I00091

A REVIEW OF THE PROCEDURES FOR INTERPRETATING ASSAY RESULTS AND ORE RESERVE ESTIMATION FOR THE DOLPHIN MINE AT KING ISLAND

by

T. QUINLAN

Chatswood

October, 1976

160002

TABLE OF CONTENTS

σ4

INTRODUCTION

The concentration of tungstic oxide and molybdenum within the Dolphin orebody has been measured by assaying samples of drill core. Programs of check assaying have shown that the measured concentration varies not only between drill holes but between:

1. the left and right halves of drill core;
2. samples taken from the same interval of crushed and pulverized drill core;
3. samples of differing weights which may have been taken from material which has been crushed to different sizes;
4. different laboratories which may or may not use different analytical methods;
5. replicate determinations which have been made by the same laboratory.

The average value for the variability, or variance, which may be associated with any one of these five sources may be small; however the effect of each may be additive, and their sum may be a significant portion of the variance within the orebody.

This report describes an attempt to assess the variances related to these five sources and to relate them to that for the whole orebody. In order to gain confidence in the estimation of an ore resource it has now become customary to continue to reassay check samples throughout the course of an ore blocking program. There is no reason to ignore the results of the check assaying, and in fact there are strong reasons why this information should be incorporated into the estimation process on a routine basis. Thus, it is proposed that Recommended Grades, rather than the original assay values, be used for an ore resource estimate.

160005

SUMMARY

The results of the assaying and check assaying procedures for the Dolphin Mine on King Island are reviewed and discussed in relation to the ore reserve estimation procedures.

CONCLUSIONS

1. The results of replicate and check assaying samples taken from diamond drill core confirm the traditional view that there are difficulties in assaying for tungsten.

2. These difficulties and the variation in the reported tungstic oxide content of replicate samples are attributed to inhomogeneities in the distribution of mineralization through the host rock.

3. This variation consists of two components, namely the accuracy and the precision of assaying.

4. A request to a laboratory for an "accurate" determination will yield a precise result. Such precision is not apparent in the results of routine assaying.

5. The total error associated with an ore reserve computation is related to the variance components at all levels of sampling the orebody. At a high level is the placement of drill holes. The lowest level is the sampling of finely ground material for the preparation of an XRF pellet.

6. Imprecision in assaying contributes less than 10 percent to the total error associated with an ore resource computation.

7. A greater contribution to the total error comes from the variability of mineralization between adjacent metre lengths of core. This is assessed to be 60 of the total.

8. The relative magnitude of these errors can be determined fairly readily. However methods have yet to be developed to determine the absolute value for the error in estimating the grade for individual mining blocks.

9. Consequently it is not possible, at this time, to assess the financial implications of these relative levels of uncertainty.

10. The traditional view is that the absolute magnitude of sampling and assaying errors, as measured by a coefficient of variation, must be less than an arbitrary limit, say 8 percent.

11. This view is challenged in that demands for high precision:
(a) do not necessarily produce accurate assay results;
(b) do not remove the source of errors which are not conspicuous and which are more significant;
(c) unecessarily increase the cost of assaying.

12. Nevertheless gross errors cannot be tolerated.

13. Analysis and interpretation of assaying results, which is presented in this report indicates that there have been changes in the level of accuracy and precision of sampling and assaying over a period of time.

14. Some lack of precision is attributed to the quantity of material used at each stage of the process to reduce the sample weights.

15. Several methods for the estimation of grades for mining blocks were examined. A three-dimensional technique for elliptical search and weighting has been implemented.

16. It is not possible as yet to calculate the estimation errors for grades of mining blocks, nor has it been possible to provide a proper reconciliation of these estimates with mining grades.

17. The spacing between diamond drill holes is adequate for resource computations. However it is inadequate for a reserve computation, which must then include the results of grade control sampling.

RECOMMENDATIONS

1. The procedures for the quality control of assaying, which includes check and umpire assaying, used by Geopeko and the K.I.S. Assay Laboratory be maintained and consolidated. In particular:

   (a) by determining the minimum quantity of material required for each stage of the sampling procedures, for diamond drill core and for piles of broken ore, in order to achieve an appropriate standard error.

   (b) by extending the check assaying procedures to the sampling of broken ore, and to establish the components of variation in mine sampling.

   (c) monitor fluctuations in the precision of assaying samples from the orebody by the regular assaying of standard rock powders.

2. To investigate the feasibility of other assaying procedures, such as the use of energy dispersive techniques using a multi-channel analyser to determine the WO_3 concentrations of material crushed to say 3/8 inch, in order to reduce the variance due to periodic changes in bias and precision.

3. To determine the absolute error for estimates of the metal content for mining blocks.

4. To implement a routine for the reconciliation of the estimated and the mining grades for blocks.

CONCEPTS

Error is an inherent factor in all types of measurement, and the admission of its existence is not necessarily an admission of incompetence, but rather than the equipment and techniques used have practical limitations. Miesch (1966) has drawn attention to the types of error, and the concepts of accuracy, precision and bias, which are associated with the estimation of mean values or grades.

ACCURACY

The term accuracy implies close agreement between the average of a large number of estimates and the correct value. In practice, this correct value is never known, but if it were, it would be the value derived by some technique which is inherently more likely to be correct than the technique used to derive the estimate.

PRECISION

The term implies reproducibility, or close agreement among replicate measurements, and is thus a measure of the sample handling, and laboratory procedures, as well as the performance of the equipment used for measurement.

The very great reduction in sample, from 5,000 grammes of drill core to the 0.01 grammes used by conventional XRF techniques, is the procedure which is most vulnerable to error of this type. However, it can always be overcome by additional sampling and analysis.

BIAS

The difference between the average of a large number of assays, and the correct value is called bias. It may be one of two types, a systematic bias which is present in all assays, or a non-systematic bias which is present in some but not in other batches of assays.

In practice, a systematic bias in instrumental results may result from an inadequate calibration curve for the instrument. A non-systematic bias may be the result of slight changes in analytical procedures, or the operating environment which occur from time to time. For example, inadequate corrections for mass absorption coefficients may cause a non systematic bias for batches of samples which have different mineralogical compositions. Of the three types of error, it is the most difficult to detect, because the correct value is never known. However it is the easiest to correct, by using the results of umpire assays.

VARIANCE

The relative magnitude of these errors may be determined by making more than one measurement, with more than one technique, using one or more samples, which have been prepared from the same interval of drill core, which in this case is of one metre length. It can be measured by calculating the variability, or variance, of the results.

It is a statistical device for describing the degree of variation among a set of assays. If X_j is the jth measurement in a group of n assays ($1 \leq j \leq n$), then the variance is estimated by:

$$s_x^2 = \frac{1}{n-1}\sum_{j=1}^{n}(x_j - \bar{x})^2$$

where $\bar{x}$ is the average of the n values.

By using a proper statistical design it is possible to readily calculate the variance which is associated with a lack of precision or accuracy.

ESTIMATION ERRORS

The effects of imprecision or inaccuracy are additive, in that an estimate may be:

a) both precise and accurate;
b) neither precise nor accurate
c) precise but not accurate;
d) accurate but not precise.

160001

BASIC DATA

This interpretation of the accuracy and precision of assaying is based on the results of several experiments, which were provided by Geopeko, King Island. These experiments were:

1. Replicate assaying of left and right halves of drill core from 30 to 62 metres in DDH - D240/3. Each metre length of half core was crushed and then split into four portions. Each portion was then pulverized and assayed by KIS for WO_3 and Mo (Appendix A). Umpire assays for eleven portions were provided by AMDEL and ASCL laboratories.

2. Assaying of replicate samples taken from crushed half core. One sample was selected for reassaying from each group of ten samples assayed from the following drill holes:

   D00/1,2,3
   D40/1,2,3,4
   D80/1,3,4,5
   D120/1,4,5
   D160/1,2,3,4,6,7
   D200/1,2
   D240/2

   These 100 gramme samples were taken from crushed core, each sample was pulverised and split into two 50 gramme samples. The six samples were then submitted to KIS for analysis, and umpire assays were provided by AMDEL and ACSL laboratories.

3. Repeated assaying of material, crushed to 3/8" and 1/8". Replicate assays were obtained for several samples from each of the size ranges.

4. Replicate assaying of 50, 100, 150, 200 and 250 gramme samples split from pulverized material.

5. Repeated XRF readings for the same pellets for several samples.

The basic data used for the estimation of the grades for mining blocks was the set of assay results from the surface and underground diamond drilling and the grab samples of broken ore which are collected by K.I.S. for grade control purposes.

ANALYSIS OF DATA

The first step in the analysis of the assay data which was provided, is to determine the statistical frequency distribution of grade values in the orebody. From this we obtain an estimate of the mean grade and the total variance.

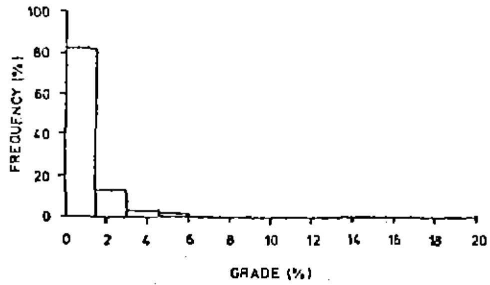
The accuracy and precision of assaying was determined by the statistical methods of principal component analysis, and analysis of variance which compares the analytical results from several laboratories for the same samples.

A regression analysis was used to determine if the KIS assaying is biased. This was done by using an estimate of the "correct" grade for each sample which was calculated during the analysis of the accuracy of assaying. The degree of confidence which can be placed on individual and the check assays is shown by the confidence limits about this bias curve.

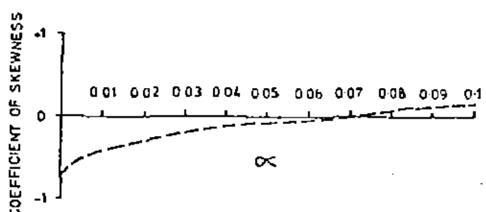
Parameters were calculated which would allow an assessment of the quantity of assaying for batches of samples. This follows the method proposed by Sichel in which he established criteria for the acceptance or rejection of the results obtained by assaying a batch of samples. An assessment was made of the variability of grade in the orebody through the use of geostatistical techniques. The results of the analysis were translated into a practical form by computing the probable errors for the estimation of the grade for mining blocks of various sizes. Finally a method is described for the computation of Recommended Grades, which makes use of the results obtained by check assaying.

STATISTICAL FREQUENCY DISTRIBUTION

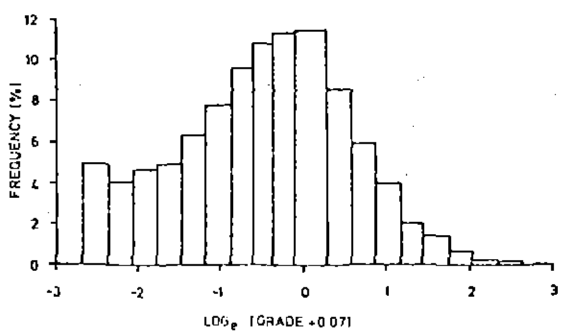
A frequency distribution was constructed for all core assays (Fig. 1A); this is markedly asymmetric, and it indicates that the assay values from the orebody are not normally distributed. Sichel (1966) has indicated that a three parameter logarithmatic normal distribution is approximate in



A. ASSUMPTION OF A NORMAL DISTRIBUTION



B. COMPUTATION OF THE CONSTANT ∝



C. ASSUMPTION OF A THREE PARAMETER NORMAL DISTRIBUTION

FREQUENCY DISTRIBUTIONS FOR ASSAY VALUES

FIG: 1

such cases. Accordingly all assay values were transformed according to the relation:

$$g = \log_e (x + \alpha)$$

where x is the original assay value and α is a constant which is to be determined.

Several trial values for α were used to calculate the coefficient of skewness, which is a measurement of the degree of symmetry of the distribution. A value of 0.07 for the constant α produces a satisfactorily small value for the coefficient of skewness (Figure 1B), and a reasonably symmetric frequency distribution. (Figure 1C). The mean value for assays of samples taken from the orebody was then estimated to be 0.56% WO_3, and their logarithmetic variance was computed to be 1.15.

ACCURACY

The term accuracy implies close agreement between the average of a large number of estimates and the correct value. In practice this correct value is never known, but if it were, it would be the value derived by some technique which is inherently more likely to be correct than the technique used to derive the estimate.

The method of principal component analysis was used to rank the assay laboratories into their order of relative accuracy (Table 2). The steps in this procedure were:

1. Computation of the correlation coefficient between each of the three laboratories (Table 1). A three parameter logarithmetic transformation was used to stabilise the variances for the computation, in which a value of 0.07 was used for the parameter alpha.

2. Extract the two most significant principal components from the correlation matrix. These components represent the coordinate ones which can be used as a framework to describe the variation between laboratories. Because the data used for the analysis consisted

160016

TABLE 1

Correlation coefficients for WO_3 determinations between analytical laboratories.

	K.I.S.	AMDEL	ACSL
K.I.S.	1.00	0.93	0.95
AMDEL	0.93	1.00	0.96
ACSL	0.95	0.96	1.00

TABLE 2

Relative accuracy and precision of assay techniques for WO_3.

LABORATORY	COMPONENT LOADINGS	
	CORRECT VALUE (ACCURACY)	PRECISION
AMDEL	0.82	-0.55
ACSL	0.77	-0.62
K.I.S.	0.58	-0.82

solely of measurements for the tungstic oxide content of samples, one component must be related to the correct values for the samples. In practice these correct values will never be known. The second component must represent the accuracy for the techniques of the three laboratories. The loadings on the component axes then can be used as weighting factors. Firstly to make estimates for the "correct" values, and secondly to rank the assaying techniques by their accuracy (Table 2). The most accurate technique will be that which has the smallest loading on the second component axis.

The "correct" value for each sample interval is a weighted average of the determinations by each laboratory, using the factor weights. A bias curve (Fig. 2) was then computed for the King Island Scheelite laboratory. The ratio of this "correct" value and the appropriate K.I.S. average for two determinations is plotted for 225 replicate samples in Figure 3. The samples are plotted in the sequence in which they were assayed, and even though this is not an absolute time scale, the resultant plot indicates a periodic change in bias from September 1974 to July 1975.

