG. VARIANCE = .70000E+01
DE CLASS INTERVAL = .10000E+01  G. VARIANCE CLASS INTERVAL = .50000E+00

** RESULTS **

ESTIMATED GRADE = 46.943
KRIG. VARIANCE = 3.950
VARIANCE OF ESTIMATED GRADES = 3.67286
VARIANCE OF KRIG. VARIANCES = .49618

BAUXITE DEPOSIT (AL2O3)

Table 3.7

** GRADE-TONNAGE TABLE **

<table>
<thead>
<tr>
<th>GRADE CLASS</th>
<th>TONS IN CLASS</th>
<th>TONS OF METAL</th>
<th>CUMMULATED TONNAGE</th>
<th>MEAN GRADE</th>
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BAUXITE DEPOSIT (AL2O3)

Table 3.8

** KRIG. VARIANCE-TONNAGE TABLE **

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<th>MEAN GRADE</th>
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Fig. 3.30.A. Centres of 250x250 m Kriged blocks of Al₂O₃ (Less than 15% SiO₂)
** INPUT PARAMETERS **

- NNIAGE FACTOR = 0.9375E+00
- BLOCK THICKNESS = 0.1000E+02
- WEST GRADE = 0.4000E+02
- HIGHEST GRADE = 0.3200E+02
- MINIMUM KRIG. VARIANCE = 0.2300E+01
- MAXIMUM KRIG. VARIANCE = 0.7900E+01
- GRADE CLASS INTERVAL = 0.1000E+01
- VARIANCE CLASS INTERVAL = 0.5000E+00

** RESULTS **

- AN ESTIMATED GRADE = 47.270
- AN KRIG. VARIANCE = 3.965
- VARIANCE OF ESTIMATED GRADES = 2.70246
- VARIANCE OF KRIG. VARIANCES = 0.51643

** GRADE-TONNAGE TABLE **

<table>
<thead>
<tr>
<th>GRADE CLASS</th>
<th>TONS IN CLASS</th>
<th>TONS OF METAL</th>
<th>CUMULATED TONNAGE</th>
<th>MEAN GRADE</th>
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<td>46.745</td>
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<td>5044.32</td>
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</table>

** KRIG. VARIANCE-TONNAGE TABLE **

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<th>MEAN GRADE</th>
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</tr>
</tbody>
</table>

Table 3.9

** AL2O3 (LESS THAN 15% SI02) **

Table 3.9

** AL2O3 (LESS THAN 15% SI02) **

** KRIG. VARIANCE-TONNAGE TABLE **

Table 3.9
Fig. 3.30.B. Grade and Kriging variance - tonnage curves of Al$_2$O$_3$ (less than 15% SiO$_2$). Los Pijigous Bauxite deposit
Al₂O₃ (Less than 10% SiO₂)

**INPUT PARAMETERS**

- Tonnage factor = .93750E+05
- Block Thickness = .10000E+02
- Lowest Grade = .46000E+02
- Highest Grade = .52000E+02
- Minimum Krig. Variance = .20000E+01
- Maximum Krig. Variance = .50000E+00
- Grade Class Interval = .50000E+00
- Krig. Variance Class Interval = .20000E+00

**RESULTS**

- Mean Estimated Grade = 48.606
- Mean Krig. Variance = 3.709
- Variance of Estimated Grades = 2.48985
- Variance of Krig. Variances = .34132

**Grade-Tonnage Table**

<table>
<thead>
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<th>Tons in Class</th>
<th>Tons of Metal</th>
<th>Commulated Tonnage</th>
<th>Mean Grade</th>
</tr>
</thead>
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<td>0</td>
</tr>
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<td>1875000.0</td>
<td>51.560</td>
</tr>
<tr>
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<td>4667500.0</td>
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<td>6562500.0</td>
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</tr>
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**Krig. Variance-Tonnage Table**

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Fig. 3.31.A. Block Kriging estimation and Kriging variances of Al₂O₃ (less than 10% SiO₂).
Fig. 3.31.B. Grade and Kriging variance-tonnage curves of Al₂O₃ (less than 10% SiO₂).
It is also very noticeable for both alumina and silica the increase on the spread of the distribution below the metre 5th in as much as the standard deviation together with the range of values increase. The absolute degree of skewness on the contrary decreases below the above mentioned depth and in particular the metres 8, 9 and 10 show very similar values.

From the geostatistical point of view it appears clear that the silica presents a much better spatial correlation because only on the metres 6 and 7 the variogram is random, in comparison with the metres 2, 5, 9 and 10 for alumina. Also, the values of C in the structured data of any particular metre is higher for silica as well as the size of the zone of internal correlation (A).

The proportional effect is well defined, in particular for silica, because the value of Co+C increases with the mean content.

A certain amount of drift (see appropriate variograms in the Appendix) appears at about a lag of 7000 m (Lag 14) in the metres 6, 9 and 10 for alumina. Also, a compound spherical scheme may be present in metres 3, 6 and 8.

For silica the variograms of metres 6, 7, 9 and 10 also show the presence of certain amount of drift at about the same lag as in the case of alumina. Compound spherical schemes could be present in metres 3, 8, 9 and 10.

When comparing contents of both alumina and silica by calculating correlation coefficients and regression lines, it appears again the top 5 metres as having fairly similar values of these parameters. This can be observed for metres 8, 9 and 10. A transition would appear to be marked by the metres 6 and 7. These relationships are better illustrated in the scatter diagrams presented in Figs. 3.32. to 3.37.

When the statistical and geostatistical parameters are considered together, the deposit can be divided into a top layer of 5 metres, a transition layer comprising the metres 6 and 7 and the bottom layer which
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**Key to Table**

M = mean  
S = standard deviation  
Range = minimum and maximum values  
SKW = degree of skewness  
Co = nugget effect  
C = sill  
A = range (in metres)  
CC = correlation coefficient  
Reg.Lin. = regression line  
l = % Al₂O₃

Table 3.11. Statistical and geostatistical parameters of Alumina and silica on one metre interval basis.
consists of the metres 8, 9 and 10.

3.5 Discussion and conclusions

The locally fitted models procedure takes into account the directional rate of change of spatial variability by using the calculated semivariance function, and estimation is carried out considering only the points that are spatially correlated. The degree of anisotropy is thus taken into account. The iso-semivariance diagram permits direct estimation of the sill, range and nugget effect values in any particular direction. Therefore, the problem of subjectivity, largely unavoidable so far in variogram fitting, is surmounted.

In addition, the clear advantage of the local model procedure is that further ore reserve estimation of sublocks located anywhere in the ore body is easily carried out by using the stored coefficients. Then, its usefulness is perhaps more readily acquired for production scheduling and blending. This is not however the case of Kriging. In this case no record is kept, remodelling can only be achieved by repeating the whole procedure - a very costly process.

As the directional spatial variability is of such great importance in prediction using this model, there is a limitation on the reliability of such predictions at the edges of the sampled area or in zones of inadequate (high clustering or irregular linearity) areal distribution of sample points. When data is arranged in peculiar sampling grids or no suitable iso-semivariance diagrams are obtained, as for instance the Bauxite deposit, conventional Kriging techniques are preferred despite its sometimes ambiguous and time consuming variogram modelling. A kriged estimated will always be produced provided there is a specified number of data points available irrespective of their position in relation to the point or block to be kriged. The kriging variances will mainly give an idea of how good the variogram parameters perform in the estimation although the fitted model may not bear resemblance whatsoever with the experimental
variogram.

Global ore reserve estimation of the Bauxite deposit indicates promise, in terms of tonnage and grades. Considered alongside for instance, ore shipped to the U.S.A. where a great part of the Bauxite used in the chemical industry exceeds 15% silica, and considering the world commercial Bauxites assay ranges from 35 to 50% alumina, this deposit appears worthy of detailed investigation.

The classification of the reserves in terms of the content of silica allowed depiction of areas of better quality Bauxite (Fig.3.31) in order to select a priority detailed drilling campaign and perhaps to establish the basis for a selective mining. The statistical and geostatistical analysis of the deposit in a metre interval allowed to distinguish 3 zones that group the top 5 metres in the first zone, the 6th and 7th metres in the second zone and the metres 8, 9 and 10 as the third zone. When local or preproduction estimation is carried out, this clear definition of the zones should be taken into consideration in as much as the quality of the Bauxite notably decreases below the first zone.
Fig. 3.33. Scatter diagrams of Silica/Alumina in metres. 3 and 4
Fig. 3.34. Scatter diagram of Silica/Alumina in metres 5 and 6.
Fig. 3.5 Scatter diagrams of Silica/Alumina in metres, 7 and 8.
Fig. 3.36. Scatter diagrams of Silica/Alumina in metres 9 and 10.
Fig. 3.37. Scatter diagram of Silica/alumina of average content in 1Qm.
CHAPTER 4

4.1. Introduction

In the search for mineral resources there is an implicit and well-defined objective: their profitable exploitation. This is the aim of either the entrepreneur, shareholders of a private corporation or any Government institution. In the latter case there might be a particular social objective but as a whole the resources are to be tapped and exploited to the full. Obviously, a fundamental philosophical confrontation arises between the theory of exhaustion which states that the present value of exploitation should be maximized and the conservationist-theory arguing that the free market system involved neglects the need of future generations. This apparent antagonism has thoroughly been discussed by Herfindahl (1962) who conclusively remarks that "the goal of society is not to leave future generations enough minerals simply to ensure that they have minerals, but rather to leave them in a position to enjoy adequate incomes. Mineral usage is therefore related to a more general problem of the optimal saving of society."

Moreover, only a society that completely deprives its members from enjoyment of modern facilities would be considered strictly conservationist. Otherwise it would cause exhaustion at an international level when buying mineral commodities from other countries.

There is on the other hand the interrelationship between technology, price level and size of the reserves. For instance, at the beginning of the 1950's the cut-off grade (Niskanen, 1975) in the U.S.A. was about 1% in copper ores and the reserves were reported to be 23 million tonnes of copper. Now, nearly 30 years later the cut-off grade is about 0.5% and the reserves are 75 millions tonnes (including new deposits). Therefore the evolution of know-how together with changing patterns of the dynamics of world economy and increasing demand for mineral supplies makes it
feasible to mine in a profitable manner, continuously lower and lower grades, preventing then the waste of reserves.

Within the context of exhaustion, the ideal model would call for an exploitation of resources using the best possible technology available at the time in order to maximize benefits. Having established adequately the natural parameters (geological control, grade variation and reserves) of a mining prospect, the possibility of it becoming a mine will have to be assessed then on the light of the available technology and considering a variety of economic factors.

It is the purpose to present in this Chapter a brief review of different methods and specific measuring indicators that are used for appraising investment and in particular of mining ventures, together with an application of the probability and sensitivity methods for economic valuation of "Los Pijiguaos" Bauxite deposit.

4.2. Financial appraisal of investment

Investment is strictly defined (Bannock et al, 1979) as the expenditure on real capital goods. It is also taken to mean purchase of any asset or undertaking any commitment which involves an initial sacrifice followed by subsequent benefits. Therefore, the appraisal of investment would be the analysis of the prospective costs and benefits derived from investments, and the evaluation of the desirability of committing resources to them.

The appraisal of investment must consider two major features:

I) The cost and benefits of an investment occur at different points in time. Therefore, comparison between cost and benefits must be carried out in relation to the period of time they occur. (The time value problem).

II) Usually the benefits and sometimes part of the costs occur in certain time in the future. This implies a factor of uncertainty in as much as parameters that affect the outcome of an investment are capable of taking
a number of forms. (The risk problem).

It is then assumed that a certain amount of money or goods is available for investment. If this is not the case, the required capital can be raised by borrowing from a bank, having a corporate funded debt, offering of bonds or debentures or selling of future production. In any case, the important fact is that money has the power of earning interest.

The concept of interest or a required rate of return is used throughout the financial world. Broadly speaking interest is the rent paid for the use of someone-else's money. The required rate of return would then be the amount of money that an investment manager needs to earn in order to justify expending in a new project.

In the early tradition of classical theory, e.g. Adam Smith, David Ricardo (in Bannock op.cit.) the rate of interest was regarded as simply the rate of return on capital invested. It was considered to be an income to capital rather like rent to land. With the subsequent development of the classical system, the nature and the determinants of the rate of interest came to be regarded in terms of a more complex pattern. The rate was arrived at by interaction of two forces operating on supply of, and the demand for, funds.

Due to its interest earning power money is worth more in the future. Alternatively, a future sum of money must be discounted at the interest rate to determine it's value today (present value). This process of determining the present value of future sums is termed discounting.

Three factors determine the amount of interest that is paid for the use of a sum of money. These are the amount of money in use (principal or present value or worth), the length of time that the money is used and the interest rate. The two basic methods for charging interest are termed "simple" and "compound". The latter is most widely used and forms the basis of modern discounting methods for investment appraisal.

The compound interest formula is given by:

\[ FV = PV(1+R)^n \]

(4.1.)
where \( FV \) = The amount of money at the end of \( n \) compounding periods
(sometimes called the "compound amount").

\( FV \) = The initial sum of money invested
\( R \) = Rate of interest applied to the given period
\( n \) = The number of compounding periods (usually measured in years)

For instance, $1 available to-day can increase to $1.10 in one year
if the interest rate is 10 per cent per year. The same amount would increase
to $1.21 in a further year and $(1+0.1)^n in \( n \) years.

4.2.1. Determinative economic parameters

In investment appraisal it is necessary to calculate several economic
parameters that help to choose amongst different investment proposals or
assess expected profits. If equation (4.1) is rearranged, the present value
of a future sum is given by:

\[
PV = \frac{FV}{(1+R)^n}
\]  

(4.2)

The factor \( \frac{1}{(1+R)^n} \) is called the "single payment present-worth factor"
(Stermole, 1976) and appropriate tables of compound interest formulas give
the values of that factor.

The present value of a future sum of money represents the current
exchange value, or price of that future sum, given the market interest rate
at which it is calculated. In other words, the present value of money in
the future is calculated by discounting it at a rate of interest equivalent
to the rate at which it could be invested. An obvious implication is that
the present value of a future sum is less, the further away in time the sum
is due.

When comparing revenues over costs it is, however, incorrect to simply
sum the net money flows over the life of the project and then subtract the
capital expenditure. It is necessary instead to take into account the time
value of money by converting the money flows into values that correspond to
the same time, so the net value of the investment can be assessed. This is
obtained by calculating the sum of the present values of the money flows. Such a sum is known as the Net Present Value (NPV) of the investment:

\[
NPV = \frac{F_1}{(1+R)} + \frac{F_2}{(1+R)^2} + \frac{F_3}{(1+R)^3} + \cdots + \frac{F_n}{(1+R)^n} - I \tag{4.3}
\]

where
- \( F_1 \) = Cash flow stream in year 1
- \( I \) = Capital investment
- \( R \) = Rate of interest

Then, the NPV is merely the sum of the net cash flows for every year in the life of the project, after being discounted at the specified rate. It includes, in a mining venture, present values of exploration and investment outlay made prior to production as well as the present values of the cash flows from operations.

The NPV is therefore a measure of the profitability of an investment in which the time value of money has been allowed for and it may be used as an indicator of project worth. A positive NPV signifies that the investment expenditures will earn a higher return than that specified by the discount rate. If the NPV is equal to zero the investment will earn the exact return specified by the discount rate. On the other hand, if the value of NPV is negative the investment does not earn as large a return as specified by the discount rate. It follows that the NPV is a valuable decision criterion for independent investment: a positive NPV indicates an acceptable project, a NPV of zero indicates a marginal project and a negative NPV indicates an acceptable project. If a set of mutually exclusive investments are to be appraised it is clear that, if maximisation of profit is the objective, the projects should be ranked in order of NPV. The project with largest NPV will be chosen, provided of course that its NPV is positive.

It should be pointed out that although, in general, both public and private sector decision makers work on the basis of a long term interest rate which is constant overtime, this may well not be the case of a particular venture. The calculation of the NPV will then be affected as
follows:

\[
NPV = \frac{F_1}{(1+R_1)} + \frac{F_2}{(1+R_1)(1+R_2)} + \cdots \frac{F_n}{(1+R_1)(1+R_2)\cdots(1+R_n)} - I \tag{4.4}
\]

where \(R_1\) to \(R_n\) are the different rates acting during the appropriate periods.

As it can be seen in equation (4.4), the adaptation of the NPV to cope with varying rates of interest with time is relatively straightforward but it is much more complex when those rates have to be forecasted.

There is another economic parameter that also measures the profitability of an investment. It is called the "Internal Rate of Return" (IRR) and it has been defined as the rate of interest which, when used to discount the money flows will set the NPV exactly to zero. This is mathematically equivalent to select the value of \(R\) such that:

\[
NPV = \frac{F_1}{(1+R)} + \frac{F_2}{(1+R)^2} + \cdots \frac{F_n}{(1+R)^n} - I = 0 \tag{4.5}
\]

In order for an investment to be attractive it must generate sufficient positive cash flows to repay the use of the money. So, the IRR can be taken as a measure of the return on the outstanding balance of the investment for each period during the life of the project. If the IRR is less than the actual market rate the project will not earn the required return and therefore is unprofitable, whereas if the IRR is greater, the project will be profitable and will earn a better return than required. When assessing mutually exclusive investments, the project with largest IRR will be chosen, provided again this is greater than the market rate of interest.

According to Bannock (op.cit.) the IRR as a measure of investment profitability suffers from two important defects: (A) a given project may have more than one interest rate which will discount its cash flows to zero, and so the method may not give a clear-cut answer. This can happen when cash inflows (profits) within the lifetime of the project are
followed by cash outflows, and (B) the method may give incorrect rankings of alternative projects in that the actual profitability of one project may be greater than that of another even though the internal rate of return is lower.

When assessing a single investment the project may present this ambiguity as is illustrated in Fig. 4.1. It can be seen in this figure that there are two (II and I2) IRR satisfying NPV equal to zero. If the market rate of interest is below II the IRR criterion would recommend though it has a negative NPV. If the market rate lies between II and I2, despite the project having a positive NPV, it would be accepted regarding I2 but unacceptable in judging from II. This situation may not appear too ambiguous when the details or circumstances of cash flows are known. For instance, the case some times arises in mining ventures when production stops due to reparation or renovation of equipment, delays in selling the product due to market undesirability or external political upheavals. The operation will have then negative cash flows followed by positive ones.

It the above mentioned ambiguity is not present the curve of NPV-rate of interest of a single investment may look like Fig. 4.2., and for mutually exclusive investments as is shown in Fig. 4.3.

Conflict between NPV and IRR when assessing mutually exclusive investments may occur when such projects have different sensitivities to the discount rate. Their NPV curves will have different slopes and intersect each other for a positive NPV. The situation usually arises when the receipts-time pattern of the two projects are different. In general, the greater the proportion of net receipts occurring later in time, the more sensitive the project will be to the discount rate, i.e. the steeper will be its NPV curve. The common objective of most projects is the maximisation of profits and the market rate of interest the factor which determines the time value of money. The NPV is usually regarded as the correct criterion for optimal decision making when assessing investments with same life and capital expenditure.
Fig. 4.1 Ambiguity of the IRR as a decision criterion.

Fig. 4.2. NPV - rate of interest curve \( f \) with no ambiguity i.e. no conflict between NPV and IRR.

Fig. 4.3. NPV - rate of interest curves for a non-conflictive situation of two (A and B) mutually exclusive projects.
Assuming no conflict between NPV and IRR, and with the idea of avoiding time consuming iteration to calculate IRR, a rough approximation can be achieved (Whitney and Whitney, 1979) by dividing the total investment expenditures by the average annual cash flow and dividing the results into 0.7, i.e.:

\[
(0.7) \div \text{(investment expenditures)} / \text{(average cash flow)} = \text{estimated IRR}.
\]

The rough estimation is not sufficiently accurate for evaluative purposes but it is a useful starting point for subsequent trial and error refinement.

The Profitability Index (PIX) is another determinative economic parameter that is usually calculated in investment appraisal. It is also known with the name of benefit-cost ratio and is equivalent to the percent value of cash flows subsequent to the capital investment, divided by that investment:

\[
\text{PIX} = \frac{\text{PV}}{\text{I}}
\]

where \( \text{PV} \) = present value of cash flows subsequent to \( I \) (initial capital investment) being \( \text{NPV} = \text{PV} - I \), then it follows:

\[
\text{PIX} = \frac{\text{NPV}}{\text{I}} + 1
\]

If \( \text{NPV} = 0 \), then \( \text{PIX} = 1 \). Therefore, a project will be accepted when PIX is greater than \( 1 \) and rejected when it is less than \( 1 \). In general, the higher the PIX, the more desirable the project is. However, when assessing investment of projects with different lives the PIX presents the same drawbacks than the NPV.

The Wealth Growth Rate (WGR) is also used in investment appraisal. It is defined as the compound interest rate of investment growth. In other words, the WGR is the interest rate which will equate the future value of the initial investment to the future value of the subsequent cash flows which are re-invested at the rate of investment established by the company. It is normally assumed that the minimum rate of return reflects the other opportunities that exist for the investment of capital, so that minimum rate
will be used in calculating the Wealth Growth Rate. It has arrived at with the development of the Growth Theory, a highly abstract branch of economics.

