67-460

MICROFILMED

MT CLEVELAND

ABERFOYLE TIN DEVELOPMENT PARTNERSHIP

CLEVELAND DEVELOPMENT PROJECT

THE USE OF COMPARATIVE SAMPLING METHODS

AT CLEVELAND MINE, TASMANIA

Roy Cox,
Resident Geologist,
Cleveland Tin N.L.,
LUINA,
via Waratah,
Tasmania.

March 1967.

179003

CONTENTS

1. ABSTRACT

In the exploration and development of new mines, large tonnages of ore are frequently outlined by means of diamond drilling. The characteristics of the ore govern the relationship between assay results from the sampling of diamond drill cores and the mean values yielded by other methods of sampling. It is desirable, especially with the evaluation of increasingly lower-grade ores, that this relationship should be established before major financing is undertaken.

During the evaluation of the tin-copper deposit at Cleveland Mine, Tasmania, by the Aberfoyle Tin Development Partnership during 1961-66, the complicated structure of the ore-bodies and the great local variation in tin-copper mineralisation, presented challenging geological and sampling problems. These problems were met by an exploratory underground development programme and the use of comparative methods of sampling under close geological control.

The comparative sampling results are presented, together with data relating to the geology, sample collection and preparation, and assaying. Groove, channel, AXT split diamond drill core and bulk sampling, as used at Cleveland Mine, all yielded closely checking average results for both tin and copper content of the ore.

2. INTRODUCTION

The Aberfoyle Tin Development Partnership secured the Cleveland Mine leases (latitude $41^{\circ}28'$ South, longitude $145^{\circ}24'$ East), in North-western Tasmania, in September 1961 and immediately undertook a thorough and systemmatic series of exploration programmes, carried through until 1966, comprising 25,207 feet of diamond drilling involving 98 holes, and 2980 feet of underground driving and cross-cutting on the 1300 ft. R. L. "Qa" adit level. This resulted in the establishment of indicated and inferred ore-reserves totalling 2,850,000 tons of average assays 1.02 % Sn and 0.43 % Cu and was followed by the flotation of Cleveland Tin N. L. on 20th September 1966, to establish operations for the

production of tin and copper concentrates. Construction of a new townsite and treatment plant, together with mine development, are currently in progress. Production is scheduled to commence in the first half of 1968.

The initial diamond drilling exploration programme, comprising 46 holes having a total footage of 12,106 feet, was completed in July 1963. The results were encouraging, suggesting that in excess of 1½ million tons of 1%Sn, 0.3%Cu ore might be inferred. The results also revealed, however, that the ore-bodies were complex both from the point of view of geological structure and local variation in tin-copper mineralisation.

In order to investigate more fully the complex geological structures and to ascertain the degree of reliability that could be placed in the tin and copper grades obtained from sampling diamond drill cores, a programme of underground development and comparative sampling was undertaken, on the 1300 ft. R. L. "Qa" adit level, during the period April 1964 - October 1965. This programme involved:-

1. 2980 ft. of driving and cross-cutting over a strike length of 1338 ft.
2. 36 AXT diamond drill holes having a total footage of 3118 ft.
3. taking 756 groove samples from development drives.
4. taking 336 groove samples over 816 ft. in 8 cross-cuts
5. taking 318 channel samples over 765 ft. in 8 cross-cuts
6. taking 346 split diamond drill core samples over 704 ft. on 21 cross-sections
7. taking 72 bulk samples, having a total weight of 311 tons, over 315 ft. in 8 cross-cuts.

In addition to achieving its prime objects, the underground programme provided (1) valuable material for milling and metallurgical research purposes, (2) an opportunity to investigate the nature of the ground with respect to competence, strength, hardness, friability, permeability and other allied factors important to future mining operations, (3) an opportunity to determine density factors for both broken and unbroken ore for future estimation and operational use, (4) an opportunity to analyse variations in lode width, tin and copper grade, chert content, and mineralogy of the ore-bodies over a reasonable strike length.

3. <u>GEOLOGICAL SUMMARY</u>

Only a brief summary of the complex geological setting of the Cleveland ore-bodies is given here, in order to allow an understanding of the sampling problem.

The ore-bodies are situated in rugged terrain covered with dense-rain forest, and outcrop on the steep southern slope of Crescent Spur, in the valley of Deep Creek, between 1100 and 1500 ft. R. L. Cassiterite and chalcopyrite occur with iron sulphides in a lode bed which is stratigraphically conformable with the enclosing rocks. The lode bed varies from less than 1 to 15 ft. thick and occurs within a series of interbedded fine grained, grey shales and grey-chocolate brown cherts, known collectively as the Lode Formation. This formation has a thickness of up to 100 ft. in the mine area, and lies between the older Mica Sandstone Formation and the younger Basic Volcanics Formation. These three formations constitute the Crescent Spur Group of the Lower Palaeozoic.

The major ore structure is the overturned S. E. limb of a S. W. -plunging anticline. The lode bed is one stratigraphic horizon which has been displaced along several major N. W. -dipping axial plane faults into a series of separate lenses. These lenses occur en echelon, increasing in depth to the S. E. Minor faulting, complimentary to the major movement, further complicates the overall structure. A typical cross-section of the deposit is shown in Figure 1.

The ore-bodies, or lenses, may have an overall width of up to 60 ft., formed by a complicated repetition of the lode bed due to folding and/or minor faulting. Thus, in any particular intersection, an ore-body or lens consists of a number of repetitions of the lode bed separated by infolded and/or infaulted bands of chert wall-rock. An ore-body thus contains an average of approximately 30-percent (by volume) included chert. Due to the complications of minor structures, the two walls of a cross-cut through a lens may have little resemblance to one another. A typical plan of a cross-cut through an ore-body is shown in Figure 2.

The average ore contains approximately 35-percent sulphides, chiefly pyrrhotite, with minor amounts of pyrite, chalcopyrite,

marcasite, arsenopyrite, sphalerite, tetrahedrite and stannite in
a quartz-dolomite groundmass containing minor amounts of calcite,
siderite, chlorite, tourmaline, fluorite, muscovite and sericite.
Tin occurs chiefly in the form of the oxide, cassiterite, with only
a small proportion in the form of the sulphide, stannite.   Copper
occurs chiefly in the form of the sulphide, chalcopyrite.

The stratiform ore is uniform in the broad sense only; in
detail it is far from uniform.   Grain size, concentration and relative
amounts of the various sulphides, and cassiterite, vary rapidly
within a few inches.   Sulphide mineralisation varies from fine dis-
seminations to irregular, massive aggregates and fracture fillings.
Cassiterite is very fine grained, averaging approximately 0. 2 mm.
across, and is rarely seen with the naked eye.   The ore is usually
well banded, hard, and generally massive.

Sampling, therefore, has to contend with rapidly changing
tin-copper content in association with variations in lode bed mineralogy
and rock type.

4.   SAMPLING METHODS

In order to determine the relationships between assay results
from samples of varying sizes for the Cleveland ore, 8 cross-cuts
(designated K, L, N, Qa, R, T and V), spaced at 120 ft. intervals
along strike, were driven at right angles to the strike through Halls
Lode, Lens "A", on the 1300 ft. R. L. "Qa" adit level .   Close
geological mapping and control was an important factor in the success
of the programme.   Both walls and the back of each cross-cut were
mapped in detail at scale 1 inch to 5 ft.

Four types of sample were taken on each of the 8 cross-sections,
(viz. groove, AXT split diamond drill core, channel and bulk).   The
data concerning the relative sizes of these samples is given in Table
1.   Groove and channel samples were cut along both walls of each
cross-cut.   A horizontal AXT diamond drill hole was drilled as close
as practical to one wall of each cross-cut, at the same elevation as
the groove/channel sample location.   A bulk sample was cut as a slot
from the back of each cross-cut.

The comparative sampling programme was based upon sound sampling principles, of which the following points should be noted:-

1.   All sampling was carried out with geological control and thorough supervision.

2.   All sample locations were carefully marked with paint and are referable to the mine survey grid.

3.   All sample locations were thoroughly washed with water before sampling to ensure that 'clean' samples were taken.

4.   All sample lengths were subject to geological control. Standard lengths of cut were never used, but each sample length was laid out on the basis of rock type and mineralisation, with the aim of achieving a high degree of uniformity within each length. Sample lengths therefore varied from a few inches up to a maximum of 48 inches. This factor is particularly important in groove and channel sampling, where even an imperfectly cut sample will tend to produce a relatively reliable result if each sample length traverses reasonably uniform material. This ensures that confidence can be placed in the weighted average grade. (This condition was not applicable to the taking of bulk samples.)

5.   All groove, channel and split core samples were taken by the same reliable sampler.

6.   All groove, channel and split core samples were placed in a new, clean canvas bag with the identification number marked on the outside in ink, and on an aluminium foil marker placed inside the bag. This system insured well against samples becoming un-identifiable.

7.   Groove and channel samples were cut from the same sample location in order to facilitate direct correlation of assays from samples of different sizes.

8.   All groove and channel cuts, and split core, was inspected before the samples were accepted for preparation and assays.

These factors ensured that reliable and representative samples were obtained, and that 'sampling errors' were kept to the minimum practical limit.

(a)   GROOVE SAMPLES, consisting of a 1 in. wide x $\frac{1}{2}$ in. deep groove, were cut by hand using a hammer and moil, from both walls of each cross-cut, over the full width of the ore-body. A total of 518 ft. of groove samples were cut from the walls of the 8 cross-cuts through Lens "A". Just under $\frac{3}{4}$ lb. of sample were obtained per foot of groove cut.

(b) CHANNEL SAMPLES, consisting of a 6 in. wide x 2 in. deep channel, were cut by hand using a hammer and moil, from both walls of each cross-cut, using the same sample locations as used for groove sampling. A total of 494 ft. of channel samples were cut from the walls of the 8 cross-cuts through Lens "A". Approximately 17 lb. of sample were obtained per foot of channel cut.

The cuttings from groove and channel samples were caught in a specially designed canvas sheet held firm by the samplers body against the cross-cut wall, thus leaving both hands free for the task of cutting.

(c) SPLIT CORE SAMPLES, were taken from AXT diamond drill holes drilled as close as practical to one wall of each cross-cut, and at the same horizon (4 ft. above rail level) as the groove and channel sample locations. Every effort was thus made to ensure that split core and groove/channel sample locations penetrated near-identical ground. This was essential if good correlations between rock type, mineralisation and tin and copper grade were to be obtained, in view of the earlier comments regarding the complicated minor structures within each lens.

A total of 273 ft. of split core samples were taken from the 8 drill holes through Lens "A". AXT core has a diameter of 1.28 in., and just under 1 lb. of sample was obtained per foot of split core. Core recovery at Cleveland Mine is consistently good, and sludge samples are not taken. Core recovery for the 8 intersections of Lens "A" was between 95 and 100 % for all drill runs.

(d) BULK SAMPLES, were bored and fired from the back of each cross-cut, and collected on a series of tightly-fitting sollar boards. Each sample was taken separately, and measured 4 ft. wide by 2 ft. 6 in. deep; the length varied between 3 ft. 6 in. and 5 ft.

66 bulk samples having a total footage of 292 ft. were taken from the 8 cross-cuts through Lens "A". Approximately 2000 lb. of sample were obtained per foot of sample length.

Bulk samples locations were marked up with paint, then bored, fired and hand-bogged under close control and supervision. Each truck of broken material was carefully identified and then weighed at the adit portal before being transported to the sample reduction plant.

5.   SAMPLE PREPARATION

All groove, channel and split diamond drill core samples were reduced in a Van Gelder 3 in. jaw crusher to approximately minus 1/8 in., prior to despatch to the assay laboratory. In addition, channel samples had to be reduced by riffling.

