The optimization of sample spacing in South African gold mines

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SYNOPSIS

A first attempt is made at optimization of sample spacing in South African gold mines. The De Wijs model, which is shown to give a good representation of the geostatistical properties of gold values, is used to analyse the influence of the distance between samples along a stope face on the error of estimation of the stope value. The influence of other factors such as the face advance before resampling, the sample density, and the stope size, is also considered. Practical conclusions are drawn concerning the optimum sampling policy to be followed in various situations, and it is shown that sampling on a regular grid does not necessarily give the best results.

SAMEVATTING

Daar word vir die erste maal 'n poging aangewend om monsterspasiëring in Suid-Afrikaanse goudmyne te optimaliseer. Die De Wijs-model, waarvan bewys word dat dit 'n goeie voorstelling van die geostatiese eienskappe van goudwaardes gee, word gebruik om die invloed van die afstand tussen monsters langs 'n afboufront op die skattingsfout van die afbouwaarde te ontleed. Die invloed van ander faktore soos die frontvordering voor hermonsterneming, die monsterdigtheid en die grootte van die afbouplek, word ook oorweeg. Daar word praktiese gevolgtrekkings gemaak oor die optimale monsternemingsbeleid wat in verskillende situasies gevolg moet word en daar word getoon dat monsterneming op 'n reëlmatige rooster nie noodwendig die beste resultate oplewer nie.

INTRODUCTION

The spacing of chip samples along an advancing stope face relative to the interval of resampling of the face, i.e., a two-dimensional pattern of sampling, has exercised the mind of mine valuators throughout the history of South African gold mining. An optimum solution to the problem will naturally depend on how the objective is set: whether the estimate of the ore broken in stoping is sought on a month-by-month basis or over a longer period of, say, one quarter or a year. A two-dimensional distribution of ore values obviously has a significant effect on the optimum pattern of sampling, as do the costs of alternative sampling for the same overall sampling density. This paper is an attempt towards a scientific solution of this problem.

METHOD AND ASSUMPTIONS

It is assumed that a rectangular stope B is considered for mining and the management is interested in obtaining the best estimate of the ore that will be broken within a fixed period of time. The face length H is known, and the expected face advance L during the period specified can be calculated. The procedure generally used for stope valuation consists in the taking of chip samples along the face at regular intervals c and periodic resampling of the face. The face advance e between resampling is kept as constant as possible. Block B is shown in Fig. 1, which represents the situation to be studied in detail in this paper.

The pattern that will result in the best block estimation, on the assumption that there are no directional ore value shoots, is obviously a uniform grid with each sample located at the centre of a square zone of influence with dimension $c=e=\sqrt{H\times L/n}$. This pattern is impossible to follow in practice because every face would have to be sampled in the middle of every measuring month.

However, various sampling patterns are possible. For example, the positions of the first and last face samplings need not necessarily correspond to the extreme sides of block B. The effective area to which a single face sampling will apply will depend on the timing of this sampling relative to the periods for which the tonnage returns are prepared. In practice, a face sampling is usually applied to the tonnage broken in the month during which the sampling was done, as well as to subsequent months until the face is resampled; that is, face sampling is usually applied partly to an area behind the advancing face but mainly to the ground ahead of the sampling position.

Furthermore, the individual sampling sections along the stope face may or may not correspond to the extremities of the face, depending on the practice adopted by the mine of where the first section is cut and whether the length of face is an exact multiple of the sampling interval. Three of the possible sampling patterns will be considered here.

The question to be answered is how many samples should be taken along the face and how often the face should be resampled so that the errors in the estimation of the ore broken are minimized for a constant sampling cost. It is difficult to estimate how the sampling cost would vary as a function of the number of samples per face and the number of faces sampled. In this first attempt to solve the problem, the constraint of constant sampling cost has been replaced by the simpler constraint of constant total number of samples n.

The method used to solve the problem is the geostatistical method developed by Dr G. Matheron[1], which is based on a study of the variogram of gold values. The problem is complicated by the lognormality of gold-value distributions[2], and simplifying assumptions had to be made to allow for it. The procedure used in this paper is as follows.

— Calculate the variogram of the logarithmic sample values (with an additive constant if necessary).
— For varying sample spacings, calculate the error when the stope average is estimated from the sample logarithmic average.