PRECISION OF ASSAYING

The term precision of assaying is used to describe the degree of similarity between several results obtained by sampling and assaying the same portion of an orebody. We know that assaying and sample handling techniques are precise if there is little difference between the assay values for replicate measurements. Such differences are due to the variability, that is a lack of homogeneity in the material taken from the orebody. There is a very real chance that scheelite grains will be more abundant in one half of drill core, or in a certain size fraction of crushed material.

Thus the total variability of assay results for an orebody is the sum of the natural variability of grade, and the variability induced by sampling. There are statistical sample designs which will yield estimates of these components of variance, and to relate them to particular aspects of the sampling and assaying procedures.

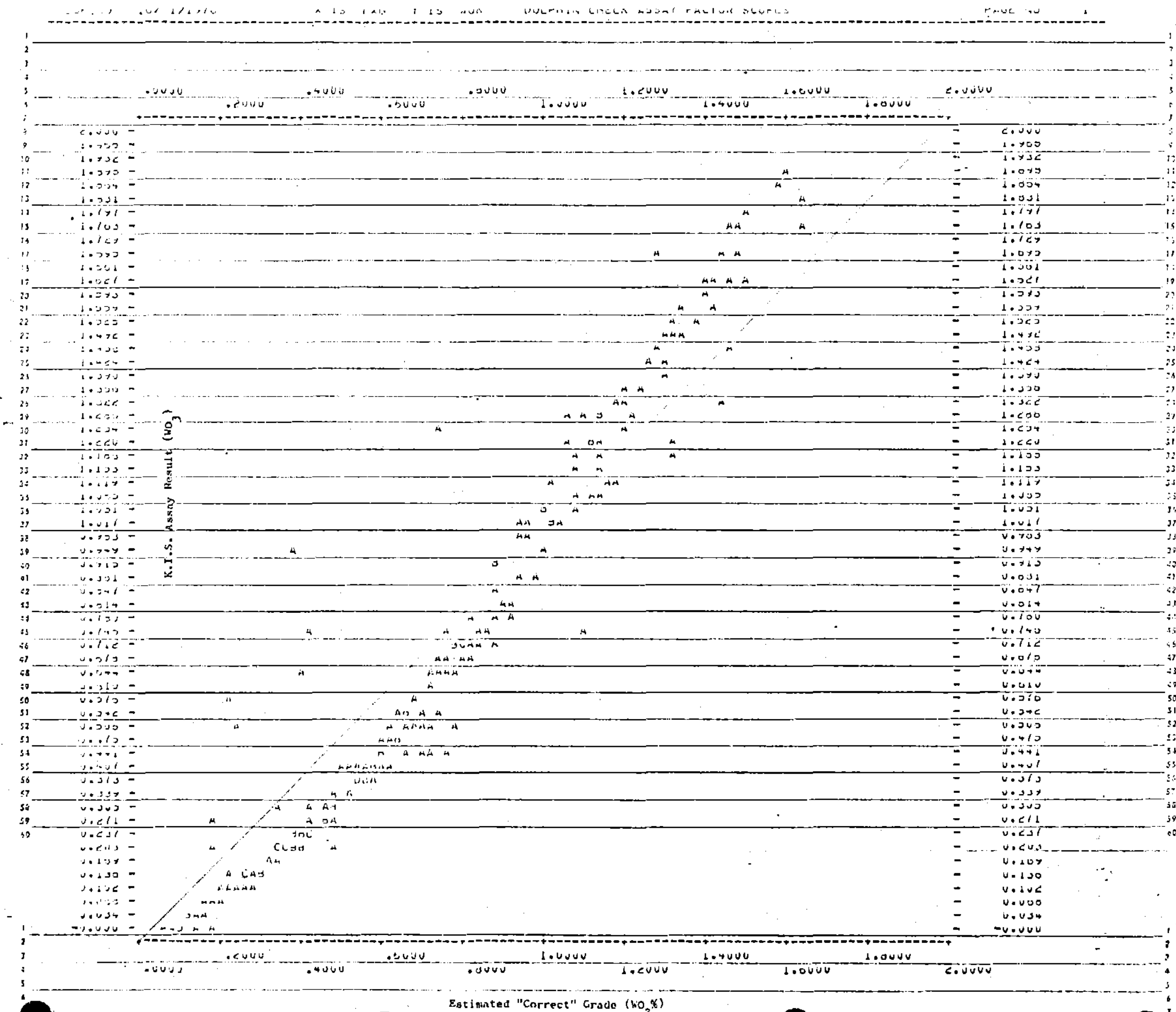


FIGURE 2. Bias Curve for K.I.S. Assaying.

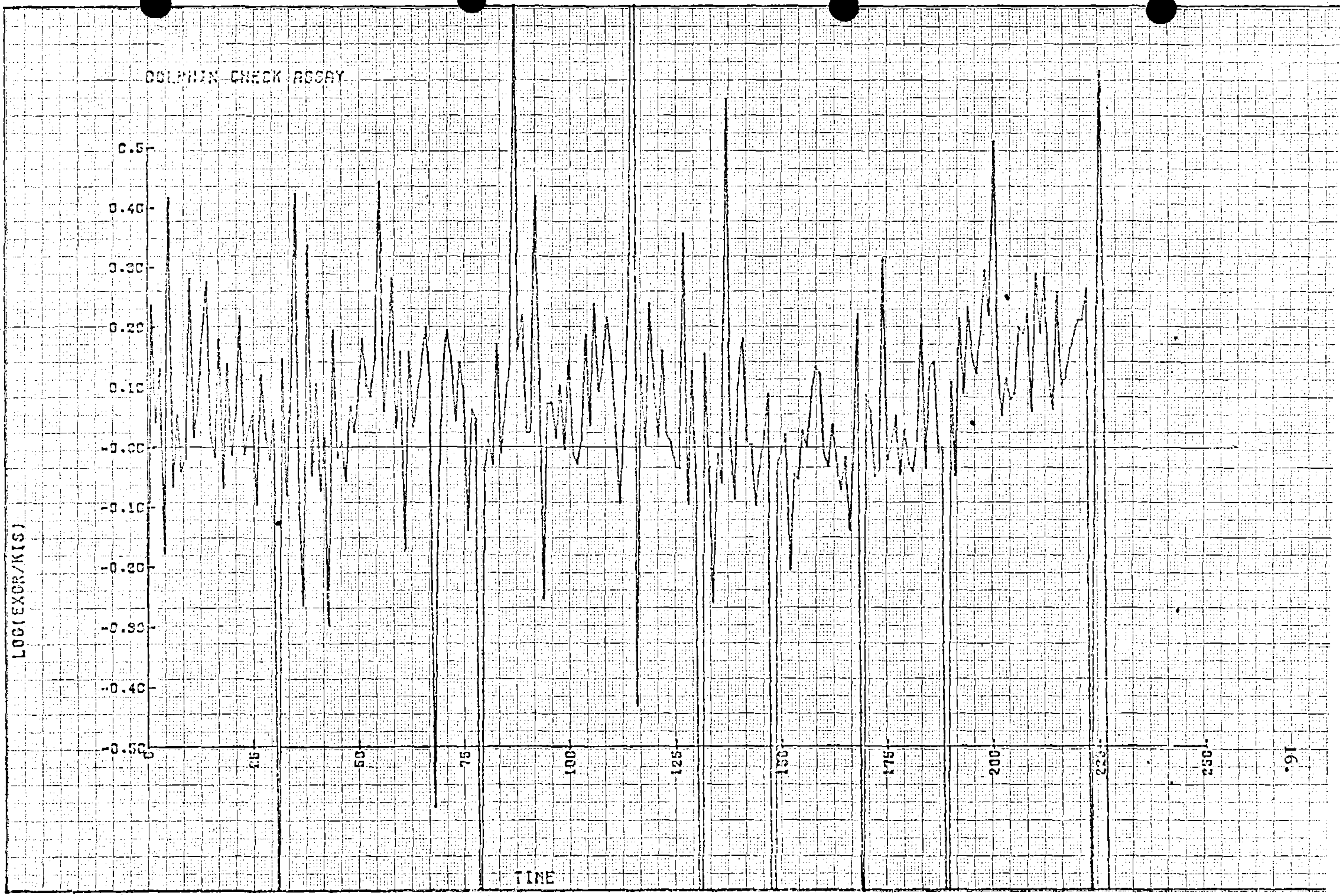
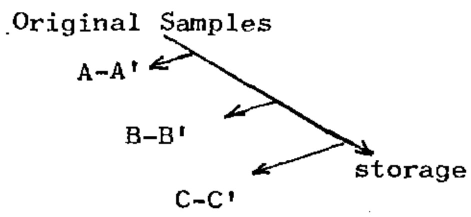


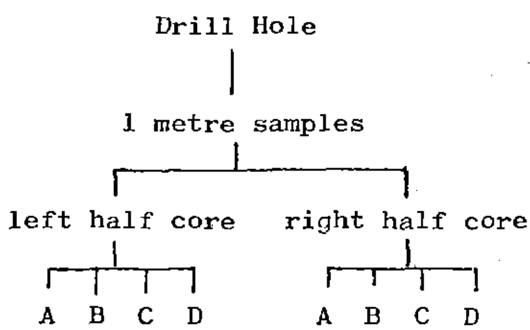
FIGURE 3. Ratios between the expected "correct" value and the K.I.S. assay for WO_3.

There were three main experiments which were designed to assess the variation, or error, associated with some of the stages of sample preparation and assaying. These are:

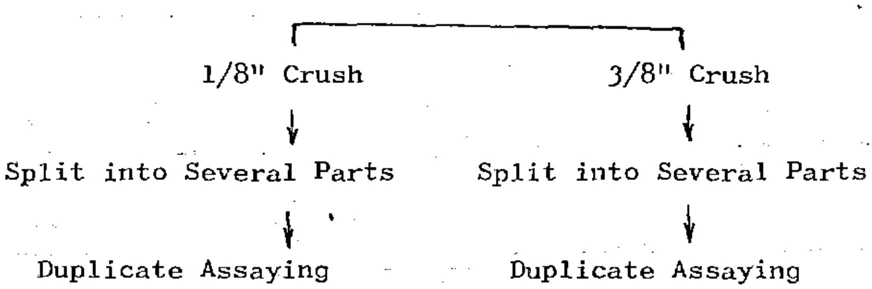
1. Repeated sampling and assaying - the results were from splitting selected samples from several drill holes and duplicate assaying of each split. The design for this experiment was:



2. Testing between left and right halves of core. In this case a single drill was divided into one metre samples and each sample was split into two (half cores) and each half core was split into four duplicates. Thus the design was:



3. Repeated assaying of a single pulverised sample. Two samples taken from two levels of crushing (i.e. 3/8", 1/8") were split into several portions and then each portion was reassayed several times:



	C-Lens Orebody	Half Core	Repeated Splitting and Sampling A-B-C	2nd Batch	New Batch	Van Gelder	Complete Experimental Design	Repeated Assays 1/8"	Repeated Assays 3/8"	Estimated Value	Frequency %	Cumulative %
Orebody	1.15						1.512			1.32	100	100.00
Drill Section												
Drill Hole												
Sample		0.8078	0.9020	1.05836	0.67352	0.77207	1.4381			.912	69	85.22
Half Core		0.759								.076	6	16.22
Crushing												
Splitting		0.0140	0.0050	0.0363			0.03611			.017	1	10.22
Pulverizing			0.0399	0.21831	0.16220	0.35376	0.03755			.113	9	9.22
Splitting								0.0005	0.0032	.002	0.2	0.22
Instrumental								0.0005	0.00002	.0003	0.02	0.02

TABLE 3. Estimates of Variance Components for the Assaying of Scheelite in the C-lens orebody within the Dolphin Mine.

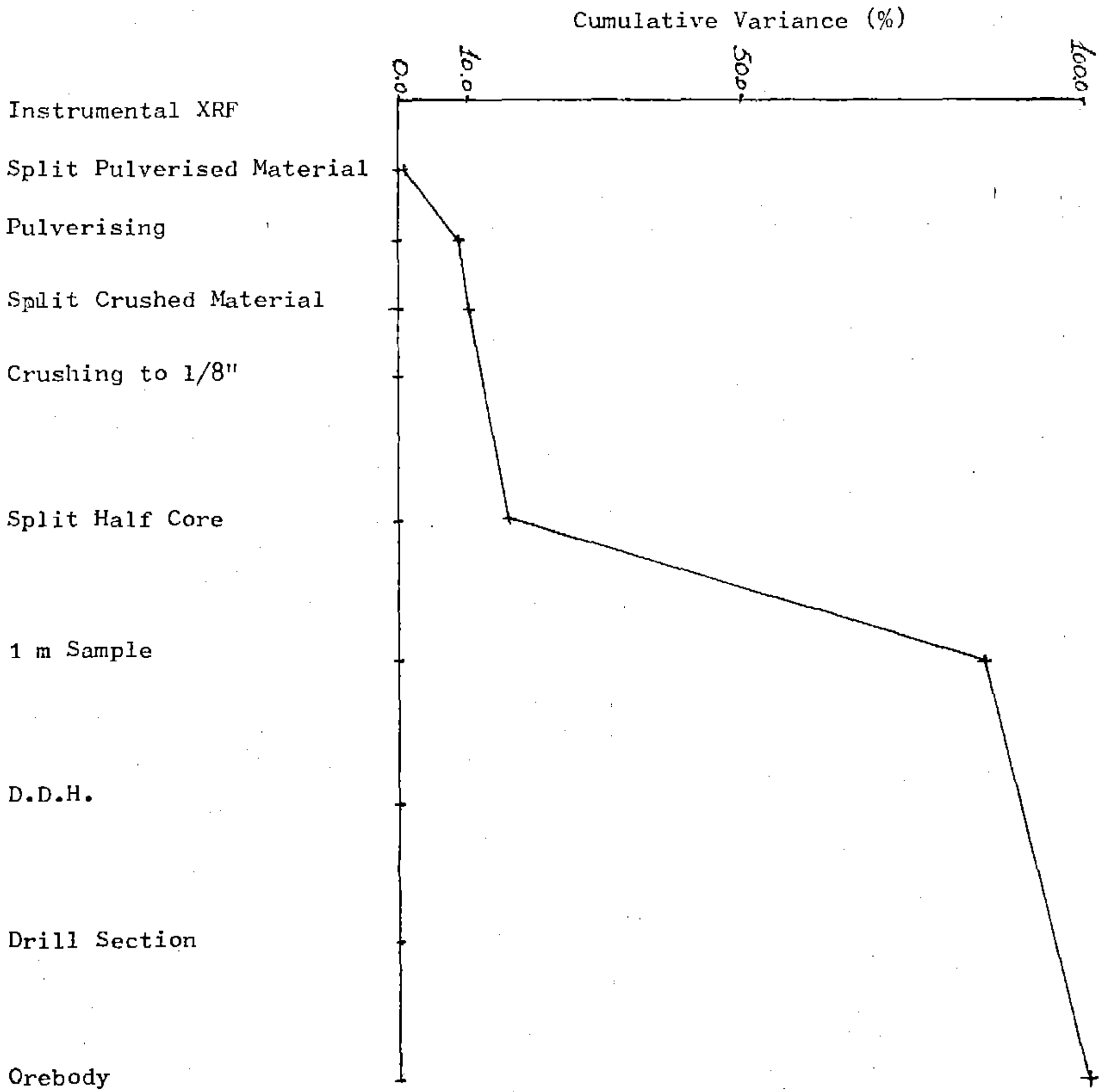


FIGURE 4. Cumulative Variance Components for Sampling and Assaying the Dolphin Orebody.