The process of sample reduction provides many potential sources where errors may be introduced. With this in mind, processing routine and layout were designed for optimum efficiency and to ensure that confusion of sample numbers did not arise. All sample preparation was carried out by one employee under strict geological control and supervision. All obvious safety factors were observed, with particular attention being paid to the thorough cleaning of machinery and equipment after each sample had been prepared.

After preparation, all samples were replaced in their original canvas bags and forwarded to the Mines Department Laboratories at Launceston, Tasmania, for assaying for tin and copper.

Bulk sample reduction was fraught with more potential sources of error than that for the other types of sample because of the larger number of employees coming into contact with the sample, and the greater number of processes involved. Strict control and supervision of all aspects of the process ensured that any potential sources of error were kept to the absolute minimum practical limit.

The flowsheet for the process of bulk sample reduction is shown in Figure 3. The final products were a 600 lb. sample for pilot plant investigations and a 200 lb. sample for assay and other

quantitative assessments.   Both samples were drummed and sealed and despatched to the Mines Department Laboratories at Launceston, Tasmania.

6.   ASSAYING

The main purpose of this article is to compare the results of various sampling methods, but assay office procedure should be mentioned, if only briefly, as it provides potential sources where errors may be introduced into the sample.

Tin and copper assays for all samples taken at Cleveland Mine during the exploration and underground development programmes were carried out by the Tasmanian Mines Department Laboratory at Launceston.   Laboratory procedure   was to reduce all samples to pass a 10 mesh (1. 67 mm.) screen.   The crushed sample was well mixed then riffled down to an assay sample of 1-2 lbs.   This sample was then pulverised in a Braun Pulveriser to all pass a 60 mesh (0. 25 mm.) screen.   The pulverised sample was mixed again then packeted for assay .

The tin assay method consisted of the following procedure:-

1.   Weigh 0. 5 to 3. 0 g. for assay.
2.   Digest with hydrochloric/nitric acid to remove acid soluble material.   Where much sulphur was present it may have had to be oxidised.
3.   Filter, wash, and burn off residue.
4.   Fuse residue with sodium peroxide in an iron crucible.
5.   Dissolve melt in 1:3 hydrochloric acid.
6.   Reduce by boiling with a nickel strip for 50 minutes in a covered beaker.
7.   While still near boiling, add a piece of marble then cool.   Add more marble during cooling as required.
8.   Titrate with standard iodine.

The copper assay method consisted of the following procedure:-

1.   Weigh 0. 5 to 2. 0 g. for assay.
2.   Acid treat as for tin, oxidising sulphur if present.
3.   Fume with 7-10 ml. of sulphuric acid.   Cool.
4.   Dissolve in water.
5.   Precipitate copper with sodium thiosulphate solution.
6.   Filter, wash and ignite the precipitate.
7.   Dissolve ignited precipitate in nitric acid, oxidise with bromine.

8.   Make just alkaline with ammonia and add 4 ml. acetic acid.

9.   Titrate with standard sodium thiosulphate.

A number of unknown check samples were introduced at irregular intervals, and on a few occasions the second half of a split-core sample was introduced as a check. Consistent results were obtained from these checks. An important point in this connection is that a composite sample prepared during 1963 from certain early drill core intersections gave good agreement with the calculated weighted averages.

7.   ANALYSIS OF COMPARATIVE SAMPLING RESULTS

The prime objective of the comparative sampling programme was to investigate the relationship between the assay results obtained from different sized samples. For evaluation purposes, the relationship between the assay results obtained from the sampling of split diamond drill core and the mean values obtained from bulk samples had to be investigated. For possible future mining operations procedure, the relationship between the assay results obtained from groove and channel samples and the mean values obtained from bulk samples had to be investigated.

As a result of the complex minor structures of the Cleveland ore-bodies two sample locations, which penetrate an ore-body and are separated by only a few feet, have not only different grades but also different overall widths. Reference to Figure 2, illustrates this point. For example, the overall horizontal width of the ore-body, as measured in groove S. W. wall, groove N. E. wall, diamond drill hole and bulk sample locations, is 283 in., 301 in., 180 in. and 313 in. in the "Qa" cross-cut. These locations are all within 12 ft. of the centre line of the cross-cut. One further point to note here is that the bulk sample slots were cut to include approximately 6 in. of barren wall-rock at each end. It was considered preferable to over-break and include a little wall-rock, so diluting the lode grade by a small percentage, rather than under-break. Over-break varied between 0.8% for the "V" cross-cut and 4.9% for the "L" cross-cut.

In estimating ore-reserve grade for this type of deposit, the weighted average grade and overall lode width are equally important. The important value for each sample intersection of an ore-body is therefore the metal content or value, where "value" = product (width x grade). In analysing the results of the comparative sampling programme, this point should be borne in mind.

The comparative sampling results for Halls Lode, Lens "A", in each of the 8 cross-cuts on the 1300 ft. R. L. "Qa" adit level are summarised in Tables 2 and 3.

(a)   COMPARISON OF WEIGHTED MEANS FOR 4 SAMPLE SIZES

The weighted mean tin and copper grades and values for groove samples (both walls individually and combined), channel samples (both walls individually and combined), AXT split core samples and bulk samples obtained from the 8 cross-cuts are analysed in Table 4. These data have been extracted from Tables 2 and 3.

The mean tin and copper grades and values for the various groove, channel and split core samples are compared with those obtained for the bulk samples, which were the largest samples it was practical to obtain. The following points are noted:

1.   for the overall lode tin grade (%Sn), the percentage difference from the bulk sample grade has a range from -7. 4% (channel samples, N. E. wall) to +12. 3% (channel samples, S. W. wall).

2.   for the overall lode copper grade (%Cu), the percentage difference from the bulk sample grade has a range from -3. 3% (groove samples, S. W. wall) to +10. 0% (D. D. H. split core samples).

3.   these differences in lode grade, as mentioned earlier, include variations in overall lode width for the different types of sample cut from near identical locations.

Thus a more meaningful result is obtained from an analysis of the relationships between tin and copper content, or value, obtained from the 4 sample sizes.

4.　for the overall lode, tin value, the percentage difference from the bulk sample value has a range from - 13. 7% (channel samples, N. E. wall) to +7, 3% (channel samples, S. W. wall). The split core, groove (average both walls) and channel (average both walls) samples show differences from the bulk sample value of -2. 1%, -6. 9% and -3. 2% respectively.

5.　for the overall lode copper value, the percentage difference from the bulk sample value has a range from -10. 0% (groove samples, S. W. wall) to +0. 1% (channel samples, S. W. wall). The split core, groove (average both walls) and channel (average both walls) samples show differences from the bulk sample value of -0. 1%, -7. 7% and -0. 8% respectively.

The significance of these means and percentage differences can be ascertained by testing for the assumption that the 4 types of sample (groove and channel, both walls averaged, split core and bulk) have equal means.

(b)　TEST FOR ASSUMPTION THAT THE 4 SAMPLE SIZES HAVE EQUAL MEANS

The F-distribution can be used to test for significant differences among two or more means. In this procedure it is necessary to estimate the variance of the sampling distribution of the means. This variance, $\dfrac{S_1^2}{N}$, contains both actual variation in the population being sampled as well as the variation due to errors in the sampling methods. The variance can be reduced by increasing N, the total number of samples, or by reducing S_1^2, which includes the errors in the sampling methods. Increasing the sampling numbers may be a costly procedure in contrast to improving the accuracy of the sampling methods.

If four random samples are taken from a perfectly homogenous population, it is known that the individual averages for the samples will not be the same because there is a variation within the parent population from which the

samples were taken. However, it is expected that the variation among the four sample averages will be of the same magnitude as the variation within the parent population. Although the actual variation within the population may not be known, it can be estimated by determining the variation within the individual items for each of the four samples. This is the within-sample variation and is a measure of the expected variation of the parent population. If the between-sample variation is significantly larger than the within-sample variation, then it can be expected that the samples were not taken from the same population but from populations whose averages are different.

In making an analysis of variance, the sum of the squares of the deviations of the items from the mean and the number of degrees of freedom for each source of variability are computed. This amounts to estimating the variance of the parent population by two separate methods and then comparing these estimates.

The first estimate of variance is made by the pooled method. For a set of four samples containing several items, the formula for the pooled estimate of the within-sample group variance is:

$$Sp^2 = \frac{\Sigma(X_1 - M_1)^2 + \Sigma(X_2 - M_2)^2 + \Sigma(X_3 - M_3)^2 + \Sigma(X_4 - M_4)^2}{n_1 + n_2 + n_3 + n_4 - 4}$$

where n_1, n_2, n_3, n_4 are the number of items in each of the four samples and $\Sigma(X_1 - M_1)^2$ equals the summation of the squares of the differences between the individual items and the mean for the first sample. This formula has been simplified to

$$Sp^2 = \Sigma X_{iT}^2 - \left(\frac{T_1^2}{n_1} + \frac{T_2^2}{n_2} + \frac{T_3^2}{n_3} + \frac{T_4^2}{n_4} \right)$$

where ΣX_{iT}^2 represents the sum of the squares of the total number of all items in the samples and T_1^2 equals the square of the total of all the n_1 items in the first sample.

If we assume that the groove, channel, split core and bulk samples represent 4 random samples taken from a perfectly homogenous population, then for the overall lode tin values for groove (average both walls), channel (average bot' walls), split core and bulk samples given in Table No. 5:

$$Sp^2 = 1158,367 + 1068,026 + 1122,785 + 1084,320 - \left(\frac{(2871)^2}{8} + \frac{(2674)^2}{8} + \frac{(2779)^2}{8} + \frac{(2810)^2}{8} \right)$$

$$= 4433,498 - 3876,483 = 557,015$$

Similarly, for the corresponding overall lode copper values,

$$Sp^2 = 620,140 - 560,292 = 59,848$$

The second estimate of the variance is made by directly computing the variance of the means. This is the variation between sample means and can be computed by the formula:

$$S_m^2 = \frac{T_1^2}{n_1} + \frac{T_2^2}{n_2} + \frac{T_3^2}{n_3} + \frac{T_4^2}{n_4} - \frac{T_T^2}{N}$$

where T_T^2 is the square of the total of all the N items in all four samples. Again from Table No. 5, for the overall lode tin values:

$$S_m^2 = \frac{(2871)^2}{8} + \frac{(2674)^2}{8} + \frac{(2779)^2}{8} + \frac{(2810)^2}{8} - \frac{(11134)^2}{32}$$

$$= 3876,483 - 3873,936 = 2547$$

Similarly, for the corresponding overall lode copper values:

$$S_m^2 = 560,292 - 559,682 = 610$$

Total variation equals the sum of the variation between means and the variation within sample groups.

$$\text{Total variation} = \sum X_{iT}^2 - \frac{T_T^2}{N}$$

For overall lode tin values,

Total variation = 4433,498 - 3873,936 = 559,562

and for overall lode copper values,

Total variation = 620,140 - 559,682 = 60,458

The number of degrees of freedom for the between-sample means variation is 3, or 1 less than the number of sample means. The number of degrees of freedom for the within-sample groups variation is 28, which is (N - K) or 4 less than the total number of items N, where K equals 4, which is the number of samples. The mean square is found by dividing the sum of squares by the degrees of freedom. These mean squares are the two estimates of the variance of the parent population. The mean square obtained from the between-sample means is always used as the numerator when calculating the F-ratio, because the means are rejected as significant only if they are more spread out than would be expected for samples from the population. These data for the overall lode tin and copper values and grades are summarised in Tables 5, 6 and 7.