The WGR is nevertheless very useful in appraising projects with the same profitability index but different lives. This can be deduced from the following equation:

\[
WGR = R + \left( \log_e(P) \right) / n
\]  

(4.6)

where \( R \) is the discount rate and \( n \) the number of years of the project. Therefore, from the two projects, the shortest lived would yield a higher WGR.

4.2.2. Controlling economic factors

Investment appraisal has been so far described in this Chapter within a very simple scheme defined by the calculation of four parameters. There is however a large number of controlling factors that affect the objective economic assessment of projects. One of the most relevant controlling factors is perhaps inflation. It has been defined as the process of steadily rising prices, resulting in diminishing purchasing power of a given nominal sum of money. One of the more accepted explanations of inflation is that governments in all countries around the world are spending more money each year than they receive in taxes, royalties or other revenues.

There is nevertheless an abundant literature on the subject with some "magic" solutions proposed, but inflation is a real problem to account for somehow when appraising projects.

A simple and straightforward method of correcting for inflation is discounting with the inflation rate. Recalling from equation 4.3.:

\[
F'1 = \frac{F1}{(1+P)}; \quad F'2 = \frac{F2}{(1+P)^2}
\]

in general \( F'n = \frac{F_n}{(1+P)^n} \)
where \( F'n \) = cash flow stream corrected for inflation rate \( P \).

Then the new \( NPV \) would be:

\[
NPV' = \frac{F'1}{(1+R)} + \frac{F'2}{(1+R)^2} + \cdots + \frac{F'n}{(1+R)^n} - I
\]  

(4.7)

this is equivalent to:

\[
NPV' = \frac{F1}{(1+P+R)} + \frac{F2}{(1+P+R)^2} + \cdots + \frac{Fn}{(1+P+R)^n} - I
\]  

(4.8)

Hosking and Green (1975) proposed several methods of inflation treatment in mining ventures:

I) Estimate costs and revenues in Dollar values at the date of the study.

It assumes:

A) Future costs and revenues inflate at same rate.

B) Capital costs quoted are less than will actually be required.

Drawbacks:

Underestimates taxes, as it ignores decline in worth of depreciation.

This method is rather unrealistic for most mining investment evaluations due to their long start-up lead times and high capital costs (Whitney, op.cit.).

II) Inflate all variables to the value expected at the start of production and thereafter use constant Dollar values as of that date.

This procedure has the advantage over I) in as much as the capital costs discrepancy is eliminated. However, the tax problems remain, e.g. still underestimates taxes. Nevertheless, the method is acceptable for preliminary financial analysis of mining projects as it is relatively simple to carry out in a reasonably realistic assessment of cash flows of the first years of the operation at least.

III) Assume a constant rate of inflation for each variable and inflate each of them at the specified rate. This implies that the internal rate of return will have an inflation rate built in. The method is being accepted as a final feasibility analysis even though it may not be very
useful in the early studies. The principle behind this method is implicit in equation (4.8).

It is obvious that the future rate of inflation is not known. There are however, no ways of forecasting price changes accurately. This does not mean we should ignore inflation, which would be equivalent to assume inflation will be zero. Allowances should be made for alternative changes in the inflation rate and attempt to hedge against them. Reference has been made above to taxes. Taxation is another controlling factor to be considered in financial analysis. In general terms, it represents a compulsory transfer of money (or occasionally of goods and services) from private individuals, institutions or groups to the government. It is then through taxation that governments finance their expenditure.

In theory, investment proposals could equally be compared by considering returns either before or after taxes have been deducted. However, the economic feasibility of a project must be studied on an after tax basis. Anyone who works for profit needs to be aware of tax concepts in order to determine how they will effect the timing in generated cash flows. A tax planning procedure requires the legitimate use of all current laws, regulations and court decisions to reduce taxable income. Every country has its own taxation laws for mining ventures. A detailed and useful account of tax planning is presented by Whitney and Whitney (1979) regarding mining activities in U.S.A.

There are, however, certain tax allowances which are common to several mineral taxation laws. For instance, depreciation, defined as the periodic write-off of the cost of items of property and other long lived assets, is sometimes accepted for tax purposes as a charge against taxable income. This is permitted on the grounds that the consumption of capital assets is one of the costs of earning the revenues. Since depreciation can only be accurately measured at the end of the life of an asset, depreciation provisions require an estimation both of the total
amount of depreciation and the asset life.

There are basically 3 methods of calculating annual depreciation: the Straight Line (SL), Double Declining Balance (DDB) and the Sum of the Years Digits (SYD). SL is a simple method of general application and is calculated by dividing the difference between the initial cost of the item and its value at the end of operation (salvage value) by the estimated useful life.

DDB and SYD are more accelerated methods of depreciation and they permit a more rapid recovery of capitalized costs than the SL method. In general, accelerated depreciation implies larger deductions in the early years and progressively smaller in the later years. In the case of mining ventures, where heavy front end investment is required, it would be desirable to depreciate with DDB or SYD methods.

The DDB depreciation allowance for year n is calculated as follows: first, subtract the accumulated depreciation to date from the cost of the asset; multiply this difference by 2.0 and divide the result by the life of the item in years. However, the item should not be depreciated below its salvage value.

The SYD is calculated by applying a changing fraction (different depreciation rate) to the original cost of the asset (less estimated salvage value). Therefore, a varying rate each year is applied to a constant amount. Then, the depreciation allowance for year n is calculated by obtaining the cost of the asset minus the salvage value, this amount is multiplied by the number of years left in the life including the current year and the result divided by the sum of the years in the life of the item. In the mining environment taxable income can sometimes be reduced when depletion allowances are advisable. This type of allowance has been granted on the grounds that mineral deposits are wasting assets that are difficult to replace and that mining ventures are highly risky. This depletion allowance is normally granted under either the cost depletion or the percentage depletion methods. The cost depletion for any year will be
determined as follows: first the undepleted cost basis of the units of mineral remaining to be recovered at the end of the year plus the number of units sold during that year. In the percentage depletion method, the allowance is expressed as a percentage of the gross income from the mine. This allowance varies with the type of mineral mined. The advantage of this latter method is that no limitations are imposed by the cost basis of the property.

Certain taxation laws allow an investment tax credit as an incentive to encourage investment. The amount of this credit and qualified assets for it varies from country to country but if available the allowance should be included in the taxation planning.

4.2.3. Methods of investment appraisal.

There are several methods of appraising the investment and perhaps the most direct is the pay-back method. The objective is to calculate the period over which the cumulative net revenue from an investment project equals the original investment. However, the method does not take into account the profits over the whole life of the investment nor the time profile of the cash flow.

One of the most widely used methods of investment appraisal is the Discounted Cash Flow (DCF) return. In this method cash receipts are treated as consisting of interest plus the recovery of the initial capital and the DCF return may be defined as the true annual rate of return in the capital outstanding on the investment. The technique involves calculating the net of tax income, since it is that which companies want to maximize and also takes full account of concessions such as investment grants and accelerated depreciation. However, the DCF return is only as accurate as the cash flows it is derived from, since it appears necessary to forecast cash flows for many years in advance. Besides, the DCF return does not define when the original investment has been recovered, and this might be an important factor in high risk projects.
Uncertainty and risk are better accounted for with the sensitivity and probability methods of investment appraisal. With the sensitivity approach the determinative parameters mentioned in 4.2.1. are calculated by a successive trials using different values of those variables that are believed to be more influential in the global outcome of the project. With the probability approach, the variables are described not as point estimates but as probability density functions. Then, a series of computer simulations in which these distributions are sampled at random and economic parameters calculated will generate not a single value for this parameter but instead a particular probability density function. So the chance of a project having one type of outcome will be expressed in terms of its probability.

The Bauxite deposit evaluated in Chapter 3 is financially assessed with the probabilistic approach using the computer program Joleval, implemented and extensively modified by Ware (1978). The details of the appraisal are presented below.

4.3. Economic valuation of "Los Pijiguaos" Bauxite deposit

The Venezuelan Aluminium Industry has been so far fulfilling its Bauxite requirements through importation. The country's main sources and prices are shown in Table 4.1. Having established the resource potential of Los Pijiguaos deposit, the "Corporacion Venezolana de Guayana" commissioned the Alusuisse Company to carry out a more detailed study of the project regarding follow up drilling, local ore reserve estimation, mining and transport equipment, infrastructure workings and required personnel. A set of costs for some of these categories has been kindly made available under strict confidentiality by the Venezuelan Institution, and the economic modelling and financial valuation of the deposit is presented below.
<table>
<thead>
<tr>
<th>COUNTRY</th>
<th>F.O.B. Cost</th>
<th>Transport cost</th>
<th>C.I.F. cost (In Venez)</th>
</tr>
</thead>
<tbody>
<tr>
<td>SURINAME</td>
<td>33.53</td>
<td>13.96</td>
<td>47.49</td>
</tr>
<tr>
<td>(Bakhuis)</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>GUAYANA</td>
<td>31.90</td>
<td>7.30</td>
<td>39.20</td>
</tr>
<tr>
<td>(Guaybau)</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>AUSTRALIA</td>
<td>17.15</td>
<td>23.82</td>
<td>40.97</td>
</tr>
<tr>
<td>(Gove)</td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

Table 4.1. Production, transport and final cost of imported Bauxite from different countries to Venezuela.
4.3.1. Operating conditions

It is considered necessary to outline very briefly the operating conditions for the project. Recalling from the previous Chapter (Fig.3.21) the deposit is located in a rather isolated area with the Orinoco River as the only means of transport to the Bauxite reduction plants at Ciudad Guayana, 700 km down the river. Fortunately this part of the river is navigable from May to December. Therefore fluvial transport of the material has been proved feasible with push units of 4 barges (200 x 30m) carrying a total of 14000 tonnes. These barges would take 6 days for loading and unloading and 7 days for river transport operations. Stockpile facilities with capacity for at least two weeks supply at the barges loading terminal has been envisaged.

A conveyor belt of 34.5 km long would transport the Bauxite from the foot of the plateau to the river. This section is relatively flat, descending only 50 m. The specification of this belt is as follows:

- **Material** .............. Bauxite 0-100mm
- **Capacity** .............. 1250 t/h
- **Belt Width** ............. 36'
- **Belt Speed** ............. 3.5 m/s
- **Sections** .............. 6, each of 5.75 km.
- **Belting** .............. Steelcord
- **Energy required** .......... 5400 Kw.

A second conveyor belt 7.75 km long down hill from the crushing area to the foot of the plateau will load the 34 km conveyor belt system.

After several mining tests, carried out by the consulting company, it was demonstrated that the ore can be stripped without explosive using only digging force i.e. hydraulic front loader with front bucket of 3.0m³ capacity. However, the sparse vegetation and the thin (0-80 cm) layer of overburden will have to be removed by bulldozers equipped with a blade.

The ore will be hauled for about 3 to 6 km to the crusher by mine
dumpers of 35 tonnes capacity. A Hazemag impact crusher with average output of 1000 t/h has been recommended. The total list of mining equipment and operating costs per ton for 3 alternatives of production is presented in Tables 4.2 and 4.3.

The project would operate with a personnel of about 450, 700 and 800 employees for the 3, 5 and 6 million tonnes production rates. Therefore, living facilities and leisure spaces for these populations and their associated families would have to be built together with offices, mechanical workshops, storage, spares and laboratory premises.

4.3.2. Financial modelling

The three alternatives of production has been firstly considered separated with their items of capital investment in mining equipment and operating costs. A period of six years (1976-1982) has been established as an exploration and preproduction stage in an annual capital expenditure of $27325581 has been reported. A production life of 20 years has been taken as the basis for the appraisal. A total of 250 simulation per computer run has been considered adequate to generate confident economic parameters. The Corporacion Venezolana is a government institution that does not pay taxes and therefore no tax credit, depreciation or depletion allowances have been declared in the analysis.

In addition, the deposit is located on National Trust territory, so property value of the land is being set to zero. A 10% yield of investment has arbitrarily been chosen to carry out the appraisal for the net present value calculation in all cases. No specific density functions has been made available for the variables. They, when appropriate, have been described as normal distributions by given a minimum and a maximum value. For instance, the grade of alumina has been defined according to the block Kriging results. i.e. a minimum of 42 and maximum of 51%. The recovery probability distribution has been defined with a
<table>
<thead>
<tr>
<th>Category</th>
<th>3mt/y $</th>
<th>3mt/y No.</th>
<th>5mt/y $</th>
<th>5mt/y No.</th>
<th>6mt/y $</th>
<th>6mt/y No.</th>
</tr>
</thead>
<tbody>
<tr>
<td>Cat. 245</td>
<td>310,000</td>
<td>3</td>
<td>930,000</td>
<td>-</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>Lib. 981</td>
<td>1,100,000</td>
<td>-</td>
<td>2,200,000</td>
<td>2</td>
<td>2,200,000</td>
<td>2</td>
</tr>
<tr>
<td>Cat. 988</td>
<td>240,000</td>
<td>2</td>
<td>480,000</td>
<td>2</td>
<td>480,000</td>
<td>3</td>
</tr>
<tr>
<td>Cat. D9</td>
<td>280,000</td>
<td>3</td>
<td>840,000</td>
<td>4</td>
<td>1,120,000</td>
<td>5</td>
</tr>
<tr>
<td>Cat. 769</td>
<td>200,000</td>
<td>19</td>
<td>3,800,000</td>
<td>-</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>Cat. 777</td>
<td>390,000</td>
<td>-</td>
<td>4,680,000</td>
<td>12</td>
<td>4,680,000</td>
<td>12</td>
</tr>
<tr>
<td>Cat. 14G</td>
<td>130,000</td>
<td>2</td>
<td>260,000</td>
<td>2</td>
<td>260,000</td>
<td>3</td>
</tr>
<tr>
<td>Pick-up/Cars</td>
<td>10,000</td>
<td>18</td>
<td>180,000</td>
<td>20</td>
<td>200,000</td>
<td>22</td>
</tr>
<tr>
<td>Lorries/Bus</td>
<td>50,000</td>
<td>8</td>
<td>400,000</td>
<td>10</td>
<td>500,000</td>
<td>12</td>
</tr>
<tr>
<td>Drilling Machines</td>
<td>100,000</td>
<td>1</td>
<td>100,000</td>
<td>2</td>
<td>200,000</td>
<td>2</td>
</tr>
<tr>
<td>Miscellaneous</td>
<td>250,000</td>
<td>-</td>
<td>350,000</td>
<td>-</td>
<td>-</td>
<td>-</td>
</tr>
</tbody>
</table>

Total: 7,240,000 9,990,000 10,860,000

Spare parts 15%: 1,086,000 1,498,000 1,629,000

Total: 8,326,000 11,488,000 12,489,000

Table 4.2. List of Mining equipment and cost per item for three production rates. (Los Pijiquaos Bauxite deposit).
<table>
<thead>
<tr>
<th>Category</th>
<th>$/h</th>
<th>h/y</th>
<th>$/y</th>
<th>h/y</th>
<th>$/y</th>
<th>h/y</th>
<th>$/y</th>
<th>h/y</th>
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</thead>
<tbody>
<tr>
<td>Cat. 245</td>
<td>45</td>
<td>9000</td>
<td>405,000</td>
<td></td>
<td>6,000</td>
<td>600,000</td>
<td>8,000</td>
<td>800,000</td>
</tr>
<tr>
<td>Lib. 991</td>
<td>100</td>
<td></td>
<td></td>
<td>6,000</td>
<td>288,000</td>
<td>7,000</td>
<td>335,000</td>
<td></td>
</tr>
<tr>
<td>Cat. 988</td>
<td>48</td>
<td>5000</td>
<td>240,000</td>
<td>9,500</td>
<td>427,000</td>
<td>11,000</td>
<td>495,000</td>
<td></td>
</tr>
<tr>
<td>Cat. D9</td>
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<td>8000</td>
<td>360,000</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Cat. 769 C</td>
<td>32</td>
<td>45000</td>
<td>1,440,000</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Cat. 777</td>
<td>55</td>
<td></td>
<td></td>
<td>30,000</td>
<td>1650,000</td>
<td>40,000</td>
<td>2,200,000</td>
<td></td>
</tr>
<tr>
<td>Cat. 14G</td>
<td>22</td>
<td>4500</td>
<td>99,000</td>
<td>5,500</td>
<td>121,000</td>
<td>6,500</td>
<td>143,000</td>
<td></td>
</tr>
<tr>
<td>Cars and Pickups</td>
<td>(18)</td>
<td>108,000</td>
<td>(20)</td>
<td>120,000</td>
<td>(22)</td>
<td>132,000</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Lorries &amp; bus</td>
<td>(8)</td>
<td>120,000</td>
<td>(10)</td>
<td>150,000</td>
<td>(12)</td>
<td>180,000</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Drill rigs</td>
<td>(1)</td>
<td>25,000</td>
<td>(2)</td>
<td>35,000</td>
<td>(2)</td>
<td>45,000</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Miscell.</td>
<td></td>
<td>70,000</td>
<td>100,000</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Total</td>
<td></td>
<td>2,867,000</td>
<td>3,491,000</td>
<td></td>
<td>4,461,000</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>$/t</td>
<td></td>
<td>0.96</td>
<td>0.78</td>
<td></td>
<td>0.74</td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

Table 4.3. Production cost estimation for three production rates of Los Pijiguaos Bauxite deposit.
minimum of 90% and a maximum of 98%.

For mining equipment (ME) and operating costs (OC) the normal distributions are described as follows:

<table>
<thead>
<tr>
<th>Prod. rates</th>
<th>ME</th>
<th>OC</th>
</tr>
</thead>
<tbody>
<tr>
<td>3mt/y</td>
<td>$12m-$14m</td>
<td>0.96 - 1.06$/t</td>
</tr>
<tr>
<td>5mt/y</td>
<td>16 – 19</td>
<td>0.78 - 0.88</td>
</tr>
<tr>
<td>6mt/y</td>
<td>19 – 22</td>
<td>0.74 - 0.84</td>
</tr>
</tbody>
</table>

Major costs of reparations and renovations of parts of the transport equipment (belts and barges) are scheduled every 8 years, as suggested by the transport suppliers companies. Therefore, in the years 14 and 22 a range of costs for those items between 6 to 8 million dollars respectively have been included as part of the capital investment. The objective of the economic valuation of the project was to examine the range of prices for the three production rates at which the Bauxite could be sold to the processing plants or for exports. Also, to examine the determinative economic parameters for which the operation is profitable. Due to the possibility of an initial low production rate and further expanding with time, the three production rates were considered together in a life production of 20 and 30 years. It was assumed a production of 3mt/y during the first 10 years, then 5mt/y in the subsequent 5 years and the remaining time up to 20 and 30 years, a 6mt/y rate. The results are discussed below.

4.3.3. Economic appraisal results

It was considered necessary to summarize the results from all the computer simulations and outputs in order to differentiate optimal operation conditions. These results are shown in Table 4.4. The values correspond to the average of the appropriate parameters.

When the internal rate of return is lower by at least 1% than the
<table>
<thead>
<tr>
<th>Price Range ($/ton)</th>
<th>3mt/y</th>
<th>5mt/y</th>
<th>6mt/y</th>
</tr>
</thead>
<tbody>
<tr>
<td>10-15</td>
<td>5.26</td>
<td>7.91</td>
<td>9.70</td>
</tr>
<tr>
<td></td>
<td>WGR</td>
<td></td>
<td>10.09</td>
</tr>
<tr>
<td></td>
<td>PIX</td>
<td></td>
<td>1.03</td>
</tr>
<tr>
<td></td>
<td>NPV</td>
<td></td>
<td>4830000</td>
</tr>
<tr>
<td>15-20</td>
<td>8.53</td>
<td>11.87</td>
<td>13.80</td>
</tr>
<tr>
<td></td>
<td>WGR</td>
<td></td>
<td>10.06</td>
</tr>
<tr>
<td></td>
<td>PIX</td>
<td></td>
<td>1.18</td>
</tr>
<tr>
<td></td>
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20 Years

30 Years

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<th>Price Range ($/ton)</th>
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Table 4.4 Economic parameters for three production rates and different price ranges (see text for description of the Table).
specified yield of investment the parameters WGR, PIX and NPV are not calculated. The table clearly defines the 6mt/y production rate as the most profitable and desirable one at any range of selling prices. When considering the three alternatives of productions together, the 30 years project obviously looks more attractive.

Discussion and Conclusions

The use of the computer program Joleval for financial analysis of mining ventures allows a great flexibility in the input parameters and is very efficient in computing time. Each set of 250 simulations together with histogram plotting takes an average of 70 seconds per run with a cost of two pounds for the analysis of single rate of production. 100 seconds and a cost of 3 pounds are needed to carry out the appraisal for the combination of the three production rates on a 30 years life of the project. The method is particularly very useful as in the present case of the Bauxite data, whereby the costs are not very well defined. However, an objective assessment of the project can be carried out by describing the variables as probability functions instead of point estimates. This is perhaps the best method of accounting for risk and uncertainty in mining ventures. By looking at the different histograms of the economic determinative parameters it can be stated that at least there is certain range of probability of particular NPV, IRR, WGR or PIX occuring.

There is, however, a need to refine the economic valuation of the Bauxite with more updated information in order to define better stream cash flows and net profitability of the project. Nevertheless, it is clear that the range of prices considered places this deposit in a very strong position for competing in the export market and, of course, it will successfully satisfy the national requirement at about half the price paid for imported Bauxite.
PROGRAM TO CALCULATE VARIOGRAMS IN THREE ORTHOGONAL PLANES.