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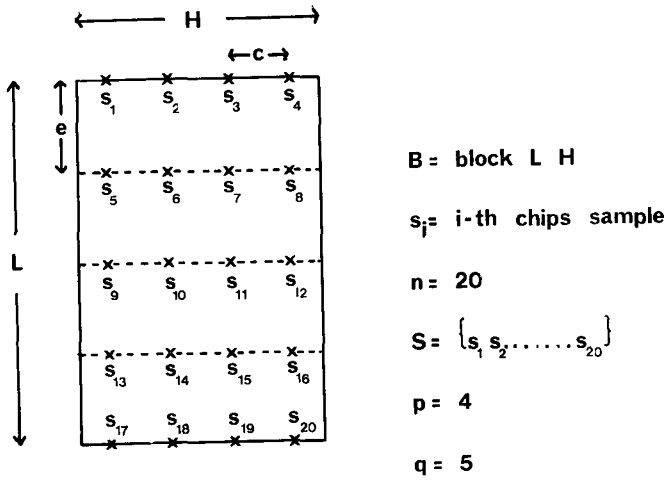


Fig. 1—Representation of stope B

— Find the sample spacing that corresponds to the minimum logarithmic error of estimation.
— Assume that this sample spacing will correspond to the minimum error of estimation of the stope arithmetic average from the sample arithmetic average.
— Repeat the procedure for various values of n, H, and L and draw practical conclusions concerning the optimum spacing.

The error of estimation of the geometric mean and the arithmetic mean are positively correlated, the correlation increasing as the number of sample values increases. The above assumption is therefore acceptable as a first approximation. However, as the stopes are valued in practice according to the arithmetic mean of the samples, an exact solution to the problem would require that the relationship between the error in the estimation of the stope arithmetic mean and the sample spacing should be analysed. The theoretical solution to this problem is very complex, and more research in this direction is necessary.

THE GEOSTATISTICAL MODEL

A section of the Hartebeestfontein Gold Mine in the Klerksdorp goldfield was used for the purpose of this analysis. In determining the statistical properties of a regionalized variable, it is recommended that the smallest possible size of support should be used. Hence, to analyse the logarithm of the gold values, individual chip samples should be considered, and the value and co-ordinates of thousands of chip samples should be recorded and stored on computer. However, in the Hartebeestfontein Gold Mine, the average gold contents of all the chip samples located within 25 ft^2 blocks forming a regular grid covering the entire mine were already available on computer[3]. These 25 ft averages were analysed, 4730 values being available in the section of the mine considered.

The cumulative distribution of the 25 ft averages was plotted, and an additive constant of 150 cm.g was found necessary for the log-normalization of these values (Fig. 2).

The relationship between variance and size for the area was then calculated. For this purpose, the average variance of the logarithm of the 25 ft averages (after the addition of 150 cm.g) was calculated within areas of different sizes S. The curve obtained is given in Fig. 3 and satisfies the De Wijs model with an absolute dispersion a=0,0215.

$$\sigma^2 \text{ (25 in } S) = N_{25} = \frac{3}{2}\, a \quad \ln\, \frac{(S)}{25\times 25}$$

N_{25}=0,310 the Nugget effect of the 25 ft averages
a=0,0215 the absolute dispersion
S=size of area, in ft^2.

These results agree with similar observations made elsewhere[4].

The De Wijs model can be used for the calculation of the theoretical semi-variogram of the log-normalized 25 ft averages. For distances d greater than 50 ft, it should be of the form[1]

$$\gamma_{25}(d) = C + 3\, a\, ln\, (d)$$

$$C = N_{25} - 3\, a\left[\ln(\lambda) - \frac{3}{2}\right]$$

λ=linear equivalent of a 25 ft square.

If 25 ft is used as the unit of length,

$$C = 0{,}310 - 3\times 0{,}0215 \times\left[\ln(2) - \frac{3}{2}\right] = 0{,}362.$$

The validity of the De Wijs model was verified by calculation of the experimental semi-variogram of the log-normalized 25 ft averages, the values obtained being plotted on Fig. 4. The theoretical semi-variogram calculated above gives a good fit to the experimental semi-variogram. The De Wijs model with an absolute dispersion a=0,0215 is therefore acceptable.

For calculation of the error of estimation in the value of block B, it is necessary to know not only the intrinsic function that characterizes the gold values,

γ(d)=3 a ln (d),

but also the Nugget effect N of the chip samples. This Nugget effect is a measurement of the sampling error and of possible micro-scale variations in gold values not represented by the De Wijs scheme. The value of N can be obtained from the variance-size of area relationship for chip samples. The contribution of the Nugget effect to the error of estimation is N/n, inversely proportional to the number of samples n. The solution to the problem considered in this paper, namely the calculation of the sample spacing that minimizes the error of estimation σ_e^2 when the number of chip samples n remains constant, is therefore independent of the Nugget effect.

Furthermore, the absolute dispersion a being only a scaling factor, the optimum sample spacing will also be independent of a.