If the 99. 5-percent confidence level is selected for rejection, there is evidence for rejection if the F-ratio is larger than that which would be due to chance 99. 5 percent of the time. The computed F-test ratios of 0. 43 and 0. 10 for the tin and copper values respectively, and 0. 41 and 0. 37 for the tin and copper grades respectively, are less than the 99. 5 percent value of 5. 32, taken from tables (Reference 8, Table 29E). Therefore the assumption of equal sample means is accepted for tin and copper grades and values, subject to the limitations of the initial assumptions.

(c)   COMPARISON OF GROOVE AND CHANNEL SAMPLE ASSAYS

Groove and channel samples were cut from identical sample locations on both walls of each of the 8 cross-cuts, in order that the actual tin and copper assays from both types of sample could be directly compared. Under ideal conditions, the channel samples had a volume 24 times greater than the corresponding groove samples.

Figures 4 and 5 are scatter diagrams with channel sample assays (tin and copper respectively) plotted as the independent variable (X) and groove sample assays plotted as the dependent variable (Y). The data employed are the 128 lode band samples from the 8 cross-cuts through Lens "A" on the 1300 ft. R. L. adit level. Inspection of the scatter diagrams shows a linear trend in both cases, which approximates to a straight line with the equation $Y = a + bX$, where a and b are constants. Furthermore, examination shows $a = 0$ approximately, and $b = 1$ approximately in both cases.

These data may be analysed in four stages, (see Appendices B, C, D and E):-

(1)   The method of least squares provides a convenient way for obtaining an objective fit of a straight line trend line to a series of data. It accomplishes two objectives :

(a)   the sum of the vertical deviations of the observed values from the fitted line equals zero, and

(b)   the sum of the squares of all these deviations is less than the sum of the squared vertical deviations from any other straight line.

For tin assays, the method of least squares gives an estimating equation,

$$Y_c = 0.01 + 0.96X$$

where units of X are in %Sn.

Similarly, for copper assays the method of least squares gives an estimating equation,

$$Y_c = 0.06 + 0.78X$$

where units of X are in %Cu.

(2)   The standard error of estimate, $(S_{Y \cdot X})$ is a measure of the divergence of the actual values of the dependent variable (Y) from their computed values (Y_c). That is, it is the standard deviation of the Y values around the estimating equation $Y_c = a + bX$, and is a measure of the dependability of the estimates, in absolute terms. It yields an estimate of the range above and below the line of estimation within which 68.27 percent of the items may be expected to fall if the scatter is normal.

For tin assays, $\quad S_{Y \cdot X} = 0.61\ \%\ Sn.$

For copper assays, $\quad S_{Y \cdot X} = 0.18\ \%Cu.$

(3)   The coefficient of correlation, (r), is a measure of the degree or closeness of relationship between the variables, independent of the units in which they were originally measured.   It is a number varying from +1, through zero, to -1.   The magnitude of the coefficient indicates the degree of association;   when r equals zero there is absolutely no relationship.

For tin assays, $\quad r = +0.7172$

For copper assays, $\quad r = +0.7415$

(4)   The coefficient of determination, (r^2) indicates the relative amount of variation in the dependent variable which has been explained by the estimating equation. For tin assays, $r^2 = 0.514.$ This indicates that 51.4 percent of the total variation is explained by the estimating equation.   For copper assays, $r^2 = 0.550.$ This indicates that 55.0 percent of the total variation is explained by the estimating equation.

From these analyses we conclude that the assays from the groove samples provide as reliable an estimate of the tin grade as the assays from the larger channel samples.   That is, the estimating equation, $Y_c = 0.01 + 0.96X$ is equivalent to $Y_c = X$ for all practical purposes.   The same conclusion does not hold good in the case of the copper grade, however, where the estimating equation is $Y_c = 0.06 + 0.78X.$

(d)   <u>ANALYSIS OF ASSAYS FROM LODE AND CHERT BANDS</u>

The distribution characteristics of the tin and copper mineralisation with respect to the lode and chert bands were analysed by plotting separate frequency distributions for these assays from groove, channel and AXT split drill core samples.

To illustrate the results obtained from these analyses, the frequency distributions of tin and copper assays from lode and chert bands for groove samples only are presented in Figure 6.   These distributions show conclusively that both

tin and copper mineralisation are confined to the lode beds within the overall lode, with trace (less than 0.1%) mineralisation only in the included chert bands. Further analyses (Reference No. 2) indicate that the chert bands comprise approximately 30% by volume of the ore-bodies.

These analyses were particularly rewarding, not only from the geological point of view. They indicate that in future sampling operations, the chert bands included in the overall lode need not be sampled. Particularly important during the evaluation stage of the whole operation was the fact that approximately 30% of an ore body consists of chert material barren of metal values. This factor was important during mineral dressing design, and the mill flow-sheet has been designed to include a heavy-media separator to reject the chert at an early stage.

8. CONCLUSIONS

The conclusions reached as a result of the exploratory underground development programme at Cleveland Mine, Tasmania, may be summarised as follows:-

(1)   The complex geological structure was more fully appreciated, thus allowing geological interpretations to be completed with a high degree of confidence.

(2)   The relationships of assays from samples of various sizes was appreciated:

(a)   the mean tin and copper grades and values for groove, AXT split core and channel samples showed small percentage differences from bulk sample results which due to their large size, were employed in the capacity of referee;

(b)   at the 99. 5 per cent confidence level, the test for assumption of equal means was accepted for the mean tin and copper grades and values for the 4 sizes of sample taken;

(c)   the direct comparison, by regression analysis, of tin and copper assays for groove and channel samples cut from the same lode band locations indicates a good degree of correlation throughout the range of values;

(d)   the analysis of tin and copper assays from lode and chert bands for groove, AXT split-core and channel samples indicates that the metal values are confined to the lode bands, with only trace (less than 0. 10%) amounts in the chert bands.

These results indicate that ore-reserve estimates based largely on the results of assays from AXT split diamond drill core would be reliable within close practical limits, and the application of any sort of correction factor to such estimates would not be required for the ground tested.

(3)   The value of the results of an involved programme of evaluation depends enormously on the quality of the data obtained. The strict control and supervision of all aspects of sampling and sample preparation, and the practical steps taken to ensure that a minimum number of errors were introduced into the samples from these sources, contributed in large measure to the success of the programme.

9. ACKNOWLEDGMENTS

10. REFERENCES

1. Cox, R., Jan. 1965   "Report of the Resident Geologist upon Completion of Exploratory Underground Development, R. L. 1300 ft. Level", A. T. D. P. (C. D. P.) unpublished report.

2. Cox, R., 13-9-1965   "Report on the Geological Nature of Hall's Lode, Lens "A" at Cleveland Mine, Tasmania", A. T. D. P. (C. D. P.) unpublished report.

3. Cox, R., 30-11-1965   "Report Upon Completion of the 1300ft. R. L. "Qa" Adit Level Exploration Programme, Cleveland Mine, Tasmania", A. T. D. P. (C. D. P.) unpublished report.

4. Cox, R., 30-11-65   "Report on Sample Data Analysis for Hall's Lode, Lens "A", etc. ", A. T. D. P. (C. D. P.) unpublished report.

5. Cox, R., 1-3-1966   "Report of the Resident Geologist Upon Completion of Diamond Drilling Programme, Cleveland Mine, Tasmania", A. T. D. P. (C. D. P.) unpublished report.

6. Gilberthorpe, N. A., July 1965   "Bulk Sampling Operation, Exploratory Underground Development, R. L. 1300 ft. Level", A. T. D. P. (C. D. P.) unpublished report.

7. Hazen, Scott W., Feb. 1958   "A Comparative Study of Statistical Analysis and Other Methods of Computing Ore Reserves, Utilizing Analytical Data from Maggie Canyon, Artillery Mountains Region, Mohave County, Ariz. ", U. S. Bureau of Mines R. I. 5375

8. Mason, A. A. C., Nov. 1963   "Outline of Proposed Exploratory Development and Ore Testing Programme", A. T. D. P. (C. D. P.) unpublished report.

TABLE NO. 1

RELATIVE SIZES OF 4 TYPES OF SAMPLE TAKEN DURING COMPARATIVE SAMPLING PROGRAMME, 1300 ft. R. L. "Qa" ADIT LEVEL

Type of Sample	Volume of Sample per foot of cut	Weight* of Sample per foot of cut	Volume of Sample per foot ÷ Volume of Bulk Sample per foot
Groove	6 cu. in	0. 7 lb.	1/2880
AXT split core	7. 7 cu. in	0. 9 lb.	1/2240
Channel	144 cu. in	17 lb.	1/120
Bulk	17, 280 cu. in (= 10 cu. ft)	2, 000 lb.	1

*   Average density of ore = 11.0 cu. ft. per ton.

TABLE NO. 2

COMPARATIVE SAMPLING DATA - OVERALL LODE VALUE

HALLS LODE, LENS "A" ON 1300 ft. R. L. "Qa" ADIT LEVEL

	GROOVE						CHANNEL					
Cross-section	LODE VALUE (Product W x G)						LODE VALUE (Product W x G)					
	S. W. wall		N. E. wall		Av. both walls		S. W. wall		N. E. wall		Av. both walls	
	Sn	Cu	Sn	Cu	Sn	Cu	Sn	Cu	Sn	Cu	Sn	Cu
V	190.86	84.82	92.13	93.15	141.50	88.99	292.70	102.76	107.18	100.91	199.94	101.84
T	383.73	115.79	383.73* 115.79* Not taken		383.73	115.79	342.85	112.93	342.85* 112.93* Not taken		342.85	112.93
R	480.46	145.08	390.61	118.08	435.54	131.58	507.50	149.92	473.18	142.64	490.34	146.28
Qa	248.50	125.60	337.50	163.70	293.00	144.65	285.94* 179.70* Not taken		285.94	179.70	285.94	179.70
P	430.62	189.60	410.74	222.69	420.68	206.15	455.26	188.85	371.44	170.00	413.35	179.43
N	528.30	158.06	494.76	123.82	511.53	140.94	541.02	148.04	503.68	146.31	522.35	147.18
L	50.36	46.04	69.52	16.13	59.94	31.09	71.32	46.38	99.72	55.52	85.52	50.95
K	473.35	107.08	379.49	169.78	426.42	138.43	585.36	153.08	292.72	153.42	439.04	153.25
Total	2786.18	972.07	2558.48	1023.14	2672.34	997.62	3081.95	1081.66	2476.71	1061.43	2779.33	1071.56
Mean	348.27	121.51	319.81	127.89	334.08	124.70	385.24	135.21	309.59	132.68	347.42	133.95
Total Width	3387 in		3312 in		3349½ in		3405 in		3312 in		3358½ in	
Av. Width	423½ in		414 in		418½ in		425½ in		414 in		420 in	
Wt. Mean Assay	0.82%Sn	0.29%Cu	0.77%Sn	0.31%Cu	0.80%Sn	0.30%Cu	0.91%Sn	0.32%Cu	0.75%Sn	0.32%Cu	0.83%Sn	0.32%Cu

*   Samples not taken: values from other wall are employed.