THE SEMIVARIANCE VALUE FOR EACH DISTANCE CALCULATION IS ASSIGNED TO THE APPROPRIATE ANGULAR SECTOR IN EACH PLANE.

VARIOGRAMS ARE PLOTTED FOR ALL SECTORS (OF SPECIFIED SIZE) IN ALL THREE ORTHOGONAL PLANES.

CONTROL CARDS: (TO BE INSERTED AT FRONT OF DATA DECK)

CARD 1: NO. OF DEPENDENT VARIABLES (UP TO 5) (FORMAT: I 11)
NEXT : TITLE CARD FOR EACH DEPENDENT VARIABLE (FORMAT: A 8)
NEXT : VARIABLE FORMAT CARD FOR INPUT VARIABLES (FORMAT: A 10)
NEXT : IF DATA IS TO BE READ FROM FILE, TYPE 3 (FORMAT: I 11)
NEXT : IF DATA IS TO BE READ FROM CARDS, TYPE 5 (FORMAT: I 11)
NEXT : RADIUS OF VARIOGRAM SEARCH SPHERE (FORMAT: I 10, 0)
NEXT : SIZE OF EACH SEARCH SECTOR IN DEGREES (FORMAT: I 10, 0)
NEXT : TYPE OF VARIOGRAMS TO BE PLOTTED (FORMAT: I 11)
FOR 1 OR 2 DIMENSIONAL VARIOGRAMS IN X-Y PLANE, TYPE 0
FOR 1 OR 2 DIMENSIONAL VARIOGRAMS IN Y-Z PLANE, TYPE 1
FOR 1 OR 2 DIMENSIONAL VARIOGRAMS IN Z-X PLANE, TYPE 2
FOR 3 DIMENSIONAL VARIOGRAMS, TYPE 3
NEXT : FORM OF VARIOGRAMS TO BE PLOTTED (FORMAT: I 11)
FOR ARITHMETIC SCALE VARIOGRAMS, TYPE 1
FOR ARITHMETIC & SEMI-LOG SCALE VARIOGRAMS, TYPE 2
FOR ARITHMETIC, SEMI-LOG, & LOG-LOG VARIOGRAMS, TYPE 3
NEXT : TITLE CARD FOR DATA (FORMAT: I 11)
NEXT : DATA DECK, INCLUDING 3 INDEPENDENT VARIABLES (X, Y, Z), AND UP TO 5 DEPENDENT VARIABLES (ASSAY VALUES), WITH LETTER Punched IN LAST COLUMN OF LAST DATA CARD (FORMAT: USER SPECIFIED, SEE ABOVE)

TO INCREASE THE NUMBER OF SEARCH SECTORS, & HENCE THE NUMBER OF VARIOGRAMS FOR EACH PLANE, THE DIMENSIONS OF: MIX, MIY, MIZ, MAXX, MAXY, MAXZ, AX, AY, AZ, KX, KY, KZ, HX, HY, & HZ MUST BE INCREASED BUT ONLY IN THE DIMENSION STATEMENT OF THE MAIN PROGRAM.


TO INCREASE THE NUMBER OF POINTS PLOTTED ON EACH VARIOGRAM, DECREASE RADIUS OF SEARCH SPHERE.

SUBROUTINE SECTOR LIMITS THE VARIOGRAM PLOT TO 100 POINTS

THE PROGRAM IS CURRENTLY SET UP TO USE THE PLOTTING ROUTINES DEVELOPED BY CULHAM LABORATORIES (CGHOST), AND PLOTTING MAY BE PERFORMED BY EITHER GRAPH PLOTTER OR LINEPRINTER.

IF ANOTHER PLOTTING ROUTINE IS USED, THE FOLLOWING CARDS MUST BE DELETED FROM THE MAIN PROGRAM:

CALL PAPER(1)
CALL STOP(1PAGE)
CALL GREN

DIMENSION C(5,700), X(700), Y(700), Z(700), B(700), FMT(8),
1X(18,100), YX(18,100), Z(18,100), MAXX(18), MAXY(18), MAXZ(18),
2XX(18,100), KY(18,100), KZ(18,100), AX(18,100), AY(18,100), AZ(18,100),
3DTITLE(5)
COMMON /A/TITLE(5)
COMMON ANGLIAN, NVGH, LVG, IBLANK, LVAR, DTITLE
LOGICAL LASTPT
LASTPT=.FALSE.

READ NUMBER OF DEPENDENT VARIABLES (IVAR)

READ(5,2) IVAR
FORMAT(II)
WRITE(6,4)
4 FORMAT(141)
WRITE(6,7) IVAR
7 FORMAT(///I0X,27HNO. OF DEPENDENT VARIABLES:,I2///)
DO 15 KA=1,IVAR
READ(5,3) TITLE(KA)
3 FORMAT(A8)
WRITE(6,8) KA,TITLE(KA)
8 FORMAT(10X,18HDEPENDENT VARIABLE,I2,10X,A8)
CONTINUE
READ VARIABLE FORMAT CARD (FMT)
READ CODE NUMBER FOR INPUT DEVICE (J)
READ(5,1) FMT
1 FORMAT(5A10)
WRITE(6,9) FMT
9 FORMAT(///I0X,17HDATA CARD FORMAT:,15A10///)
READ(5,2) J
READ RADIUS OF SEARCH SPHERE (ANR)
READ SEARCH ANGLE (ANGL)
READ CODE NUMBER FOR TYPE OF VARIOGRAM TO BE PLOTTED (NVGM)
READ CODE NUMBER FOR FORM OF VARIOGRAMS TO BE PLOTTED
READ(5,1000) ANR
READ(5,1000) ANGL
READ(5,2) NVGM
READ(5,2) LVG
1000 FORMAT(F10.0)
WRITE(6,10) ANR,ANGL
10 FORMAT(10X,24HSEARCH ANGLE (DEGREES):,F10.2//)
INITIALISE NUMBER OF SEARCH SECTORS IN X-Y PLANE (MIX),
IN Y-Z PLANE (MIY), AND IN Z-X PLANE (MIZ).
MIX=180.0/ANGL
MIY=180.0/ANGL
MIZ=180.0/ANGL
WRITE(6,169) MIX,MIY,MIZ
1695 FORMAT(10X,35HNO. OF SEARCH SECTORS IN X-Y PLANE:,I4/
10X,35HNO. OF SEARCH SECTORS IN Y-Z PLANE:,I4/
210X,35HNO. OF SEARCH SECTORS IN Z-X PLANE:,I4///)
READ TITLE CARD OF DATA DECK (DTITLE)
READ(5,11) DTITLE
WRITE(6,4)
WRITE(6,12) DTITLE
11 FORMAT(5A10)
12 FORMAT(///,2X,5A10///)
WRITE(6,199) (TITLE(KA),KA=1,IVAR)
199 FORMAT(///,F10.2//)
READ X,Y,Z CO-ORDINATES, & UP TO 5 DEPENDENT VARIABLES
ON EACH DATA CARD.
NUM=0
DO 25 N=1,700
READ(J,FMT) X(N),Y(N),Z(N),(C(I,N),I=1,IVAR),LASTPT
WRITE(6,2000) N,X(N),Y(N),Z(N),(C(I,N),I=1,IVAR)
2000 FORMAT(10X,15,3F11.2,5(3X,F8.2))
NUM=NUM+1
25 CONTINUE
IF(LASTPT) GO TO 35
35 WRITE(6,6) NUM
6 FORMAT(///20X,21HNUMBER OF DATA CARDS:,I5///)
WRITE TITLE PAGE FOR EACH DEPENDENT VARIABLE
DO 45 L=1,IVAR
WRITE(6,4)
WRITE(6,3) TITLE(L)
45 FORMAT(///,14HVARIOGRAMS OF ,A8,7H VALUES///)
CALCULATE & PLOT VARIOGRAMS FOR EACH DEPENDENT VARIABLE

DO 55 M=1,NUM
B(M)=C(L,M)
55 CONTINUE

SWITCH ON GRAPH- PLOTTER (OR LINEPRINTER)
CALCULATE & INITIALISE NO. OF PAGES TO BE USED BY PLOTTER (IPAGE)

CALL PAPER(I)

MI=36G,0/ANGL
IF(NVGM,LT,3) GO TO 200
IPAGE=3*MI*LVG*IVAR
GO TO 105
200 IPAGE=MI*LVG*IVAR
105 CONTINUE

CALL GPSTOP(IPAGE)

CALL MAIN SUBROUTINE TO CALCULATE & PLOT VARIOGRAM
VALUES FOR EACH DEPENDENT VARIABLE

LVAR=L
CALL VECTPR(B,X,Y,Z,NUM,MIY,MIZ,MAXX,MAXY,MAXZ,1KX,KY,KZ,AX,AY,AZ,HX,HY,HZ)

CONTINUE

SWITCH OFF PLOTTER (OR LINEPRINTER)
CALL GREND
STOP
END

SUBROUTINE VECTPR(AS,X,Y,Z,NUM,MIY,MIZ,MAXX,MAXY,MAXZ,1KX,KY,KZ,AX,AY,AZ,HX,HY,HZ)
SUBROUTINE TO CALCULATE VECTOR-DISTANCE, VECTOR-ANGLE,
AND SEMIVARIANCE BETWEEN ALL POINT-PAIRS.

VECTPR RESOLVES THE VECTOR RELATIONSHIP BETWEEN POINT-PAIRS
INTO DISTANCE & ANGLE COMPONENTS IN 3 ORTHOGONAL PLANES.

SUBROUTINE VECTPR CALLS OTHER SUBROUTINES, INCLUDING:
ANGLE : CALCULATES VECTOR ANGLE COMPONENT IN 3 ORTHOGONAL PLANES
SECTOR : ASSIGNING VECTOR ANGLE COMPONENT TO SECTOR OF SEARCH CIRCLE
SEMVAR: CALCULATES SEMIVARIANCE BETWEEN ALL POINT-PAIRS
MLIST : LISTS SEMIVARIANCE VALUES FOR EACH SECTOR IN 1 PLANE
VGMPLT : CALLING SUBROUTINE FOR PLOTTING SUBROUTINE (PINIST)

INPUT VALUES:
AS : ASSAY VALUE AT EACH POINT
X : X CO-ORDINATE OF EACH POINT (EASTING)
Y : Y CO-ORDINATE OF EACH POINT (NORTHING)
Z : Z CO-ORDINATE OF EACH POINT (VERTICAL DEPTH)
ANG : RADIUS OF SEARCH SPHERE
ANGL : SEARCH ANGLE (DEGREES)

DIMENSION KX(MIX,100),KY(MIY,100),KZ(MIZ,100),1AX(MIX,100),AY(MIY,100),AZ(MIZ,100),MAXX(MAX1),MAXY(MAXM),MAXZ(MAXZ),2HX(MIX,100),HY(MIY,100),HZ(MIZ,100),3AS(NUM),X(NUM),Y(NUM),Z(NUM)
COMMON ANGL,ANK,NVGM,LVG

CLEAN OUT ARRAYS: HX,HY,HZ,AX,AY,AZ,KX,KY,KZ,MAXX,MAXY,MAXZ.
SET UP LOOP TO MEASURE THE RELATION BETWEEN ALL POINT PAIRS

DO 30 J=1,MIZ
MAXZ(J)=0
HZ(J,JJ)=-100.00
KZ(J,JJ)=0
AZ(J,JJ)=0.0
30 CONTINUE

40 CONTINUE

DO 80 JA=1,NUM
DO 60 JC=1,NUM
IF(JA.EQ.JC) GO TO 60

CALCULATE SQUARED DIFFERENCES IN DEPENDENT VARIABLE (ASSAY),
CALCULATE DISTANCE BETWEEN ALL POINT PAIRS (D)

ASSAY=(AS(JA)-AS(JC))**2
D=SQRT(((X(JA)-X(JC))**2)+((Y(JA)-Y(JC))**2)+((Z(JA)-Z(JC))**2))

TEST FOR DUPLICATE CARDS IN DATA DECK.

IF(D.EQ.0.0) GO TO 77
GO TO 66
77 WRITE(6,7000) JA, JC
7000 FORMAT(1H///5X,3H*WARNING*///10X,15HDATA CARD NUMBER,15,///
152H HAS THE SAME X,Y,Z CO-ORDINATES AS DATA CARD NUMBER,15,///)
GO TO 88

CALCULATE DISTANCE BETWEEN POINT PAIRS IN X-DIRECTION (XX)
CALCULATE DISTANCE BETWEEN POINT PAIRS IN Y-DIRECTION (YY)
CALCULATE DISTANCE BETWEEN POINT PAIRS IN Z-DIRECTION (ZZ)

66 XX=X(JA)-X(JC)
YY=Y(JA)-Y(JC)
ZZ=Z(JA)-Z(JC)

CALCULATE COMPONENT OF DISTANCE-VECTOR (D) IN X-Y PLANE (DX)
CALCULATE COMPONENT OF DISTANCE-VECTOR (D) IN Y-Z PLANE (DY)
CALCULATE COMPONENT OF DISTANCE-VECTOR (D) IN Z-X PLANE (DZ)

1 DX=SQRT((XX**2)+(YY**2))
DY=SQRT((YY**2)+(ZZ**2))
DZ=SQRT((ZZ**2)+(XX**2))

ASSIGN DISTANCE COMPONENT (DX) TO AN INTEGER CLASS-INTERVAL (INTX)
ASSIGN DISTANCE COMPONENT (DY) TO AN INTEGER CLASS-INTERVAL (INTY)
ASSIGN DISTANCE COMPONENT (DZ) TO AN INTEGER CLASS-INTERVAL (INTZ)

INTX=DX/ANR
INTY=DY/ANR
INTZ=DZ/ANR

INCREMENT CLASS-INTERVAL VALUES.

INTX=INTX+1
INTY=INTY+1
INTZ=INTZ+1

CALL SUBROUTINE TO CALCULATE VECTOR-ANGLE COMPONENTS IN
X-Y PLANE (BEAR), Y-Z PLANE (DECLY), X-Z PLANE (DECLX).

CALL ANGLE(XX,YY,ZZ,BEAR,DECLY,DECLX,JA,JC)

CALL SUBROUTINE TO ASSIGN VECTOR ANGLE COMPONENTS
IN EACH PLANE TO AN INTEGER SECTOR OF THAT PLANE

CALL SECTOR(ASSAY,INTX,MAXX,ISECTX,AX,KX,BEAR,MIX)
CALL SECTOR(ASSAY,INTY,MAXY,ISECTY,AY,KY,DECLY,MIX)
CALL SECTOR(ASSAY,INTZ,MAXZ,ISECTZ,AZ,KZ,DECLX,MIZ)
88 CONTINUE
60 CONTINUE
80 CONTINUE

CALL SUBROUTINE TO CALCULATE SEMIVARIANCE (H)
FOR ALL CLASS INTERVALS IN EACH SECTOR.
THIS SUBROUTINE IS CALLED FOR EACH ORTHOGONAL PLANE.

CALL SEMVAR(MIX,MAXX,KX,HX,AX)
CALL SEMVAR(MIX,MAXY,XY,HY,A)
CALL SEMVAR(MIX,MAXZ,KZ,HZ,AZ)

PRINT NUMBER OF CLASS INTERVALS IN EACH SECTOR
WRITE(6,1025) 1025 FORMAT(5X,4OHNUMBER OF CLASS INTERVALS IN EACH SECTOR///
       11X,6HSECTOR,20X,9HX-Y PLANE,21X,9HY-Z PLANE,21X,9HX-Z PLANE/)  
GO TO 140 L=1,40  
WRITE(6,1030) L,MAXX(L),MAXY(L),MAXZ(L)  
1030 FORMAT(12X,12,3(25X,18))  
IF(L.LT.MIX) GO TO 90  
IF(L.LT.MIY) GO TO 90  
IF(L.LT.MIZ) GO TO 90  
GO TO 110  
90 CONTINUE  
100 CONTINUE  
C  CALL SUBROUTINE TO LIST SEMIVARIANCES (H-VALUES)
   FOR ALL CLASS INTERVALS IN EACH SECTOR.
   THIS SUBROUTINE IS CALLED FOR ORTHOGONAL PLANE SPECIFIED BY NVGM
C  CALL SUBROUTINE TO PLOT ARITHMETIC-SCALE VARIOGRAMS
   FOR EACH SECTOR IN ONE PLANE.
   THIS SUBROUTINE IS CALLED FOR ORTHOGONAL PLANES SPECIFIED BY NVGM
C  TEST WHETHER 1, 2, OR 3-DIMENSIONAL VARIOGRAMS REQUIRED

110 IF(NVGM.EQ.1 .OR.NVGM.EQ.2) GO TO 300.

WRITE(6,1050) MIX  
1050 FORMAT(111///50X,2E0HX-Y PLANE VARIOGRAMS///
       147X,23HNUMBER OF VARIOGRAMS = ,12///
       234X,52HSECTORS INCREASE ANTI-CLOCKWISE FROM POSITIVE X-AXIS///)
WRITE(6,1035)  
1035 FORMAT(///41X,37HSEMIVARIANCE (H-VALUES) FOR X-Y PLANE//)
WRITE(6,1070)  
CALL HLIST(MIX,MAXX,HX)  
CALL PAIRLIST(MIX,MAXX,KX)  
CALL VGMPLT(MIX,MAXX,HX)

300 IF(NVGM.EQ.3 .OR.NVGM.EQ.2) GO TO 400

WRITE(6,1055) MIY  
1055 FORMAT(111///50X,2E0HY-Z PLANE VARIOGRAMS///
       147X,23HNUMBER OF VARIOGRAMS = ,12///
       38X,256HSECTORS INCREASE CLOCKWISE (DOWN) FROM POSITIVE Y-AXIS///)
WRITE(6,1040)  
1040 FORMAT(///41X,37HSEMIVARIANCE (H-VALUES) FOR Y-Z PLANE//)
WRITE(6,1070)  
CALL HLIST(MIY,MAXY,HY)  
CALL PAIRLIST(MIY,MAXY,KY)  
CALL VGMPLT(MIY,MAXY,HY)

400 IF(NVGM.EQ.0 .OR.NVGM.EQ.1) GO TO 990

WRITE(6,1060) MIZ  
1060 FORMAT(111///50X,2E0HX-Z PLANE VARIOGRAMS///
       147X,23HNUMBER OF VARIOGRAMS = ,12///
       256HSECTORS INCREASE ANTI-CLOCKWISE (UP) FROM POSITIVE Z-AXIS///)
WRITE(6,1045)  
1045 FORMAT(///41X,37HSEMIVARIANCE (H-VALUES) FOR X-Z PLANE//)
WRITE(6,1070)  
CALL HLIST(MIZ,MAXZ,HZ)  
CALL PAIRLIST(MIZ,MAXZ,KZ)  
CALL VGMPLT(MIZ,MAXZ,HZ)

990 RETURN  
END

SUBROUTINE ANGLE(XX,YY,ZZ,BEAR,DECLY,DECLX,JA,JC)

CALCULATION OF A VECTOR BETWEEN A PAIR OF POINTS.

COMMENCE TREE-SEARCH TO CALCULATE THREE VECTOR COMPONENTS
OF ALL POSSIBLE POINT-PAIR COMBINATIONS.

CALCULATE VECTOR-ANGLE COMPONENT IN X-Y PLANE (BEAR)
   RELATIVE TO POSITIVE X-AXIS,
CALCULATE VECTOR-ANGLE COMPONENT IN Y-Z PLANE (DECLY)
   RELATIVE TO POSITIVE Y-AXIS,
CALCULATE VECTOR-ANGLE COMPONENT IN X-Z PLANE (DECLX)
RELATIVE TO POSITIVE Z-AXIS.