A conventional analysis of variance could not be used for this, because the results were provided by six different experiments. Thus the components of variance had to be determined separately for each level of assaying and sampling and then combined. The process by which this was done was not a rigorous one, and could be criticised on theoretical grounds. Nevertheless it is believed that the results are valid, and that they indicate the correct order of magnitude for the variability and the precision at each level for sampling and assaying the orebody.

The result is shown in the column "Estimated Value" of Table 3 and in the graph of Figure 4. A steep rise in the graph at the 1 metre sample level dramatically illustrates the variability in grade between adjacent lengths of core.

CONTROL OF THE QUALITY OF ASSAYING

Attention has been drawn to differences in the reported grades for the assaying of the same samples, and it has been shown that the magnitude of these differences is less than 10 percent of the total variability within the orebody.

Under normal operating conditions the variance of replicate assaying should be one twentieth that of the variance of the orebody. It should then have a value of 0.05.

H.S. Sichel has described (Coxon and Sichel 1959, Rowland and Sichel 1960) the methods used in the South African Gold Mines to control the quality of routine sampling and assaying in order that the results are acceptable within such a variance. The basis of the method is the continued scrutiny ratio between the logarithm of the original assay value and the logarithm of the replicate assay value.

Sichel established the form of the exact sampling distribution to establish warning and action limits to the ratios for unacceptable assay values. The mathematical form of the distribution is rather difficult to deal with, so these limits were set in this case by an heuristic method. A random number generator was used to generate 1000 assay values with a mean and variance equal to that of the whole orebody. For each of the 1000 values a replicate value was generated at random with a variance of 0.05, which is equal to the value expected from the splitting of 1/8 inch crusted material. A cumulative frequency distribution was constructed for the 1000 ratios of the logarithms for the two values, and the values for the ratios were determined for the chances of 1 in 1000 and 1 in 40 that a ratio is unacceptable.

If the ratio of the logarithms for the two assay values is greater than 0.94 and is less than 1.14 then the precision assaying of these two samples is within the normal limits. A value of the ratios between 0.84 and 0.94 or between 1.14 and 1.22 indicates that the precision of assaying is less than to be expected under normal conditions, this should occur for fewer than 1 in 40 assays. There is normally a chance of 1 in 1000 that the value of the ratio would be less than 0.84 or greater than 1.22.

GRADE CONTROL SAMPLING

King Island Scheelite have made available the results of assaying the samples which have been taken for grade control purposes. Between two and twelve samples have been taken from ore broken in most of the firings or rounds. Contingency tables and histograms (Figures 5, 6) were constructed for samples taken in the Wedge Block Undercut. The asymmetric shape of the histograms for individual samples and for the arithmetic mean for each round (Figure 5) indicates that the arithmetic mean is a poor estimate of the grade. Similar histograms which were constructed for the logarithms of the values are reasonably symmetric. Accordingly the geometric mean, for the values obtained from each firing, has been used for the computer work performed by Geopeko.

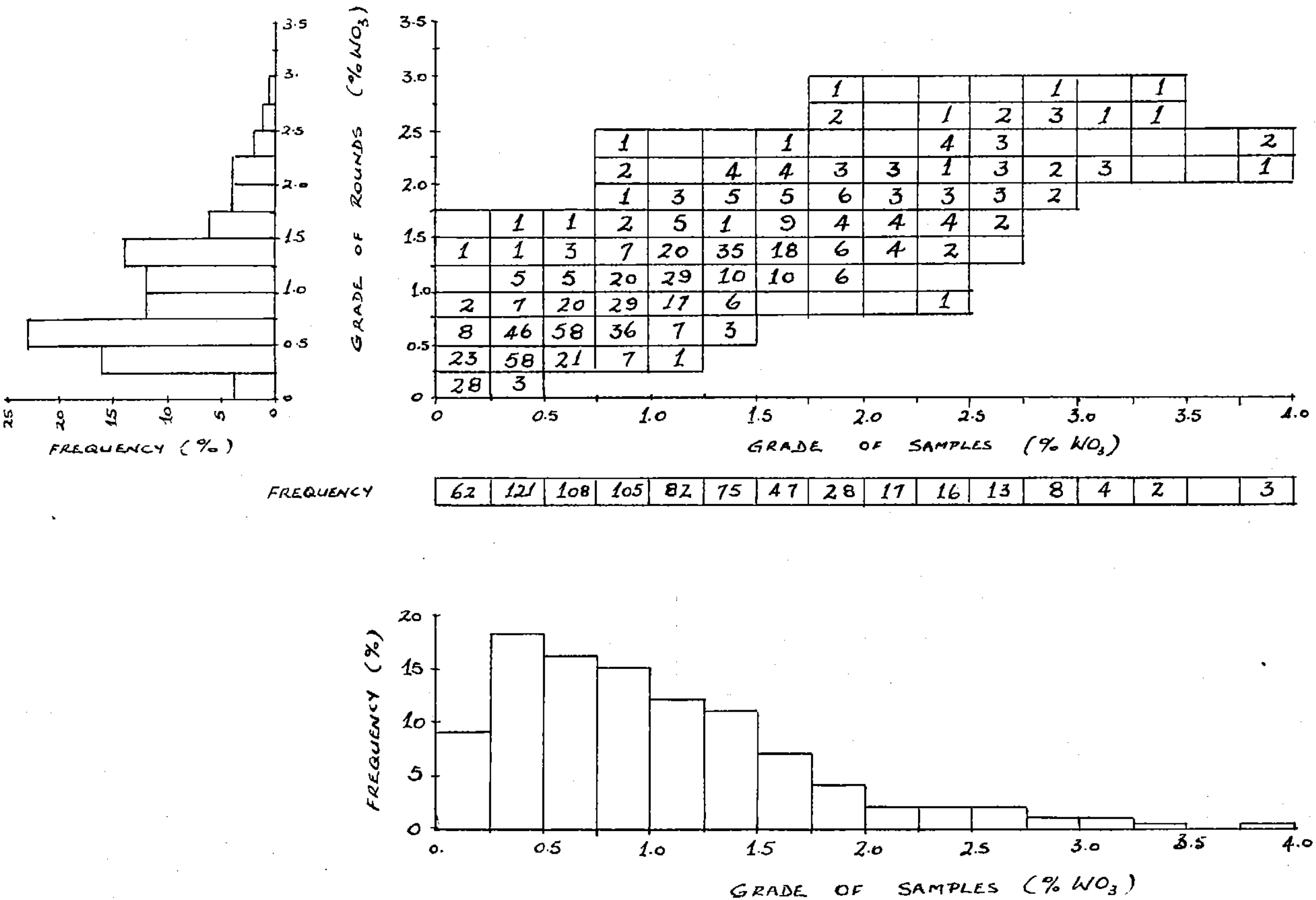


FIGURE 5. Arithmetic Frequency Distributions for Grade Control Samples, RL-125 to RL-130, Wedge Block, Dolphin Mine.

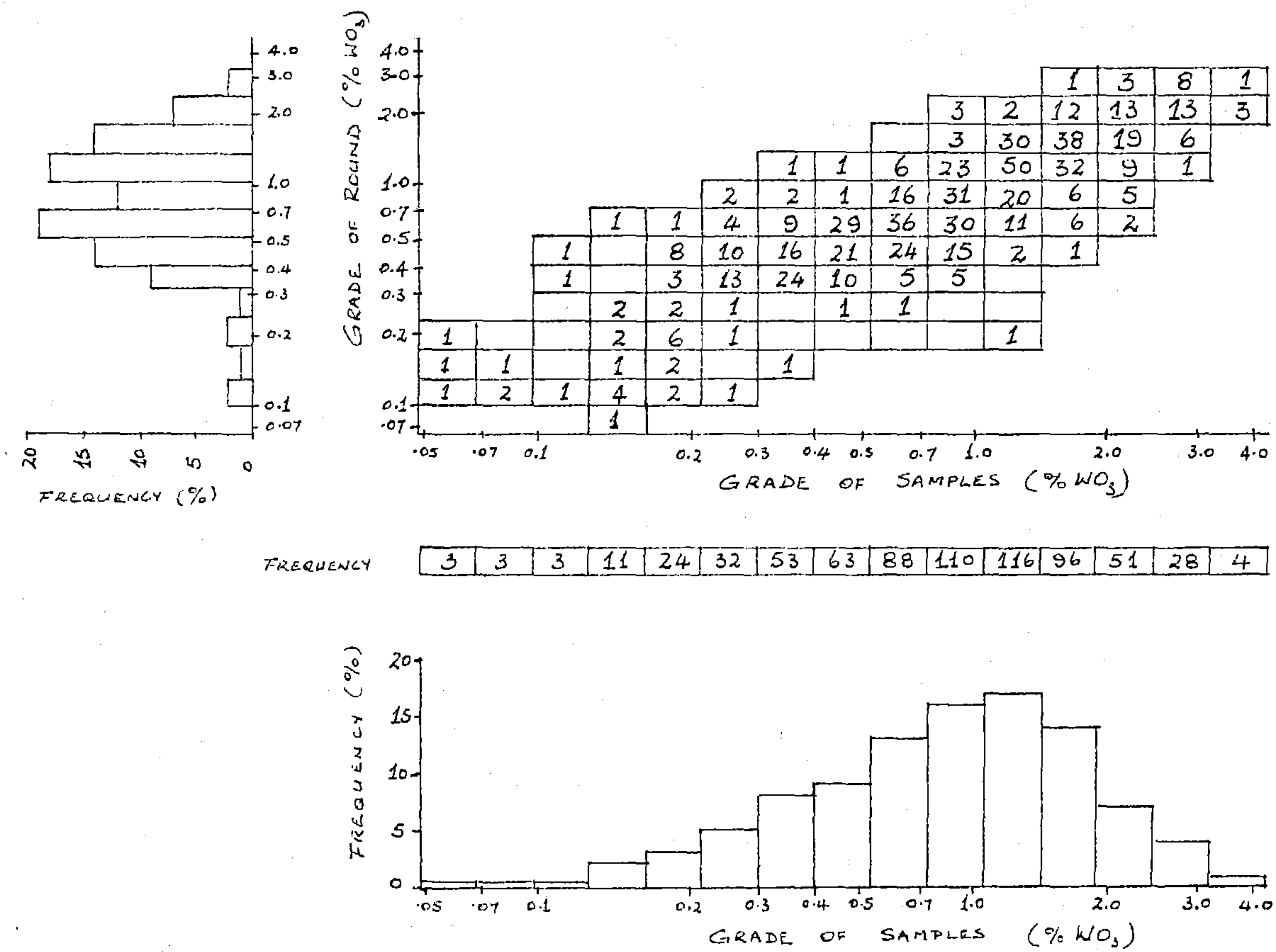


FIGURE 6. Logarithmetic Frequency Distributions for Grade Control Samples, RL-125 to RL-130, Wedge Block, Dolphin Mine.

SAMPLE SIZE

P. Gy has described the relationship which exists between the standard deviation of sampling errors, the characteristics of the ore samples, and the weight of the sample. The computation of the minimum weight of samples which should be used, in order to achieve a desired standard deviation for the sampling errors, requires a knowledge of the grade for various size fractions. This information is not available for Dolphin ore.

Another method was proposed by Ingamells (1974), in which a number of sub-samples, each of a constant weight, are taken from broken ore and assayed. This exercise would be repeated for other sets of sub-samples of different weights. The results are plotted on a diagram to illustrate the decrease of sampling errors with increasing sample size.

The results of two sets of experiments, which were undertaken by the K.I.S. assay laboratory have been combined to produce a similar diagram. The first set of experiments were performed in 1971, and the exact conditions are not known. They were combined with the 1976 experiment in order to produce a meaningful result, but it is one which cannot be considered to be definitive.