	SPLIT CORE		BULK	
	LODE VALUE		LODE VALUE	
	Sn	Cu	Sn	Cu
V	382.55	139.80	285.60	128.52
T	301.84	153.86	363.54	109.56
R	453.91	140.83	461.70	159.60
Qa	249.35	129.19	286.20	178.08
P	396.00	180.70	503.04	212.22
N	506.01	158.76	512.46	135.78
L	137.42	47.06	109.44	41.76
K	382.84	129.76	349.32	115.02
Total	2809.92	1079.96	2871.30	1080.54
Mean	351.24	135.00	358.91	135.07
Total Width	3276 in		3501 in	
Av. Width	409½ in		438 in	
Wt. Mean Assay	0.86%Sn	0.33%Cu	0.81%Sn	0.30%Cu

TABLE NO. 3

COMPARATIVE SAMPLING DATA - OVERALL LODE GRADE
HALLS LODE, LENS "A" ON 1300 ft. R. L. "Qa" ADIT LEVEL

A   GROOVE

Cross-section	S. W. wall			N. E. wall			Av. both walls		
	length (in.)	%Sn	%Cu	length (in.)	%Sn	%Cu	length (in.)	%Sn	%Cu
V	717	0.27	0.12	595	0.15	0.16	656	0.22	0.14
T	485	0.79	0.24	485	0.79* not	0.24* taken	485	0.79	0.24
R	571	0.84	0.25	544	0.72	0.22	557½	0.78	0.24
Qa	283	0.88	0.44	301	1.12	0.54	292	1.00	0.50
P	409	1.05	0.46	408	1.01	0.55	408½	1.03	0.50
N	433	1.22	0.37	431	1.15	0.29	432	1.18	0.33
L	140	0.36	0.33	145	0.48	0.11	142½	0.42	0.22
K	349	1.36	0.31	403	0.94	0.42	376	1.13	0.37
Total	3387	-	-	3312	-	-	3349½	-	-
Average	423½	-	-	414	-	-	418½	-	-
Wt. Av.	-	0.82 %Sn	0.29 %Cu	-	0.77 %Sn	0.31 %Cu	-	0.80 %Sn	0.30 %Cu

B   CHANNEL

Cross-section	S. W. wall			N. E. wall			Av. both walls		
	length (in.)	%Sn	%Cu	length (in.)	%Sn	%Cu	length (in.)	%Sn	%Cu
V	717	0.41	0.14	595	0.18	0.17	656	0.30	0.16
T	485	0.71	0.23	485	0.71* not	0.23* taken	485	0.71	0.23
R	571	0.89	0.26	544	0.87	0.26	557½	0.88	0.26
Qa	301	0.95* not	0.60* taken	301	0.95	0.60	301	0.95	0.60
P	409	1.11	0.46	408	0.91	0.42	408½	1.01	0.44
N	433	1.25	0.34	431	1.17	0.34	432	1.21	0.34
L	140	0.51	0.33	145	0.69	0.38	142½	0.60	0.36
K	349	1.68	0.44	403	0.73	0.38	376	1.17	0.41
Total	3405	-	-	3312	-	-	3358½	-	-
Average	425½	-	-	414	-	-	420	-	-
Wt. Av.	-	0.91 %Sn	0.32 %Cu	-	0.75 %Sn	0.32 %Cu	-	0.83 %Sn	0.32 %Cu

*   Samples not taken: values from other wall employed.

TABLE NO. 3

Continued

Cross-	SPLIT CORE			BULK		
section	length(in)	%Sn	%Cu	length(in)	%Sn	%Cu
V	702	0.54	0.20	714	0.40	0.18
T	511	0.59	0.30	498	0.73	0.22
R	584	0.78	0.24	570	0.81	0.28
Qa	180	1.39	0.72	318	0.90	0.56
P	406	0.98	0.45	393	1.28	0.54
N	421	1.20	0.38	438	1.17	0.31
L	133	1.03	0.35	144	0.76	0.29
K	339	1.34	0.47	426	0.82	0.27
Totals	3276	-	-	3501	-	-
Averages	409½	-	-	437½	-	-
Weight Averages	-	0.86%Sn	0.33%Cu	-	0.81%Sn	0.30%Cu

TABLE NO. 4

COMPARATIVE SAMPLING DATA - OVERALL LODE GRADE
AND VALUE, HALLS LODE, LENS "A" ON 1300 ft. R. L. "Qa"
ADIT LEVEL: ANALYSIS OF WEIGHTED MEANS FOR
4 TYPES OF SAMPLE FROM 8 CROSS-SECTIONS

Types of Samples	Weighted Mean Grade %Sn	Percent difference from bulk sample grade	Weighted Mean Grade %Cu	Percent difference from bulk sample grade	Weighted Mean Value Sn	Percent difference from bulk sample value	Weighted Mean Value Cu	Percent difference from bulk sample value
Bulk	0.81	-	0.30	-	358.91	-	135.07	-
DDH S/C	0.86	+ 6.1	0.33	+10.0	351.24	- 2.1	135.00	- 0.1
Groove SW	0.82	+ 1.2	0.29	- 3.3	348.27	- 3.0	121.51	-10.0
Groove NE	0.77	- 4.9	0.31	+ 3.3	319.81	-10.9	127.89	- 5.3
Groove Av. both	0.80	- 1.2	0.30	Zero	334.08	- 6.9	124.70	- 7.7
Channel SW	0.91	+12.3	0.32	+ 6.7	385.24	+ 7.3	135.21	+ 0.1
Channel NE	0.75	- 7.4	0.32	+ 6.7	309.59	-13.7	132.68	- 1.8
Channel Av. both	0.83	+ 2.5	0.32	+ 6.7	347.42	- 3.2	133.95	- 0.8

TABLE NO. 5

COMPARATIVE SAMPLING DATA - OVERALL LODE VALUE,
HALLS LODE, LENS "A" ON 1300 ft. R. L. "Qa" ADIT LEVEL:
TEST FOR ASSUMPTION THAT SEVERAL SAMPLES HAVE
EQUAL MEANS

Cross-Section	BULK		GROOVE		CHANNEL		SPLIT-CORE	
	Sn value	(Sn value)^2	Sn value	(Sn value)^2	Sn value	(Sn value)^2	Sn value	(Sn value)^2
V	286	81,796	142	20,164	200	40,000	383	146,689
T	364	132,496	384	147,456	343	117,649	302	91,204
R	462	213,444	436	190,096	490	240,100	454	206,116
Qa	286	81,796	293	85,849	286	81,796	249	62,001
P	503	253,009	421	177,241	413	170,569	396	156,816
N	512	262,144	512	262,144	522	272,484	506	256,036
L	109	11,881	60	3,600	86	7,396	137	18,769
K	349	121,801	426	181,476	439	192,791	383	146,689
Totals	2871	1158,367	2674	1068,026	2779	1122,785	2810	1084,320

Cross-Section	Cu value	(Cu value)^2	Cu value	(Cu value)^2	Cu value	(Cu value)^2	Cu value	(Cu value)^2
V	129	16,641	89	7,921	102	10,404	140	19,600
T	110	12,100	116	13,456	113	12,769	154	23,716
R	160	25,600	132	17,424	146	21,316	141	19,881
Qa	178	31,684	145	21,025	180	32,400	129	16,641
P	212	44,944	206	42,436	179	32,041	181	32,761
N	136	18,496	141	19,881	147	21,609	159	25,281
L	42	1,764	31	961	51	2,601	47	2,209
K	115	13,225	138	19,044	153	23,409	130	16,900
Totals	1082	164,454	998	142,148	1071	156,549	1081	156,989

TABLE NO. 6

COMPARATIVE SAMPLING DATA - OVERALL LODE GRADE,
HALLS LODE, LENS "A" ON 1300 ft. R. L. "Qa" ADIT LEVEL:
TEST FOR ASSUMPTION THAT SEVERAL SAMPLES HAVE
EQUAL MEANS

Cross-Section	BULK		GROOVE		CHANNEL		SPLIT-CORE	
	Sn grade	(Sn grade)^2	Sn grade	(Sn grade)^2	Sn grade	(Sn grade)^2	Sn grade	(Sn grade)^2
V	0.22	0.0484	0.30	0.0900	0.54	0.2916	0.40	0.1600
T	0.79	0.6241	0.71	0.5041	0.59	0.3481	0.73	0.5329
R	0.78	0.6084	0.88	0.7744	0.78	0.6084	0.81	0.6561
Qa	1.00	1.0000	0.95	0.9025	1.39	1.9321	0.90	0.8100
P	1.03	1.0609	1.01	1.0201	0.98	0.9604	1.28	1.6384
N	1.18	1.3924	1.21	1.4641	1.20	1.4400	1.17	1.3689
L	0.42	0.1764	0.60	0.3600	1.03	1.0609	0.76	0.5776
K	1.13	1.2769	1.17	1.3689	1.34	1.7956	0.82	0.6724
Totals	6.55	6.1875	6.83	6.4841	7.85	8.4371	6.87	6.4163

	Cu grade	(Cu grade)^2	Cu grade	(Cu grade)^2	Cu grade	(Cu grade)^2	Cu grade	(Cu grade)^2
V	0.14	0.0196	0.16	0.0256	0.20	0.0400	0.18	0.0324
T	0.24	0.0576	0.23	0.0529	0.30	0.0900	0.22	0.0484
R	0.24	0.0576	0.26	0.0676	0.24	0.0576	0.28	0.0784
Qa	0.50	0.2500	0.60	0.3600	0.72	0.5184	0.56	0.3136
P	0.50	0.2500	0.44	0.1936	0.45	0.2025	0.54	0.2916
N	0.33	0.1089	0.34	0.1156	0.38	0.1444	0.31	0.0961
L	0.22	0.0484	0.36	0.1296	0.35	0.1225	0.29	0.0841
K	0.37	0.1369	0.41	0.1681	0.47	0.2209	0.27	0.0729
Totals	2.54	0.9290	2.80	1.1130	3.11	1.3963	2.65	1.0175

179031

TABLE NO. 7

COMPARATIVE SAMPLING DATA - OVERALL LODE
HALLS LODE, LENS "A", 8 CROSS-SECTIONS ON 1300 ft. R. L.
"Qa" ADIT LEVEL
TEST FOR ASSUMPTION THAT THE 4 SAMPLE SIZES HAVE
EQUAL MEANS

	TIN VALUES				COPPER VALUES			
	Sum of Squares	Degrees of freedom	Mean square	F-ratio	Sum of Squares	Degrees of freedom	Mean square	F-ratio
Sm^2	2547	3	849	$F=\dfrac{849}{1989}=0.43$	610	3	203	$F=\dfrac{203}{2137}=0.10$
Sp^2	557015	28	1989	$F_{0.995}(3-28)$ $=5.32$	59,848	28	2137	$F_{0.995}(3-28)$ $=5.32$
Total	559562	31			60,458	31		

	TIN GRADES				COPPER GRADES			
Sm^2	0.1210	3	0.0403	$F=\dfrac{0.0403}{0.0975}$ $=0.41$	0.0230	3	0.0077	$F=\dfrac{0.0077}{0.0208}$ $=0.37$
Sp^2	2.7287	28	0.0975	$F_{0.995}(3-28)$ $=5.32$	0.5825	28	0.0208	$F_{0.995}(3-28)$ $=5.32$
Total	2.8497	31			0.6055	31		

APPENDIX A

COMPUTATIONS FOR TEST FOR ASSUMPTION THAT THE FOUR SAMPLE SIZES HAVE EQUAL MEANS: GRADE (TIN AND COPPER)

1. **TIN GRADE**

   (a) $S_p^2 = 27.5250 - \left(\dfrac{(6.55)^2}{8} + \dfrac{(6.83)^2}{8} + \dfrac{(7.85)^2}{8} + \dfrac{(6.87)^2}{8} \right)$

   $= 27.5250 - (5.3628 + 5.8311 + 7.7028 + 5.8996)$

   $= 27.5250 - 24.7963 = 2.7287$

   (b) $S_m^2 = 24.7963 - \dfrac{(28.10)^2}{32} = 24.7963 - 24.6753 = 0.1210$

   (c) Total Variation $= 27.5250 - 24.6753 = 2.8497$

   (d)

	Sum of Squares	Degrees of Freedom	Mean Square	F-ratio
S_m^2	0.1210	3	0.0403	$F = \dfrac{0.0403}{0.0975} = 0.41$
S_p^2	2.7287	28	0.0975	$F_{0.995}(3\text{-}28) = 5.32$
Totals	2.8497	31		