PI = 3.14159

7 IF (XX) 30, 20, 10

10 ALPHA = ATAN2 (YY, XX) + PI
   IF (YY) 120, 110, 100
20 IF (YY) 220, 210, 200
30 ALPHA = ATAN2 (YY, XX) + PI
   IF (YY) 320, 310, 300

100 BETA = ATAN2 (ZZ, YY) + PI
   IF (ZZ) 500, 600, 500
110 IF (ZZ) 700, 490, 750
120 BETA = ATAN2 (ZZ, YY) + PI
   IF (ZZ) 500, 600, 500
300 BETA = ATAN2 (ZZ, YY) + PI
   IF (ZZ) 500, 600, 500
310 IF (ZZ) 320, 310, 300

30 GAMMA = ATAN2 (XX, ZZ) + PI
   BEAR = ALPHA
   DECLY = BETA
   DECLX = GAMMA
   GO TO 45

500 GAMMA = ATAN2 (XX, ZZ) + PI
   BEAR = ALPHA
   DECLY = BETA
   DECLX = GAMMA
   GO TO 45

600 BEAR = ALPHA
   DECLY = BETA
   DECLX = PI/2.0
   GO TO 45

650 BEAR = ALPHA
   DECLY = BETA
   DECLX = 3.0*PI/2.0
   GO TO 45

700 GAMMA = ATAN2 (XX, ZZ) + PI
   BEAR = ALPHA
   DECLY = PI/2.0
   DECLX = GAMMA
   GO TO 45

750 GAMMA = ATAN2 (XX, ZZ) + PI
   BEAR = ALPHA
   DECLY = 3.0*PI/2.0
   DECLX = GAMMA
   GO TO 45

800 GAMMA = ATAN2 (XX, ZZ) + PI
   BEAR = PI/2.0
   DECLY = BETA
   DECLX = GAMMA
   GO TO 45

850 GAMMA = ATAN2 (XX, ZZ) + PI
   BEAR = 3.0*PI/2.0
   DECLY = BETA
   DECLX = GAMMA
   GO TO 45

450 BEAR = PI
   DECLY = 10.0
   DECLX = 3.0*PI/2.0
   GO TO 45

480 BEAR = 2.0*PI
   DECLY = 10.0
   DECLX = PI/2.0
   GO TO 45

490 BEAR = 3.0*PI/2.0
   DECLY = PI
   DECLX = 10.0
   GO TO 45

500 BEAR = PI/2.0
   DECLY = 2.0*PI
DECLX=10.0
GO TO 45
C 400 BEAR=10.0
DECLY=3.0*PI/2.0
DECLX=PI
GO TO 45
C 420 BEAR=10.0
DECLY=PI/2.0
DECLX=2.0*PI
GO TO 45
C 410 BEAR=10.0
DECLY=10.0
DECLX=10.0

PRINT ERROR MESSAGE.
WRITE (6,7000) JA, JC
7000 FORMAT(1//,50X,11H*WARNING*///,10X,16HDATA CARD NUMBER, I3,
152H HAS THE SAME X,Y,Z CO-ORDINATES AS DATA CARD NUMBER, I3)
GO TO 45

CONVERT ANGLES FROM TO RADIANS TO DEGREES.
45 BEAR=BEAR*57.29578
DECLY=DECLY*57.29578
DECLX=DECLX*57.29578
IF(BEAR.GT.180.) BEAR=BEAR-180.
IF(DECLY.GT.180.) DECLY=DECLY-180.
IF(DECLX.GT.180.) DECLX=DECLX-180.

RETURN
END

SUBROUTINE SECTOR(ASSAY, INT, MX, ISECT, AA, KK, VECT, M)

SUBROUTINE TO ASSIGN COMPONENT OF VECTOR ANGLE IN A PLANE (VECT)
TO AN INTEGER SECTOR OF THAT PLANE (ISECT), TO INCREMENT THE
DISTANCE CLASS-INTERVAL COUNTER (KK) FOR THAT SECTOR, AND TO
INCREMENT THE RUNNING TOTAL (AA) OF SQUARE-DIFFERENCES OF
METAL VALUES (ASSAY) BETWEEN POINT-PAIRS FOR THAT SECTOR.

INPUT VALUES: VECT = COMPONENT OF VECTOR ANGLE IN A PLANE.
ASSAY = SQUARE-DIFFERENCES IN METAL VALUE
BETWEEN A PAIR OF POINTS.
INT = CALCULATED DISTANCE CLASS-INTERVAL.
MX = MAX. NO. OF CLASS-INTERVALS IN EACH SECTOR.
(ARRAY) AA = LIST OF TOTAL SQUARE-DIFFERENCES IN METAL
VALUES BETWEEN POINT-PAIRS IN EACH SECTOR.
(ARRAY) KK = LIST OF NUMBER OF POINT-PAIR DISTANCES IN
EACH CLASS-INTERVAL OF EACH SECTOR.

OUTPUT VALUES: ISECT = INTEGER NUMBER FOR EACH SECTOR.
DIMENSION MX(M), KK(M, 100), AA(M, 100)
COMMON ANGL, ANR, NVGM

TEST FOR ZERO VECTOR COMPONENT.
IF (VECT.GT.5.00.0) GO TO 999

CALCULATE WHICH SECTOR (ISECT) CONTAINS VECTOR-COMPONENT (VECT)
INCREMENT ISECT, & CLASS-INTERVAL COUNTER (KK) FOR THAT SECTOR.
ADD VALUE OF SQUARE-DIFFERENCES OF METAL VALUE (ASSAY)
TO LIST (AA) FOR THAT CLASS-INTERVAL AND SECTOR.
ISECT=VECT/ANGL
ISECT=ISECT+1

TEST WHETHER CLASS-INTERVAL NO. (INT) WITHIN DIMENSION RANGE
IF (INT.GT.100) GO TO 120
GO TO 25

WRITE ERROR MESSAGE
120 IFACT=(INT/100)+1
WRITE (6,2000) IFACT
2000 FORMAT(1//,50X,9H*WARNING*///,10X,
13H RADIUS OF SEARCH SPHERE IS TOO SMALL,///,10X,
23H INCREASE RADIUS BY A FACTOR OF, I5,////)
SUBROUTINE SEMVAR(M,MX,KK,HH,AA)

SUBROUTINE TO CALCULATE SEMIVARIANCE.

SEMIVARIANCE (H) CALCULATED FROM HALF THE SUM OF SQUARED DIFFERENCES OF ASSAY VALUES (A), DIVIDED BY THE NUMBER OF POINT-PAIRS (K) FOR EACH SECTOR.

I.E. H(I,J) = A(I,J)/(K(I,J)*2.0)

INPUT VALUES: M = NO. OF SECTORS
MX = MAX. NO. OF CLASS INTERVALS IN EACH SECTOR.
(ARRAY) KK = LIST OF NUMBER OF PAIRS IN EACH SECTOR.
(ARRAY) AA = LIST OF SQUARED DIFFERENCES IN EACH SECTOR.
(ARRAY) HH = LIST FOR CALCULATED SEMIVARIANCE VALUES IN EACH SECTOR.

I = SECTOR NO. (= ISECT)
J = CLASS INTERVAL NO. (= INT)

DIMENSION MX(M),KK(M,100),AA(M,100),HH(M,100)

CALCULATE SEMIVARIANCE (HH).

DO 40 I=1,M
MAX=MX(I)
25 DO 20 J=1,MAX
IF(KK(I,J).EQ.0) GO TO 100
B=KK(I,J)
HH(I,J)=AA(I,J)/(B*2.0)
GO TO 80
100 HH(I,J)=-100.00
20 CONTINUE
40 CONTINUE
RETURN
END

SUBROUTINE HLIST(M,MX,HH)

SUBROUTINE TO LIST VALUES OF SEMIVARIANCE (HH) FOR EACH SECTOR IN A PLANE.

INPUT VALUES: M = NO. OF SECTORS
MX = MAX. NO. OF CLASS INTERVALS IN EACH SECTOR.
(ARRAY) HH = LIST OF CALCULATED SEMIVARIANCE VALUES IN EACH SECTOR.

DIMENSION MX(M),HH(M,100)
COMMON ANGL

INITIALIZE LOOP-COUNTER (N).
INITIALIZE MAX. NO. OF CLASS-INTERVALS IN LAST SECTOR (MAXS).

MAXS=5
DO 10 J1=1,M
IF(MX(J1).GT.MAXS) MAXS=MX(J1)
10 CONTINUE
WRITE(4,1000)M,MAXS,ANGL
N=1
I=1
J=1
LIST VALUES OF SEMIVARIANCE (HH) IN EACH SECTOR.

WRITE(6,1000)N,(HH(K,J),K=1,M)
WRITE(4,1000)N,(HH(K,J),K=1,M)
SUBROUTINE VGMPLT(M,MX,HH)

CALLING SUBROUTINE FOR MAIN PLOTTING SUBROUTINE (PINIST)

IN EACH SECTOR, VALUES FOR SEMIVARIANCE (HH) ARE ASSIGNED TO
CORRECT CLASS-INTERVAL NO. (J) AND STORED
IN TWO ARRAYS (DIN(1) & DIN(2))

VGMPLT THEN CALLS SUBROUTINE TO PLOT VARIOGRAM
FOR EACH SECTOR (PINIST)

VARIOGRAMS ARE PLOTS OF SEMIVARIANCE VS. INCREASING LAG
(INCREASING CLASS-INTERVAL NUMBERS)

VGMPLT ALSO DETERMINES WHETHER ARITHMETIC,
SEMI-LOG, OR LOG-LOG VARIOGRAMS ARE TO BE PLOTTED.

ANY PLOTTING ROUTINE MAY BE USED TO PLOT SEMIVARIANCE
(DIN(2,J)) AS THE Y-COMPONENT, & LAG (DIN(1,J)) AS THE X-COMPONENT

LOG-TRANSFORMS OF SEMIVARIANCE VALUES, OR BOTH SEMIVARIANCE
& LAG VALUES MAY BE USED TO GIVE SMOOTHER VARIOGRAM PLOTS

INPUT VALUES: M = NO. OF SECTORS.
MX = MAX. NO. OF CLASS INTERVALS IN EACH SECTOR.
(ARRAY) HH = LIST OF CALCULATED SEMIVARIANCE VALUES
IN EACH SECTOR.

DIMENSION MX(M),HH(M,100),DIN(2,100)
COMMON IBLANK(3),LVG,NSECT,LVAR

STORE CLASS-INTERVAL NUMBERS (J) IN ARRAY DIN(1,J)
STORE VALUES OF SEMIVARIANCE (HH) IN ARRAY DIN(2,J)

DO 40 I=1,M
MAX=MX(I)
NSECT=I
DO 20 J=1,MAX
DIN(1,J)=J
DIN(2,J)=HH(I,J)
20 CONTINUE

PLOT ARITHMETIC VARIOGRAM

LVG=1
CALL PINIST(DIN,MX)
LVG=N

TEST WHETHER SEMI-LOGARITHMIC VARIOGRAM REQUIRED

IF(LVG.EQ.1) GO TO 40

CALCULATE LOG-SEMIVARIANCE VALUES

DO 60 J=1,MAX
Q=DIN(2,J)
IF (Q.EQ.-100.00) GO TO 60
IF (Q.EQ.0.00) Q=1.0
DIN(2,J)=ALOG(Q)
60 CONTINUE

PLOT SEMILOGARITHMIC VARIOGRAM

LVG=2
CALL PINIST(DIN,MX)
LVG=N
TEST WHETHER LOG-LOG VARIOGRAM REQUIRED

IF(LVG.EQ.2) GO TO 40

CALCULATE LOG-DISTANCE VALUES

DO 80 J=1,MAX
   DIN(1,J)=ALOG(DIN(1,J))
80 CONTINUE

PLOT LOG-LOG VARIOGRAM

LVS=3
CALL PINIST(DIN,MAX)
LVS=N

40 CONTINUE

RETURN

END

SUBROUTINE PINIST(DIN,MAX)

SUBROUTINE TO PLOT ARITHMETIC, SEMI-LOGARITHMIC, OR LOG-LOG VARIOGRAMS.

SUBROUTINE PINIST USES A SERIES OF PLOTTING ROUTINES DEVELOPED
BY CULHAM LABORATORIES (CGHOST), AND THIS ROUTINE (PINIST)
MAY ONLY BE USED WHERE CGHOST FACILITIES ARE IN OPERATION

OUTPUT FROM A DIGITAL GRAPH- PLOTTER MAY BE OBTAINED BY USE
OF THE APPROPRIATE CONTROL CARDS IN THE JCL DECK

INPUT VALUES : MAX = MAX NO. OF CLASS INTERVALS IN EACH SECTOR
ARRAY) DIN(1) = LIST OF CLASS INTERVAL NUMBERS (LAG VALUES)
ARRAY) DIN(2) = LIST OF SEMIVARIANCE VALUES

DIMENSION DIN(2,100),XX(100),YY(100),DTITLE(5)
COMMON /A/ TITLE(5)
COMMON /B/ TITLE(5)
COMMON I8,blank(3),LVS,NSECT,LVAR,OTITLE

DO 40 I=1,MAX
   XX(I)=DIN(1,I)
   YY(I)=DIN(2,I)
40 CONTINUE

ATITLE=TITLE(LVAR)
YMAX=0.00
YMIN=0.00
DO 30 I=1,MAX
   YI=YY(I)
   IF(YI.EQ.-1.00) GO TO 33
   IF(YI.GT.YMAX) YMAX=YI
   IF(YI.LT.YMIN) YMIN=YI
30 CONTINUE

TEST WHETHER ARITHMETIC VARIOGRAM REQUIRED

IF(LVG.NE.1) GO TO 10

PLOT ARITHMETIC SEMIVARIANCE VS. ARITHMETIC LAG

XMAX=FLOAT(MAX)
CALL PSPACE(0.1,0.65,0.2,0.8)
CALL MAP(0.0,XMAX,YMIN,YMAX)
CALL WINDOW(0.0,XMAX,YMIN,YMAX)
CALL CTRMAG(8)
DO 50 I=1,MAX
   YI=YY(I)
   IF(YI.EQ.-1.00) GO TO 83
50 CONTINUE

CALL PIPLLOT(XX,YY,I,MAX,45)
CALL PIPLLOT(XX,YY,I,MAX,-45)
CALL AXES

CALL PLACE(57,10)
CALL TYPECS(22H ARITHMETIC VARIOGRAM ,22)
CALL PLACE(3,10)
CALL TYPECS(12H SEMIVARIANCE,12)
CALL PLACE(5,12)
CALL TYPECS(9H GAMMA H),9)
CALL PLACE(62,12)
CALL TYPECS(12H SECTOR NO. ,12)
CALL PLACE(74,12)
CALL TYPECS(NSECT)
CALL PLACE(35,74)
CALL TYPECS(15H LAG (DISTANCE),16)
CALL PLACE(20,77)
CALL TYPECS(0TITLE,50)
CALL PLACE(40,79)
CALL TYPECS(0TITLE,8)
CALL FRAME

TEST WHETHER SEMI-LOGARITHMIC VARIOGRAM REQUIRED
10 IF(LVG.NE.2) GO TO 20

PLOT LOG-SEMIVARIANCE VS. ARITHMETIC LAG
XMAX=FLOAT(MAX)
CALL MAP(0.5,XMAX,YMIN,YMAX)
CALL WINDOW(0.0,XMAX,YMIN,YMAX)
DO 60 I=1,MAX
YI=YY(I)
IF(YI.GT.-100.00)GO TO 90
60 CONTINUE
90 CALL PTPLOT(XX,YY,I,MAX,45)
CALL PTPLOT(XX,YY,I,MAX,-45)
CALL AXES

CALL PLACE(53,10)
CALL TYPECS(27H SEMI-LOGARITHMIC VARIOGRAM,27)
CALL PLACE(3,10)
CALL TYPECS(16H LOG SEMIVARIANCE,16)
CALL PLACE(5,12)
CALL TYPECS(14H (LOG GAMMA H),14)
CALL PLACE(74,12)
CALL TYPECS(12H SECTOR NO.,12)
CALL PLACE(74,12)
CALL FRAME

TEST WHETHER LOG-LOG VARIOGRAM REQUIRED
20 IF(LVG.NE.3) GO TO 99

PLOT LOG-SEMIVARIANCE VS. LOG-LAG
XMAX=FLOAT(MAX)
XMAXL=ALOG(XMAX)
CALL MAP(0.5,XMAXL,YMIN,YMAX)
CALL WINDOW(0.0,XMAXL,YMIN,YMAX)
DO 70 I=1,MAX
YI=YY(I)
IF(YI.GT.-100.00)GO TO 100
70 CONTINUE
100 CALL PTPLOT(XX,YY,I,MAX,45)
CALL PTPLOT(XX,YY,I,MAX,-45)
CALL AXES

CALL PLACE(57,10)
CALL TYPECS(18H LOG-LOG VARIOGRAM,18)
CALL PLACE(3,10)
CALL TYPECS(16H LOG SEMIVARIANCE,16)
CALL PLACE(5,12)
CALL TYPECS(14H (LOG GAMMA H),14)
CALL PLACE(74,12)
CALL TYPECS(12H SECTOR NO.,12)
CALL PLACE(74,12)
CALL FRAME

99 RETURN
END

SUBROUTINE PAIRLST(M,MX,KK)

SUBROUTINE TO LIST THE NUMBER OF PAIRS(KK) OF SAMPLES IN
DIMENSION MX(M), KK(M, 16)

WRITE (6, 4)
WRITE (6, 1004)
WRITE (6, 1070)
MAXS=0
DO 10 J1=1, M
IF (MX(J1) > MAXS) MAXS=MX(J1)
10 CONTINUE
N=1
I=1
J=1
WRITE (6, 1005) N, (KK(K, J), K=1, M)
4 FORMAT (1H1)
1004 FORMAT (///30X, 30H NUMBER OF PAIRS IN EACH SECTOR///)
1070 FORMAT (1X, 6H CLASS, 45X, 13H SECTOR NUMBER, /, 1X, 8H INTERVAL, /, 1X, 3H NO.)
1005 FORMAT (1X, 13, 10I12/4X, 10I12///)

TEST IF ALL NUMBER OF PAIRS PRINTED

30 MAX=MX(I)
IF (J LT MAX) GO TO 40
IF (MAX .EQ. MAXS) GO TO 999
I=I+1
GO TO 30

40 J=J+1
N=N+1
GO TO 20
999 RETURN
END

15.09.52. UCLP, 22, 1.092 KLNS.
PROGRAM TRPOCA (OUTPUT=201, TAPE4=201, TAPE6=OUTPUT)

A PROGRAM TO TRANSFORM POLAR TO CARTESIAN COORDINATES
FOR FURTHER ANALYSIS OF THE VARIOGRAM DATA SET OUTPUT

INPUT PARAMETERS ARE:
- ANGL=SIZE OF SEARCH SECTOR (IN DEGREES)
- NY=NUMBER OF SECTORS
- NX=MAXIMUM NUMBER OF CLASS INTERVALS (LAGS)
- CC=SEMIVARIANCE VALUES

DIMENSION X(3000), Y(3000), Z(3000), CC(18, 100)
COMMON ANGL, READ(4, 1001) NY, NX, ANGL
DO 1 J=1, NX
1 READ(4, 1002) N, (CC(K, J), K=1, NY)
1001 FORMAT(2I3, F8.2)
1002 FORMAT(1X, I3, 10F12.2)

ALPHA=ANGL
ALPHA=ALPHA/57.29578
Phi=ALPHA/2.0
KNY=NY/2
/*
* DO 8 INT=1, KNY
* DO 7 JNT=1, NX
* N=N+1
* X(N)=JNT*COS(Phi)
* Y(N)=JNT*SIN(Phi)
* Z(N)=CC(INT, JNT)
* ANX=JNT*COS(Phi)
* ANY=JNT*SIN(Phi)
* ANZ=Z(N)
* IF(ANZ.EQ.-10.0) GO TO 7
* WRITE(6, 1003) X(N), Y(N), Z(N), PHI, INT, JNT
* N=N+1
* X(N)=ANX-(2.0*ANX)
* Y(N)=ANY-(2.0*ANY)
* Z(N)=CC(INT, JNT)
* WRITE(6, 1003) X(N), Y(N), Z(N), PHI, INT, JNT
* CONTINUE
* PHI=PHI+ALPHA
*/
8 CONTINUE
CPHI=3.141592-(Phi)
INT=INT+1
/*
* DO 10 JNT=1, NX
* N=N+1
* XT=JNT*COS(CPHI)
* X(N)=XT-(2.0*XT)
* Y(N)=JNT*SIN(CPHI)
* Z(N)=CC(INT, JNT)
* BNX=X(N)
* BNY=JNT*SIN(CPHI)
* BNZ=Z(N)
* IF(BNZ.EQ.-10.0) GO TO 10
* WRITE(6, 1003) X(N), Y(N), Z(N), CPHI, INT, JNT
* N=N+1
* X(N)=-1.0*BNX
* Y(N)=BNY-(2.0*BNY)
* Z(N)=CC(INT, JNT)
* WRITE(6, 1003) X(N), Y(N), Z(N), CPHI, INT, JNT
* CONTINUE
* CPHI=CPHI+ALPHA
*/
INT=INT+1
10 CONTINUE
/*
* DO 9 INT=1, KNY
* N=N+1
* X(N)=-1.0*BNX
* Y(N)=BNY-(2.0*BNY)
* Z(N)=CC(INT, JNT)
* WRITE(6, 1003) X(N), Y(N), Z(N), CPHI, INT, JNT
* CONTINUE
* CPHI=CPHI+ALPHA
*/
INT=INT+1
9 CONTINUE
1003 FORMAT(10X, 2F6.2, F12.2, F6.2, 2I3)
10 STOP
END
PRGS2A I CONTOUR (INPUT=101, OUTPUT=201, TAPE3=101, TAPE4=201,  
1 TAPE5=INPUT, TAPE6=OUTPUT)  
DIMENSION A(45, 45), X(978), Y(978), Z(978), H(10)  
READ(5, 3)(H(I), I=1, 10)  
READ(3, 1)(X(I), Y(I), Z(I), I=1, 978)  
WRITE(3, 4)(X(I), Y(I), Z(I), I=1, 978)  
CALL PS2AD (7, X, Y, Z, -50, 81, 50, 81, -18.35, 18.35, 45, 45, A, IEXIT)  
IF (EXIT.EQ.0) GO TO 10  
WRITE (5, 2) EXIT  
STOP  
10 CONTINUE  
CALL PSPACE (0.10, 0.95, 0.50, 0.8069)  
CALL MAP (-50, 50, -18.35, 18.35)  
call CONTRA (A, 1, 45, 45, 1, 45, 45, h, 1, 10)  
CALL CTRIAG (5)  
CALL PTPULL (X, Y, i, 78, 45)  
CALL BORDER  
call GRATIE  
CALL CTSMAG(3)  
CALL PLACE (50, 50)  
CALL DENSITY (3)  
call TYPESG (30, HLAG= 10 F NO. OF SECTORS= 18,30)  
CALL PLACE (50, 50)  
CALL TYPECS (42, BASE CONTOUR = 0.3 CONTOUR INTERVAL = 0.3,42)  
CALL PLACE (55, 55)  
CALL TYPECS (39, FIG. -ISO-SEMIVARIANCE MAP,30)  
CALL PLACE (60, 60)  
CALL TYPECS (23, COPPER ON BENCH 11 EAST,23)  
CALL G END  
STOP  
1 FORMAT (10x, 2F6.2, F12.2, 46X)  
2 FORMAT (1X, 15H FAILURE WITH EXIT = , I5)  
3 FORMAT (F10.2)  
4 FORMAT (20X, 3F10.2)  
5 FORMAT (10X, 10F10.2)  
END
PROGRAM LOCMODL (INPUT=101, OUTPUT=201, TAPE5(INPUT, ITAPE3=101, TAPE6(OUTPUT, TAPE4=201).