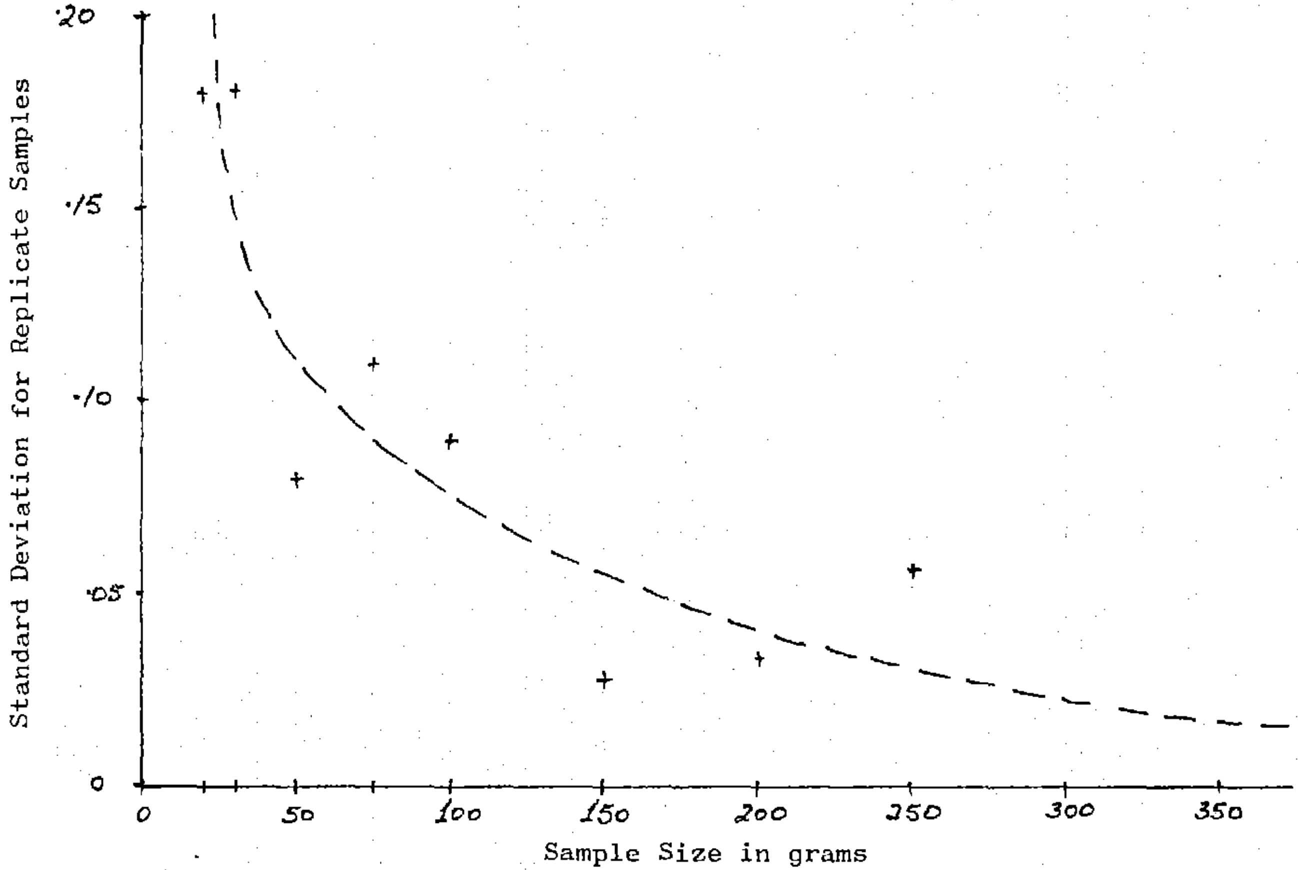


FIGURE 7. Relation between Standard Deviation of Sampling Errors and Sample Weight.

VARIOGRAMS

Two methods were used to compute variograms for the orebody. For the first method the orebody was divided into a number of grid cells, with dimensions of 4 metres by 4 metres by 1 metre. The variogram was then computed by summing the square of the differences between grades in two grid cells. The second method is based on a suggestion which was made by the "Matheron Group". Variograms were computed for each drill hole, and these were then grouped and averaged, according to the direction in which the hole was drilled, to form composite variograms for the various directions.

The variogram function was computed by the first method for each of the three main directions - east, north and the vertical. That for the northerly and vertical directions were reasonably satisfactory (Figure 8) while that for the easterly direction was discarded because assay values are clustered about drill sections which are 40 metres apart.

The interpretation of these variogram curves should be approached with caution, because we are not able as yet to correct for any regional trends in the grade of the orebody, or for its geological structure. The mine coordinates were used to compare the differences between pairs of assays, and as the geological boundaries transgress the mine grid such comparisons will be frequently between assays in different rock units. A variogram taken in a horizontal direction will then contain a significant component of the variance between grades in adjacent rock units.

The basic assumptions used for this interpretation were that:

a) it will apply to the "C-lens" orebody;
b) there is a hole effect in the vertical direction, and that this has a range between 3 and 5 metres;
c) the curves of figure 2 are compound ones, in which the "steps" represent natural periodicities of grade in the orebody;
d) the vertical thickness of the "C-lens" is approximately 40 metres;
e) the three dimensional variogram function is anisotropic, in that within the interval 15 to 40 metres the height of the steps in the two directions are different.

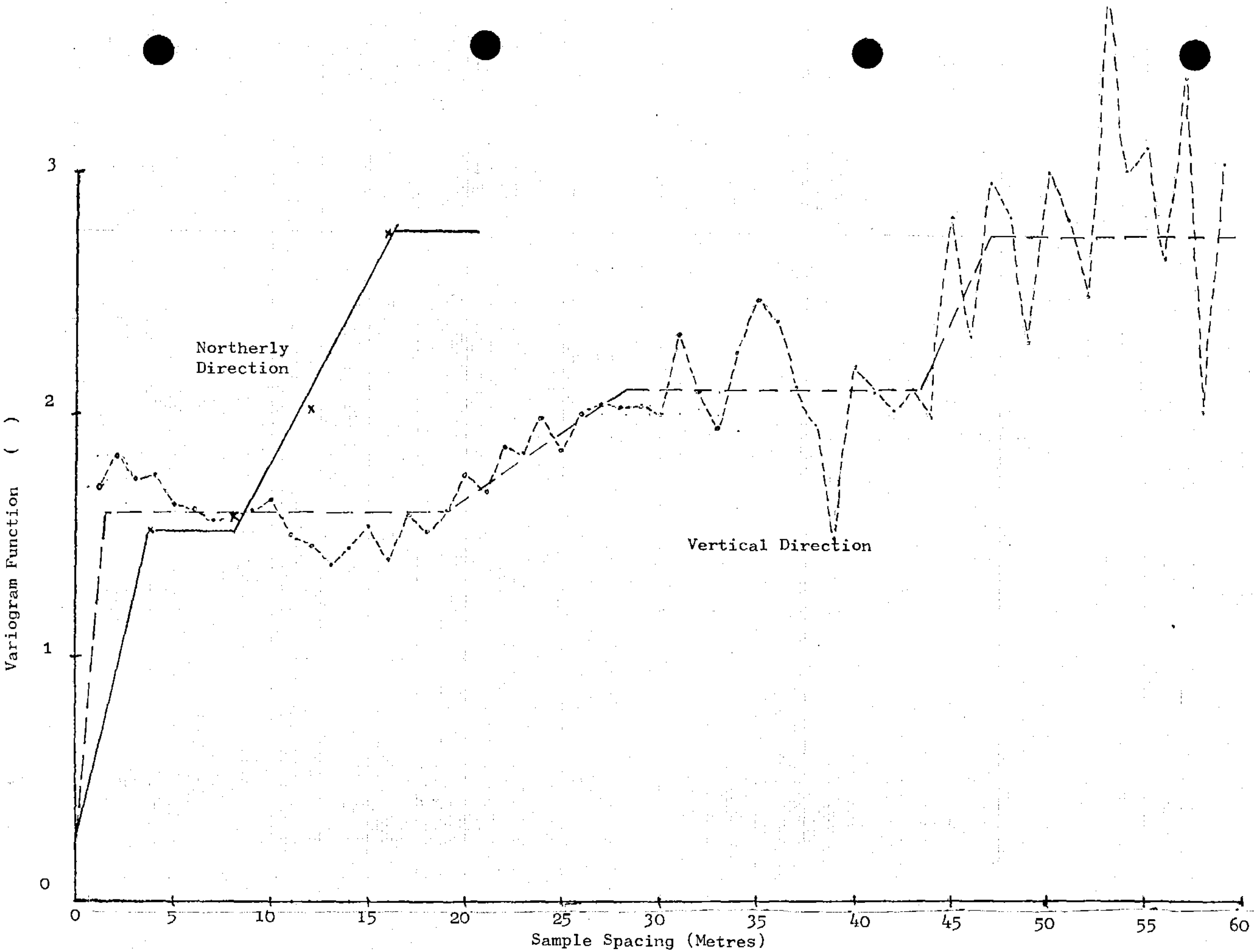


FIGURE 8. Variograms for the Dolphin Orebody

160032

FIGURE 9. Variograms for a Spherical Model of the Dolphin Orebody.

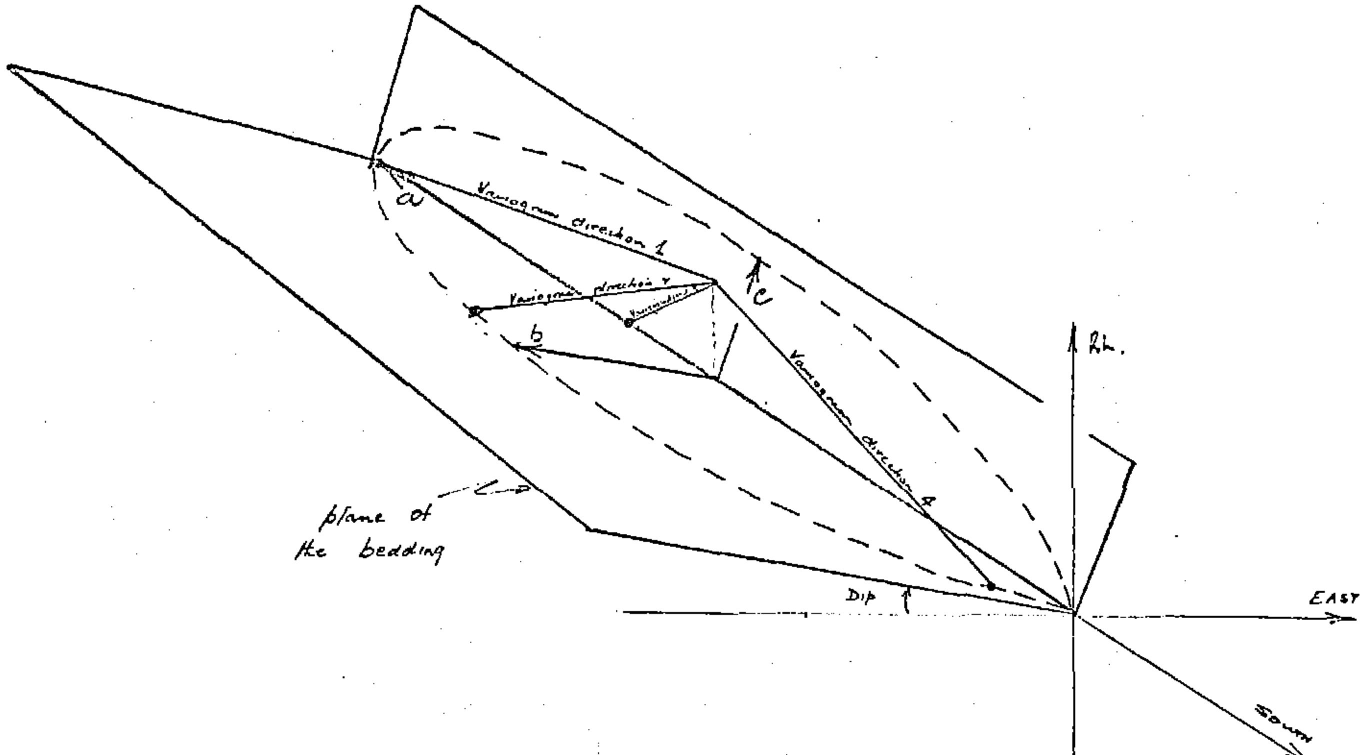
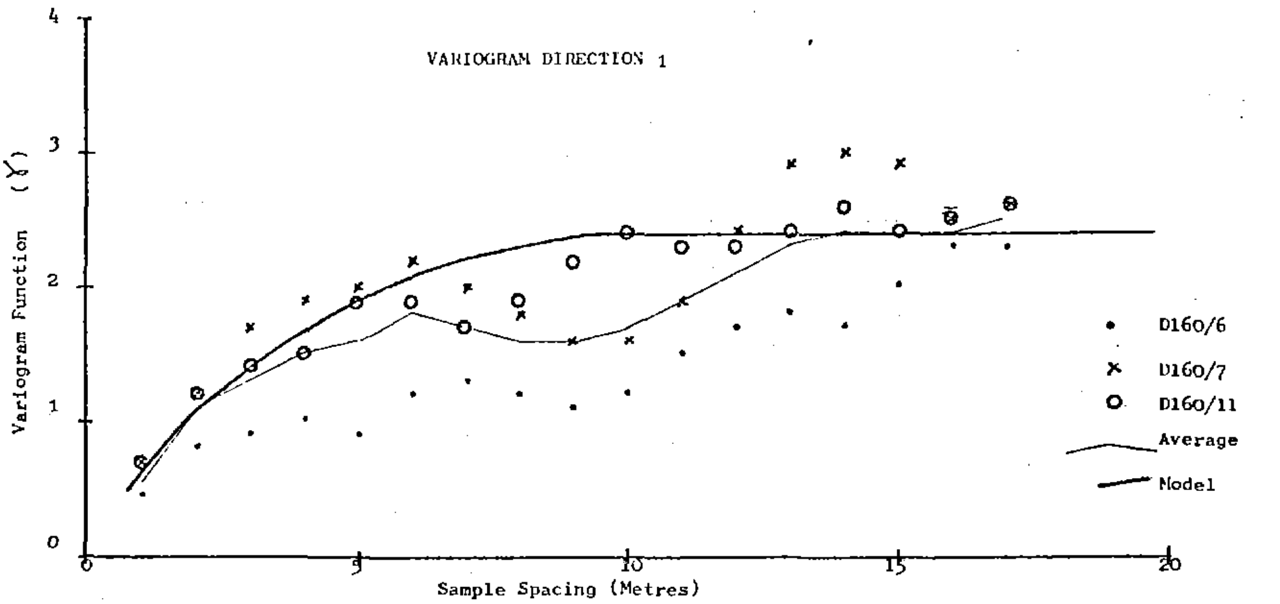