2. **COPPER GRADE**

   (a) $S_p^2 = 4.4558 - (0.8065 + 0.9800 + 1.2090 + 0.8778)$

   $= 4.4558 - 3.8733 = 0.5825$

   (b) $S_m^2 = 3.8733 - \dfrac{(11.10)^2}{32} = 3.8733 - 3.8503 = 0.0230$

   (c) Total Variation $= 4.4558 - 3.8503 = 0.6055$

   (d)

	Sum of Squares	Degrees of Freedom	Mean Square	F-ratio
S_m^2	0.0230	3	0.0077	$F = \dfrac{0.0077}{0.0208} = 0.37$
S_p^2	0.5825	28	0.0208	$F_{0.995}(3\text{-}28) = 5.32$
Totals	0.6055	31		

APPENDIX B

DETERMINATION OF VALUES USED IN COMPUTING THE ESTIMATING EQUATION, STANDARD ERROR OF ESTIMATE AND COEFFICIENT OF CORRELATION FOR GROOVE AND CHANNEL TIN GRADE FROM 128 LODE SAMPLE LOCATIONS

Sample No.	Tin Grade (% Sn) of Groove Sample	Tin Grade (% Sn) of Channel Sample	Product					
	Y	X	XY	X^2	Y^2	Y_c	$Y-Y_c$	$(Y-Y_c)^2$
Q29	0.36	0.78	0.281	0.608	0.130	0.7588	0.3988	0.1590
Q31	0.70	0.69	0.483	0.476	0.490	0.6724	0.0276	0.0008
Q33	4.03	1.30	5.239	1.690	16.241	1.2580	2.7720	7.6840
Q35	1.02	1.16	1.183	1.346	1.040	1.1236	0.1036	0.0107
Q36	1.03	1.12	1.154	1.254	1.061	1.0852	0.0552	0.0030
Q37	0.80	1.08	0.864	1.166	0.640	1.0468	0.2468	0.0609
Q38	2.15	1.53	3.290	2.341	4.623	1.4788	0.6712	0.4505
Q39	1.17	1.18	1.381	1.392	1.369	1.1428	0.0272	0.0007
N19	0.64	1.12	0.717	1.254	0.410	1.0852	0.4452	0.1982
N21	1.59	1.32	2.099	1.742	2.528	1.2772	0.3128	0.1291
N22	1.25	1.13	1.413	1.277	1.563	1.0948	0.1552	0.0241
N24	1.00	1.83	1.830	3.349	1.000	1.7668	0.7668	0.5880
N26	1.48	1.47	2.176	2.161	2.190	1.4212	0.0588	0.0035
N28	Tr	Tr	Nil	Nil	Nil	0.0100	0.0100	0.0001
N30	0.93	1.01	0.939	1.020	0.865	0.9796	0.0496	0.0025
N31	1.18	2.01	2.372	4.040	1.392	1.9396	0.7596	0.5770
N33	1.74	1.70	2.958	2.890	3.028	1.6420	0.0980	0.0096
N35	2.50	1.98	4.950	3.920	6.250	1.9108	0.5892	0.3472
N36	1.73	1.81	3.131	3.276	2.993	1.7476	0.0176	0.0003

APPENDIX B

Continued

	Y	X	XY	X^2	Y^2	Y_c	$Y-Y_c$	$(Y-Y_c)^2$
N 2	1.61	1.70	2.737	2.890	2.592	1.6420	0.0320	0.0010
N 4	1.26	1.13	1.424	1.277	1.588	1.0948	0.1652	0.0273
N 5	1.43	1.19	1.702	1.416	2.045	1.1524	0.2776	0.0771
N 7	1.81	1.74	3.149	3.028	3.276	1.6804	0.1296	0.0167
N 8	1.64	1.38	2.263	1.904	2.690	1.3348	0.3052	0.0931
N 9	1.51	1.58	2.386	2.496	2.280	1.5268	0.0168	0.0003
N11	1.36	1.35	1.836	1.823	1.850	1.3060	0.0540	0.0029
N13	1.71	1.63	2.787	2.657	2.924	1.5748	0.1352	0.0183
N15	1.41	1.76	2.482	3.098	1.988	1.6996	0.2896	0.0839
N16	0.69	0.71	0.490	0.504	0.476	0.6916	0.0016	Nil
P 1	0.55	0.54	0.297	0.292	0.303	0.5284	0.0216	0.0005
P 2	1.69	1.63	2.755	2.657	2.856	1.5748	0.1152	0.0133
P 3	1.83	1.55	2.837	2.403	3.349	1.4980	0.0520	0.0027
P 4	1.30	1.05	1.365	1.103	1.690	1.0180	0.2820	0.0795
P 6	0.96	0.77	0.739	0.593	0.922	0.7492	0.2108	0.0444
P 8	1.71	1.07	1.830	1.145	2.924	1.0372	0.6728	0.4527
P10	0.57	0.80	0.456	0.640	0.325	0.7780	0.2080	0.0433
P12	1.43	1.39	1.988	1.932	2.045	1.3444	0.0856	0.0073
P14	1.60	1.44	2.304	2.074	2.560	1.3924	0.2076	0.0431
P17	1.07	0.59	0.631	0.348	1.145	0.5764	0.4936	0.2436
P19	0.50	0.76	0.380	0.578	0.250	0.7396	0.2396	0.0574
P20	0.95	1.14	1.083	1.300	0.903	1.1044	0.1544	0.0238
P21	2.02	2.00	4.020	4.000	4.080	1.9300	0.0900	0.0081
P23	1.56	1.83	2.855	3.349	2.434	1.7668	0.2068	0.0428
P25	1.56	1.77	2.761	3.133	2.434	1.7092	0.1492	0.0223
P26	1.64	1.87	3.067	3.497	2.690	1.8052	0.1652	0.0273
P27	1.61	1.44	2.318	2.074	2.592	1.3924	0.2176	0.0473
P29	0.37	0.52	0.192	0.270	0.137	0.5092	0.1392	0.0194

APPENDIX B

Continued

	Y	X	XY	X^2	Y^2	Y_c	$Y-Y_c$	$(Y-Y_c)^2$
R56	1.84	1.53	2.815	2.341	3.386	1.4788	0.3612	0.1305
R55	1.05	1.45	1.523	2.103	1.103	1.4020	0.3520	0.1239
R53	0.70	1.16	0.812	1.346	0.490	1.1236	0.4236	0.1794
R51	1.51	1.55	2.341	2.403	2.280	1.4980	0.0120	0.0001
R49	0.90	0.84	0.756	0.706	0.810	0.8164	0.0836	0.0070
R48	0.71	1.18	0.838	1.392	0.504	1.1428	0.4328	0.1873
R46	1.76	1.91	3.362	3.648	3.098	1.8436	0.0836	0.0070
R44	0.76	1.03	0.783	1.061	0.578	0.9988	0.2388	0.0570
R42	1.00	0.98	0.980	0.960	1.000	0.9508	0.0492	0.0024
R40	0.98	0.55	0.539	0.303	0.960	0.5380	0.4420	0.1954
R38	1.35	1.07	1.445	1.145	1.823	1.0372	0.3128	0.0978
R36	0.95	1.30	1.235	1.690	0.903	1.2580	0.3080	0.0949
R 2	0.21	1.63	0.342	2.657	0.044	1.4933	1.2833	1.6469
R 4	1.28	1.91	2.445	3.648	1.638	1.8436	0.5636	0.3176
R 6	0.83	0.92	0.764	0.846	0.689	0.8932	0.0632	0.0040
R 8	1.16	1.23	1.427	1.513	1.346	1.1908	0.0308	0.0009
R 9	0.74	0.67	0.496	0.449	0.548	0.6532	0.0868	0.0075
R11	1.13	1.33	1.503	1.769	1.277	1.2868	0.1568	0.0246
R13	1.80	2.13	3.834	4.537	3.240	2.0548	0.2548	0.0649
R15	1.52	1.20	1.824	1.440	2.310	1.1620	0.3580	0.1282
R17	1.01	0.44	0.444	0.194	1.020	0.4324	0.5776	0.3336
R18	1.09	1.37	1.493	1.877	1.188	1.3252	0.2352	0.0553
R19	1.46	1.42	2.073	2.016	2.132	1.3732	0.0868	0.0075
R22	1.01	0.94	0.949	0.884	1.020	0.9124	0.0976	0.0095
R24	1.32	1.47	1.940	2.161	1.742	1.4212	0.1012	0.0102
L 1	0.45	0.52	0.234	0.270	0.203	0.5092	0.0592	0.0035
L 2	0.75	1.14	0.855	1.300	0.563	1.1044	0.3544	0.1256
L 3	0.28	0.53	0.148	0.281	0.078	0.5188	0.2388	0.0570
L 5	1.17	1.29	1.509	1.664	1.369	1.2484	0.0784	0.0061

034

APPENDIX B

Continued

	Y	X̄	XY	X²	Y²	Y_c	Y-Y_c	(Y-Y_c)²
L 6	0.24	0.80	0.192	0.640	0.058	0.7780	0.5380	0.2894
L 7	0.14	0.65	0.091	0.423	0.020	0.6340	0.4940	0.2440
L 8	0.54	0.30	0.162	0.090	0.292	0.2980	0.2420	0.0586
L10	0.97	0.63	0.611	0.397	0.941	0.6148	0.3552	0.1262
L11	0.26	0.14	0.036	0.020	0.068	0.1444	0.1156	0.0134
T24	4.65	2.54	11.811	6.452	21.623	2.4484	2.2016	4.8470
T23	1.70	1.20	2.040	1.440	2.890	1.1620	0.5380	0.2894
T21	0.98	0.88	0.862	0.774	0.960	0.8548	0.1252	0.0157
T19	1.08	1.52	1.642	2.310	1.166	1.4692	0.3892	0.1515
T16	1.37	1.47	2.014	2.161	1.877	1.4212	0.0512	0.0026
T14	0.86	1.05	0.903	1.103	0.740	1.0180	0.1580	0.0250
T12	0.13	0.26	0.034	0.068	0.017	0.2596	0.1296	0.0168
T11	0.96	0.70	0.672	0.490	0.922	0.6820	0.2780	0.0773
T10	0.29	0.47	0.136	0.221	0.084	0.4612	0.1712	0.0293
T 9	0.62	0.73	0.453	0.533	0.384	0.7108	0.0908	0.0082
K43	1.38	1.16	1.601	1.346	1.904	1.1236	0.2564	0.0657
K42	0.39	0.71	0.277	0.504	0.152	0.6916	0.3016	0.0910
K40	0.14	0.48	0.067	0.230	0.020	0.4708	0.3308	0.1094
K38	0.36	0.97	0.349	0.941	0.130	0.9412	0.5812	0.3378
K36	0.54	0.75	0.405	0.563	0.292	0.7300	0.1900	0.0361
K34	5.71	1.43	8.165	2.045	32.604	1.3828	4.3272	18.7247
K33	0.44	0.97	0.427	0.941	0.194	0.9412	0.5012	0.2512
K31	0.93	0.80	0.744	0.640	0.865	0.7780	0.1520	0.0231
K30	0.68	0.74	0.503	0.548	0.462	0.7204	0.0404	0.0016
K19	4.75	4.75	22.563	22.563	22.563	4.5700	0.1800	0.0324
K18	1.70	2.77	4.709	7.673	2.890	2.6692	0.9692	0.9393
K17	1.77	2.70	4.779	7.270	3.133	2.6020	0.8320	0.6922
K16	0.41	0.37	0.152	0.137	0.168	0.3652	0.0448	0.0020
K15	1.31	1.90	2.489	3.610	1.716	1.8340	0.5240	0.2746
K14	0.33	0.54	0.178	0.292	0.109	0.5284	0.1984	0.0394
K13	1.61	2.51	4.041	6.300	2.592	2.4196	0.8096	0.6555