****************************************************************************************************

THE PROGRAM LOCMODL IS A PROGRAM TO FIT LOCAL MODELS TO DATA SUB-SETS IN TWO DIMENSIONS

SUB-SETS ARE DEFINED BY DATA POINTS FALLING WITHIN THE ZONE OF INTER-NAL CORRELATION OBTAINED FROM THE ISO-SEMIVARIANCE DIAGRAM. DATA POINTS SUB-SETS ARE WEIGHTED ACCORDING TO THE COVARIANCE FUNCTION IN RELATION TO LOCAL MODEL CENTRE (VARIGRAMS ORIGIN)

A FIRST ORDER POLYNOMIAL IS FITTED TO THE SMOOTHED SUB-SET AND THE CALCULATED COEFFICIENTS ARE USED FOR ESTIMATING MEAN GRADES OF BLOCKS.

IF A SUB-BLOCK ESTIMATION IS ALSO REQUIRED, ADDITIONAL INFORMATION SHOULD BE PROVIDED AND THIS PROGRAM IS RUNNED IN CONJUNCTION WITH THE PROGRAM "PREPRO".

A MAXIMUM OF 100 DATA POINTS IS ALLOWED WITHIN EACH LOCAL MODEL.

THE PROCEDURE CAN BE DEFINED AS A "COVARIANCE-WEIGHTED MOVING AVERAGE TRENDSURFACE" OF ORE RESERVE ESTIMATION. THE CENTRE OF EACH LOCAL MODEL SHOULD PREFERABLY COINCIDE WITH THE CENTER OF THE BLOCK WHICH MEAN GRADE WE WANT TO ESTIMATE.

****************************************************************************************************

INPUT DATA

1.- NUMBER OF LOCAL MODELS IN THE X (EAST) AND Y (NORTH) DIRECTION.
2.- DISTANCE IN THE X AND Y DIRECTION BETWEEN MODEL CENTRES.
3.- X AND Y CO-ORDINATES OF FIRST MODEL (CENTRE OF FIRST BOTTOM LEFT HAND CORNER BLOCK)
4.- LONGER AXE OF ZONE OF INTERNAL CORRELATION.
5.- SIZE OF SEARCH SECTORS
6.- NUMBER OF DATA POINTS
7.- PARAMETERS (RANGE, SILL AND NUGGET EFFECT) OF VARIOGRAMS IN 9 OR LESS SECTORS.
8.- X, Y CO-ORDINATES AND DEPENDENT VARIABLE DATA SET.

N.B. EAST AND NORTH CO-ORDINATES OF DATA POINTS MUST, RESPECTIVELY, INCREASE TO THE RIGHT AND TO THE TOP OF MAP AREA.

****************************************************************************************************

CONTROL CARDS

CARD 1: TITLE CARD FOR PROJECT FORMAT(8A10)
NEXT: DATA LISTING AND SUB-BLOCK ESTIMATION OPTIONS: (FMT: 2I1)
IF DATA LISTING DESIRED TYPE 1
IF DATA LISTING IS NOT DESIRED TYPE 0
IF ESTIMATION REQUIRED TYPE 1
IF ESTIMATION IS NOT REQUIRED TYPE 0 (SKIP NEXT CONTROL CARD)
NEXT: SUB-BLOCK CONTROL CARD: (FORMAT(I3,2X,13,I3,1X,4F10.0)
-NU. OF SUB-BLOCKS (PER BLOCK) IN X-DIRECTION
-NU. OF SUB-BLOCKS (PER BLOCK) IN Y-DIRECTION
-DISTANCE BETWEEN SUB-BLOCK CENTRES (X-DIRECTION)
-DISTANCE BETWEEN SUB-BLOCK CENTRES (Y-DIRECTION)
-X AND Y COORDINATES OF FIRST SUB-BLOCK CENTER (BOTTOM LEFT)
NEXT: LOCAL MODELS CONTROL CARD INCLUDING: (FORMAT:2I3,3X,5F10.2)
-NUMBER OF MODELS IN X AND Y DIRECTION
-DISTANCE BETWEEN MODELS IN X AND Y DIRECTION
-X AND Y CO-ORDINATES OF FIRST MODEL (CENTRE OF BOTTOM LEFT HAND CORNER BLOCK)
-SIZE OF ZONE OF INTERNAL CORRELATION MAJOR AXE
NEXT: VARIOGRAMS PARAMETERS CARD INCLUDING: (FORMAT:I1,F5.0,4X,7A10)
-INPUT DEVICE NO. FOR VARIOGRAMS PARAMETERS (TYPE 3 FOR DATA ON FILE; TYPE 5 FOR DATA ON CARDS)
- SIZE OF SEARCH SECTORS IN WHICH VARIOGRAMS ARE AVAILABLE. VARIABLE FORMAT FOR VARIOGRAMS PARAMETERS (FMT1)
NEXT: SAMPLE DATA CARD INCLUDING: (FORMAT:2I3,3X,5F10.2)
-NUMBER OF SAMPLE POINTS (NO MORE THAN 100)
-VARIABLE FORMAT CARD FOR DATA SET (FMT12)
NEXT: VARIOGRAM PARAMETERS (FORMAT: FMT1)
-SECTOR NO. JOINED ANTI-CLOCKWISE FROM X AXIS RANGE: CUT-OFF LAG (FROM I.S.V.D.)
SILL: SEMI-VARIANCE CUT-OFF (FROM I.S.V.O)

ANUGGT: NUGGET EFFECT (FROM I.S.V.O.)

NEXT: X, Y CO-ORDINATES AND DEPENDENT VARIABLE (ASSAY VALUE), (FMT 2)

LIST OF ARRAYS (ARR), CONSTANTS (CT) AND VARIABLES (VAR)

X (ARR): X-COORDINATES OF DATA POINTS
Y (ARR): Y-COORDINATES OF DATA POINTS
XM (CT): X-COORDINATE OF FIRST MODEL
YM (CT): Y-COORDINATE OF FIRST MODEL
XMOD1 (VAR): X-COORDINATE OF SUCCESSIVE MODEL CENTRES
YMOD1 (VAR): Y-COORDINATE OF SUCCESSIVE MODEL CENTRES
XMOD (ARR): X-COORDINATE OF EVERY MODEL
YMOD (ARR): Y-COORDINATE OF EVERY MODEL
NX (CT): NUMBER OF MODELS IN EAST DIRECTION
NY (CT): NUMBER OF MODELS IN NORTH DIRECTION
DX (CT): DISTANCE BETWEEN MODEL CENTRES IN EAST DIRECTION
DY (CT): DISTANCE BETWEEN MODEL CENTRES IN NORTH DIRECTION
XINC (CT): INCREMENT TO DISPLACE DATA POINTS THAT COINCIDES WITH MODEL CENTRES

ANGL (CT): SIZE OF SEARCH SECTORS
GR (ARR): ASSAY VALUES OF RAW DATA
COEFF (ARR): NUMBER OF DATA POINTS IN EACH MODEL
GF (ARR): MODELS' GOODNESS OF FIT
CC (ARR): MODELS' CORRELATION COEFFICIENT
NM (CT): TOTAL NUMBER OF MODELS
GLGR (ARR): GLOBAL ESTIMATED GRADE OF BLOCKS

VECTOR X, Y COORDINATES AND DEPENDENT VARIABLE (ASSAY VALUE), (FMT 2)

READ PROJECT TITLE (DTITLE)
READ SUB-BLOCK ESTIMATION OPTION AND APPROPRIATE INFORMATION
READ NUMBER OF MODELS IN X(NX) AND Y(NY) DIRECTION
READ DISTANCE BETWEEN MODELS IN X(DX) AND Y(DY) DIRECTION
READ X (X1I) AND Y (YM) CO-ORDINATES OF FIRST MODEL
READ MAJOR RANGE OR AXE OF ZONE OF INTERNAL CORRELATION (AXE)

READ INPUT DEVICE NO. FOR VARIOGRAM PARAMETERS (INDEVI)
READ SIZE OF SEARCH SECTORS (ANGL)
READ FORMAT FOR VARIOGRAM PARAMETERS (FMT1)

READ INPUT DEVICE NO. FOR DATA SET (INDEVI)
READ NUMBER OF SAMPLE POINTS (NS)
READ FORMAT FOR DATA SET (FMT2)

READ SAMPLE CO-ORDINATES (X,Y) AND GRADE (GR)

DO 15 I=1,NS
  READ (INDEVI,FMT2) X(I),Y(I),GR(I)
15 CONTINUE

LIST (IF SELECTED) INPUT DATA SET

IF (IOPTL.EQ.0) GO TO 17

DO 16 I=1,NS
  WRITE (6,540) I,X(I),Y(I),GR(I)
16 CONTINUE
WRITE(6,560)

READ AND WRITE VARIOGRAMS PARAMETERS
17 DO 25 I=1,NSECT
   READ(INOEVI,FMT1)ISECT,RANGE(ISECT),SILL(ISECT),ANUGGT(ISECT)
   WRITE(6,580)ISECT,RANGE(ISECT),SILL(ISECT),ANUGGT(ISECT)
25 CONTINUE

*******************************************************************************
STARTING OF MAIN DO LOOP TO FIND THE CENTRE OF EACH BLOCK IN TURN AND OBTAIN LOCAL MODELS.
*******************************************************************************

NMODS=0
DO 25 J=1,NY
   YM001= YM+(J-1)*DY
   NM0DS=NM00S+1
   CALL OF MAIN SUBROUTINE SEARCH TO LOOK FOR POINTS FALLING WITHIN THE ZONE OF INTERNAL CORRELATION CENTRED AT XM001(K) AND YM001(J)
   CALL SEARCH(X,Y,GR,XMOO1,YMOD1,RANGE,SILL,ANUGGT,NS,NSMOD,MLIST,COEFF,NSECT,NM,GF,GC,STE)
25 CONTINUE

L=NMODS
CALL SUBROUTINE TO CARRY OUT GLOBAL ESTIMATION OF BLOCK GRADE
   CALL GLOBAL(XMOO,YMOO,COEFF,MLIST,nx,0Y,L,GLGR)
WRITE OUT THE FILE CONTAINING X AND Y COORDINATES, NUMBER OF DATA POINTS, COEFFICIENTS, STANDARD ERRORS AND ESTIMATED MEAN BLOCK GRADE OF EACH LOCAL MODEL
   WRITE (6,582)
   WRITE (6,584)
   WRITE (6,586)

LL=0
DO 55 I=1,L
   IF(MLIST(I).LE.4) GO TO 55
   WRITE(6,590)XMOO(I),YMOD(I),MLIST(I),GF(I),CC(I),GLGR(I),STE(I)
   WRITE(6,600) (COEFF(I,J),J=1,10)
   LL=LL+1
55 CONTINUE

WRITE INTO TAPE 4 THE REQUIRED INFORMATION FOR SUB-BLOCK ESTIMATION
   WRITE (4,3)LL,NSX,NSY,NX,NY,SBDX,S30Y,XSD1,YSB1,0X,DY
   DO 65 I=1,L
      IF(MLIST(I).LE.4) GO TO 65
      WRITE(4,590)XMOO(I),YMOD(I),MLIST(I),GF(I),CC(I),GLGR(I),STE(I)
      WRITE (4,600) (COEFF(I,J),J=1,10)
   65 CONTINUE

STOP
1 FORMAT(2I1)
2 FORMAT(13,2X,I3,1X,4F10.0)
3 FORMAT(14,13,2X,I3,1X,4F10.0)
10 FORMAT (8A10)
20 FORMAT (2I3,3X,5F10.2)
30 FORMAT (11,F5.1,4X,7A10)
A subroutine to sort the data points that fall within the zone of internal correlation

This subroutine calls other subroutines:

ANGLE: to calculate azimuth between model center and each data point.

TEST: to test for points (NPM) to be included in the model. It also calculates the covariance function to weight these points and select the order of the model accordingly, e.g.:

IF: NPM \leq 4 NO MODEL IS FITTED
4 \lt NPM \leq 9 A FIRST ORDER MODEL IS FITTED
9 \lt NPM \leq 15 A 2ND ORDER MODEL IS FITTED
15 \lt NPM A 3RD ORDER IS FITTED

POLYNUM: to fit a 1ST ORDER TREND SURFACE POLYNOMIAL TO THE SELECTED COVARIANCE WEIGHTED POINTS AND CALCULATE THE STANDARD ERRORS AND COEFFICIENTS

DIMENSION XC(100), YC(100), GR(100), GF(1000), CC(1000), STE(1000), DATA(100, 5), COEFF(1000, 10), RANGE(NSECT), SILL(NSECT), ANUGGT(NSECT), MLIST(NM), B(30)

COMMON ANGL, DTITLE(8), AXE, XINC

ZERO ARRAYS: DATA
DO 5 L=1,100
DATA(L,1)=0.00
DATA(L,2)=0.00
DATA(L,3)=0.00
5 CONTINUE

CALCULATE VECTOR RELATIONSHIP BETWEEN EACH SUCCESSIVE MODEL CENTER AND EACH SAMPLE POINT

N=0
SUMRD=0.00
SUMMD=0.00

DO 25 K=1,NS
6 VCOMP1=XMOO1 - XC(K)
VCOMP2=YMOO1 - YC(K)

SKIP DATA POINTS THAT ARE LOCATED AT A LONGER DISTANCE FROM MODEL CENTER THAN LONGER RANGE

D=SQRT((VCOMP1**2)+(VCOMP2**2))
IF(D,GT,AXE) 50 TO 25

IF DATA POINT COINCIDES WITH MODEL CENTER, DATA POINT IS SHIFTED BY 1/1000 OF DISTANCE BETWEEN MODEL CENTERS ALONG EAST DIRECTION

IF (D,NE,0.00)GO TO 7
XC(K)=XC(K)+XINC
GO TO 6

CALL SUBROUTINE TO CALCULATE AZIMUTH(AZMTH) BETWEEN MODEL CENTER AND EACH SAMPLE POINT

7 CALL ANGLE(VCOMP1,VCOMP2,AZMTH)
ASSIGN AZIMUTH TO AN INTEGER SECTOR (KSECT)

IF(AZMTH .GT. 183.) AZMTH = AZMTH - 180.
KSECT = AZMTH / ANGL
KSECT = KSECT + 1

CALL SUBROUTINE OTEST TO TEST WHETHER DATA POINT FALL WITHIN
THE RANGE OF THE ZONE OF INTERNAL CORRELATION IN THE APPROPRIATE SECTOR.
CALL OTEST(GR,0,RANGE,SILL,ANUGGT,KSECT,GRMW,K,NSECT,NS)

RH = RANGE(KSECT)
IF(RH .LT. D) GO TO 25

N = N + 1
DATA(N,1) = XC(K)
DATA(N,2) = YC(K)
DATA(N,3) = GRMW
SUMR = SUMR + GR(K)
SUMWD = SUMWD + DATA(N,3)

25 CONTINUE

ASSIGN COUNTER OF NO. OF DATA POINTS WITHIN ZONE OF INTERNAL CORRELATION TO LIST OF COUNTERS (MLIST) FOR EACH MODEL

MLIST(NMODS) = N

CHECK FOR THE NUMBER OF POINTS TO BE INCLUDED IN
THE MODEL AND SELECT ORDER ACCORDINGLY. IF FOUR OR LESS
DATA POINTS A MESSAGE IS PRINTED AND NO MODEL IS FITTED. ALSO,
THE VALUES OF GOODNESS-OF-FIT, CORRELATION COEFFICIENT AND
TREND EQUATION COEFFICIENTS ARE SET TO 0.0

IF(N .GT. 4) GO TO 27
QF(NMODS) = 0.0
G2(NMODS) = 0.0
DO 26 I = 1, NMODS
      COEFF(NMODS,I) = 0.0
26 CONTINUE

27 IORD = 0
IQ = 4
IP = 9
IQ = 15

IF(N-IQ) 430, 400, 100
100 IF(N-IP) 500, 500, 200
200 IF(N-IQ) 600, 600, 300
300 IORD = IORD + 3
GO TO 610
400 WRITE(6, 3) N, NMODS, XM01, YM01
RETURN
500 IORD = IORD + 1
GO TO 610
600 IORD = IORD + 2

CALCULATE MEAN VALUE OF RAW AND SMOOTHED DATA

AMEAN = SUMR / FLOAT(N)
AMEANW = SUMWD / FLOAT(N)

CALCULATE RATIO OF RAW TO SMOOTHED DATA AND ADJUST
VALUES TO OBTAIN AN EQUAL MEAN POLYNOMIAL MODEL

RATIO = (AMEAN/AMEANW)
DO 50 I = 1, N
      DATA(I,3) = DATA(I,3) * RATIO
50 CONTINUE

CALL SUBROUTINE TO FIT A LOCAL POLYNOMIAL OF IORDER

CALL POLYNOM(IORD,DATA,B,N,R2,R)

CALCULATE STANDARD ERROR (STE)

SE = FLOAT(N)
STOE = (1-R**2)/SQRT(SE)
ASSIGN THE CALCULATED LOCAL MODEL COEFFICIENTS (B) TO ARRAY COEFF(NMODS,10)

DO 60 I=1,10
   COEFF(NMODS,I)=B(I)
60 CONTINUE

ASSIGN MODEL FITTING PARAMETERS AND STANDARD ERROR TO ARRAYS GF, CC AND STE

GF(NMODS)=R2
CC(NMODS)=R
STE(NMODS)=STDE

3 FORMAT(///,5X,14HWARNING: ONLY, I4, 14H DATA POINTS
112HA00UT MODEL, I4, 5X, 4HC=, F10.2,
25X, 4H YC=, F10.2, 5X, 22H**NO MODEL IS FITTED**)  

RETURN
END

SUBROUTINE ANGLE(VCOMP1, VCOMP2, AZMTH)

C
C
C A SUBROUTINE TO CALCULATE THE VECTOR ANGLE (ANTICLOCKWISE) BETWEEN THE MODEL CENTER AND SAMPLE POINT

PI=3.14159

10 IF(VCOMP1) 100,40,100
20 IF(VCOMP2) 50,60,70
30 IF(VCOMP2) 100,30,100

100 AZMTH=ATAN2(VCOMP2, VCOMP1)+PI
GO TO 200
40 AZMTH=0.0
GO TO 200
50 AZMTH=PI/2.0
GO TO 200
60 AZMTH=10.0
GO TO 200
70 AZMTH=3.0/(PI*2.0)
GO TO 200
80 AZMTH=PI

C
C CONVERT ANGLE(AZIMUTH) FROM RADIANS TO DEGREES

200 AZMTH=AZMTH*57.29578

RETURN
END

SUBROUTINE OTEST(GR, DK, RANGE, SILL, ANUGGT, KSEC, GRMW, IM, NSECT, NS)

C
C A SUBROUTINE TO TEST WHETHER SAMPLE POINT FALLS WITHIN ZONE OF INTERNAL CORRELATION OF THE APPROPRIATE SECTOR. THEN, IT CALCULATES THE COVARIANCE FUNCTION BETWEEN SAMPLE POINT AND MODEL CENTER AND WEIGHT THE SAMPLE VALUE ACCORDINGLY.

DIMENSION GR(NS), RANGE(NSECT), SILL(NSECT), ANUGGT(NSECT)

RH= RANGE(KSEC)
IF(RH.LT.DK) GO TO 100

SLOPE=(SILL(KSEC)-ANUGGT(KSEC))/RH
COV=SILL(KSEC)-ANUGGT(KSEC)-SLOPE*DK
GRMW=GR(M)*COV

100 RETURN
END

SUBROUTINE POLYNOM(IORD, DATA, B, N, R2, R)
SUBROUTINE TO COMPUTE A POLYNOMIAL TREND SURFACE OF DEGREE IORD.
INPUT DATA MATRIX IS N BY 3, WHERE N IS THE NUMBER OF
OBSERVATIONS. THE FIRST COLUMN CONTAINS Y1 (EAST-WEST OR
ACROSS THE MAP) COORDINATE, THE SECOND COLUMN CONTAINS X2
(NORTH-SOUTH OR DOWN THE MAP), AND THE THIRD COLUMN CONTAINS
THE DEPENDENT VARIABLE.