Diagram illustrating the three-dimensional Variogram Model



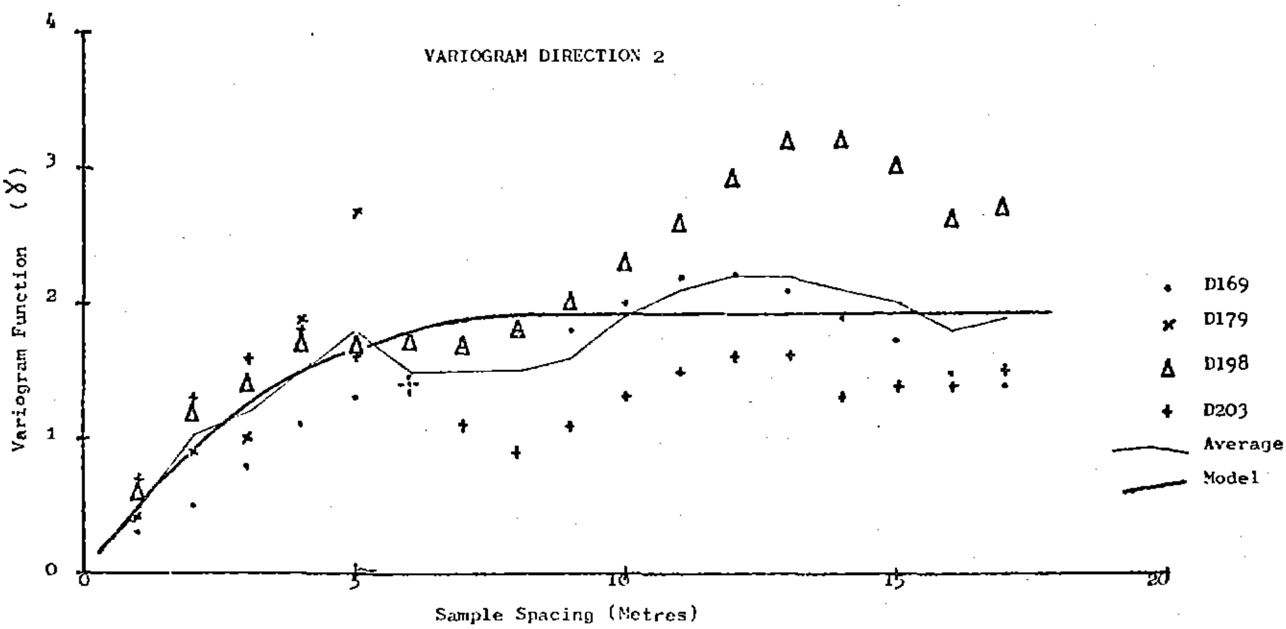
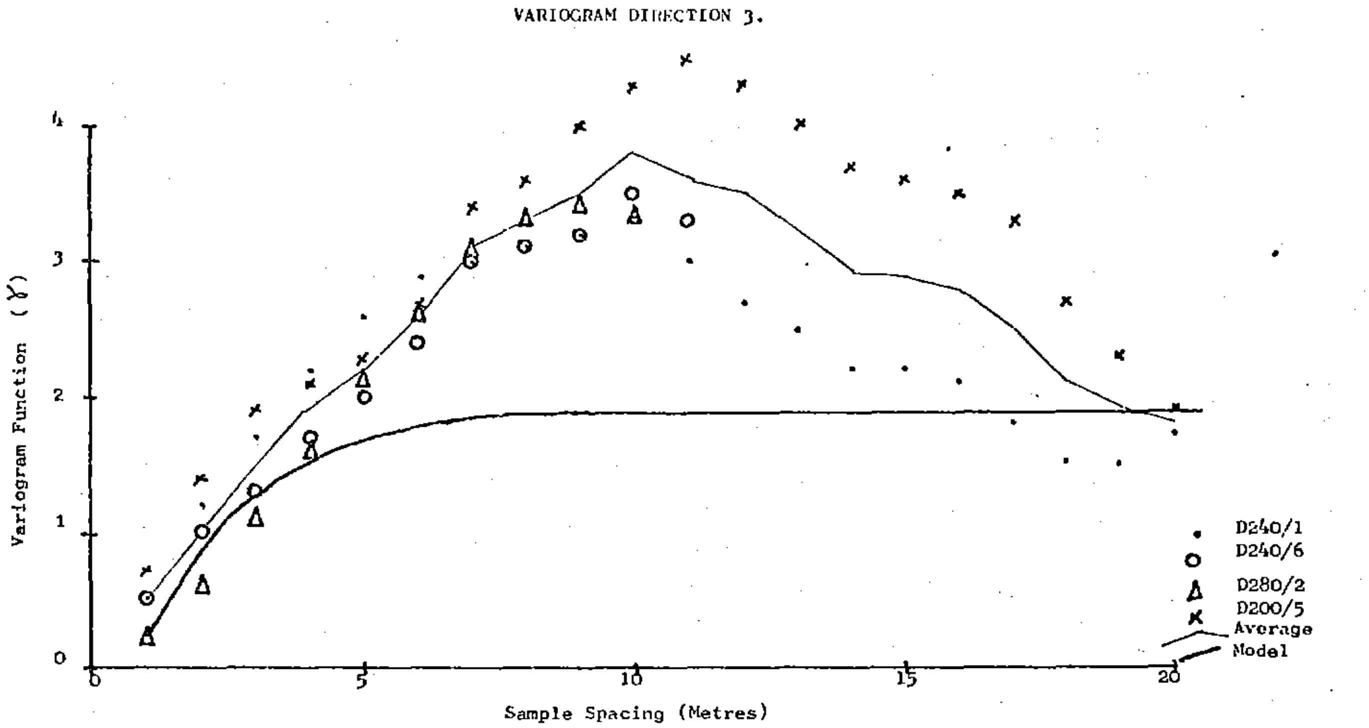


FIGURE 9. Variograms for a Spherical Model of the Dolphin Orebody.



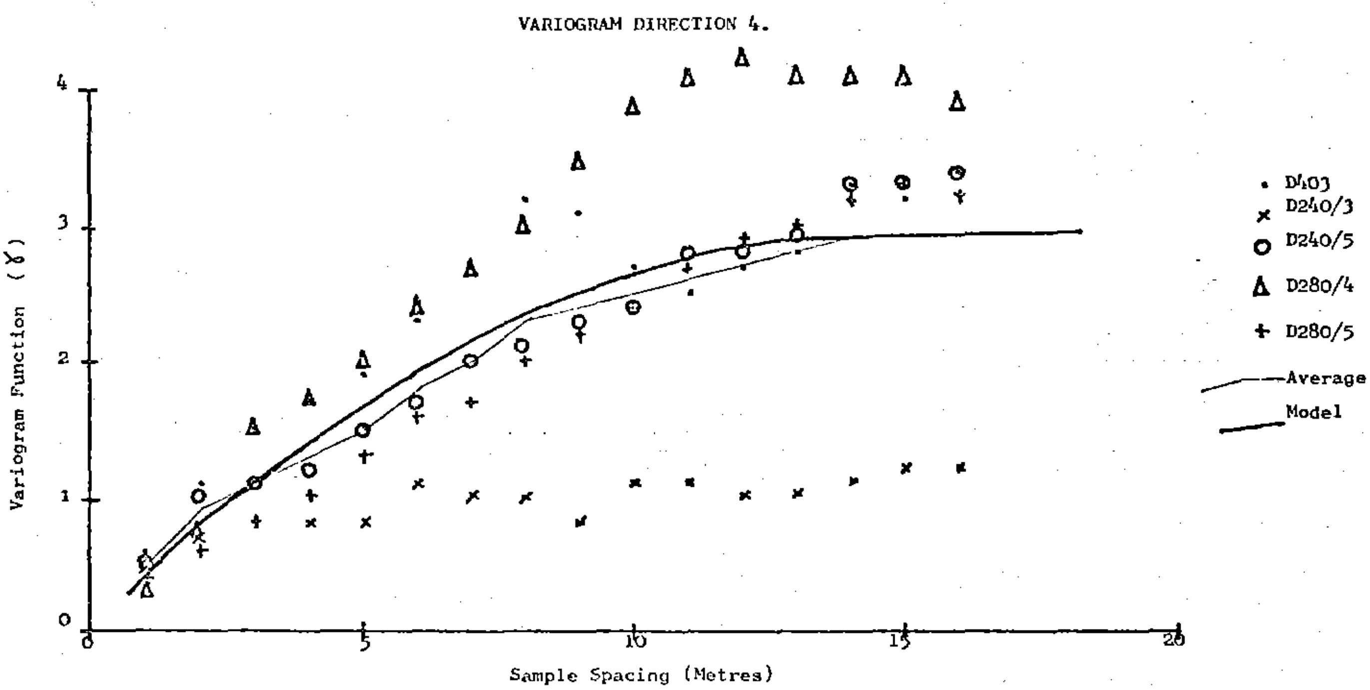


FIGURE 9. Variograms for a Spherical Model of the Dolphin Orebody

5 cm

The results of this method of computation were useful but they were not entirely satisfactory.

An attempt was made to try and resolve this problem by only calculating variograms down drill holes and then to relate these variograms to various components (i.e. to relate them either to mine grid or to bedding). To try and reduce the random noise component it was decided to consider only a small part of the orebody, from the top of the Pgh to the base of C-lens within the Wedge Block area.

The curves for fifteen drill holes are shown in Figure 9 together with the average value for each of four directions. Hole effects are apparent in most of the variogram curves, and this is considered to be a reflection of a banding of the mineralization.

ORE RESOURCE AND RESERVE ESTIMATION - METHODS

The purpose of sampling and assaying portions of an orebody is to estimate the quantities of metal which exist within specified volumes. The distinction between resource and reserve estimation is merely the basis for the specification of the volume. Resource estimates are concerned with geological boundaries to the volumes, and reserves are estimated for volumes which are to be removed by mining. There should then be no difference in the computational methods which are used to derive the estimates. A number of methods are available (Knudsenetal, 1975). These include:

The Polygonal Method

This method has been most widely used in the past for porphyry copper deposits in the U.S. The method has been used for total ore reserve estimation (i.e., total tonnages and the average grade) and for individual block grade estimation.

In this method, the assay grade of a drill hole is extended halfway to any adjacent hole, thereby defining an extent of the influence this hole exerts. This procedure results in each block receiving the grade of the nearest hole. The method has a certain practical appeal in that: (1) it is simple because no weighting of different holes are required, (2) suitable for updating of ore reserves when additional drill holes are drilled.

However, some difficulty arises in the use of this method when there are insufficient number of drill holes in the area, thus resulting in a rather extensive "area of influence" of a drill hole. The common practice has been to choose an arbitrary distance called "the radius of influence" and to limit the grade assignment of individual blocks to only those that lie within this radius of influence. This difficulty is avoided in other conventional ore reserve estimation methods as well as in the geostatistical methods.

<u>Inverse of the Distance Squared</u>

With the introduction of a computer in ore reserve computation, the use of many variations of distance weighting schemes as well as the geostatistical methods became practical. The use of many variations of distance weighting schemes is to recognize that there are certain spatial relationships between the grade of a hole and the grade of adjoining blocks and that these relationships are some function of the distance between the two. In contrast to the polygon method, these methods utilise the grades from all or nearly all surrounding holes by giving certain weights that are determined by the distance between the drill hole and the block under consideration. Obviously, the sum of weights must add up to one, thereby forming the linear (convex) combination of the grades of surrounding samples. The particular weights to form the linear combination are inversely proportional to the distance squared, which are calculated from the central point of the block under consideration. The decision as to whether or not the distance should be squared, cubed, or even raised to 2/3 power is either based on past experience or an arbitrary one.

Just as in the polygon method, there are some practical difficulties as well as shortcomings of this method. One difficulty is in deciding (or defining) the surrounding holes. Similarly, one shortcoming is that the directional effect (i.e., geologic anistropy) of surrounding drill holes are ignored in the weighting scheme.

Elliptical Weighting

This is an improvement on the previous method, in that the weighting scheme is based on the square of an elliptical distance from a central point. The geological anistropy of the orebody is used to define the parameters of the ellipse which is used.

Kriging

Kriging is the geostatistical technique to find an estimate of the true grade of a block as a linear combination of all the available samples, such that the estimate is unbiased and has minimum variance. Simply stated, kriging is a technique to find the set of weights that minimises the extension variance, according to the geometrical characteristics of the problem. Kriging assigns low weights to distant samples and vice versa, but also takes into account the relative positions of the samples with respect to the block and each other. Thus samples close to the block may form a screen that lessens the influence of samples farther away. In spite of the unbiased condition, assumed in the derivation of the weights there appears to be a bias in the results, in that the metal content of lower grade blocks is over-estimated and that for the higher grade blocks is under-estimated. There are normally more blocks with low grades than there are with high areas, so the result is an over-estimate of the total contents of metal in the orebody. Moreover the method ensures that the estimated grades will not rise above the largest observed value, nor will they pass below the lowest. There is then a weakness in the method, as there is no certainty that drill holes will intersect the highest and the lowest grades throughout the orebody.

COMPARISON BETWEEN METHODS

A comparative test of the four methods was described by Knudsen et. al. (1975). In their view the total metal content is over-estimated by all four methods by the following amounts:

Method	Percentage Over-Estimation
Elliptical Weighting	10%
Kriging	13%
Inverse Square of the Distance	17%
Polygonal Method	20%

The four methods derive estimates for the metal content of volumes directly from the original assay values. It is proposed that this is the basis for the bias, and that it would be preferable to estimate the metal content by numerical integration of grades at points throughout the orebody. For this it is necessary to know or to estimate the grades at regular intervals, which may be done by interpolation. This method was used to estimate the reserves in the Warrego Gold Pod, and an example from the results is shown in Figure 10.

Grades at twelve points bounding a block are shown, as are estimates for its metal content. It is a considered opinion that the estimate by numerical integration is closest to the grade which was recovered in the mill.

INTERPRETATION OF ASSAY RESULTS

The differences in the reported values (Appendix A) for the WO_3 content of half core prompted the institution of routine procedures for check and umpire assaying of one in ten samples. The laboratories used for the umpire assaying are AMDEL and ACSL in Adelaide.