For these reasons, in the following calculations the assumption N=0 was made. Also, the relative error of estimation $\sigma_e^2/3a$ was calculated, rather than σ_e^2. Consequently, *the results obtained can be applied without changes to any bedded deposit that satisfies the De Wijs model.*

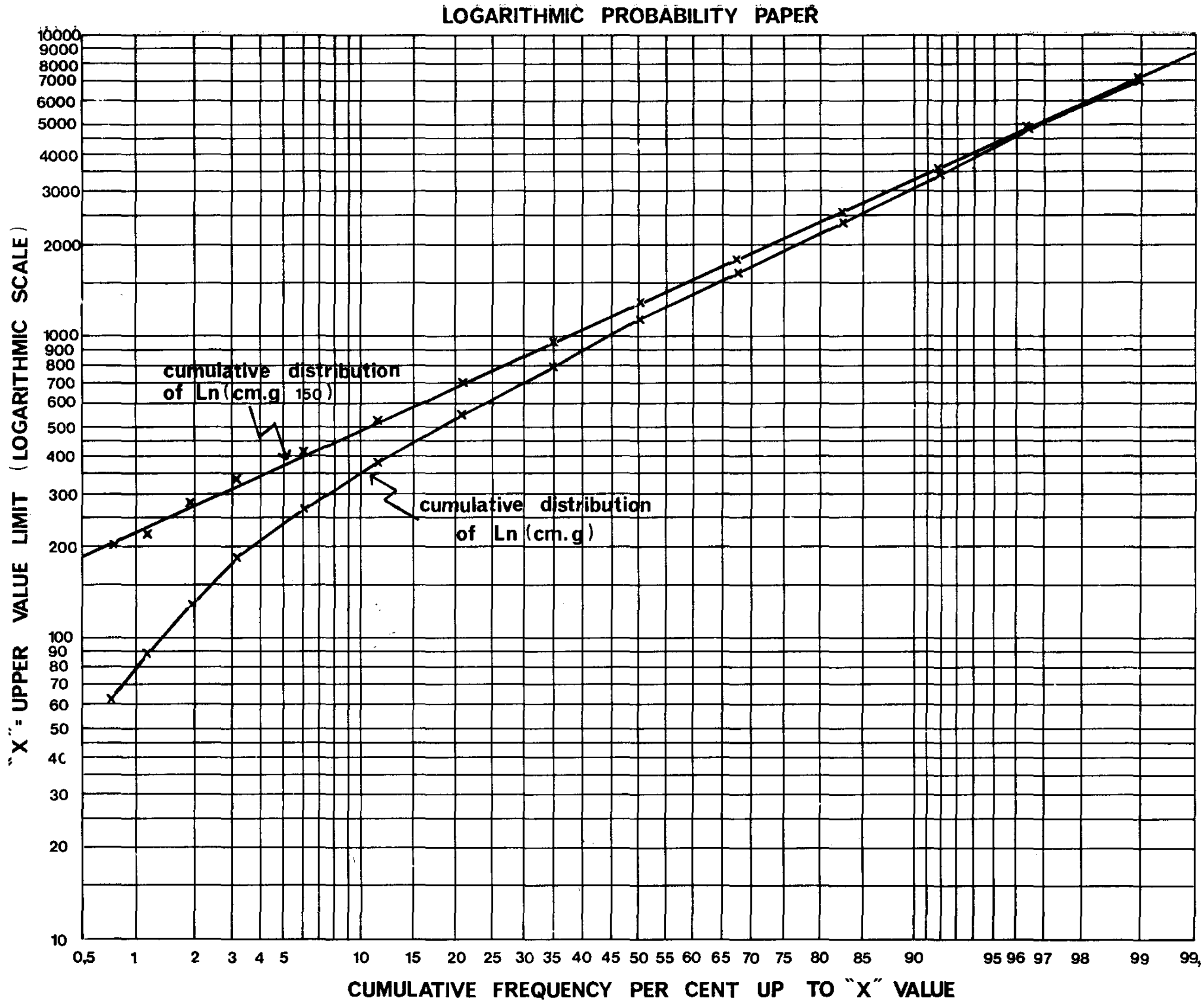


Fig. 2—Cumulative distribution of 25 ft averages

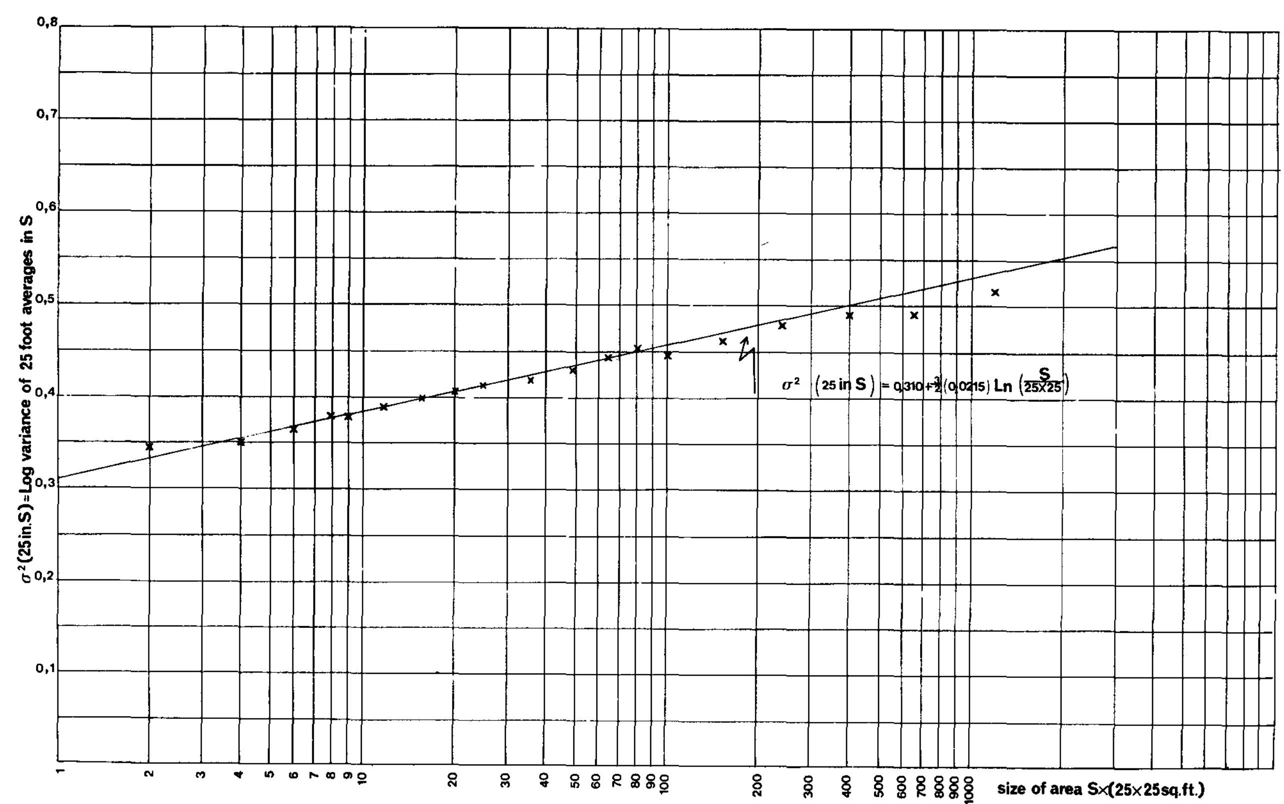


Fig. 3—Relationship between variance and size of area

is given by

$$\sigma_e^2 = E\left[(G^* - G)^2\right]$$

$$= 2\,\bar{\gamma}\,(S,B) - \bar{\gamma}\,(S,S) - \bar{\gamma}\,(B,B),$$

where $\bar{\gamma}(V', V'')$ is the average value of the semi-variogram $\bar{\gamma}(M'M'')$ when M' is any point in V' and M'' any point in V'' (Fig. 5).

If

- p=number of samples along a face and
- q=number of faces sampled ($p\,q = n$),

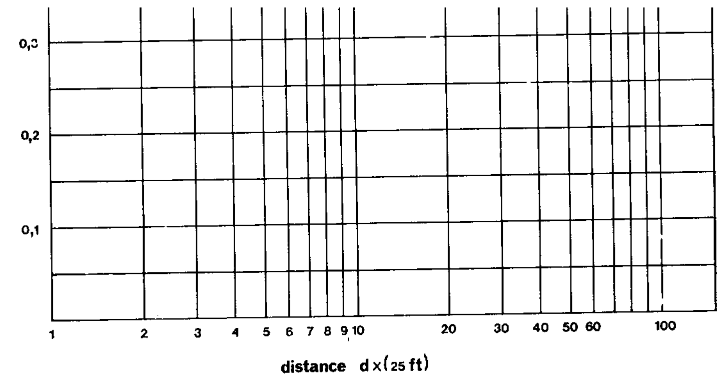
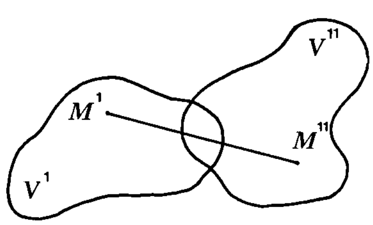


Fig. 4—Semi-variogram of 25ft averages