*DIMENSION A(30,30),B(30),C(30),DATA(100,5)
MM=30
C(1)=1.0

... CALCULATE NUMBER OF COEFFICIENTS
IORD2=(IORD+1)*(IORD+2)/2

... ZERO SLE MATRIX
DO 100 I=1,30
B(I)=0.0
DO 100 J=1,30
A(I,J)=0.0
100 CONTINUE

... CALCULATE SLE MATRIX
DO 111 I=1,N
JB=1
DO 102 J=1,IORD
DO 103 K=1,J
KB=JB+1
C(JB)=C(KB)*DATA(I,1)
103 CONTINUE
JB=JB+1
C(JB)=C(KB)*DATA(I,2)
102 CONTINUE
DO 114 J=1,100
B(J)=3.0+C(J)*DATA(I,3)
DO 104 K=1,IORD2
A(J,K)=A(J,K)+C(J)*C(K)
104 CONTINUE
114 CONTINUE

... SOLVE SYSTEM OF LINEAR EQUATIONS
CALL LINEQUA(A,B,IORD2,MM,1.0E-38)

... CALCULATE ESTIMATED VALUE AND DEVIATION FOR EACH OBSERVATION
DO 105 I=1,N
JB=1
DO 106 J=1,IORD
DO 107 K=1,J
KB=JB+1
C(JB)=C(KB)*DATA(I,1)
107 CONTINUE
JB=JB+1
C(JB)=C(KB)*DATA(I,2)
106 CONTINUE
DATA(I,5)=0.0
DO 108 J=1,IORD2
DATA(I,6)=DATA(I,5)+B(J)*C(J)
108 CONTINUE
DATA(I,5)=DATA(I,3)-DATA(I,4)
105 CONTINUE

WRITE OUT DATA AND COEFF ARRAYS FOR CHECKING
DO 50 I=1,N
WRITE(6,200)(DATA(I,J),J=1,5)
50 CONTINUE
WRITE(6,210)(B(K),K=1,10)

... CALCULATE ERROR MEASURES
SY=0.0
SYY=0.0
SYC=0.0
SYYC=0.0
DO 111 I=1,N
SY=SY+DATA(I,3)
SYY=SYY+DATA(I,3)**2
SYC=SYC+DATA(I,4)
SYYC=SYYC+DATA(I,4)**2
111 CONTINUE
SST=SY-SY*SY/FLOAT(N)
SSR=SYYC-SYC*SYC/FLOAT(N)
SSD=SST-SSR
NDF=IORDD-1
AMSR=SSR/FLOAT(NDF1)
NDF2=N-IORDD2
AMSD=SSD/FLOAT(NDF2)
R2=SSR/SST
R=SQRT(R2)
F=AMSR/AMSD
NDF3=N-1
WRITE OUT STANDARD ERRORS AND FITTING PARAMETERS
WRITE (6,220 ) R2,R,SSR,SSD,SST,NDF1,NDF2,NDF3,F
220 FORMAT(2X,5F10.5)
RETURN

ENG
SUBROUTINE LINEQUA(A,B,N,N1,ZERO)
DIMENSION A(N1,N1),B(N1)
DO 100 I =1,N
DIV=A(I,I)
IF(ABS(DIV)-ZERO) 99,99,1
 1 DO 10 J=1,N
A(I,J)=A(I,J)/DIV
10 CONTINUE
B(I)=B(I)/DIV
100 CONTINUE
RETURN
99 CALL EXIT
END

SUBROUTINE GLOBAL(X,Y,CF,MLIST,DX,DY,L,GLGR)
DIMENSION X(1000),Y(1000),CF(1000,10),MLIST(1000),GLGR(1000)
S Docker = DX/2.0
SDY = DY/2.0
DO 56 I=1,L
XC=X(I)
YC=Y(I)
X1=XC-SDX
Y1=YC-SDY
X2=X1+SDX
Y2=Y1+SDY
X3=X1+SDX
Y3=Y1+SDY
X4=X2
Y4=Y3
A=CF(I,1)
B=CF(I,2)
C=CF(I,3)
D=CF(I,4)
E=CF(I,5)
F=CF(I,6)
G=CF(I,7)
H=CF(I,8)
V=CF(I,9)
W=CF(I,10)

IF NO MODEL HAS BEEN FITTED ON THE BLOCK
NO ESTIMATION IS CARRIED OUT AND GLOBAL GRADE IS
SET TO 0.0

IF(MLIST(I).LE.4) GO TO 40

Z1C=A+X1*B+Y1*C+X1**2*D+X1*Y1*E+Y1**2*F+
   1X1*3*G+X1**2*H+X1*Y1**2*V+Y1**3*W
Z2C=A+X2*B+Y2*C+X2**2*D+X2*Y2*E+Y2**2*F+
   1X2**3*G+X2**2*H+X2*Y2**2*V+Y2**3*W
Z3C=A+X3*B+Y3*C+X3**2*D+X3*Y3*E+Y3**2*F+
   1X3**3*G+X3**2*H+X3*Y3**2*V+Y3**3*W
Z4C=A+X4*B+Y4*C+X4**2*D+X4*Y4*E+Y4**2*F+
   1X4**3*G+X4**2*H+X4*Y4**2*V+Y4**3*W
ZC=A+X*C+B+Y*C+X**2*D+X*Y*C+E+Y**2*F+
   1X**3*G+X**2*H+X*Y**2*V+Y**3*W

SUM=Z1C+Z2C+Z3C+Z4C+ZC
GLGR(I)=SUM/5.00
GO TO 50

40 GLGR(I)=0.0
50 CONTINUE
RETURN
END
PROGRAM PREPROD (OUTPUT=201, TAPE4=201, TAPE6=OUTPUT)

A PROGRAM TO ESTIMATE PREPRODUCTION GRADE OF SUB-BLOCKS WITHIN FITTED LOCAL MODELS. INPUT DATA IS OBTAINED FROM PROGRAM "AXMSBES". FIVE VALUES (FOUR CORNERS AND THE CENTER) OF EACH SUB-BLOCK ARE ESTIMATED AND A MEAN VALUE OF THE MODEL IS ASSIGNED TO SUB-BLOCK GRADE.

IF NO MODEL HAS BEEN PREVIOUSLY FITTED TO A PARTICULAR BLOCK NO SUB-BLOCK ESTIMATION IS CARRIED OUT.

A MAXIMUM OF 100 BLOCKS, EACH ONE WITH 10 SUB-BLOCKS IS ALLOWED.

PROGRAM TO ESTIMATE PREPRODUCTION GRADE OF SUB-BLOCKS WITHIN FITTED LOCAL MODELS. INPUT DATA IS OBTAINED FROM PROGRAM "AXMSBES". FIVE VALUES (FOUR CORNERS AND THE CENTER) OF EACH SUB-BLOCK ARE ESTIMATED AND A MEAN VALUE OF THE MODEL IS ASSIGNED TO SUB-BLOCK GRADE.

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A PROGRAM TO ESTIMATE PREPRODUCTION GRADE OF SUB-BLOCKS WITHIN FITTED LOCAL MODELS. INPUT DATA IS OBTAINED FROM PROGRAM "AXMSBES". FIVE VALUES (FOUR CORNERS AND THE CENTER) OF EACH SUB-BLOCK ARE ESTIMATED AND A MEAN VALUE OF THE MODEL IS ASSIGNED TO SUB-BLOCK GRADE.

IF NO MODEL HAS BEEN PREVIOUSLY FITTED TO A PARTICULAR BLOCK NO SUB-BLOCK ESTIMATION IS CARRIED OUT.

A MAXIMUM OF 100 BLOCKS, EACH ONE WITH 10 SUB-BLOCKS IS ALLOWED.
SBGR(NSB) = SUM/5.0
GO TO 70
65 SBGR(NSB)=-10000.0

C 70 CONTINUE
C 80 CONTINUE
C 90 CONTINUE
100 CONTINUE
C WRITE OUT PRE-PRODUCTION RESULTS
C WRITE(6,4) NSB
C DO 200 I=1,NSB
C   WRITE(6,300) XSB(I),YSB(I),SBGR(I)
200 CONTINUE
C STOP
C 3 FORMAT(I4,I3,2X,I3,1X,2I3,6F10.3)
C 4 FORMAT(I3)
C 5 FORMAT(2F10.0)
C 300 FORMAT(5X, 2F10.2,5X,F10.2,///
C 500 FORMAT(32X,I4,65X)
C 600 FORMAT(5X,1<12.5,///
C END

15.06.05.UCLP, 22, 0.252KLNS.
PROGRAM RAND5ST (INPUT, OUTPUT, TAPES=INP1, TAPE3, 1, TAPES=OUTPUT)

A PROGRAM TO ESTIMATE THE MEAN GRADE OF A SUB-BLOCK OR SET OF SUB-BLOCKS LOCATED AT RANDOM WITHIN THE AREA IN WHICH LOCAL MODELS HAVE BEEN FITTED TO MAJOR SUB-BLOCKS.

INPUT DATA (COEFFICIENTS AND LOCAL MODELS' COORDINATES) IS OBTAINED FROM PROGRAM 'LOCMOOL'. ALSO, THE USER SHOULD PROVIDE NUMBER OF SUB-BLOCKS TO BE ESTIMATED TOGETHER WITH THEIR CENTER'S COORDINATES AND DIMENSIONS IN EAST AND NORTH DIRECTIONS.

UP TO 100 RANDOM SUB-BLOCKS ARE ALLOWED.

DIMENSION MLIST(1000), CF(1000,10), DIST(500), XSB(100), YSB(100), XMOD(1000), YMOD(1000), NP(500), SSBX(100), SSBY(100), XFM(500), YFM(500), CFFM(500,10)

READ RANDOM SUB-BLOCK INFORMATION AND LOCAL MODELS.

READ(5,10)NSB,NMOODS
DO 23 I=1,NSB
READ(5,3C)XSB(I), YSB(I), SBOX(I), SBODY(I)
20 CONTINUE

READ LOCAL MODELS INFORMATION

READ(3,3) L1, L2, L3, L4, L5, D1, D2, D3, D4, D5
DO 40 I=1,NMOODS
READ(3,590)XMOD(I), YMOD(I), MLIST(I)
READ(5,590) (CF(I,J), J=1,10)
40 CONTINUE

SORT OUT THE INFORMATION OF LOCALLY-FITTED MODELS

KK=0
DO 60 I=1,NMOODS
IF(MLIST(I).LE.4) GO TO 60
KK=KK+1
XFM(KK)=XMOD(I)
YFM(KK)=YMOD(I)
NP(KK)=MLIST(I)
DO 50 J=1,10
CFFM(KK,J)=CF(I,J)
50 CONTINUE
60 CONTINUE

CALCULATE DISTANCE BETWEEN EACH RANDOM SUB-BLOCK CENTER AND FITTED MODELS CENTERS. FIND THE NEAREST MODEL AND USE ITS COEFFICIENTS FOR ESTIMATING SUB-BLOCK MEAN GRADE

DO 100 I=1,NSB
DO 95 L=1,500
DIST(L)=0.00
95 CONTINUE

DO 90 M=1,KK
VCOMP1=XSB(I)-XFM(M)
VCOMP2=YSB(I)-YFM(M)
DIST(M)=SQRT((VCOMP1**2)+(VCOMP2**2))
90 CONTINUE

SELECT MINIMUM DISTANCE TO FITTED MODEL CENTERS

DMIN=1.0E+60
DO 88 MN=1,KK
DIST(MN)=DIST(MN)
IF(DIST(MN).LT.DMIN) DMIN=DIST(MN)
88 CONTINUE

CHECK FOR APPROPRIATE COEFFICIENTS OF NEAREST MODEL AND ESTIMATE SUB-BLOCK MEAN GRADE

DO 83 JJ=1,KK
IF(DIST(JJ).NE.DMIN) GO TO 83
A=CFFM(JJ,1)
B=CFFM(JJ,2)
C=CFFM(JJ,3)
D=CFFM(JJ,4)
83 CONTINUE
E=CFFM(JJ,5)
F=CFFM(JJ,6)
G=CFFM(JJ,7)
H=CFFM(JJ,8)
V=CFFM(JJ,9)
W=CFFM(JJ,10)
83 CONTINUE
X0=XSB(I)
Y0=YSB(I)
X1=XSB(I)-(S30X(I)/2.0)
X2=X1+S30X(I)
Y1=YSB(I)-(S30Y(I)/2.0)
Y2=Y1
X3=X1
Y4=X3+S30X(I)
Y3=Y1+S30Y(I)
Y4=Y3
Z1C=A+X1*B+Y1*C+X1*2*G+X1*Y1*E+Y1*2*F+
1*X1*3*G+X1**2*H+X1*Y1**2*V+Y1**3*W
Z2C=A+X2*B+Y2*C+X2*2*G+X2*Y2*E+Y2*2*F+
1*X2*3*G+X2**2*H+X2*Y2**2*V+Y2**3*W
Z3C=A+X3*B+Y3*C+X3*2*G+X3*Y3*E+Y3*2*F+
1*X3*3*G+X3**2*H+X3*Y3**2*V+Y3**3*W
Z4C=A+X4*B+Y4*C+X4*2*G+X4*Y4*E+Y4*2*F+
1*X4*3*G+X4**2*H+X4*Y4**2*V+Y4**3*W
ZC=A+X5*B+Y5*C+X5*2*G+X5*Y5*E+Y5*2*F+
1*X5*3*G+X5**2*H+X5*Y5**2*V+Y5**3*W
SUM=Z1C+Z2C+Z3C+Z4C+ZC
SBGR(I)=SUM/5.0
WRITE(6,615) DMIN
WRITE(6,620) XG,YC,XI,Y1,X2,Y2,X3,Y3,X4,Y4
90 CONTINUE
WRITE OUT RANDOM SUB-BLOCK'S ESTIMATED MEAN GRADE
DO 110 IJ=1,NSB
WRITE(6,610) XSB(IJ),YSB(IJ),SBGR(IJ)
110 CONTINUE
STOP
15.04.2C.UCLP, 22, 0.252KLNS.
PROGRAM E5TDISP(INPUT, OUTPUT, TAPE3, TAPE5=INPUT, 1TAPE5=OUTPUT)
A PROGRAM TO DISPLAY ORE RESERVE ESTIMATION RESULTS FROM LOCAL MODELS METHOD (LOCMOOL), USING GHOST PLOTTING SUBROUTINES.

TWO TONNAGE TABLES ARE PRODUCED: ONE CONSIDERING ACCUMULATION OF TONNES IN DECREASING ORDER OF GRADES AND THE OTHER IN AN INCREASING ORDER OF STANDARD ERRORS. ALSO, A GRAPH IS PRODUCED FOR EACH OF THIS TABLES BY TYPING 1 IN THE FIRST FIELD OF GRADE-TONNAGE CONTROL CARD, IF PLOTTING IS NOT REQUIRED 0 MUST BE TYPED INSTEAD.

INPUT DATA IS THE FILE CONTAINING LOCAL MODELS RESULTS AND VALUES OF TONNAGE FACTOR, BLOCK THICKNESS, RANGE AND CLASS INTERVAL OF GRADES OF INTEREST, MINIMUM AND MAXIMUM STANDARD ERROR TO BE CONSIDERED TOGETHER WITH ITS CLASS INTERVAL.

DIMENSION X(500), Y(500), EG(500), STE(500), TO(400), XG(400), YG(400), EGG(400), SEG(400)
COMMON DTITLE(5)

READ TITLE OF PROJECT AND GRADE-TONNAGE CONTROL CARD:
TF : TONNAGE FACTOR
THK: BLOCK THICKNESS
GL : LOWEST GRADE
GH : HIGHEST GRADE
CL : GRADE CLASS INTERVAL
STEL : MINIMUM STANDARD ERROR
STEH : MAXIMUM STANDARD ERROR
CLIE : STANDARD ERROR CLASS INTERVAL

READ(5,1) DTITLE
WRITE(6,6)
READ(5,12) DTITLE
WRITE(6,12) DTITLE
READ(5,*) IPS, TF, THK, GL, GH, CL, STEL, STEH, CLIE
WRITE(6,2) TF, THK, GL, GH, CL, STEL, STEH, CLIE

READ LOCAL MODEL RESULTS FILE
READ(3,3)L
DO 10 I=1,L
READ(3,590) X(I), Y(I), EG(I), STE(I)
10 CONTINUE

SORT ADEQUATE DATA, STORE IT IN NEW ARRAYS AND CALCULATE MEAN GRADE, MEAN STANDARD ERROR AND BLOCK TONNAGE.
N=0
SUMG=0.0
SUME=0.0
DO 20 I=1,L
IF(EG(I).LT.GL) GO TO 20
IF(EG(I).GT.GH) GO TO 20
IF(STE(I).GT.STEH) GO TO 20
N= N+1
XG(N)= X(I)
YG(N)= Y(I)
EGG(N)= EG(I)
SEG(N)= STE(I)
TO(N)= TF*THK
SJMG= SJMG+EGG(N)
SJME= SJME+SEG(N)
20 CONTINUE

WRITE(6,9)
GM=SJMG/FLOAT(N)
EM=SJME/FLOAT(N)
WRITE(6,4)GM, EM

CALCULATE VARIANCES OF SELECTED GRADES AND STANDARD ERROR
VG=0.0
VSE=0.0
DO 30 K=1,N
VG=VG+(EGG(K)-GM)*(EGG(K)-GM)
VSE=VSE+(SEG(K)-EM)*(SEG(K)-EM)
30 CONTINUE
VG=VG/FLOAT(N-1)
VSE=VSE/FLOAT(N-1)
WRITE (6,5) VG, VSE
PRINT OUT SELECTED BLOCKS TO BE INCLUDED IN THE RESERVES
WRITE (6,6)
WRITE (6,12) DTITLE
WRITE (6,7)
DO 40 L = 1, N
WRITE (6,9) L, XG(L), YG(L), TO(L), EGG(L), SEG(L)
40 CONTINUE
SWITCH ON GRAPH PLOTTER (OR LINEPRINTER)
CALL PAPER(1)
CALL SUBROUTINE GRACUM TO ACCUMULATE TONNAGE FROM THE HIGHEST GRADE TO THE LOWEST
CALL GRACUM(N, TO, EGG, GL, CLIG, IPS)
IPS=IPS+1
CALL SUBROUTINE STECLAS TO CLASSIFY RESERVES IN TERMS OF THE STANDARD ERROR.
CALL STECLAS(N, TO, EGG, SEG, STEL, CLIE, IPS)
SWITCH OFF GRAPH PLOTTER (OR LINEPRINTER)
CALL GRENDO
STOP
1 FORMAT (5A10)
2 FORMAT (15X, 16HTONNAGE FACTOR =, E12.5, 5X, 14HLONGEST GRADE =, E12.5, 5X, 22HCLASS INTERVAL =, E12.5, 5X, 53HSTANDARD ERROR CLASS INTERVAL =, E12.5, ///)
3 FORMAT (14, 75X)
4 FORMAT (10X, 22HMMEAN ESTIMATED GRADE =, F10.3, //, 110X, 21HMEEAN STANDARD ERROR =, F10.3, ///)
5 FORMAT (10X, 23HVARIANCE OF ESTIMATED GRADES =, F10.5, //, 110X, 22HVARIAIANCE OF STANDARD ERRORS =, F10.5, ///)
6 FORMAT (14, 75X)
7 FORMAT (30X, 37H ** BLOCKS INCLUDED IN THE RESERVES **, ///, 15X, 9HBLOCK NO., 7X, 9H-COORD, 13X, 9HY-COORD, 13X, 9HTONNAGE, 214X, 9HGRADE, 10X, 14HSTANDARD ERROR, ///)
8 FORMAT (5X, 14, 5(I4), F10.2, //, 110X, 21HMEAN ESTIMATED GRADE =, F10.3, //, 110X, 21HMEEAN STANDARD ERROR =, F10.3, ///)
9 FORMAT (10X, 31HVARIANCE OF ESTIMATED GRADES =, F10.5, //, 110X, 22HVARIAIANCE OF STANDARD ERRORS =, F10.5, ///)
50G FORMAT (5X, 2F10.2, 46X, 2(5X, F10.2), ///)
END
SUBROUTINE GRACUM(N, T, G, GMIN, CL, IPS)
C
C A SUBROUTINE TO ACCUMULATE TONNAGE FROM THE HIGHEST GRADE TO THE LOWEST
C
DIMENSION T(N), G(N), AC(50), NPG(30), ACT(50), XX(50), YY(50)
COMMON UTITLE(5)
C
C ACT(L): ACCUMULATED TONNAGE OF BLOCKS IN GRADE GROUP L
AC(L): ACCUMULATED ACCUMULATIONS OF BLOCKS IN GRADE GROUP L
NPG(L): NO. OF BLOCKS IN GRADE GROUP L

DO 10 L = 1, 50
AC(L) = 0.0
ACT(L) = 0.0
NPG(L) = 0
10 CONTINUE

JA: NUMBER OF THE GRADE GROUP = (GRADE-GMIN)/CL+1

DO 20 I = 1, N
JA = (G(I)-GMIN)/CL+1
IF(JA.LT.1) JA=1
IF(JA.GT.50) JA=50
NPG(JA) = NPG(JA)+1
ACT(JA) = ACT(JA)+T(I)
AC(JA) = AC(JA)+G(I)*T(I)
20 CONTINUE
C
WRITE (6,6)
WRITE (6,7) DTITLE
WRITE(6,8)
WRITE(6,13)
WRITE(6,14)

CALCULATE THE SUM OF THE ACCUMULATIONS (A1), THE TOTAL METAL IN THE RESERVES (TM1), MEAN GRADE OF RESERVES (G1), SUM OF THE TONNAGES IN DESCENDING ORDER OF GRADE (TT).