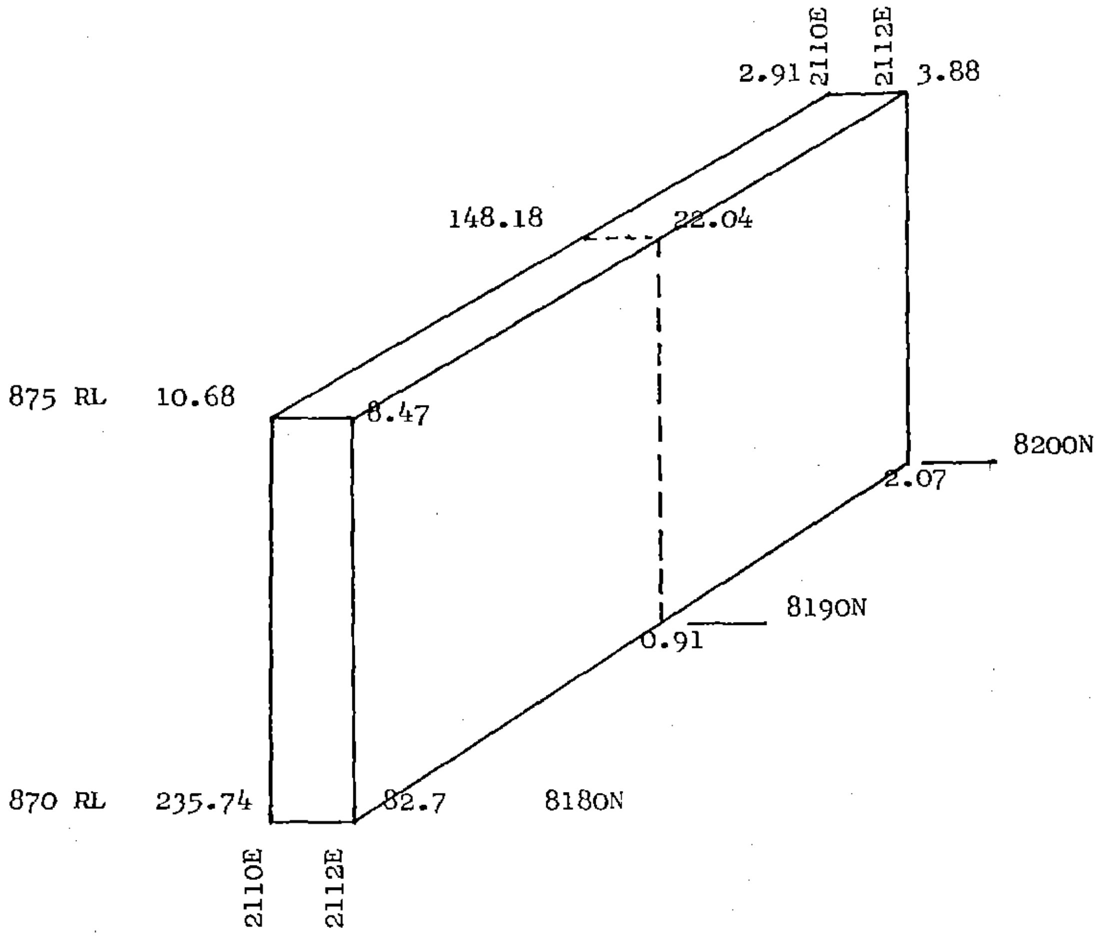
These differences raise the two questions -

(a) what is their cause?

and (b) are they significant, and materially affect an ore resource computation?

It is proposed that these differences are introduced by -

(i) the taking of 100 gram samples from some 5000 grams of crushed core, and is attributed to the heterogeneity of the material;



BLOCK GRADE

Arithmetic Mean	43 gm/tonne
Sichel Method	27 gm/tonne
Kriging	42 gm/tonne
Numerical Integration	21 gm/tonne

FIGURE 10. Computation of Block Grades - A comparison of methods.

(ii) the presence of a bias in the K.I.S. assaying technique (Figure 2).

(iii) periodic changes in this bias (Figure 4).

The cumulative value of the sample handling variance component (Table 3) is 0.1323, and this includes variability from these three sources. An important feature of this component is that it was estimated from determinations which were made over a long period of time, between the 30th September, 1974 and 24th July, 1975. Moreover the determination of the sample splitting and instrumental variances was done over a much shorter time interval, during which conditions external to these experiments should be constant.

Thus it can be argued that the difference between the two

Total sample handling variance		0.1323
Variance due to sample splitting (Fig.4)	0.0064	
Variance due to pellet preparation (Table 3)	0.0020	
Instrumental variance (Table 3)	0.0003	0.0087
Difference		0.1236

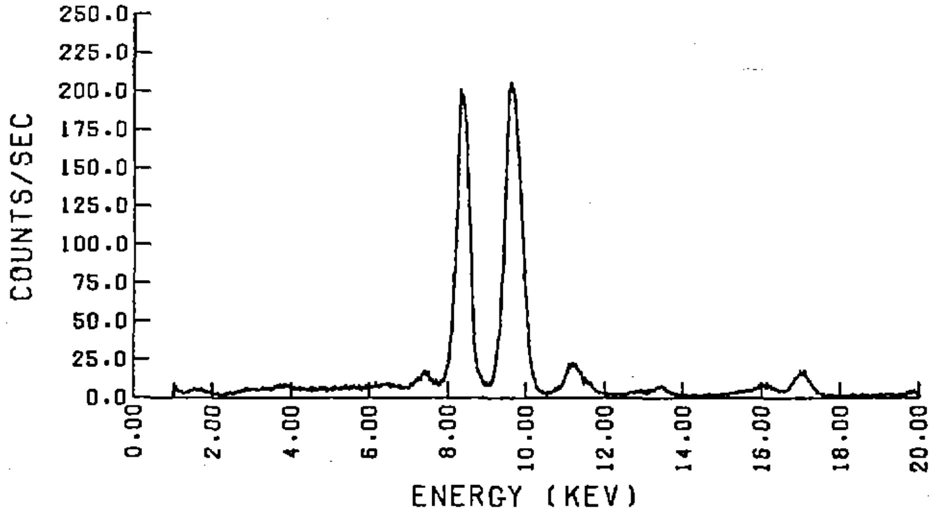
represents the variance component for periodic changes in the bias and precision of the laboratory procedures, it is approximately 9% of the total variance within the orebody. A reduction of this value to 5% is believed to be feasible.

If the sample handling techniques are precise, what then is the cause of these significant changes in precision: The Geopeko Ortec multichannel analyser was used to produce the three XRF Spectra in Figure 11. There are two main differences in the three:

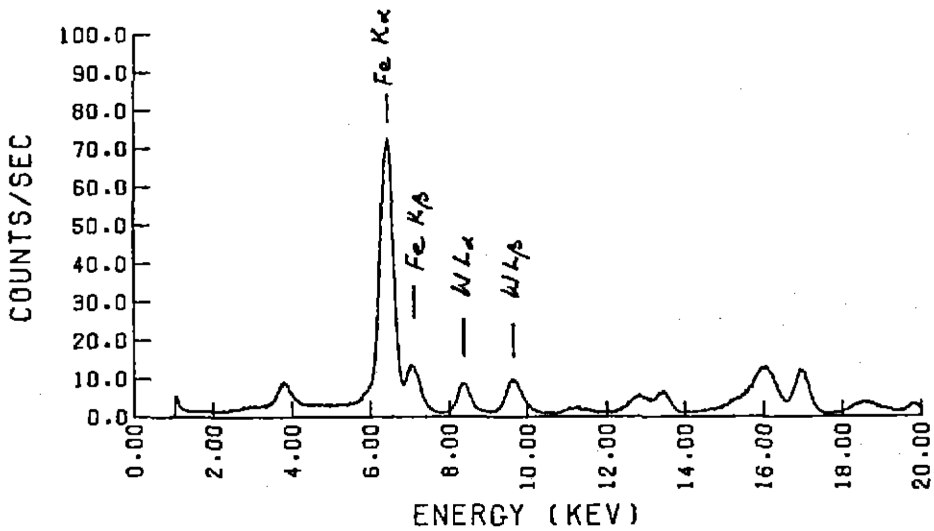
(a) the marked difference in the areas of the tungsten peaks for the samples;

(b) the peak areas of the matrix and compton affects are of an order of magnitude greater than that for tungsten.

XRF SPECTRA. SCHEELITE CONCENTRATE.



XRF SPECTRA. C LENS SKARN - 1.30% WO_3



XRF SPECTRA. C LENS SKARN - 0.18% WO_3

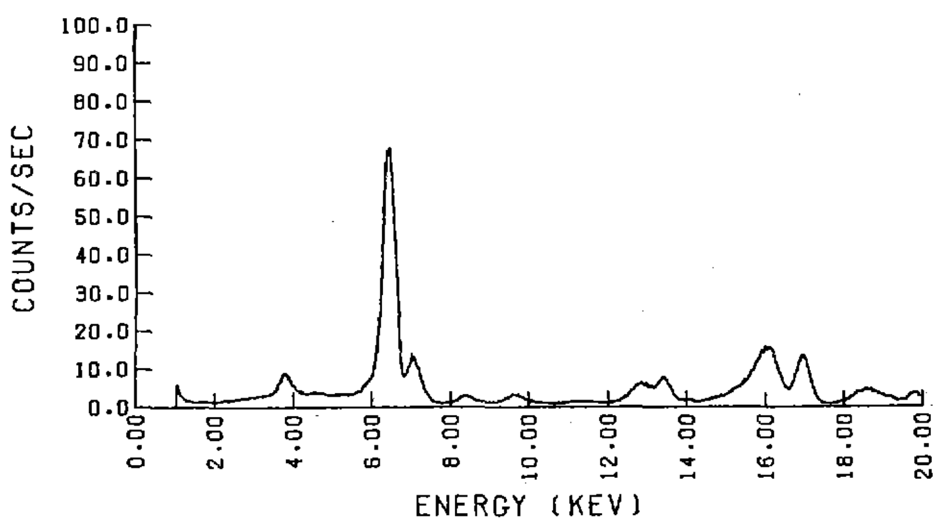


FIGURE 11. *XRF Spectra for Samples from Dolphin.*

It could be concluded that such fluctuations in precision are due to difficulties in positioning the three detectors in the K.I.S. Philips Machine, in order to correct for matrix affects, and to maintain its calibration.

The largest source of variation is between adjacent metre lengths of drill core, which is 69% of the total, and it is felt that this is too high to permit these assays to be used, with numerical interpolation procedures, to define grade variations within the orebody. Methods by which this component might be reduced were investigated, that which was adopted is now described.

The shape of the variograms in Figure 8 indicates a strong nugget phenomenon in the distribution of mineralization, within the range 2 to 4 metres. The magnitude of this nugget effect is approximately 58% of the sill height, which is comparable with the cumulative variance compound at the level of metre lengths of core.

Consequently it was decided that interpolation should be based on recommended grades for lengths of core between 2 and 4 metres. The average length for a drill hole intersection in rectangular volumes of 6 x 8 x 3 metres and 8 x 8 x 3 metres is 3.65 metres. Moreover it would be desirable that any interval used for a recommended grade should be wholly within one stratigraphic unit. Accordingly grades were computed within each stratigraphic unit for a number of intervals, of equal lengths which are closest to 3.65 metres.

Accordingly the drill hole intersections for each of the stratigraphic units were divided into a number of intervals and recommended grades were computed for each. Within a unit the intervals are of lengths closest to 3.65 metres which will yield an integer number of intervals. The variance components associated with these recommended grades have yet to be determined. Similar procedures such as the traditional compositing of samples, or the geostatistical technique of regularization of grades, are frequently used within the industry.

NUMERICAL INTERPOLATION

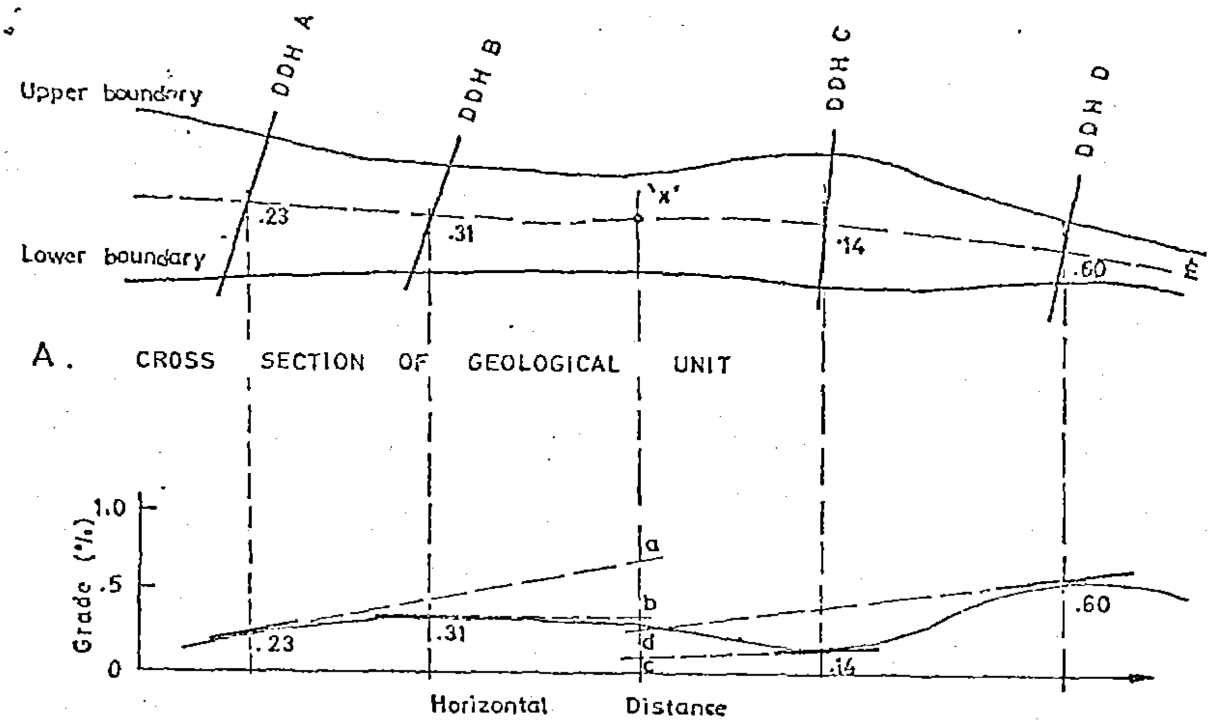
Interpolation is the insertion of intermediate terms in a mathematical series, which in a geological context means the computation of values at intermediate points between sampling localities, using the sampled values. We know that the distribution of mineralisation varies in the three dimensions of the body, and that this variation is complex. It follows that the calculations required are also complex, and cannot be done without computing machines.