$$\bar{\gamma}\,(B,B) = \frac{1}{(LH)^2} \int_B dM' \int_B dM''\, \ln\,(M'M'')$$

$$\bar{\gamma}\,(S, S) = \frac{1}{n^2} \sum_{i=1}^{n} \sum_{j=1}^{n} \bar{\gamma}\,(s_i, s_j)$$

$$\bar{\gamma}\,(S, B) = \frac{1}{n} \sum_{i=1}^{n} \bar{\gamma}\,(s_i, B).$$



$$\bar{\gamma}(V', V'') = \frac{\int_{M' \in V'} dM' \int_{M'' \in V''} dM''\, \gamma\,(M'M'')}{\int_{M' \in V'} dM' \int_{M'' \in V''} dM''}$$

Fig. 5—Definition of $\bar{\gamma}(V', V'')$

Provided the distance d_{ij} between the two samples s_i and s_j is large compared with the section of a chip sample,

$$\bar{\gamma}(s_i, s_j) = \ln(d_{ij}) \text{ for } i \neq j.$$

When $i=j$, $\bar{\gamma}(s_i, s_i)$ is the average value of ln (d), where d is the distance between any two points in the sample. If a is the width of the chip sample along the face and b its depth normal to the face,

$$\bar{\gamma}\,(s_i, s_i) = \ln\,(\tau) - \frac{3}{2}$$

$\tau = a + b$ the linear equivalent of a chip sample.