A1=0.0
TM1=0.0
TT=0.0
G1=0.0

DO 30 M=1,50
K=51-M
F=GMIN+FLOAT(K-1)*CL
IF(NPG(K).EQ.Q) GO TO 25
A1=A1+AC(K)
TM1 = A1/1GG .C
TT=TT+ACT(K)
G1=A1/TT
30 WRITE (6,12)F,ACT(K),TM1,TT,G1

DO 30 CONTINUE

ASSIGN GRADE CLASS INTERVAL (F) AND ACCUMULATED TONNAGE (TT) TO ARRAYS XX AND YY FOR PLOTTING GRADE-TONNAGE CURVE

XX(K)=K
YY(K)=TT

35 CALL PLOT(XX,YY,CL,IPS)

RETURN
END
SUBROUTINE STECLAS(N,T,A,D,C,S,IPS)

A SUBROUTINE TO CLASSIFY RESERVES OF SELECTED GRADE ACCORDING TO THE STANDARD ERROR, ACCUMULATING FROM THE LOWEST TO THE HIGHEST STANDARD ERROR.

DIMENSION T(N),A(N),D(N),M(50),TH(50),AC1(50),
1XX(50),YY(50)
COMMON DTITLE(5)

CLEAR ARRAYS
DO 10 I=1,25
M(I)=0
TH(I)=0.0
AC1(I)=0.0
10 CONTINUE

DO 20 L=1,N
JA = (L-1)*S+1
IF(JA.LT.1)JA=1
IF(JA.LT.25)JA=25
M(JA)=M(JA)+1
TH(JA)=TH(JA)+T(L)
AC1(JA)=AC1(JA)+A(L)*T(L)
20 CONTINUE
C 20 CONTINUE
  WRITE(6,6)
  WRITE(6,7) DTITLE
  WRITE(6,8)
  WRITE(6,13)
  WRITE(6,14)
  DO 30 J=1,25
  F= C+F*FLOAT(J)*S
  IF (M(J).EQ.0) GO TO 22
  A1=A1+AC1(J)
  TM1= A1/100.0
  TT=TT+TH(J)
  G1=1/TT
  22 WRITE(6,5)F,TH(J),TM1,TT,G1
C 30 CONTINUE
C TEST IF PLOTTING IS REQUIRED
  IF(IPS.NE.2) GO TO 35
C CALL SUBROUTINE PLOT TO PRODUCE STANDARD ERROR-TONNAGE CURVE
  CALL PLOT(XX,YY,S,IPS)
C A SUBROUTINE TO PLOT GRADE-TONNAGE AND STANDARD ERROR-TONNAGE CURVES USING GHOST PLOTTING ROUTINES
C ARRAY Y CONTAINS ACCUMULATED TONNAGE
  DIMENSION X(50),Y(50)
  COMMON DTITLE(5)
  IF(IPS.EQ.1) NX=50
  IF(IPS.NE.1) NX=25
  PX=FLOAT(NX)
C FIND THE MAXIMUM VALUE OF Y
  YMAX=0.0
  DO 1 I=1,NX
    Y=Y(I)
    IF(Y.GT.YMAX) YMAX=Y
  CONTINUE
C TEST FOR TYPE OF PLOTTING REQUIRED
  IF(IPS.NE.1) GO TO 50
C PLOT GRADE - TONNAGE CURVE
  DO 2 I=1,50
    Y=FLOAT(I)
    IF(Y.GT.0.0) GO TO 3
  CONTINUE
  2 CALL PTPL0T(X,Y,1,NX,45)
C CALL PTPLÜT(X,Y,1,NX,-45)
  CALL AXES
  CALL PLACE(36,10)
  CALL TYPECS(19'H GRADE-TONNAGE CURVE',19)
  CALL PLACE(3,10)
  CALL TYPECS(7 'H TONNAGE',7)
CALL PLACE(35,74)
CALL TYPECS(16, HGRADE INTERVAL = , 16)
CALL PLACE(52,74)
CALL TYPENF(CI, 3)
CALL PLACE(20,77)
CALL TYPECS(DTITLE, 50)

GO TO 100

50 CALL FRAME

PLOT STANDARD ERROR - TONNAGE CURVE

DO 60 I = 1, NX
  YI = Y(I)
  IF(YI.GT.0.0) GO TO 65
60 CONTINUE
65 CALL PTPL0T(X, Y, 1, NX, 45)
   CALL PTPL0T(X, Y, 1, NX, -45)
   CALL AXES
   CALL PLACE(50, 10)
   CALL TYPECS(28, STANDARD ERROR - TONNAGE CURVE, 28)
   CALL PLACE(3, 10)
   CALL TYPECS(7, TONNAGE, 7)
   CALL PLACE(25, 74)
   CALL TYPECS(25, STANDARD ERROR INTERVAL = , 25)
   CALL PLACE(51, 74)
   CALL TYPENF(CI, 3)
   CALL PLACE(20, 77)
   CALL TYPECS(DTITLE, 50)
C
100 RETURN
END

15.08.00, UCLP, 22, 0.504KLNS.
PROGRAM JOLEVAL (INPUT=301, OUTPUT=301, TAPE8)

VERSION OF 15 SEPTEMBER 1978.

PROGRAM JOLEVAL IS A PROBABALISTIC MODEL FOR FINANCIALLY EVALUATING PROPOSED MINING INVESTMENTS.

********* TO RUN THIS PROGRAM ON THE UNIVERSITY OF LEICESTER CYBER 72 THE FOLLOWING JOB CONTROL IS REQUIRED: *********

JOB CARD: (IDENTIFIER, CM12000, TIME.)
USER CARD
CHARGE CARD.
GET, JOLEVAL.
GET, DATAFILE.
RFL, 60000.
MMF, I=JOLEVAL, OR FTN, I=JOLEVAL.
ATTACH, HBOOK, UTILIB/UN=LIBS.
LOSET, LI=HBOOK/UTILIB.
LOG, DATAFILE.
END OF RECORD CARD.
ALTERNATIVELY J0LSUB2 MAY BE USED.

********* INPUT TO THIS MODEL CAN BE ANY COMBINATION OF POINT ESTIMATES NORMAL DISTRIBUTIONS, AND/OR DENSITY FUNCTIONS. THE SIMULATOR CAN BE USED FOR EITHER SENSITIVITY OR PROBABALISTIC RISK ANALYSIS. *********

********* INPUT DATA REQUIREMENTS. *********

OPTION CARD - ONE REQUIRED FOR EACH SET OF SIMULATIONS

COL 1-2 LIFE OF OPERATION IN YEARS (45 YEARS MAXIMUM)
BLANK = END OF JOB.

COL 5-8 NUMBER OF SIMULATIONS DESIRED (MAXIMUM=1000).

COL 11 METHOD OF DEPRECIATION OPTION.
FOR STRAIGHT LINE
FOR DOUBLE DECLINING BALANCE
FOR SUM OF THE YEARS DIGITS

COL 14-23 PROPERTY VALUE (VALUE OF LAND)

COL 26-29 NUMBER OF LIST SUMMARIES REQUIRED.

COL 32 OPTION TO CLEAR OR RETAIN PARAMETER VALUES AT THE END OF A SET OF SIMULATIONS.
RETAIN VALUES
CLEAR VALUES

COL 35 INVESTMENT TAX CREDIT OPTION
INVESTMENT TAX CREDIT
NO INVESTMENT TAX CREDIT

COL 38 TAXATION OPTION
FOR MONTANA STATE TAX SYSTEM
FOR FAVORABLE TAX SYSTEM
FOR UNFAVORABLE TAX SYSTEM

COL 40-50 DESIRED YIELD ON INVESTMENT FOR NET PRESENT VALUE CALCULATION

COL 77 BLANK
COL 78 PLUS SIGN
COL 79 PERIOD (FULL STOP)
COL 80 ASTERISK

********* PARAMETER CARDS: A MAXIMUM OF FORTY-FIVE INPUT PARAMETERS CAN BE ENTERED FOR EACH YEAR. FOR THE FIRST OPTION OR SET OF SIMULATIONS ALL INPUT PARAMETERS HAVE INITIAL VALUES OF ZERO. PARAMETER VALUES ASSIGNED DURING ANY SET OF SIMULATIONS MAY BE OPTIONALLY RETAINED FOR SUBSEQUENT SIMULATIONS *********
THE LAST PARAMETER ENTRY IN EACH OPTION OR SET OF SIMULATIONS MUST BE FOLLOWED BY A BLANK CARD TO BEGIN THE SIMULATION PHASE. THIS IN TURN IS FOLLOWED BY ANOTHER OPTION CARD, WHICH IS REPLACED BY A BLANK CARD TO INDICATE THE END OF THE JOB.

THE CATEGORIES FOR GRADE, RECOVERY, AND PRICE ARE NOT MUTUALLY EXCLUSIVE. THIS ALLOWS UP TO FIVE DIFFERENT VALUES FOR THESE PARAMETERS TO BE CONSIDERED SIMULTANEOUSLY, OR THE VALUES TO BE VARIED WITH THE LIFE OF THE MINE. THE CATEGORIES FOR TONNEAGE ARE MUTUALLY EXCLUSIVE AND ONLY ALLOW VARYING MINING RATES TO BE CONSIDERED DURING THE LIFE OF THE MINE.

<table>
<thead>
<tr>
<th>COL 1-2 CATEGORY</th>
<th>LIST OF CATEGORIES:</th>
</tr>
</thead>
<tbody>
<tr>
<td>BLANK = BEGIN SIMULATION.</td>
<td></td>
</tr>
<tr>
<td>1-5 = GRADE</td>
<td></td>
</tr>
<tr>
<td>6-10 = RECOVERY</td>
<td></td>
</tr>
<tr>
<td>11-15 = PRICE</td>
<td></td>
</tr>
<tr>
<td>16-20 = TONNEAGE</td>
<td></td>
</tr>
<tr>
<td>21-25 = CAPITAL INVESTMENT (MINING)</td>
<td></td>
</tr>
<tr>
<td>26-30 = CAPITAL INVESTMENT (PROCESSING)</td>
<td></td>
</tr>
<tr>
<td>31-34 = WORKING CAPITAL</td>
<td></td>
</tr>
<tr>
<td>35-39 = OPERATING COST (MINING)</td>
<td></td>
</tr>
<tr>
<td>40-45 = OPERATING COST (PROCESSING)</td>
<td></td>
</tr>
</tbody>
</table>

| COL 5-44 ALPHABETIC DESCRIPTION OF CATEGORY |
| COL 45-49 FIRST YEAR OF ESTIMATE |
| COL 50-54 LAST YEAR OF ESTIMATE |
| COL 55-56 NUMBER OF DATA POINTS |
| FOR POINT ESTIMATE = 1 |
| FOR NORMAL DISTRIBUTION = 2 |
| FOR DENSITY FUNCTION = 3-25 (REQUIRES 3 TO 25 CARDS) |

| COL 57-58 YEARS CAPITAL INVESTMENT DEPRECIATED |
| COL 61-70 POINT ESTIMATE VALUE OR MINIMUM VALUE OF PARAMETER |
| COL 71-80 MAXIMUM VALUE OF PARAMETER FOR NORMAL DISTRIBUTION |

**X-Y CARDS:** ONE CARD REQUIRED FOR EACH (X,Y) PAIR. IT IS RECOMMENDED THAT THE DENSITY FUNCTIONS BE DESCRIBED WITH AT LEAST FIVE (X,Y) POINTS, THREE OF WHICH MUST HAVE POSITIVE Y-VALUES.

**NOTE:** THE SUMMATION OF ALL THE Y-VALUES USED IN DESCRIBING THE FUNCTION MUST EQUAL ONE, AS Y IS A PROBABILITY FUNCTION.

**NOTE:** IN THE OUTPUT LIST SUMMARY ALL VALUES FOR TONNEAGE THROUGH TO CASH FLOW ARE IN THOUSANDS.

**LIST OF VARIABLES USED IN PROGRAM JOLEVAL**

<table>
<thead>
<tr>
<th>ASTP:</th>
<th>AN ASTERISK USED IN PLOTTING</th>
</tr>
</thead>
<tbody>
<tr>
<td>ATAX:</td>
<td>SUMMATION OF LOCAL OR STATE TAXES</td>
</tr>
<tr>
<td>BLNK:</td>
<td>A BLANK USED IN PLOTTING</td>
</tr>
<tr>
<td>CARY:</td>
<td>TAX LOSS CARRIED FORWARD</td>
</tr>
<tr>
<td>CASH:</td>
<td>CASH FLOW FOR A PARTICULAR YEAR</td>
</tr>
</tbody>
</table>
13.6H YEARS / 25H PROPERTY VALUE
1.15H MONETARY UNITS / 25H NUMBER OF SIMULATIONS:

IF (METH = 1) 100, 120, 140
GO TO 160

120 PRINT 130
130 FORMAT (1H+, 25X, 13H STRAIGHT LINE)
GO TO 160

140 IF (ITC LT 1) GO TO 170
PRINT 170
170 FORMAT (IH, INVESTMENT TAX CREDIT)

* READ PARAMETER CARD

190 READ 190, NCAT, NAME, NFYR, NTYR, N, L, X(1), Y(1)
190 FORMAT (I2, 2X, 2U2, 4(I2, 2X), 2F10.0)
IF (NCAT) 70, 250, 200

* DETERMINE COEFFICIENTS FOR POINT ESTIMATE OR NORMAL DISTRIBUTION

200 DO 220 J=NFYR, NTYR
   LL=NCAT-2
   IF (LL.LT.1) GO TO 210
   IF (LL.GT.12) GO TO 210
   YEAK(LL, J) = L
210 IF (N.GT.2) GO TO 230
   COEF(NCAT, J, 1) = X(1)
   COEF(NCAT, J, 2) = Y(1)
220 CONTINUE

* LINEAR REGRESSION TO COMPUTE COEFFICIENTS FOR DENSITY FUNCTION

230 READ 240, (X(I), Y(I), I=1, N)
240 FORMAT (2F10.0)
CALL DENFUNK(X, Y, N, BLNK, PERD, PLUS, ASTR, COEF, YCAL, WT, Z, ZN,
   SUMY, R, NFYR, NTYR, NCAT, NAME)
GO TO 180

* DETERMINE RANDOM VALUES FOR SIMULATION

250 DO 260 I=1, 123
   PT(I) = 0.0
260 CONTINUE
   PCT = 0.0
   IF (LIFE.GT.7) GO TO 330
   GO TO (270, 270, 270, 280, 280, 290, 290), LIFE
270 PCT = 0.0
   GO TO 300
280 PCT = 0.0
   GO TO 300
290 PCT = 0.46667
   ZN = 0.0
   DO 310 J = 1, LIFE
   SLVG = 0.0
   ATAX = 0.0
   STAX = 0.0
   FTAX = 0.0
   DO 310 J = 1, LIFE
   DEPR(I, J) = 0.0
   DEPR(2, J) = 0.0
   CARY(J) = 0.0
   VALU(J) = 0.0
310 CONTINUE
   CUSH = 0.0
   SUMC = 0.0
   CUPX = 0.0
   SUMCUP = 0.0
   SUMCUPX = 0.0
   DO 490 J = 1, LIFE

DO 350 I=1,45
IF(COEF(I,J,2)) 340,320,330

* POINT ESTIMATE
320 VALU(I)=COEF(I,J,1)
   GO TO 350

* NORMAL DISTRIBUTION
330 STDV=(COEF(I,J,2)-COEF(I,J,1))/6.0
   Z=0.0
   YRAN=RANF(K)
   Z=Z+YRAN*6.0
   VALU(I)=(Z-3.0)*STDV+(COEF(I,J,1)+COEF(I,J,2))/2.0
   GO TO 350

* DENSITY FUNCTION
340 YRAN=RANF(K)
   VALU(I)=(-ALOG(1.0/YRAN-1.0)-COEF(I,J,2))/COEF(I,J,1)
   IF(VALU(I),LE,0.0) GO TO 340
   CONTINUE

* DEPRECIATION SCHEDULES
360 IF(METH=1)350,370,380
   CALL STRAIT(YEAR,VALU,SLVG,LIFE,DEPR,J)
   GO TO 390
370 CALL DOUBLE(YEAR,VALU,SLVG,LIFE,DEPR,J)
   GO TO 390
380 CALL SOYD(YEAR,VALU,SLVG,LIFE,DEPJ,PROP,J)

* SELECTION OF DESIRED TAX ROUTINE
390 IF(TAX=1)400,410,420
400 CALL MONTAX(ATAX,CUMI,CUPR,CUMC,DEPL,DEPR,FTAX,ITC,J,
   PCT,PROP,TINC,STAX,CARY,VALU,WORK)
   GO TO 430
410 CALL NOTAX(ATAX,CUMI,CUPR,CUMC,DEPL,DEP2,FTAX,J,
   PROP,TINC,STAX,VALJ,WORK)
   GO TO 430
420 GO TO 400

* COMPUTE CASH FLOW
430 CASH(J)=TINC*DEPR(1,J)*DEPR(2,J)+DEPL-FTAX-
   VALU(21)-VALU(27)-VALU(33)-VALU(34)+WORK
   IF(J,GT.1) GO TO 440
   CASH(J)=CASH(J)-PROP
440 IF(LIFF,GT.J) GO TO 450
   CASH(J)=CASH(J)+SLVG+CJWC

* LIST YEARLY SUMMARY
450 IF(LS,LE,6) GO TO 490
   CALL RATE(CASH,RR,J)
   IF(J,GT.1) GO TO 470
   PRINT 460
460 FORMAT(1H1,1* YEAR GRADE RECOV PRICE MINCOST*,
   2* PROD TONNAGE DEPLET ATAX STAX*,
   2* FTAX INVEST DEPEX CSFLO RETURN*)
470 VALU(16)=VALU(16)/1000.0
   CMIN=VALU(35)+VALU(36)+VALU(37)+VALU(38)+VALU(39)
   CPRO=VALU(40)+VALU(41)+VALU(42)+VALU(43)+VALU(44)+
   VALU(45)
   DEPL=DEPL/1003.0
   ATAX=ATAX/1000.0
   STAX=STAX/1000.0
   FTAX=FTAX/1000.0
   XX=(VALU(21)+VALU(27))/1000.0
   SX=(DEPR(1,J)+DEPR(2,J))/1000.0
   SY=CASH(J)/1000.0
   CUSH=CUSH+SY
**EXPECTED RATE OF RETURN HISTOGRAM**

510 CALL RATE(CASH,RR,LIFE)
JS=RR+1.5
PT(JS)=PT(J5)+1.0
ZN=ZN+RR
DO 520 J=1,LIFE
SUMDCF=SUMDCF+(CASH(J)/(1+DRR)**(J-1))
520 CONTINUE.

IF(SUMDCF)<0,9,9
PRINT 6
FORMAT(///10X,3H*WARNING* : THE SUM OF DCF IS NEGATIVE)
7 PRINT 8
8 FORMAT(///10X,3H*WARNING* : THE SUM OF DCF IS ZERO)
GO TO 570

9 SZ=SUMDCF/1000.0
PROI=SUMDCF/CUMI+1
WGR=DRR*(LOG(PROI)/LIFE)
IF(LS.LE.0) GO TO 560
PRINT 530,SZ,WGR

530 FORMAT(///1H,3HNET PRESENT VALUE OF INVESTMENT = ,F10.0,
6H FOR ,F6.4,19H X100 PERCENT YIELD)
540 FORMAT(///1H,22HPROFITABILITY INDEX = ,F10.4)
550 FORMAT(1H,21HWEALTH GROWTH RATE = ,F12.4,
4HFOR ,F6.4,18HX100 PERCENT YIELD)
560 RRR(I)=RR
WGRR(I)=WGR*100
PROIR(I)=PROI
NPV1(I)=SZ

570 CONTINUE.
Z=0.0
ZN=ZN/NS
PRINT 580,NS

580 FORMAT(///H,33X,16HEXPECTED RATE OF RETURN DISTRIBUTION//
3X,4HPERCENT RATE OF RETURN HAS OF OCCURRING BASED ON,
5X,14H SIMULATIONS**/)
DO 620 I=1,125
PT(I)=PT(I)/NS*100.0
 IF(PT(I).LE.0.0) GO TO 600
JS=PT(I)+0.5
Z=Z+PT(I)
J=I-1
PRINT 590,J,PT(I),(ASTR,K=1,JS)
600 IF(Z.GE.9999) GO TO 630
610 FORMAT(1H )
620 CONTINUE.
630 PRINT 640,ZN
640 FORMAT(///H,22HMEAN RATE OF RETURN = ,F6.2)

**RATE OF RETURN, WEALTH GROWTH RATE, PROFITABILITY INDEX, AND NET PRESENT VALUE HISTOGRAMS.**

CALL HIST(NS,RR,WGR,PRO1,HPV)

RETURN TO READ NEW OPTION CARD
**MAIN SUBROUTINE FOR PLOTTING OF HISTOGRAMS**

******************************************************************************

COMMON XTITLE(4)
DIMENSION RR1(1000), WGR1(1000), PRO11(1000), A(4,1000), B(4),
1 XCHAN(4), XNCOL(4), AMIN(4), WEIGHT(4), TITLE(8),
2 TITLED(8,4), NPV1(1000)

INTEGER XCHAN,XNCOL
REAL NPV1

SET NO OF CARDS (NC) EQUAL TO NUMBER OF SIMULATIONS; NUMBER OF VARIABLES TO FOUR.