A computer is a complex calculating machine, which should be capable of interpolating ore reserves in three dimensions. However, it is not yet possible to completely define in detail and in terms of a machine language, all of the logic and procedures to be used. It has been found that the logic which produced adequate results for the massive sulphide bodies at Tennant Creek was totally inadequate for the Dolphin orebody. Consequently, we developed the logic to perform machine interpolation in three dimensions, and the results which are now presented are based on this method. However it is believed that, while these results are reasonable, it will be necessary to further develop the method so that the orebody is "unfolded" and so that grades are not extrapolated into barren portions of the marble marker.

The basic assumptions which have been used for the machine interpolation are that grade varies systematically in all directions, along the strike, down dip, and stratigraphically across the orebody.

The rate of changes in grade can be defined by the slope of a simple plane of best fit to the concentrations in neighbouring intervals. The basis for the selection of the points to be used is that they be evenly distributed, about the point for which a value is required. This is achieved by a search technique which selects the two closest within each of 48 sections of an ellipsoid. The estimated value for grade at a point is a weighted average of the grades for neighbouring intervals, projected to the point along each of the planes of best fit. An example of the computation in two dimensions is shown in Figure 12.

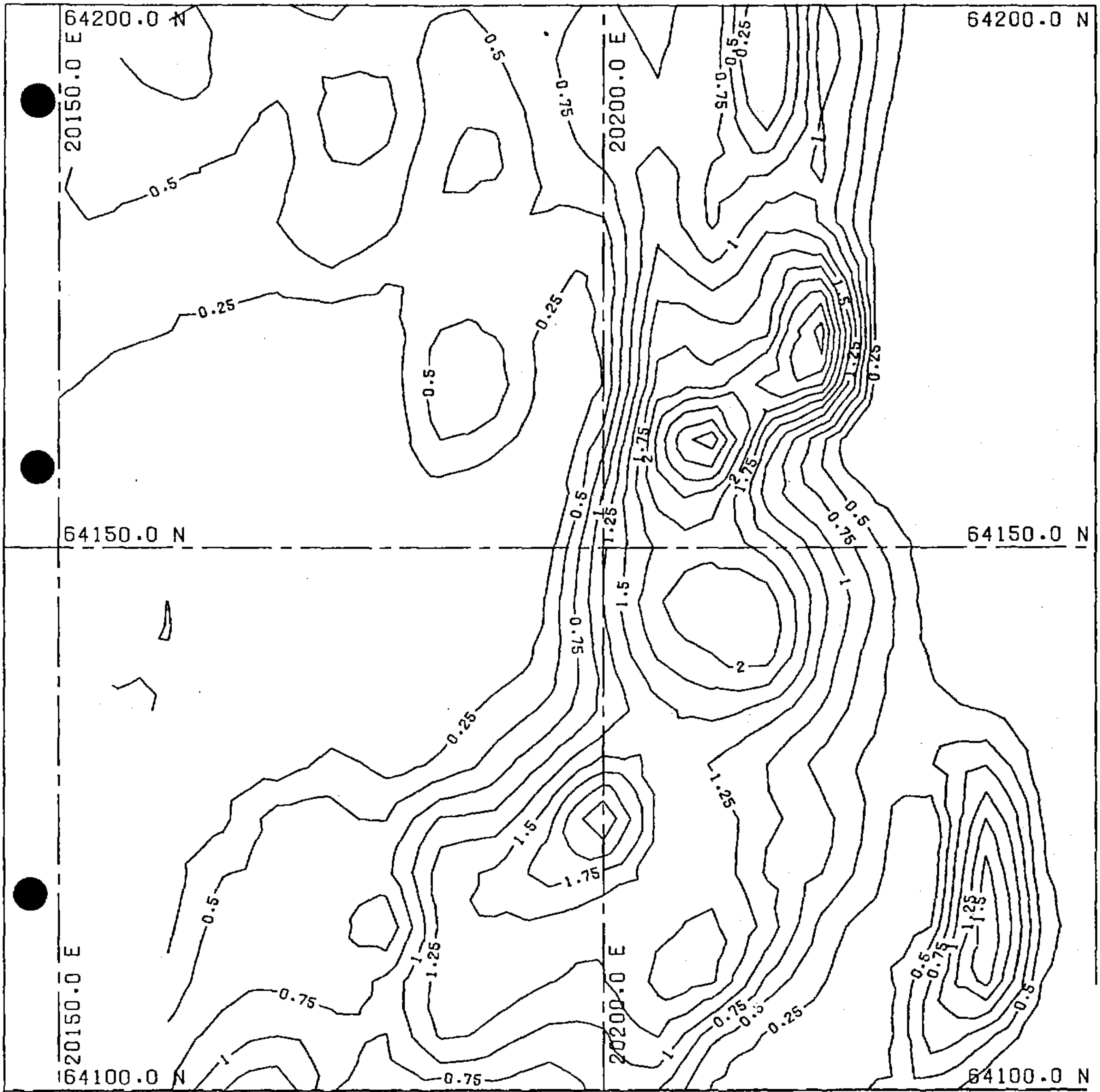
An appropriate method of weighting is by the inverse square of the elliptical distance between two points.



A. CROSS SECTION OF GEOLOGICAL UNIT

B. CROSS SECTION OF THE HYPOTHETICAL GRADE SURFACE

FIGURE 12. DIAGRAM TO ILLUSTRATE THE METHOD USED TO INTERPOLATE A VALUE FOR THE GRADE AT A POINT X, FROM THE MEAN VALUES FOR INTERSECTIONS IN THE SURROUNDING DRILL HOLES. THIS IS THE WEIGHTED AVERAGE OF THE VALUES a, b, c, d WHICH WERE COMPUTED AS THE PROJECTIONS OF THE TANGENT PLANES TO THE HYPOTHETICAL GRADE SURFACE OF BEST FIT TO THE MEAN GRADES IN DRILL HOLES A, B, C, D.



GEOPEKO LIMITED

SCALE 1: 500

DATE: 28/10/76

GEOLOGIST:

DRAWN:

CHECKED:

DOLPHIN WEDGE BLOCK -119RL GRADES

FIGURE 13.

GEOPEKO LIMITED

SCALE 1: 500

DATE: 28/10/75

GEOLOGIST:

DRAWN:

CHECKED:

DOLPHIN BLOCK GRADES

WEDGE BLOCK -116 TO -119

FIGURE 14.

ORE RESERVE ESTIMATES

Files are maintained on the discs of the computer. These are for estimates of grades at points on a three-dimensional array of points through the orebody. These can be displayed as grade contours in plan and section (Fig. 13), or as maps showing estimates of mining grades for rooms in one lift (Fig. 14).

A reconciliation of predicted and mining grades is necessary to assess the value of the techniques, however there is some difficulty in establishing a value for the mining grade of a room. There is some objection to the use of the grade control assays, on the basis of sampling theory and because every firing is not sampled, as an unbiassed and efficient estimator. It would appear that the best estimate would be the result of numerical interpolation and integration, using diamond drill hole assays together with grade control assays for the lifts below, within and above that being used for a reconciliation. This has been done for the Undercut and the 1st lift using those rooms which were mined as ore and for which all estimates are available.

For the Undercut (Table 4) two predictions for block grades are shown, one based on diamond drill hole assays and one using both diamond drill hole assays and the first batch of grade control sample assays. These may be compared with Sichel estimates for block grades using grade control samples, and the final results obtained by numerical interpolation using grade control assays between -130 and -119 RL together with those from diamond drilling. It is proposed that these latter be used as best estimates for the metal content.

The tabulated results may be interpreted as follows:-

(i) predictions based on diamond drill results alone yield an average grade for the lifts which is a reasonable estimate for the resource grade. However the high standard deviation for the differences between prediction and the final estimate indicates that they are inadequate as ore reserve estimates.

TABLE 4 160048

RECONCILIATION OF GRADE ESTIMATES FOR WEDGE BLOCK UNDERCUT,

-130 to -125 RL , DOLPHIN MINE

	GRADES (% WO_3) ESTIMATED FROM				DIFFERENCES		
ROOM NO.	1 DDH ASSAYS	2 DDH + GRADE CONTROL ASSAYS	3 GRADE CONTROL ASSAYS (Sichel Est)	4 FINAL INTERPOLATION	1-4	2-4	3-4
W72-28	1.12	1.47	1.11	1.14	-.02	+.33	-.03
29	.79	1.50	1.15	1.29	-.05	+.21	-.14
30	.95	1.49	1.42	1.63	-.68	-.14	-.21
31	1.25	1.70	2.10	1.93	-.68	-.23	+.17
32	1.62	1.89	1.32	1.65	-.03	+.24	-.33
33	1.88	1.84	1.77	1.68	+.20	+.16	+.09
W73-28	.99	1.20	2.09	1.22	-.23	-.02	+.87
30	1.07	1.53	1.96	1.62	-.55	-.09	+.34
32	1.48	1.67	1.31	1.34	+.14	+.33	-.03
W74-26	..51	..85	1.20	.85	-.34	.00	+.35
27	.64	.73	.53	.76	-.12	-.03	-.23
28	.77	.88	.54	.65	+.12	+.23	-.11
29	.91	1.10	1.26	1.02	-.11	+.08	+.24
30	1.13	1.36	1.19	1.38	-.25	-.02	-.19
31	1.43	1.65	1.36	1.55	-.12	+.10	-.19
32	1.28	1.41	.99	1.22	+.06	+.19	-.23
33	.95	1.12	.37	.68	+.27	+.44	-.31
W75-26	.32	.70	.74	.81	-.49	-.11	-.07
28	.57	.57	.57	.65	-.08	-.08	-.08
30	1.18	1.13	1.27	.91	+.21	+.22	+.36
32	1.09	1.25	1.59	1.44	-.35	-.19	+.15
34	.35	.47	.81	.55	-.20	-.08	+.26
W76-25	.25	.56	.71	.50	-.25	+.06	+.21
26	.19	.58	.42	.57	-.38	+.01	+.15
27	.15	.32	.95	.72	-.57	-.40	+.23
28	.38	.38	.46	.61	-.23	-.23	-.15
29				WASTE			
30	1.30	1.01	.74	.58	+.72	+.43	+.16
31	1.25	1.13	1.19	1.21	+.04	-.08	-.02
32	.68	1.05	1.74	1.46	-.78	-.41	+.28
W77-24	.35	.43	.47	.41	-.06	-.02	+.06
26	.12	.51	.56	.38	-.26	[illegible]13	+.18
28	.28	.30	.16	.29	-.01	[illegible]01	-.13
30				WASTE			
32	.83	.87	.94	1.10	-.27	-.[illegible]3	-.16
W78-24	.45	.42	.45	.29	+.16	+.[illegible]	+.16
25	.32	.54	.40	.42	-.10	+.1[illegible]	+.02
26	.19	.46	.42	.42	-.23	+.0[illegible]	.00
29	.80	.83	.48	.62	+.18	+.21	-.14
30	1.44	1.14	.83	.86	+.58	+.28	-.03
31	1.46	1.04	1.15	.76	+70	+.28	+.39
32	1.00	.72	.46	.54	+.46	+.18	-.08

160049

TABLE 4 (Contd.)

Room No.	Grades (% WO_3) Estimated From				Differences		
	1 DDH Assays	2 DDH + Grade Control Assays	3 Grade Control Assays (Sichel Est)	4 Final Interpolation	1-4	2-4	3-4
W80-24	.73	.51	.28	.42	+.31	+.09	-.14
25	.95	.57	.45	.46	+.49	+.11	-.01
26	.86	.54	.39	.49	+.37	+.05	-.10
27	.52	.48	.60	.53	-.01	-.15	+.07
28	.43	.52	.53	.54	-.11	-.02	-.01
29	1.04	.88	.49	.92	+.12	-.04	-.43
30	1.70	1.42	2.40	1.68	+.02	-.26	+.72
31	2.06	1.09	.86	1.08	+.98	+.01	-.22
32	2.16	.44	.34	.29	+.97	+.15	+.05
W81-22	.42	.69	.28	.57	-.15	+.12	-.29
24	1.16	.58	.67	.60	+.56	-.02	+.07
26	1.30	.63	.53	.74	+.56	-.11	-.21
28	.61	.66	.49	.52	+.09	+.14	-.03
30	2.25	1.33	1.41	1.35	+.90	-.02	+.06
32	1.59	.99	.42	.72	+.87	+.27	-.30
W82-22	.41	.82	.84	.77	-.36	+.05	+.07
23	.39	.77	.99	.77	-.38	.00	+.22
24	.99	.63	.43	.71	+.28	-.08	-.28
25	1.47	.53	.90	.83	+.64	-.30	+.07
26	1.25	.66	.86	.85	+.40	-.19	+.01
27	.89	.76	.88	.72	+.17	+.04	+.16
28	.72	.78	.57	.75	-.03	+.03	-.18
29	1.19	1.07	1.04	1.13	+.06	-.06	-.09
30	2.01	1.17	.93	.99	+1.02	+.18	-.06
31	2.13	.75	.52	.46	+1.47	+.29	+.06
W83-22	.34	.71	.50	.78	-.44	-.07	-.28
24	.82	.63	.85	.69	+.13	-.06	+.16
26	.90	.67	.35	.73	+.17	-.06	-.38
28	1.08	.87	1.43	.99	+.09	-.12	+.44
30	1.69	.87	.31	.60	+1.09	+.27	-.29
W84-24	.57	.54	.51	.60	-.03	-.06	-.09
25	.68	.45	.38	.50	+.18	-.05	-.12
26	.84	.65	.68	.84	.00	-.19	-.16
27	.93	.90	1.33	1.23	-.30	-.33	+.10
28	1.11	.84	.89	1.16	-.05	-.32	-.27
29	1.40	.82	.36	.89	+.51	-.07	-.53
Average	.97	.88	.86	.86	.092	.022	.002
Std. Dev.					.450	.189	.246