The average $\bar{\gamma}(s_i, B)$ can be expressed as follows (Fig. 6):

$$\bar{\gamma}\,(s_i, B) = \frac{1}{LH}\Big[y_i\,x_i\,\bar{\gamma}\,(s_i, B_1) + (L - y_i)\,x_i\,\bar{\gamma}\,(s_i, B_2) + y_i\,(H - x_i)\,\bar{\gamma}\,(s_i, B_3) + (L - y_i)\,(H - x_i)\,\bar{\gamma}\,(s_i, B_4\Big].$$

To calculate the value of the averages, calculation of the following auxiliary functions is necessary (Fig. 7):

$F(L,H)=\bar{\gamma}(B, B)$, where B is a rectangle of size $L \times H$

$Q(L,H)=\bar{\gamma}(s,B)$, where s is a punctual sample taken at the corner of B.

The following expressions are obtained by integration of ln(d) in the rectangle[5]:

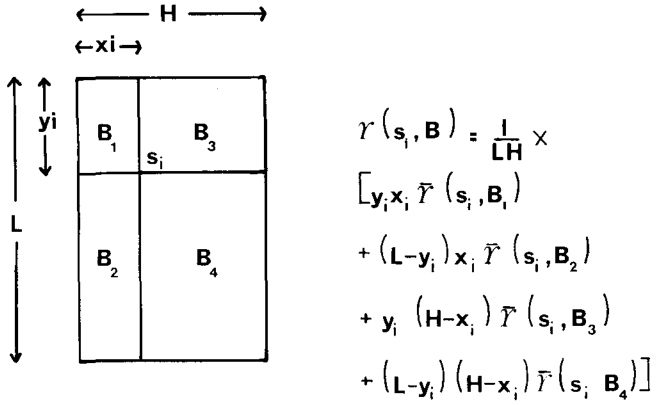


Fig. 6—Calculation of $\bar{\gamma}(S_i, B)$

$$F(L,H)=\ln(L)-\ln(\sin\theta)-\frac{25}{12}+\frac{\tan^2\theta}{6}\ln(\sin\theta)+\frac{\ln(\cos\theta)}{6\tan^2\theta}+\frac{2}{3}\frac{\theta}{\tan\theta}+\frac{2}{3}\left(\frac{\pi}{2}-\theta\right)\tan\theta$$

$$Q(L,H)=\ln(L)-\ln(\sin\theta)-\frac{3}{2}+\frac{\theta}{2\tan\theta}+\frac{1}{2}\left(\frac{\pi}{2}-\theta\right)\tan\theta,$$

where $\tan\theta=L/H$.

Computerization of the above formulae is straightforward, and a short program was written to calculate the error of estimation $\sigma_e^2/3a$ for changing values of L, H, and n.

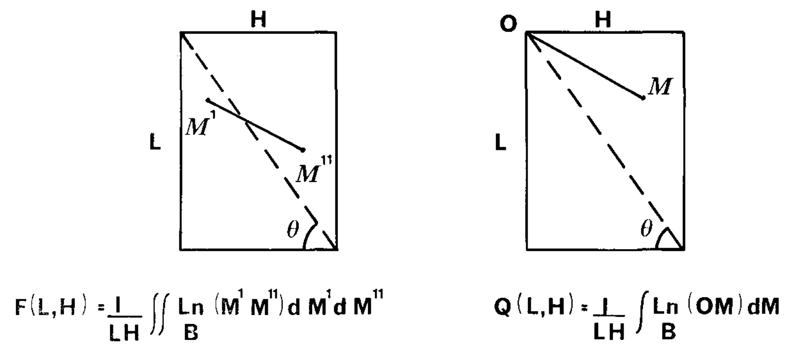


Fig. 7—Definition of the auxiliary functions $F(L\ H)$ and $Q(L, H)$

RESULTS

The length of a typical stope face at Hartebeestfontein Gold Mine is 30 m. A typical stope has a total length of 150 m and is mined in two blocks of length 75 m. The average face advance per quarter is 30 m. The effect of changes in the sampling pattern was analyzed for $H=30$ m and $L=30$ m, $L=75$ m, and $L=150$ m. For each one of these block sizes, the variance of estimation $\sigma_e^2/3a$ was calculated for various numbers of samples n and various distances between samples. The relationship between the error of estimation and the distance c between samples along the face or the distance e between faces is shown in Figs. 8 to 13. The value of c or e that corresponds to the minimum error of estimation was obtained graphically.