APPENDIX B

Continued

	Y	X	XY	X^2	Y^2	Y_c	$Y-Y_c$	$(Y-Y_c)^2$
K12	0.44	0.68	0.299	0.462	0.194	0.6628	0.2228	0.0496
K11	2.02	0.95	1.919	0.903	4.080	0.9220	1.0980	1.2056
V19	0.87	0.87	0.757	0.757	0.757	0.8452	0.0248	0.0006
V23	0.21	0.41	0.086	0.168	0.044	0.4036	0.1936	0.0375
V25	Tr	0.15	Nil	0.023	Nil	0.1540	0.1540	0.0237
V27	Tr	Nil	Nil	Nil	Nil	0.0100	0.0100	0.0001
V29	1.07	1.21	1.295	1.464	1.145	1.1716	0.1016	0.0103
V31	0.51	1.03	0.525	1.061	0.260	0.9988	0.4888	0.2389
V33	0.22	0.92	0.202	0.846	0.048	0.8932	0.6732	0.4532
V35	0.41	0.87	0.357	0.757	0.168	0.8452	0.4352	0.1894
V37	0.62	0.58	0.360	0.336	0.384	0.5668	0.0532	0.0028
V38	0.27	1.25	0.338	1.563	0.073	1.2100	0.9400	0.8836
V 2	Tr	Tr	Nil	Nil	Nil	0.0100	0.0100	0.0001
V 4	Tr	0.12	Nil	0.014	Nil	0.1252	0.1252	0.0157
V 5	0.71	0.41	0.291	0.168	0.504	0.4036	0.3064	0.0939
V 7	0.21	0.24	0.050	0.058	0.044	0.2404	0.0304	0.0009
V 8	0.22	0.32	0.070	0.102	0.048	0.3172	0.0972	0.0094
V11	0.49	0.61	0.299	0.372	0.240	0.5956	0.1056	0.0112
V13	0.35	0.52	0.182	0.270	0.123	0.5092	0.1592	0.0253
V15	0.47	0.56	0.263	0.314	0.221	0.5476	0.0776	0.0060
V16	Tr	Tr	Nil	Nil	Nil	0.0100	0.0100	0.0001
Totals 128	139.70	143.83	209.704	216.650	250.672			47.3625

APPENDIX C

COMPUTATIONS OF THE ESTIMATING EQUATION, THE STANDARD ERROR OF ESTIMATE AND THE COEFFICIENT OF CORRELATION BY THE METHOD OF LEAST SQUARES

1. The Estimating Equation

    (a) $\Sigma Y = Na + b\Sigma X$

    (b) $\Sigma XY = a\Sigma X + b\Sigma X^2$

    These two normal equations are solved simultaneously, using data from Appendix "B":-

    (a) $139.70 = 128a + 143.83b$

    (b) $209.704 = 143.83a + 216.650b$

    from which a = 0.01 and b = 0.96 . The Estimating Equation is therefore: $Y_c = 0.01 + 0.96 X$, where X units are percent Sn.

2. The Standard Error of Estimate, $(S_{Y.X})$

    $$S_{Y.X} = \sqrt{\frac{\Sigma(Y - Y_c)^2}{N}} = \sqrt{\frac{47.3625}{128}} = \sqrt{0.3700}$$

    $S_{Y.X} = 0.61$

3. The Coefficient of Correlation, (r)

    $$r = \frac{N.\Sigma XY - (\Sigma X)(\Sigma Y)}{\sqrt{\left[N.\Sigma X^2 - (\Sigma X)^2\right]\left[N.\Sigma Y^2 - (\Sigma Y)^2\right]}}$$

    $$= \frac{(128 \times 209.704) - (143.83 \times 139.70)}{\sqrt{\left[(128 \times 216.650) - (143.83)^2\right]\left[(128 \times 250.672) - (139.70)^2\right]}}$$

    $$= \frac{+674.9061}{\sqrt{885436.3259}} = \frac{+674.9061}{940.98} = +0.7172$$

4. The Coefficient of Determination, (r^2)

    $$r^2 = (0.7172)^2 = 0.514$$

    i.e. 51.4% of the total variation is explained by the Estimating Equation.

APPENDIX D

DETERMINATION OF VALUES USED IN COMPUTING THE ESTIMATING EQUATION, STANDARD ERROR OF ESTIMATE AND COEFFICIENT OF CORRELATION FOR GROOVE AND CHANNEL COPPER GRADE FROM 128 LODE SAMPLE LOCATIONS

Sample No.	Copper Grade (%Cu) of Groove sample	Copper Grade (%Cu) of Channel Sample	Pro-duct					
	Y	X	XY	X^2	Y^2	Y_c	$Y-Y_c$	$(Y-Y_c)^2$
Q29	1.19	1.95	2.321	3.803	1.416	1.5810	0.3910	0.1529
Q31	0.53	0.62	0.329	0.384	0.281	0.5436	0.0136	0.0002
Q33	0.68	0.33	0.224	0.109	0.462	0.3174	0.3626	0.1315
Q35	0.44	0.47	0.207	0.221	0.194	0.4266	0.0134	0.0002
Q36	0.78	0.72	0.562	0.518	0.608	0.6216	0.1584	0.0251
Q37	0.49	0.57	0.279	0.325	0.240	0.5046	0.0146	0.0002
Q38	0.73	0.97	0.708	0.941	0.533	0.8166	0.0866	0.0075
Q39	0.66	0.55	0.363	0.303	0.436	0.4890	0.1710	0.0292
N19	Tr	0.15	Nil	0.023	Nil	0.1770	0.1770	0.0313
N21	0.38	0.31	0.118	0.096	0.144	0.3018	0.0782	0.0061
N22	0.28	0.32	0.090	0.102	0.078	0.3096	0.0296	0.0009
N24	0.81	0.58	0.470	0.336	0.656	0.5124	0.2976	0.0886
N26	0.38	0.35	0.133	0.123	0.144	0.3330	0.0470	0.0022
N28	0.20	0.37	0.074	0.137	0.040	0.3486	0.1486	0.0221
N30	0.46	0.34	0.156	0.116	0.212	0.3252	0.1348	0.0182
N31	0.23	0.24	0.055	0.058	0.053	0.2472	0.0172	0.0003
N33	0.53	0.52	0.276	0.270	0.281	0.4656	0.0644	0.0041
N35	0.63	0.62	0.391	0.384	0.397	0.5436	0.0864	0.0075
N36	0.79	0.74	0.585	0.548	0.624	0.6372	0.1528	0.0233

APPENDIX D

Continued

	Y	X	XY	x^2	Y^2	Y_c	$Y-Y_c$	$(Y-Y_c)^2$
N 2	0.65	0.61	0.397	0.372	0.423	0.5358	0.1142	0.0130
N 4	0.72	0.77	0.554	0.593	0.518	0.6606	0.0594	0.0035
N 5	0.52	0.41	0.213	0.168	0.270	0.3798	0.1402	0.0197
N 7	Nil	0.34	Nil	0.116	Nil	0.3252	0.3252	0.1058
N 8	Nil	0.35	Nil	0.123	Nil	0.3330	0.3330	0.1109
N 9	0.28	0.35	0.098	0.123	0.078	0.3330	0.0530	0.0028
N11	0.41	0.45	0.185	0.203	0.168	0.4110	0.0010	0.0000
N13	0.47	0.40	0.188	0.160	0.221	0.3720	0.0980	0.0096
N15	0.27	0.27	0.073	0.073	0.073	0.2706	0.0006	0.0000
N16	0.15	0.22	0.033	0.048	0.023	0.2316	0.0816	0.0067
P 1	0.44	0.34	0.150	0.116	0.194	0.3252	0.1148	0.0132
P 2	0.50	0.46	0.230	0.212	0.250	0.4188	0.0812	0.0066
P 3	0.50	0.47	0.235	0.221	0.250	0.4266	0.0734	0.0054
P 4	0.77	0.60	0.462	0.360	0.593	0.5280	0.2420	0.0586
P 6	0.98	0.63	0.617	0.397	0.960	0.5514	0.4286	0.1837
P 8	0.60	0.50	0.300	0.250	0.360	0.4500	0.1500	0.0225
P10	0.65	0.56	0.364	0.314	0.423	0.4968	0.1532	0.0235
P12	0.75	0.64	0.480	0.410	0.563	0.5592	0.1908	0.0364
P14	0.78	0.48	0.374	0.230	0.608	0.4344	0.3456	0.1194
P17	0.97	0.48	0.466	0.230	0.941	0.4344	0.5356	0.2869
P19	0.57	0.96	0.547	0.922	0.325	0.8088	0.2388	0.0570
P20	0.55	0.51	0.281	0.260	0.303	0.4578	0.0922	0.0085
P21	0.42	0.48	0.202	0.230	0.176	0.4344	0.0144	0.0002
P23	0.69	0.62	0.428	0.384	0.476	0.5436	0.1464	0.0214
P25	0.61	0.49	0.299	0.240	0.372	0.4422	0.1678	0.0282
P26	0.39	0.54	0.211	0.292	0.152	0.4812	0.0912	0.0083
P27	0.25	0.39	0.098	0.152	0.063	0.3642	0.1142	0.0130
P29	0.55	0.46	0.253	0.212	0.303	0.4188	0.1312	0.0172

APPENDIX D

Continued

	Y	X	XY	X^2	Y^2	Y_c	$Y-Y_c$	$(Y-Y_c)^2$
R56	0.30	0.27	0.081	0.073	0.090	0.2706	0.0294	0.0009
R55	0.22	0.21	0.046	0.044	0.048	0.2238	0.0038	0.0000
R53	0.17	0.22	0.037	0.048	0.029	0.2316	0.0616	0.0038
R51	0.30	0.39	0.117	0.152	0.090	0.3642	0.0642	0.0041
R49	0.32	0.28	0.090	0.078	0.102	0.2784	0.0416	0.0017
R48	0.42	0.51	0.214	0.260	0.176	0.4578	0.0378	0.0014
R46	0.55	0.30	0.165	0.090	0.303	0.2940	0.2560	0.0655
R44	0.35	0.34	0.119	0.116	0.123	0.3252	0.0248	0.0006
R42	0.35	0.48	0.168	0.230	0.123	0.4344	0.0844	0.0071
R40	0.50	0.29	0.145	0.084	0.250	0.2862	0.2138	0.0457
R38	0.42	0.47	0.197	0.221	0.176	0.4266	0.0066	0.0000
R36	0.22	0.29	0.064	0.084	0.048	0.2862	0.0662	0.0044
R 2	0.35	0.41	0.144	0.168	0.123	0.3798	0.0298	0.0009
R 4	0.25	0.34	0.085	0.116	0.063	0.3252	0.0752	0.0057
R 6	0.15	0.15	0.023	0.023	0.023	0.1770	0.0270	0.0007
R 8	0.35	0.33	0.116	0.109	0.123	0.3330	0.0170	0.0003
R 9	0.25	0.34	0.085	0.116	0.063	0.3252	0.0752	0.0057
R11	0.13	0.12	0.016	0.014	0.017	0.1536	0.0236	0.0006
R13	0.53	0.41	0.217	0.168	0.281	0.3798	0.1502	0.0225
R15	0.15	0.41	0.062	0.168	0.023	0.3798	0.2298	0.0528
R17	Nil	0.14	Nil	0.020	Nil	0.1692	0.1692	0.0286
R18	0.35	0.55	0.193	0.303	0.123	0.4890	0.1390	0.0193
R19	0.38	0.54	0.205	0.292	0.144	0.4812	0.1012	0.0102
R22	0.40	0.36	0.144	0.130	0.160	0.3408	0.0592	0.0035
R24	0.35	0.57	0.200	0.325	0.123	0.5046	0.1546	0.0239
L 1	Tr	0.12	Nil	0.014	Nil	0.1536	0.1536	0.0236
L 2	0.19	0.75	0.143	0.563	0.036	0.6450	0.4550	0.2070
L 3	Tr	0.21	Nil	0.044	Nil	0.2238	0.2238	0.0501
L 5	0.70	1.28	0.896	1.638	0.490	1.0584	0.3584	0.1285