160050

TABLE 5

RECONCILIATION OF GRADE ESTIMATES FOR WEDGE BLOCK 1ST LIFT, -125 to -122 RL , DOLPHIN MINE

Room No.	Grades ($\% WO_3$) estimated from			Differences	
	1 DDH + Grade Control Assays (1.4.76)	2 Grade Control Assays (Sichel Est.)	3 Final Interpolation (8th Oct 76)	1-3	2-3
W69-34	1.0	1.32	1.30	-.20	.02
W70-31	1.08	1.78	1.29	-.21	.49
32	1.05	1.08	1.33	-.28	-.25
33	1.01	1.00	1.26	-.25	-.26
34	.93	1.00	1.26	-.33	-.26
35	.35	1.57	.80	-.45	.77
W71-32	1.36	1.79	1.50	-.14	.29
34	.91	2.87	1.30	-.39	1.57
W72-28	1.26	2.11	1.56	-.30	.55
30	1.79	1.12	1.20	.59	-.08
31	1.50	1.00	1.15	.35	-.15
32	1.48	1.34	1.22	.26	.12
33	1.70	1.53	1.54	.24	-.01
34	.91	2.36	1.28	-.37	1.08
W73-26	.92	.59	.80	.12	-.21
28	1.29	2.46	1.61	-.32	.85
30	1.60	1.71	1.49	.11	.22
32	1.44	.59	.87	.57	-.58
W74-27	.82	1.01	.98	-.16	.03
28	.98	.75	1.10	-.12	-.35
29	1.23	1.15	1.18	.05	.03
30	1.41	1.30	1.34	.07	-.04
31	1.28	.95	1.10	.18	-.15
32	1.10	.25	.65	.45	-.40
W75-28	.64	.83	.64	.00	.19
30	1.30	.87	.99	.31	-.12
32	1.10	.83	.64	.46	.19
W76-24	.48	.59	.30	.18	.29
30	1.16	.33	.81	.35	-.48
32	1.06	1.01	.78	.28	.23
W77-24	.40	.32	.24	.16	.08
30	1.03	1.14	1.02	.01	.12
32	.69	.53	.62	.07	-.09
W78-24	.38	.23	.16	.22	.07
29	.90	.83	.89	.01	-.06
30	1.16	1.35	1.19	-.03	.16
32	.30	.49	.39	-.09	.10
W79-28	.57	.46	.64	-.07	-.18
30	1.34	1.93	1.31	.03	.62

TABLE 5 (Contd) 160051

Room No.	Grades (% WO_3) Estimated From 1 DDH + Grade Control Assays (1.4.76)	2 Grade Control Assays (Sichel Est.)	3 Final Interpolation (8th Oct 76)	Differences 1-3	2-3
W80-23	.45	.46	.32	.13	.14
24	.44	.34	.29	.15	.05
25	.36	.45	.41	-.05	.04
26	.32	.91	.45	-.13	.46
28	.68	.72	.83	-.15	.11
29	1.45	1.14	1.71	-.26	.57
30	1.11	1.49	1.33	-.22	.16
31	.46	.34	.45	.01	-.11
W81-22	.61	.50	.46	.15	.04
24	.58	.65	.51	.07	.14
28	.42	.40	.90	-.48	-.50
W82-21	.87	.60	.66	.21	-.06
22	.83	.67	.62	.21	.05
23	.65	.54	.63	.02	-.09
24	.65	.88	.69	-.04	.19
25	.61	.68	.64	-.03	.04
26	.60	.54	.58	.02	-.04
27	.66	.40	.58	.08	-.18
28	.97	1.34	1.01	-.04	.33
29	1.11	1.50	1.40	-.29	.10
30	.60	.84	.93	-.33	-.09
Average	.92	.996	.92	.0065	.095
Std. Dev.				.025	.369

(ii) there is a marked difference in the quality of the predictions for the first lift, and for the Undercut. This can be attributed to an increase in the amount of grade information and improvements to the numerical techniques between January and April, 1976.

(iii) the quality of the predictions is best within the orebody and is poor on the margins, because the existing logic of the computer programs cannot recognise property changes in dip and rock units. Attempts are being made to rectify this deficiency.

(iv) the tabulated values do not include dilution by mine fill.

REFERENCES

COSON, C.H. and SICHEL, H.S. 1959. Quality control of routine mine assaying and its influence on underground evaluation. J. Sth. African Institute of Mining and Metallurgy. V.59 p489-515.

INGAMELLS, C.O., 1974. Control of geochemical error through sampling and subsampling diagrams. Geochimica et Cosmochimica Acta V.38, p1225-1227.

KNUDSEN, H.P., KIM, Y.C. and MUELLER, E., 1976. A Comparative study of the geostatistical ore reserve estimation method over conventional methods. APCOM 1976.

MIESCH, A.T., 1967. Theory of error in geochemical data. U.S. Geol. Survey Prof. Paper. 574-A.

ROWLAND, R. St. J., and SICHEL, H.S., 1960. Statistical quality control of routine underground sampling. J. Sth. African Institute of Mining and Metallurgy. V.60, p251-284.

SICHEL, H.S., 1966. The estimation of means and associated confidence limits for small samples from lognormal populations. Sth African Institute of Mining and Metallurgy.

APPENDIX A

Some Results for Check and Umpire Assaying
Dolphin Orebody

Abbreviations Used

WOR	One Metre Recommended Grade
WOK	K.I.S. Assay for WO_3
STR	Stratigraphic Unit
MOK	K.I.S. Assay for Mo
WOA	AMDEL Assay for WO_3
MOA	AMDEL Assay for Mo
WOC	ACSL Assay for WO_3
MOC	ACSL Assay for Mo

160055

COORDINATION FOR D000/3

DEPTH (M)	SAMPLE NO. TP.	ASSAY VALUES WOR	WOK	STR	MOK	WOA	MOA	WOC
4.00								
	C 902HC	12.705	19.600	6.000				
	C 2116HC		19.100		.800	17.280	.400	16.800
	C 2388HC		19.600		.850			
	C 1748HC		20.900		.720			
	C 2229HC		20.900		.860			
5.00								
12.00								
	C 2117HC	2.288	2.700		.100			
	C 1746HC		2.450		.080			
	C 2216HC		2.450		.070			
	C 910HC		2.410	6.000				
	C 2389HC		2.720		.110			
13.00								
31.00								
	C 917HC	.090	.120	6.000				
	C 2186HC		.120					
	C 1756HC		.100					
	C 2390HC		.130					
	C 2118HC		.120					
	C 2201HC		.100					
31.50								
59.00								
	C 1757HC	2.714	2.900		.100			
	C 2187HC		3.100		.110			
	C 929HC		2.900	6.000				
	C 2119HC		3.600		.070			
	C 2391HC		2.520		.120			
	C 2204HC		3.100		.110			
60.00								
70.00								
	C 940HC	.231	.260	6.000				
	C 2120HC		.250		.010			
	C 2189HC		.240		.010			
	C 1758HC		.270					
	C 2222HC		.270		.010			
	C 2392HC		.250		.020			
71.00								
81.00								

160056

COORDINATION FOR D080/5

DEPTH (M)	SAMPLE NO. TP.	ASSAY VALUES WOR	WOK	STR	MOK	WOC	MOC	WOA
60.00								
	C 1852HC	.116	.170	6.000				
			.210					
	C 1901HC		.190					
	C 1065HC		.160		.290			
			.200					
61.00								
	C 1853HC	.318	.310	6.000				
	C 2074HC		.300		.010	.330	.012	.440
62.00								
66.00								
	C 2075HC	.609	.720		.030			
	C 1858HC		.660	6.000				
67.00								
	C 1859HC	1.073	.990	6.000				
			1.090					
	C 1066HC		1.400		.050			
			1.400					
	C 1902HC		.880					
	C 1051HC					.830	.024	1.510
68.00								

COORDINATION FOR 0120/4

DEPTH (M)	SAMPLE NO. TP.	ASSAY VALUES WOR	WOK	STR	MOK	WOC	MOC	WOA
13.00								
	C 1501HC	1.647	2.460	6.000				
			2.740					
	C 1052HC					2.010	.054	2.590
	C 1061HC		2.710		.120			
			3.600					
	C 1903HC		2.360					
14.00								
	C 1502HC	2.391	2.120	6.000				
	C 2076HC		2.560		.120			
15.00								
23.00								
	C 1904HC	1.039	1.050					
	C 1053HC					1.250	.047	1.390
	C 1062HC		1.230		.040			
			1.420					
	C 1508HC		.900	6.000				
			.970					
24.00								
28.00								
	C 2078HC	.647	.760		.030			
	C 1513HC		.680	6.000				
29.00								
33.00								
	C 1063HC	.402	.090		.110			
			.210					
	C 1905HC		.670					
	C 1518HC		.700	6.000				
			.780					
34.00								
	C 2077HC	5.306	6.300		.250	5.250	.100	5.670
	C 1519HC		6.000	6.000				
35.00								
43.00								
	C 1528HC	.790	.780	6.000				
			.880					
	C 1064HC		.860		.040			
			1.000					
	C 1906HC		.880					
44.00								
52.00								
	C 1532HC	.584	.680	6.000				
	C 2079HC		.620		.010			
53.00								

160058

COORDINATION FOR D160/6

DEPTH (M)	SAMPLE NO. TP.	ASSAY VALUES WOR	WOK	STR	MOK	WOC	MOC	WOA
1.00								
	C 1090HC	.294	.490		.040			
			.510					
	C 1914HC		.440					
	C 1654HC		.430					
2.00								
58.00								
	C 2088HC	.872	.050		.170			
	C 1591HC		1.890	6.000				
59.00								
64.00								
	C 2087HC	.368	.410		.010			
	C 1597HC		.410	6.000				
65.00								
66.00								
	C 1599HC	1.309	1.380	6.000				
			1.440					
	C 1091HC		1.520		.040			
			1.500					
	C 1915HC		1.450					
67.00								
71.00								
	C 1604HC	.265	.300	6.000				
	C 2089HC		.290		.010			
72.00								
76.00								
	C 1092HC	1.529	1.650		.060			
			1.650					
	C 1609HC		1.760	6.000				
			1.850					
	C 1916HC		1.650					
77.00								
80.00								
	C 1613HC	2.593	2.720	6.000				
	C 2090HC		3.050		.080			
81.00								
86.00								

COORDINATION FOR D160/6 100059

DEPTH (M)	SAMPLE NO. TP.	ASSAY VALUES WOR	WOK	STR	MOK	WOC	MOC	WOA
	C 1619HC	2.897	3.500	6.000				
			3.500					
	C 1093HC		2.450		.140			
			2.700					
	C 1917HC		3.600					
87.00								
89.00								
	C 1622HC	5.437	5.800	6.000				
	C 2093HC		6.300		.310			
90.00								
95.00								
	C 1628HC	1.277	.980	6.000				
			1.100					
	C 1094HC		1.600		.050			
			1.580					
	C 1918HC		1.600					
	C 1055HC					1.400	.034	1.510
96.00								
100.00								
	C 1633HC	1.442	1.580	6.000				
	C 2092HC		1.760		.070	1.390	.035	1.770
101.00								
105.00								
	C 1095HC	.780	.880		.040			
			.900					
	C 1919HC		.930					
	C 1638HC		.750	6.000				
			.820					
106.00								
111.00								
	C 1644HC	2.825	.900	6.000				
	C 2091HC		5.390		.120			
112.00								
115.00								
	C 1920HC	274.461	.660					
	C 1096HC		.750		.030			
			.700					
	C 1648HC		.580	6.000				
			.630					
	C 1056HC					.720	.027	.080
116.00								

160060

COORDINATION FOR D200/1

DEPTH (M)	SAMPLE NO. TP.	ASSAY VALUES WOR	WOK	STR	MOK	WOC	MOC	WOA
5.00								
	C 1197HC	3.579	5.600	4.000				
			5.600					
	C 1070HC		6.200		.210			
			6.200					
	C 1058HC					5.660	.120	4.850
	C 1924HC		5.200					
6.00								
18.00								
	C 1803HC	.566	.610	4.000				
	C 2103HC		.650		.030			
19.00								
33.00								
	C 1925HC	.018	.020					
	C 1071HC		.020					
			.030					
	C 1807HC		.010	4.000				
			.020					
34.00								
58.00								
	C 1811HC	.400	.390	6.000				
	C 2102HC		.500		.030			
59.00								
60.00								
	C 1926HC	.280	.060					
	C 1072HC		.810		.030			
			.800					
	C 1813HC		.050	6.000				
			.090					
61.00								
64.00								
	C 1817HC	2.548	2.500	6.000				
	C 2105HC		3.170		.140			
65.00								
73.00								
	C 1821HC	3.265	3.700	6.000				
			3.700					
	C 1067HC		3.800		.120			
			3.800					

COORDINATION FOR D200/1

DEPTH (M)	SAMPLE NO. TP.	ASSAY VALUES WOR	WOK	STR	MOK	WOC	MOC	WOA
	C 1921HC		3.400					
74.00								
78.00								
	C 1826HC	.732	.700	6.000				
	C 2100HC		.930		.030			
79.00								
83.00								
	C 1831HC	1.945	2.250	6.000				
			2.500					
	C 1068HC		2.200		.040			
			1.220					
	C 1922HC		2.480					
84.00								
87.00								
	C 2104HC	5.212	6.000		.230			
	C 1835HC		5.600	6.000				
88.00								
93.00								
	C 1841HC	.033	.010	6.000				
			.020					
	C 1069HC		.040					
			.100					
	C 1923HC		.030					
94.00								
96.00								
	C 2101HC	.255	.270		.010	.370	.022	.350
	C 1844HC		.240	6.000				
97.00								

160062

COORDINATION FOR D240/2

DEPTH (M)	SAMPLE NO. TP.	ASSAY VALUES WOR	WOK	STR	MOK	WOA	MOA	WOC
11.00								
	C 1993HC	.121	.200	4.000				
	C 174HC					.190	.020	
	C 173HC							.240
	C 175HC		.120		.030			
12.00								
25.00								
	C 171HC	1.447				1.830	.080	
	C 170HC							1.950
	C 1997HC		1.500	6.000				
	C 172HC		1.160		.090			
26.00								
35.00								
	D 167HC	.917						.980
	D 2007HC		.990	6.000				
	D 168HC					1.080	.065	
	D 169HC		1.030		.090			
36.00								
64.00								
	D 164HC	.652						.710
	D 2036HC		.700	6.000				
	D 165HC					.760	.040	
	D 166HC		.730		.060			
65.00								
74.00								
	D 161HC	.025						.040
	D 2046HC		.010	6.000				
	D 162HC					.030	.015	
	D 163HC		.030		.030			
75.00								
82.00								
	D 159HC	.339				.370	.025	
	D 2054HC		.320	6.000				
	D 160HC		.340		.040			
	D 158HC							.480
83.00								