Given a block size B, it is possible to plot the relationship between c and e that will give the minimum error of estimation for a given sampling density. This relationship is plotted on Fig. 14. If the desired sampling density is 1 sample per 40 m² and the mine policy is to obtain the best block estimates on a quarterly basis ($H=30$ m and $L=30$ m), the sample spacing along the face should be 7,7 m and the face advance before resampling 5,2 m. In practice, the face being 30 m long, 4 samples should be taken along the face at a distance of 7,5 m and the face advance should be 5,0 m, resulting in a sampling density of 1 sample per 37,5 m².

One interesting result of this analysis is that, in spite of the complexity of the functions that were used, the final results as indicated on Fig. 14 are remarkably simple: the relationship between the optimal values of c and e is linear and has the form $c=K\,e$, where K is a function of the block size only. The value of K is independent of the sampling density, and K is never equal to 1.

These results can be explained by a border effect. The samples taken on the two extreme faces give information only on one side, while all the samples taken on faces inside the block give information in all directions. Consequently, an effort should be made to take more samples inside the block and fewer on the border. This can be done only by an increase in the number of faces. This border effect becomes less significant as the block becomes longer.

EFFECT OF ANISOTROPIES

In the above analysis, the variogram did not vary with the direction considered, the section of Hartebeestfontein analysed being characterized by such an isotropic variogram. Other sections of Hartebeestfontein

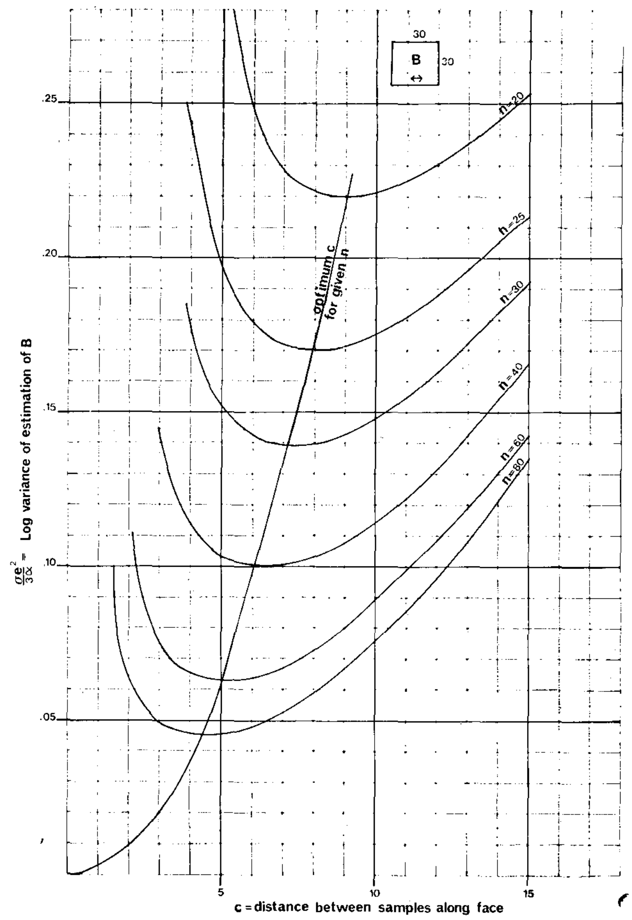


Fig. 8—Influence of spacing between samples along the face of a 30 m by 30 m block

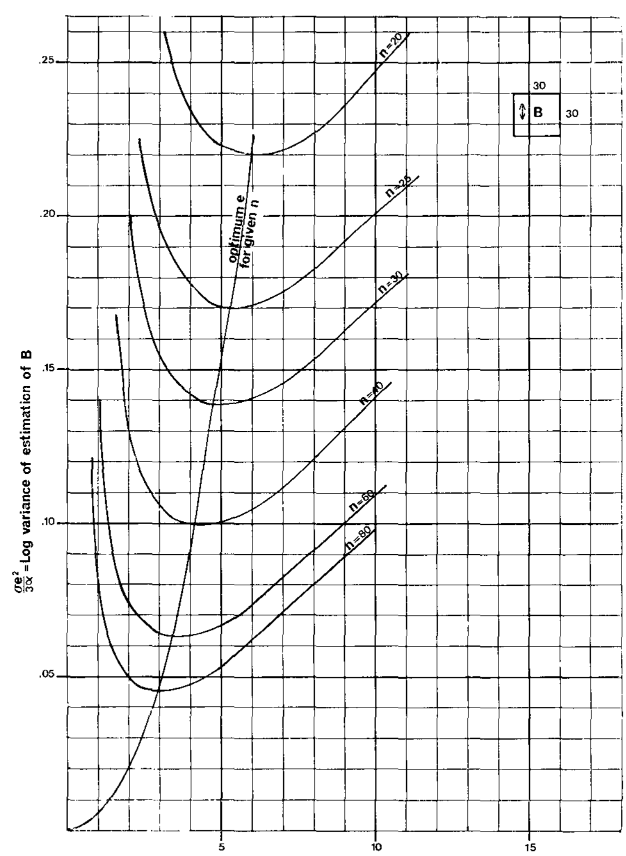


Fig. 9—Influence of spacing between faces of a 30 m by 30 m block

where there are significant pay shoots have anisotropic variograms, the gold variability in the direction of the shoots being lower than in the direction normal to the shoots. The usual effect of geometric anisotropies on the semi-variogram is that, in the direction of the shoots, one observes

$\gamma_1\,(d)=N_1+3\,a\ln\,(d)$;

and, in the direction normal to the shoots,

$\gamma_2\,(d)=N_2+3\,a\ln\,(d)$,

where N_2 is greater than N_1, the absolute dispersion a remaining constant.