NCENS
NV=4

SET VARIABLE NAMES.
B(1)="RR1"
B(2)="WGR1"
B(3)="PRO11"
B(4)="NPV1"
XTITLE(1)="HISTOGRAMS"
XTITLE(2)="OF FINANCE"
XTITLE(3)="IAL PARAMETERS"
XTITLE(4)="TERTS."
DO 26 IB=1,NV
WRITE(8,13) (XTITLE(IB), IB)
13 FORMAT(4A10,30X,A10)
26 CONTINUE

REWIND 8
DO 49 IC=1,NV
READ(8,30) (TITLEB(IX,IC), IX=1,8)
30 FORMAT(8A10)
49 CONTINUE

NCOL=2
NCHAN=50
VALMIN=0.000
WEIGHT=10.0
DO 59 J=1,NV
XNCOL(J)=NCOL
XCHAN(J)=NCHAN
AMIN(J)=VALMIN
WEIGHT(J)=WEIGHT
59 CONTINUE

PRINT 60, (B(J), J=1,4)
60 FORMAT(4H1,12X,*14,4(8X,A5))

FIRST SHIFT ORIGIN OF BLANK COMMON TO 5.

CALL HI5TG0(5)

CALL HISPREP(JL,NC,A,TITLE,NCHAN,XMIN,XMAX,WEIGHT)

TO BOOK AND FILL HISTOGRAMS

DO 120 I=1,NC
PRINT 110, I, (A(NV,I), NV=1,4)
110 FORMAT(1H,10X,I3,3(3X,F10.5),1X,F14.3)
120 CONTINUE

PRINT 130
130 FORMAT(1H,18HRR1=RATE OF RETURN1H,23HWGR1=WEALTH GROWTH RATE, 1
1H,1H,25HPRO11=PROFITABILITY INDEX,
2  IH ,22HNPV1=NET PRESENT VALUE)

CALL STANDARD HISTOGRAM PLOTTING PACKAGE
CALL HISPLTNV,XNCOL)
RETURN
END
SUBROUTINE HISPREP(ID,N,A,TITLE,NCHAN,XMIN,XMAX,WEIGHT)

COMPILATION OF STANDARD ROUTINES TO BOOK AND FILL HISTOGRAMS

FOR FURTHER INFORMATION SEE:
HPLOT USERS GUIDE FOR VERSION 1.5 CERN NOV 1976
HBOOK USERS GUIDE FOR VERSION 2.0 CERN MAY 1975

DIMENSION A(4,1000),TITLE(8)
I=0
BOOKING
CALL HB001(ID,TITLE,NCHAN,XMIN,XMAX)
FILLING FROM DATA ARRAY A(I,J)
DO 20 J=1,N
   DO 10 I=1,4
      CALL HFILL(I,A(I,J),0.0,WEIGHT)
   CONTINUE
20 CONTINUE
RETURN
END

SUBROUTINE HISPLOT(NV,XNCOL)

COMPILATION OF STANDARD SUBROUTINES TO PLOT HISTOGRAMS

INTEGER XNCOL
DIMENSION XNCOL(4)

EDITING
CALL INDEX
DO 10 ID=1,NV
   NCOL=XNCOL(ID)
   CALL HBIGBI(ID,NCOL)
10 CONTINUE
PRINTING OF HISTOGRAMS
CALL HPRINT(ID)
RETURN
END

SUBROUTINE OENFUN(X,Y,N,BLNK,PERO,PLUS,ASTR,COEF,YCAL,WT,Z,ZN,1
SUMY,R,NFYR,NTYR,NCAT,NAM3)
DIMENSION X(25),Y(25),COEF(45,45,2),YCAL(25),WT(45),SUMY(2),
R(2,4),PT(35),NAME(20)

SUBROUTINE OENFUN USES LINEAR REGRESSION TO COMPUTE THE COEFFICIENTS
FOR DENSITY FUNCTIONS IN INPUT DATA, AND PLOTS A GRAPH OF CUMULATIVE
PROBABILITY VERSUS VALUE.

Z=0.0
DO 20 I=1,N
   Z=Y(I)+Z
   IF(Z.LE.0.0) GO TO 10
   IF(Z.GE.1.0) GO TO 10
   YCAL(I)=ALOG(Z/(1.0-Z))
   WT(I)=Z*(1.0-Z)+2
   GO TO 20
10 YCAL(I)=0.0
   WT(I)=0.0
20 CONTINUE
SUMY(1)=0.0
SUMY(2)=0.0
R(1,1)=0.0
R(2,2)=0.0
DO 30 I=1,N
   SUMY(1)=SUMY(1)+YCAL(I)*WT(I)
   SUMY(2)=SUMY(2)+YCAL(I)*WT(I)*X(I)
   R(1,1)=R(1,1)+WT(I)*X(I)
   R(1,2)=R(1,2)+WT(I)*X(I)
   R(2,2)=R(2,2)+X(I)**2
30 CONTINUE
R(2,1) = R(1,2)
R(2,2) = R(2,2) + W T(I) * X(I) * X(I)

30 CONTINUE
R(1,3) = 1.0
R(2,4) = 1.0
DO 70 I = 1,2
LL = I + 1
DO 40 J = LL,4
R(I,J) = R(I,J) / R(I,I)
40 CONTINUE
DO 60 J = 1,2
IF (J .EQ. I) GO TO 60
DO 50 K = LL,4
R(J,K) = F M J, K) - R(J,I) * R(I,K)
50 CONTINUE
60 CONTINUE
70 CONTINUE
Z = R(1,3) * SUMY(1) + R(1,4) * SUMY(2)
ZN = R(2,3) * SUMY(1) + R(2,4) * SUMY(2)
DO 80 J = NFYR, NTYR
COEF(NCAT, J, 2) = Z
COEF(NCAT, J, 1) = ZN
80 CONTINUE

INSTRUCTIONS FOR DOUBLE CURVE GRAPH

Z = X(1)
ZN = X(N)
PRINT 100, NAME
100 FORMAT (1H1, 45X, 2GA2)
DO 110 I = 1, N
X(I) = X(I) - Z
110 CONTINUE
SY = 0.0
DO 120 I = 1, N
IF (SY .GE. YCAL(I)) GO TO 120
SY = YCAL(I)
120 CONTINUE
SXY = 11A, 00 /X(N)
SY = 50.0 / SY
DO 220 J = 1, N
PT(1) = PERD
DO 140 J = 2, 119
IF (I .LT. 51) GO TO 130
PT(J) = PERD
GO TO 140
PT(J) = BLNK
130 CONTINUE
IF (MOD(I, 10) .NE. 1) GO TO 160
PT(121) = PERD
PT(123) = PERD
IF (I .NE. 1) GO TO 150
PT(124) = "1"
PT(126) = "0"
GO TO 160
PT(122) = "0"
IF (I .EQ. 1) PT(124) = "8"
IF (I .EQ. 21) PT(124) = "6"
IF (I .EQ. 41) PT(124) = "2"
IF (I .EQ. 51) PT(124) = "0"
160 CONTINUE
IF (MOD(I, 10) .NE. 1) PT(121) = BLNK
IF (MOD(I, 10) .NE. 1) PT(123) = BLNK
IF (MOD(I, 10) .NE. 1) PT(124) = BLNK
IF (MOD(I, 10) .NE. 1) PT(124) = BLNK
DO 200 J = 1, N
JS = YCAL(J) * SY + 1.5
JF = X(J) * SXY + 1.5
IF (JS .GT. 0) GO TO 170
JS = 1
JF = 52 - JS
IF (JF .LT. 194, 186, 190)
PT(J) = PLUS
190 CONTINUE
JS = Y(J) * SY + 1.5
JF = 52 - JS
IF (JF .LT. 194, 186, 190)
PT(J) = PLUS
DIMENSION Y E A R (12,45) , V A L U (45) , D E P R (2,45)
***********************************************************
C DOUBLE DECLINING BALANCE METHOD
DO 30 I = 1 , 12
XX=YEAR(I,J)
IF (XX .LE .0 ) GO TO 30
OLD=VALU(I+20)
SLVG=SLVG+OLD
JJ=I+I/12
JI=XX
JJ=J+JI-1
IF (LIFE .GE .JI) GO TO 10
JJ=LIFE
10 HOLD=OLD/XX
DO 20 L = J , J I
DEPR(JJ,L)=DEPR(JJ,L)+HOLD
OOPL=OLD*HOLD
DEPR(JJ,L)=DEPR(JJ,L)+WORK
SLVG=SLVG-WORK
CONTINUE
20 CONTINUE
RETURN
END
DIMENSION Y E A R (12,45), VALU(45), DEPR(2,45)
***********************************************************
C SUM OF THE YEARS DIGITS METHOD
DO 50 I = 1 , 12
XX=YEAR(I,J)
1 IF (XX .LE .0 ) GO TO 50
OLD=VALU(I+20)
SLVG=SLVG+OLD
JJ=I+I/12
JI=XX
JJ=J+JI-1
IF (LIFE .GE .JI) GO TO 10
JJ=LIFE
10 WORK=0.0
N=XX
DO 20 K = 1 , N
XX=K
WORK=WORK+XX
20 CONTINUE
SUBROUTINE RATE (CASH, RR, NY)

SUBROUTINE RATE CALCULATES THE RATE OF RETURN ON INVESTMENT BY USING THE BISECTION METHOD.

DIMENSION CASH(45)
RX=.000000
HOLD=0.0
PREV=0.0
10 Z=0.0
IF(RR.LE.1.18) GO TO 30
PRINT 20
20 FORMAT(10X,34H RATE OF RETURN EXCEEDS 118 PERCENT)
GO TO 90
30 DO 40 J=1,NY
Z = Z + CASH(JJ)*(EXP(RR) - 1.)/(RR*EXP(RR*JJ))
40 CONTINUE
IF(Z .LT. 90.90, 70, 40
RATE ESTIMATE IS HIGH (GREATER THAN 118 PERCENT)
50 IF(HOLD.GT.0.0) GO TO 60
P%=C.0
GO TO 90
60 PREV=HOLD
HOLD=RR
AVL=ABS(RR-PREV)
IF(AVL.LE.0.005) GO TO 90
PREV=AVL/2.0
GO TO 10
RATE ESTIMATE IS LOW (NEGATIVE OR ZERO)
70 IF(PREV.GT.0.0) GO TO 80
HOLD=RR
RR=RR+1.
GO TO 10
80 PREV=HOLD
HOLD=RR
AVL=ABS(RR-PREV)
IF(AVL.LE.0.005) GO TO 90
RR=RR+AVL/2.0
GO TO 10
90 RR=RR*100.0
RETURN
END

SUBROUTINE MONTAX (ATAK, CUMI, CUPR, CUWC, DEPL, DEPR, FTAX, ITC, J, PCT, PCTT, TINC, STAX, CARRY, VALU, WORK)

DIMENSION CARY(45), CPT(45), DEPJ(2,45), VALU(45), TEMP(5)

MONTANA TAX ROUTINE -- REFERENCE MONTANA STATE TAX COMMISSION

DO 30 I=1,35,45
COST=COST+VALU(I)
30 CONTINUE
COST=COST*VALU(16)
DO 20 I=1,5
TEMP(I)=VALU(I)*VALU(I+5)*VALU(I+10)*VALU(16)
20 CONTINUE

GROS = TEMP(1) + TEMP(2) + TEMP(3) + TEMP(4) + TEMP(5)

REAL ESTATE TAX

ATAX = PROP * YVEL

METALLIFEROUS MINES TAX

WORK = GROS
HOLD = WORK - 500000
IF (HOLD <= 0.5) GOTO 30
ATAX = ATAX + 0.125 * HOLD
WORK = 500000
HOLD = WORK - 400000
IF (HOLD <= 0.5) GOTO 40
ATAX = ATAX + 0.125 * HOLD
WORK = 400000
HOLD = WORK - 300000
IF (HOLD <= 0.5) GOTO 50
ATAX = ATAX + 0.125 * HOLD
WORK = 300000
HOLD = WORK - 250000
IF (HOLD <= 0.5) GOTO 60
ATAX = ATAX + 0.125 * HOLD

PERSONAL PROPERTY TAX

XX = 0.12
ATAX = ATAX + XX * CUMI * YVEL
HOLD = ATAX

IMPROVEMENT TAX

IF (J > 3) GOTO 70
XX = 0.02
ATAX = ATAX + XX * CUPR * YVEL
WORK = GROS - COST - 0.024 * CUPR - DEPR(1, J)
IF (WORK <= 3.0) GOTO 90
ATAX = ATAX + WORK - YVEL

PERCENTAGE DEPLETION ALLOWANCE

DEPL = 0.1
TINC = GROS - DEPR(1, J) - DEPR(2, J) - COST - ATAX
IF (TINC <= 0.0) GOTO 100
IF (0.5 * TINC <= 0.15 * GROS) GOTO 90
DEPL = 0.15 * TINC
GOTO 100
DEPL = 0.0
TINC = TINC + DEPL

CORPORATION LICENSE TAX

HOLD = (GROS - HOLD - DEPL) * 0.045
IF (HOLD <= 0.0) GOTO 110
ATAX = ATAX + HOLD
TINC = TINC + HOLD

UNUSED TAX LOSS CARRIED FORWARD

110  WORK = J, J
IF (TINC) 230, 240, 120
120  IF (J = LE 1) GOTO 150
JJ = J - 1
IF (J = LE 6) GOTO 130
J1 = J - 5
130  DO 150 1 = J1, JJ

150  END
IF(TINC.GT.CARY(I)) GO TO 140
   CARY(I)=CARY(I)-TINC
   GO TO 240
TINC=TINC+CARY(I)
   WORK=WORK+CARY(I)
   CARY(I)=0.0
CONTINUE

MONTANA STATE TAX
***************************************************************************
   STAX=0.055*TINC
   TINC=TINC-STAX
***************************************************************************
FEDERAL INCOME TAX
***************************************************************************
IF(TINC.GT.25000.) GO TO 160
   FTAX=0.22*TINC
   GO TO 170
160   FTAX=5500.+.48*(TINC-25000.)
***************************************************************************
INVESTMENT TAX CREDIT
***************************************************************************
170   IF(I.T.EQ.0) GO TO 260
   J=1
180   IF(J.LE.7) GO TO 180
   J=J-6
   XX=FTAX
190   DO 210 I=J,J
      IF(XX.GT.CRDT(I)) GO TO 200
      CRDT(I)=CRDT(I)-XX
      FTAX=(FTAX-25000.)/2.0
   200      CRDT(I)=0.0
      CONTINUE
   IF(FTAX.LE.25000.) GO TO 220
   XX=XX+(FTAX-25000.)/2.0
220   FTAX=XX
   GO TO 260
230   CARY(J)=-TINC
240   STAX=0.0
250   FTAX=0.0
260   RETURN
END

SUBROUTINE NOTAX(ATAX,CUMI,CUPR,CUWC,DEPL,DEPR,FTAX,J,
   1     PROP,TINC,STAX,VALU,WORK)
***************************************************************************
   DIMENSION DEPR(2,45),VALU(5),TEMP(5)
***************************************************************************
SUBROUTINE IS A ROUTINE WHICH SETS ALL TAXES TO ZERO
***************************************************************************
   VALU(21)=VALU(21)+VALU(22)+VALU(23)+VALU(24)+VALU(25)+VALU(26)
   VALU(27)=VALU(27)+VALU(28)+VALU(29)+VALU(30)+VALU(31)+VALU(32)
   CUMI=CUMI+VALU(21)
   CUPR=CUPR+VALU(27)
   CUWC=CUWC+VALU(34)+VALU(33)
   COST=0.0
   DO 10 I=35,45
      COST=COST+VALU(I)
   10   CONTINUE
   VALU(16)=VALU(16)+VALU(17)+VALU(18)+VALU(19)+VALU(20)
   COST=COST+VALU(16)
   DO 20 I=1,5
      TEMP(I)=VALU(I)*VALU(I+5)*VALU(I+10)*VALU(16)
   20   CONTINUE
   GROS=0.0
   GROS=GROS+TEMP(1)+TEMP(2)+TEMP(3)+TEMP(4)+TEMP(5)
   YVEL=0.0
   ATAX=YVEL*PROP
   STAX=YVEL*GROS
   FTAX=YVEL*GROS
   WORK=0.0
   TINC=GROS-DEPR(1,J)-DEPR(2,J)-COST-ATAX
   DEPL=.0
   TINC=TINC-STAX-FTAX+DEPL
   RETURN
END
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The table represents a series of values in five columns, each with a range of values from 0 to 1.50, increasing by 0.10 in each column.
\[ \text{Log } = 500 \text{ m} \]

Graphs of \( \text{Al}_2\text{O}_3 \) and \( \text{SiO}_2 \) concentrations for two different samples (m1 and m2) are shown. The x-axis represents time, and the y-axis represents the concentration levels.
$\log = 500m$

$\text{Al}_2\text{O}_3 - \text{m5}$

$\text{SiO}_2 - \text{m5}$

$\text{Al}_2\text{O}_3 - \text{m6}$

$\text{SiO}_2 - \text{m6}$
$\text{Al}_2\text{O}_3 - m7$

$\text{SiO}_2 - m7$

$\text{Al}_2\text{O}_3 - m8$

$\text{SiO}_2 - m8$
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**CUMULATIVE CASH FLOW = 487342.**

**NET PRESENT VALUE OF INVESTMENT = 35642. FOR 1000 X 100 PERCENT YIELD**

**PROFITABILITY INDEX = 1.1867**

**WEALTH GROWTH RATE = 10.65% FOR 1000 X 100 PERCENT YIELD**

**EXPECTED RATE OF RETURN DISTRIBUTION**

**CURVE REPRESENTS THE PROBABILITY A LISTED PERCENT RATE OF RETURN HAS OF OCCURRING BASED ON 250 SIMULATIONS**

11 OCCURS 21.2 PCT
12 OCCURS 78.8 PCT

**MEAN RATE OF RETURN = 11.72%**
HISTOGRAMS OF FINANCIAL PARAMETERS.

**HISTORY**

**ID = 1**

**DATE 22/02/80.**

**NO = 1**

| CHANNE LS | 10 | 1 | 0 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 | 12 | 13 | 14 | 15 | 16 | 17 | 18 | 19 | 20 | 21 | 22 | 23 | 24 | 25 | 26 | 27 | 28 | 29 | 30 | 31 |
|-----------|----|---|---|---|---|---|---|---|---|---|---|----|----|----|----|----|----|----|----|----|----|----|----|----|----|----|----|----|----|----|----|----|----|----|----|
| CONTENTS  | 10 | 1 | 0 | 2 | 4 | 6 | 8 | 10| 12| 14| 16| 18| 20| 22| 24| 26| 28| 30| 32| 34| 36| 38| 40| 42| 44| 46| 48| 50| 52| 54| 56| 58| 60 |
| LOW EDGE  | 10 | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10| 11| 12| 13| 14| 15| 16| 17| 18| 19| 20| 21| 22| 23| 24| 25| 26| 27| 28| 29| 30| 31| 32| 33| 34 |

**ENTRIES = 1000**

**ALL CHANNELS = 9920E+04**

**UNDERFLOW = 8500E+02**

**OVERFLOW = 3**

**BIN WID = 3200E-01**

**MEAN VALUE = 1173E+02**

**R, M, S = 2481E+00**

**ABNORM CHA = 0**
HISTOGRAMS OF FINANCIAL PARAMETERS.

DATE 22/02/80.

NO = 1

CHANNELS 10

idl

CONTENTS 10

LOW EDGE 10

* ENTRIES = 750
* ALL CHANNELS = .7500E+04
* UNDERFLOW = 0.
* OVERFLOW = 0.
* BIN WID = .1280E-01
* MEAN VALUE = .1002E+02
* K.M.S. = .6825E-01
* AUNOK CNA = 0.
REFERENCES


Lepeltier, C., 1969. A simplified statistical treatment of geo-chemical data by graphical representation. Econ. Geol. 64, pp. 538-550


ABSTRACT - The data collected in projects of exploration and evaluation of mineral resources is spatially dependent. It is in this context that its analysis should be carried out. The calculation of the Geostatistical semivariance function in as many directions and lags as the data allows permits to identify different degrees of data's internal organization by using iso-semivariance diagrams. Structures within the data are outlined by these diagrams together with their relative position, dimensions and orientations. The technique is arrived at through the development of several computer programs that calculate the semivariance function, transform co-ordinates and produce contourings. Trials on simulated and geochemical exploration data show the close relationship between data's structure and derived functions.

The diagrams are also used to assess goodness-of-fit in polynomial data modelling and to develop a computerized procedure for data filtering as predictive models of spatial variability. This procedure consists of the locally fitting of covariance-weighted regression models. Its applicability as a new method of ore reserve estimation is demonstrated with bore hole data from a porphyry copper and a stratabound base metal deposit. Limitations to the technique are imposed by inadequate sampling patterns or impossibility for defining realistic iso-semivariance diagrams. A practical application of alternative conventional Kriging methods in global evaluation of a Bauxite deposit is also presented.

From these results, the application of probability analysis in financial appraisal of mining ventures clearly defines the deposit as an important and feasible project.