179042

APPENDIX D

Continued

	Y	X	XY	X^2	Y^2	Y_c	$Y-Y_c$	$(Y-Y_c)^2$
L 6	0.11	0.24	0.026	0.058	0.012	0.2472	0.1372	0.0188
L 7	0.32	0.34	0.109	0.116	0.102	0.3252	0.0052	0.0000
L 8	0.51	0.57	0.291	0.325	0.260	0.5046	0.0054	0.0000
L10	0.42	0.47	0.197	0.221	0.176	0.4266	0.0066	0.0000
L11	0.17	0.12	0.020	0.014	0.029	0.1536	0.0164	0.0003
T24	0.40	0.37	0.148	0.137	0.160	0.3486	0.0514	0.0026
T23	0.78	0.26	0.203	0.068	0.608	0.2628	0.5172	0.2675
T21	0.34	0.66	0.224	0.436	0.105	0.5748	0.2348	0.0551
T19	0.30	0.20	0.060	0.040	0.090	0.2160	0.0840	0.0071
T16	0.28	0.31	0.087	0.096	0.078	0.3018	0.0218	0.0005
T14	0.41	0.37	0.152	0.137	0.168	0.3486	0.0614	0.0038
T12	Tr	Tr	Nil	Nil	Nil	0.0600	0.0600	0.0036
T11	0.28	0.19	0.053	0.036	0.078	0.2082	0.0718	0.0052
T10	0.43	0.51	0.219	0.260	0.185	0.4578	0.0278	0.0008
T 9	0.33	0.39	0.129	0.152	0.109	0.3642	0.0342	0.0012
K43	1.48	0.81	1.199	0.656	2.190	0.6918	0.7882	0.6213
K42	0.39	0.64	0.250	0.410	0.152	0.5592	0.1692	0.0286
K40	1.54	1.35	2.079	1.823	2.372	1.1130	0.4270	0.1823
K38	0.44	0.48	0.211	0.230	0.194	0.4344	0.0056	0.0000
K36	0.46	0.48	0.221	0.230	0.212	0.4344	0.0256	0.0007
K34	0.14	0.10	0.014	0.010	0.020	0.1380	0.0020	0.0000
K33	0.34	0.38	0.129	0.144	0.116	0.3564	0.0164	0.0003
K31	0.25	0.19	0.048	0.036	0.063	0.2082	0.0418	0.0017
K30	0.39	0.43	0.168	0.185	0.152	0.3954	0.0054	0.0000
K19	0.19	0.56	0.106	0.314	0.036	0.4968	0.3068	0.0941
K18	0.61	0.41	0.250	0.168	0.372	0.3798	0.2302	0.0530
K17	0.19	0.39	0.074	0.152	0.036	0.3642	0.1742	0.0303
K16	0.16	0.17	0.027	0.029	0.026	0.1926	0.0326	0.0011
K15	0.68	0.91	0.619	0.828	0.462	0.7698	0.0898	0.0081

APPENDIX D

Continued

	Y	X	XY	x^2	Y^2	Y_c	$Y-Y_c$	$(Y-Y_c)^2$
K14	0.14	0.25	0.035	0.063	0.020	0.2550	0.1150	0.0132
K13	0.19	0.32	0.061	0.102	0.036	0.3096	0.1196	0.0143
K12	0.44	0.54	0.238	0.292	0.194	0.4812	0.0412	0.0017
K11	0.35	0.53	0.186	0.281	0.123	0.4734	0.1234	0.0152
V19	0.36	0.30	0.108	0.090	0.130	0.2940	0.0660	0.0044
V23	0.18	0.19	0.034	0.036	0.032	0.2082	0.0282	0.0008
V25	0.11	0.16	0.018	0.026	0.012	0.1848	0.0748	0.0056
V27	Tr	Tr	Nil	Nil	Nil	0.0600	0.0600	0.0036
V29	0.45	0.45	0.203	0.203	0.203	0.4110	0.0390	0.0015
V31	0.11	0.56	0.062	0.314	0.012	0.4968	0.3868	0.1496
V33	Tr	0.20	Nil	0.040	Nil	0.2160	0.2160	0.0467
V35	0.20	0.23	0.046	0.053	0.040	0.2394	0.0394	0.0016
V37	0.15	0.17	0.026	0.029	0.023	0.2096	0.0596	0.0036
V38	0.16	0.29	0.046	0.084	0.026	0.2862	0.1262	0.0159
V 2	0.23	0.22	0.051	0.048	0.053	0.2316	0.0016	0.0000
V 4	0.28	0.22	0.062	0.048	0.078	0.2316	0.0484	0.0023
V 5	0.42	0.27	0.113	0.073	0.176	0.2706	0.1494	0.0223
V 7	0.17	0.15	0.026	0.023	0.029	0.1770	0.0070	0.0000
V 8	Tr	0.11	Nil	0.012	Nil	0.1458	0.1458	0.0213
V11	0.23	0.35	0.081	0.123	0.053	0.3330	0.1030	0.0106
V13	0.27	0.30	0.081	0.090	0.073	0.2940	0.0240	0.0006
V15	0.27	0.37	0.100	0.137	0.073	0.3856	0.1156	0.0134
V16	0.30	0.35	0.105	0.123	0.090	0.3330	0.0330	0.0011
Totals 128	50.63	54.41	28.046	31.500	29.274			4.1898

APPENDIX E

COMPUTATIONS OF THE ESTIMATING EQUATION, THE STANDARD ERROR OF ESTIMATE AND THE COEFFICIENT OF CORRELATION BY THE METHOD OF LEAST SQUARES

1. The Estimating Equation

(a) $\sum Y = Na + b\sum X$

(b) $\sum XY = a\sum X + b\sum X^2$

These two normal equations are solved simultaneously, using data from Appendix D:-

(a) $50.63 = 128a + 54.41b$

(b) $28.046 = 54.41a + 31.500b$

from which $a = 0.06$ and $b = 0.78$. The Estimating Equation is therefore $Y_c = 0.06 + 0.78X$, where X units are percent Cu.

2. The Standard Error of Estimate, $(S_{Y'X})$

$$S_{Y'X} = \sqrt{\frac{\sum (Y-Y_c)^2}{N}} = \sqrt{\frac{4.1898}{128}} = \sqrt{0.03273}$$

$$S_{Y'X} = 0.18$$

3. The Coefficient of Correlation, (r)

$$r = \frac{N\sum XY - (\sum X)(\sum Y)}{\sqrt{\left|N.\sum X^2 - (\sum X)^2\right|\left|N\sum Y^2 - (\sum Y)^2\right|}}$$

$$= \frac{(128 \times 28.046) - (54.41 \times 50.63)}{\sqrt{\left|(128 \times 31.500) - (54.41)^2\right|\left|(128 \times 29.274) - (50.63)^2\right|}}$$

$$= \frac{+835.110}{\sqrt{1,268,452.92}} = \frac{+835.110}{1126.26} = +0.7415$$

4. The Coefficient of Determination, (r^2)

$$r^2 = (0.7415)^2 = 0.550$$

i.e. 55.0% of the total variation is explained by the Estimating Equation.

043

LIST OF CAPTIONS FOR FIGURES 1-6

CLEVELAND MINE

CROSS SECTION "Qa"

(LOOKING N.E.)

SCALE OF FEET

	Volcanics
	Lode Shales etc.
	Lode
	Sandstone
	Axial Plane Fault

AUGUST 1966

Figure 1   Cross section "Qa", looking North East,
Cleveland Mine.

R. Cox

3529

PLAN OF BACK OF X-CUT

·1 2 3 4 5 6 7 8 9 10 11 12 13

SCALE

5   0   5   10Ft.

5 cm
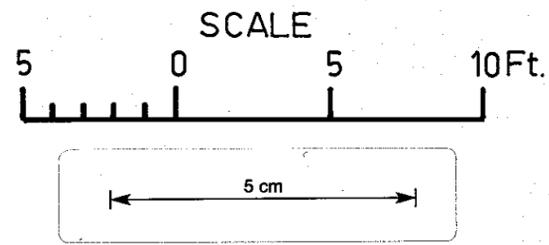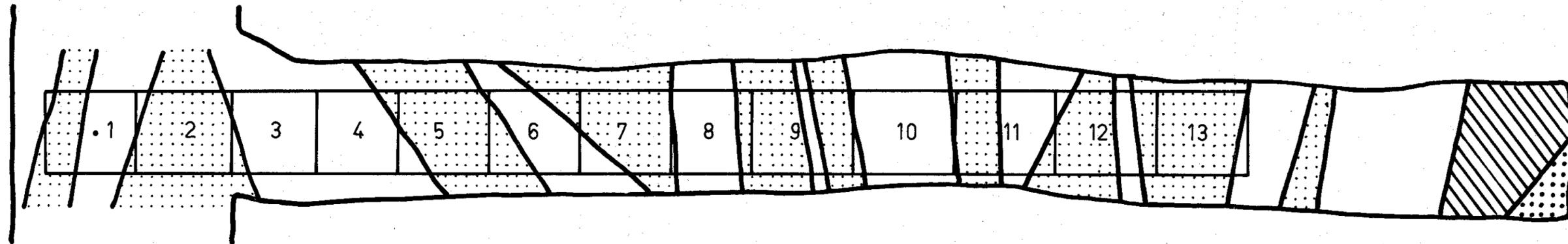
CLEVELAND   MINE   TASMANIA

"V" X-CUT 1300ft.R.L. ADIT LEVEL

SANDSTONE
SHALE
CHERT
LODE

12   BULK SAMPLE
     LOCATION No.

VIEW OF S.W. WALL LOOKING N.E.

Figure 2   Plan and section of "V" cross-cut through
           Hall's Lode, Lens "A", on 1300ft R.L.
           "Qa" adit level, Cleveland Mine.
                                    R. Cox

179047
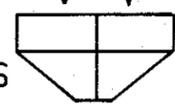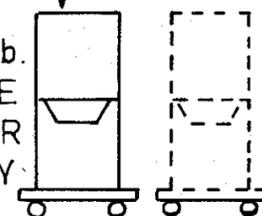
Figure 3   Flowsheet of bulk sample reduction plant,
            Cleveland Mine.