If the principal directions of anisotropy are parallel to the sides of the block to be valued, the results obtained in the above analysis can be adjusted as follows when the stope face is normal to the pay shoot. The optimum sampling distance along the face is obtained from Fig. 14 by multiplying the value c read in ordinate by $\sqrt{d_2/d_1}$, where d_1 and d_2 are such that $\gamma_1(d_1)=\gamma_2(d_2)$:

$\ln\,(d_1/d_2)=(N_2-N_1)\,/3\,a$.

The optimum distance between faces is obtained from Fig. 14 by multiplying the value e read in abscissa by $\sqrt{d_1/d_2}$. As an illustration, a section of Hartebeestfontein Gold Mine was considered[6] that presented an anisotropy with ratio $d_1/d_2=10/4$. In this mine section, if a sampling density of 1 sample per 40 m² is required, an optimum valuation of a block of size 30 m by 30 m whose face is normal to the direction of the pay shoot will be obtained for $c=7{,}7\sqrt{4/10}=4{,}87$ m and $c=5{,}2\sqrt{10/4}=8{,}22$ m. In practice, one will take 6 samples per face at 5 m intervals, and resample the face after a face advance of 7,5 m. If the block face is parallel to the direction of the pay shoot, the optimum sampling policy will consist in taking 3 samples per face at 10 m intervals and resampling the face after an advance of 3,75 m.

If the principal directions of anisotropy are at an angle with the sides of the block, the rectangle B must be transformed to a parallelogram to yield an isotropic variogram, and the solution will lie somewhere between that for the isotropic case and that given above for anisotropy parallel or at right-angles to the face.

EFFECT OF DIFFERENT SAMPLING PLANS

In the previous analysis, it was assumed that the block B was sampled as shown on Fig. 1, the first and last faces being sampled and each face being given an equal weight. In practice, a face value can be applied only to the area ahead and up to the next valuation, and there will then not be a valuation at the last face position. A study of this situation showed that, for the same block size B and the same number of samples n, the minimum error of estimation is obtained for a ratio c/e slightly higher than when the last face is sampled ($c/e=1{,}4$ for a block of size 30 m by 75 m, instead of $c/e=1{,}3$). The loss in efficiency that would result from the use of Fig. 14 in the determination of c and e is therefore negligible, particularly as the average position in practice lies somewhere between the two cases considered.

The situation where chip samples are taken at the beginning and end of each face was also considered. In that case, provided the two extreme faces of the block are also sampled, the optimum sample distribution corresponds to the uniform grid ($c=e$). For a fixed number of samples n, the error of estimation is much higher (20 per cent higher for a 30 m by 75 m block) if the samples are taken at each end of the face than it would otherwise be if the sampling pattern indicated on Fig. 1 was accepted. The latter pattern should therefore be preferred.

CONCLUSIONS

The main purpose of this paper was to show that modern geostatistical methods can be applied successfully to complex problems of gold-mine valuation. The practical application of the results is still somewhat limited, and more research is necessary before fully representative answers can be formalized. Particularly helpful would be a more detailed analysis of the geostatistical models that can be used to represent the distribution of gold values in various gold mines[6]. If the De Wijs model is proved to represent the properties of most gold mines, analyses such as these could be simplified and standardized for the gold-mining industry as a whole.