            R. Cox
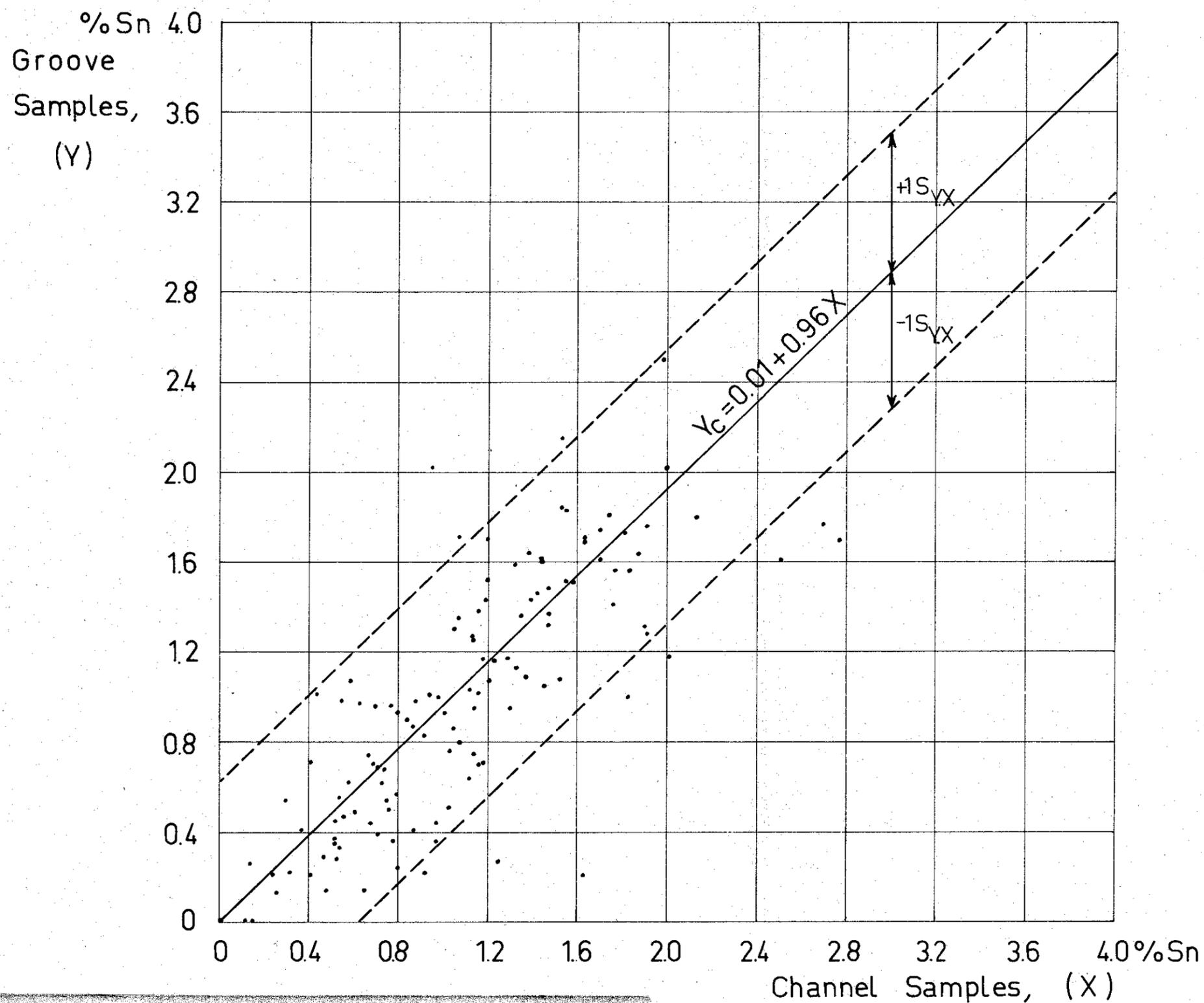
179048

Figure 4   Scatter diagram of tin grade from groove
and channel sample assays for 128 lode
sample locations: eight cross-cuts through
Hall's Lode, Lena "A", on 1300ft R.L.
"Qa" adit level, Cleveland Mine. (Four
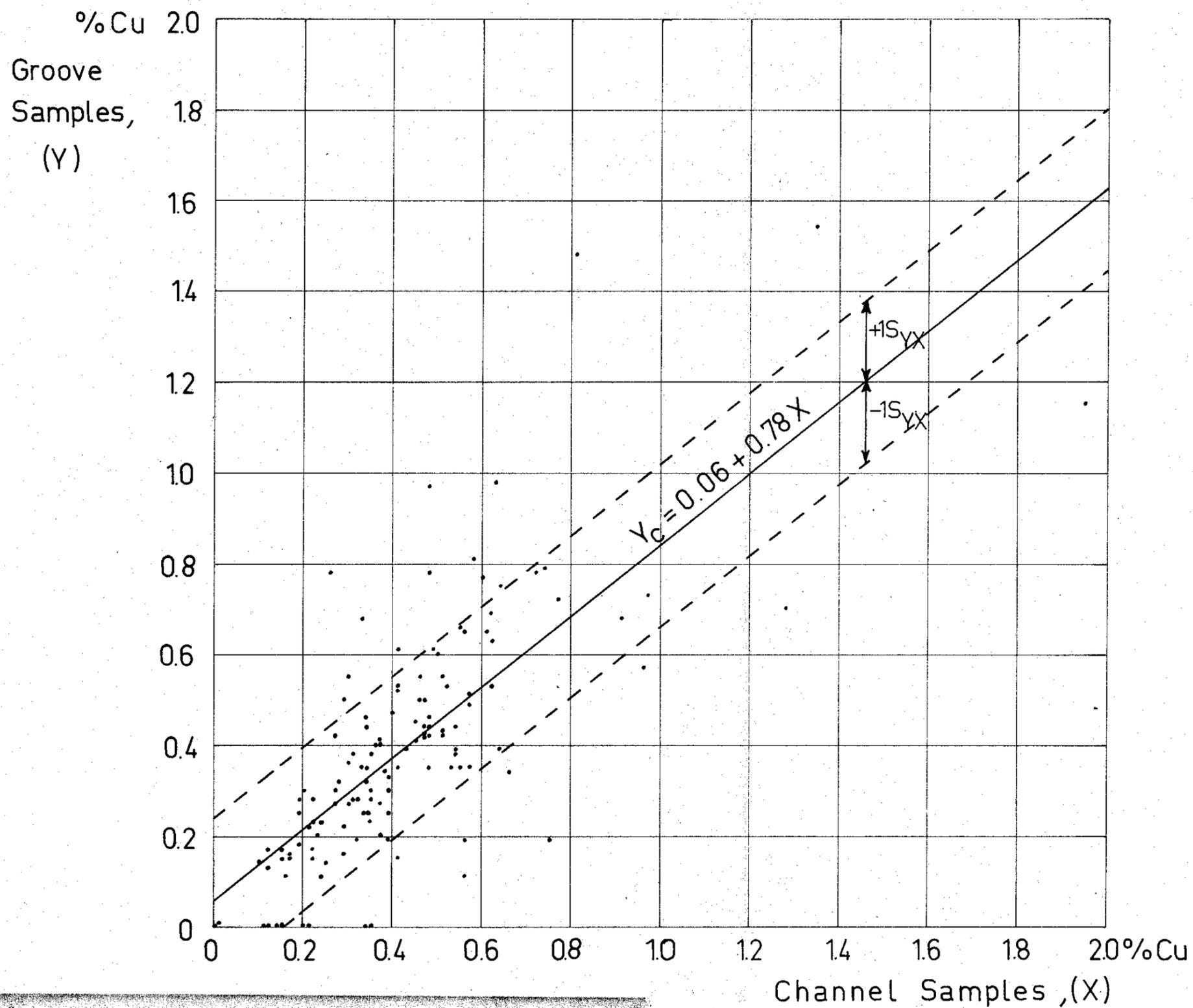points not shown).   R. Cox

179049

8531

Figure 5   Scatter diagram of copper grade from groove
and channel sample assays for 128 lode
sample locations: eight cross-cuts through
Hall's Lode, Lens "A", on 1300ft R.L.
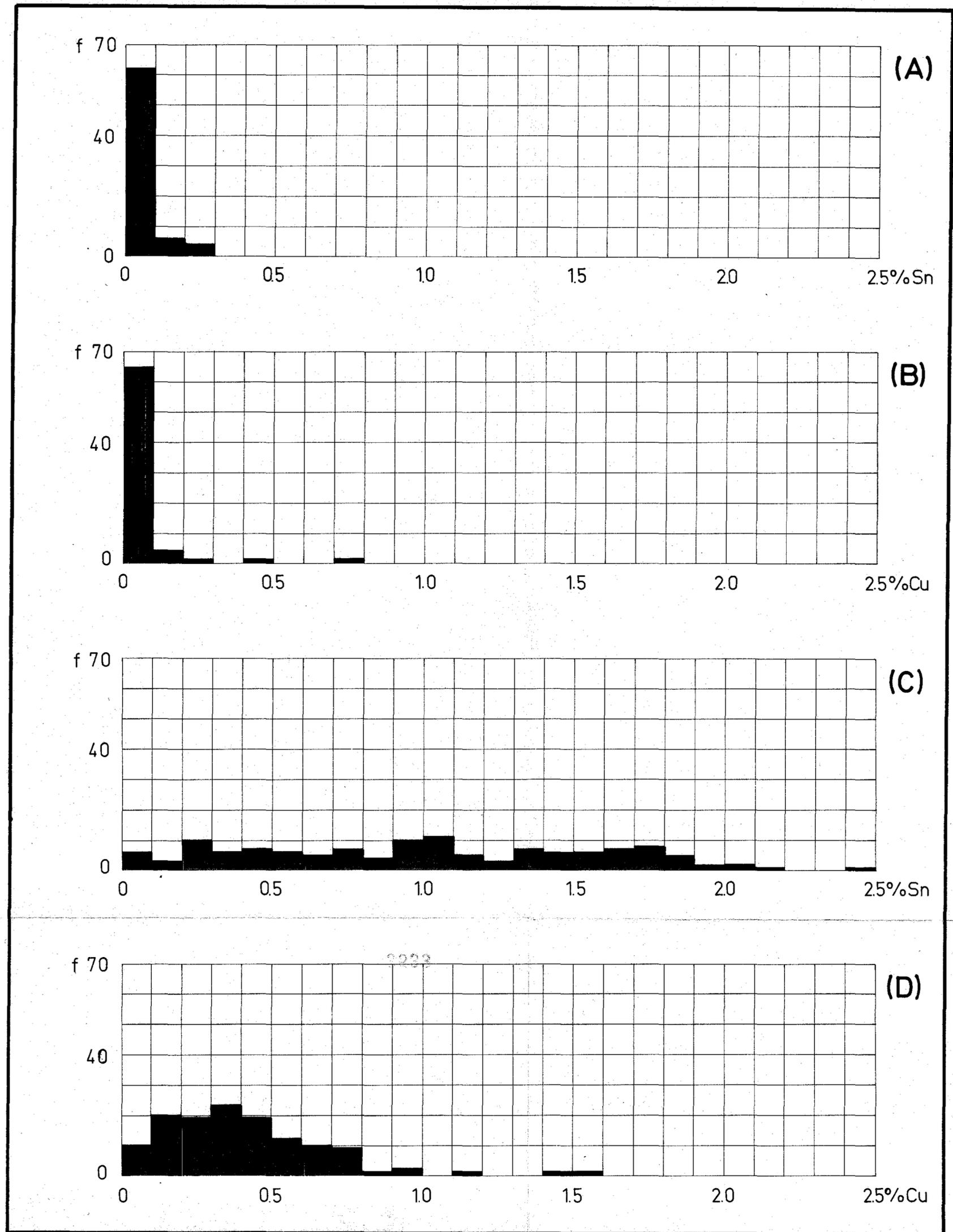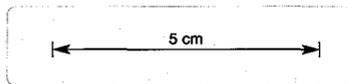"Qa" adit level, Cleveland Mine.

R. Cox

179050

3532

Figure 6   Column diagrams of frequency distributions
            for:
(A)   tin grade of 72 groove sample assays
       from chert bands
(B)   copper grade of 72 groove sample
       assays from chert bands
(C)   tin grade of 128 groove sample assays
       from lode bands
(D)   copper grade of 128 groove sample
       assays from lode bands
from eight cross-cuts through Hall's Lode,
Lens "A", on 1300ft R.L. "Qa" adit level,
Cleveland Mine.
                R. Cox

3533

179051