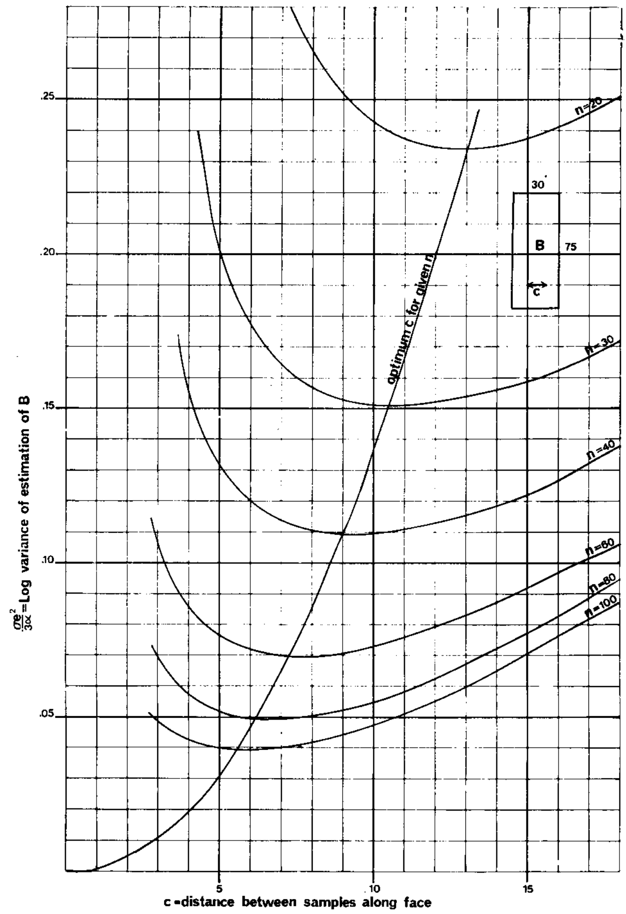


Fig. 10—Influence of spacing between samples along the face of a 30 m by 75 m block

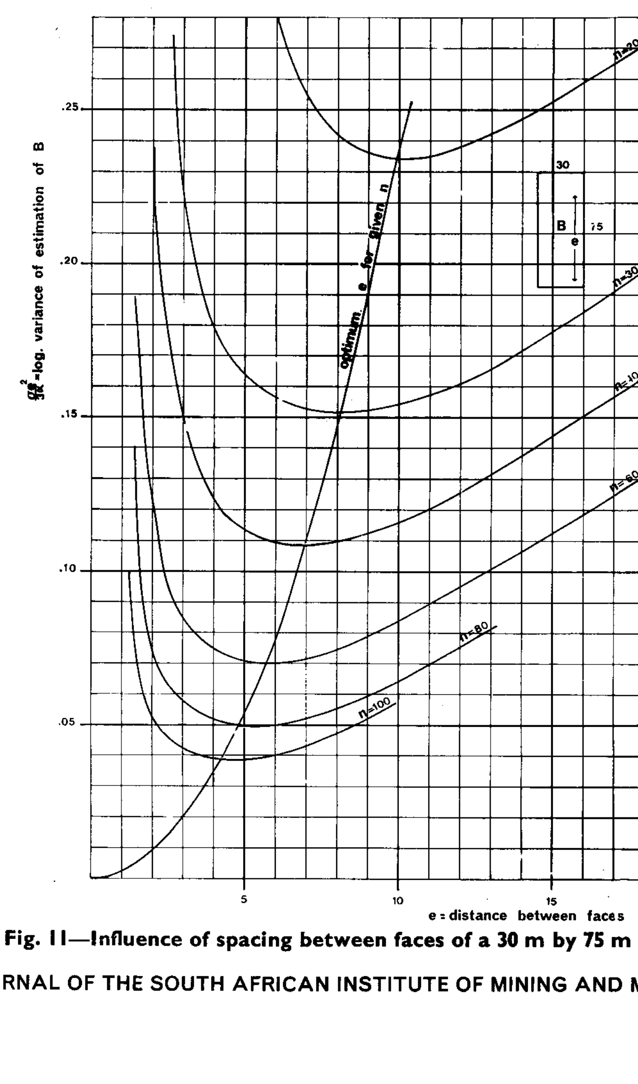


Fig. 11—Influence of spacing between faces of a 30 m by 75 m block

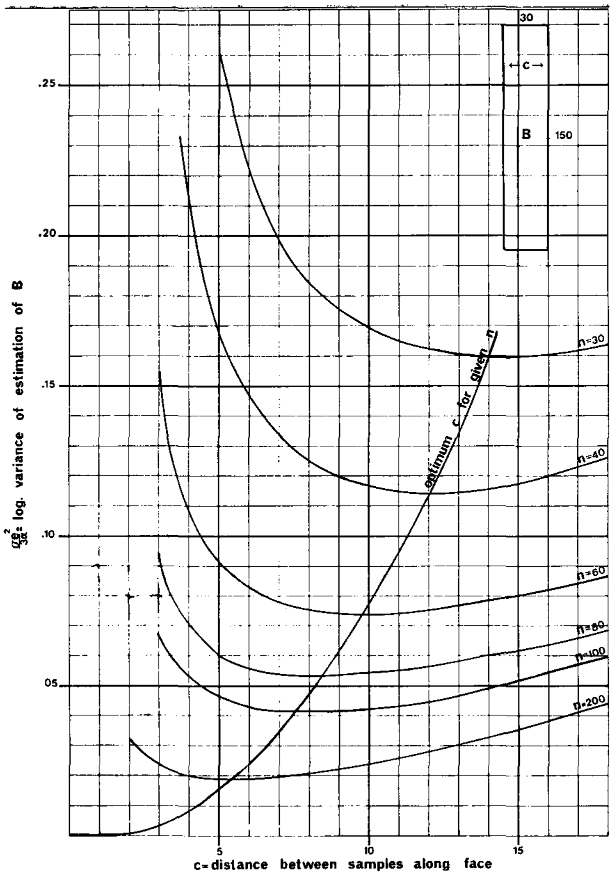


Fig. 12—Influence of spacing between samples along the face of a 30 m by 150 m block

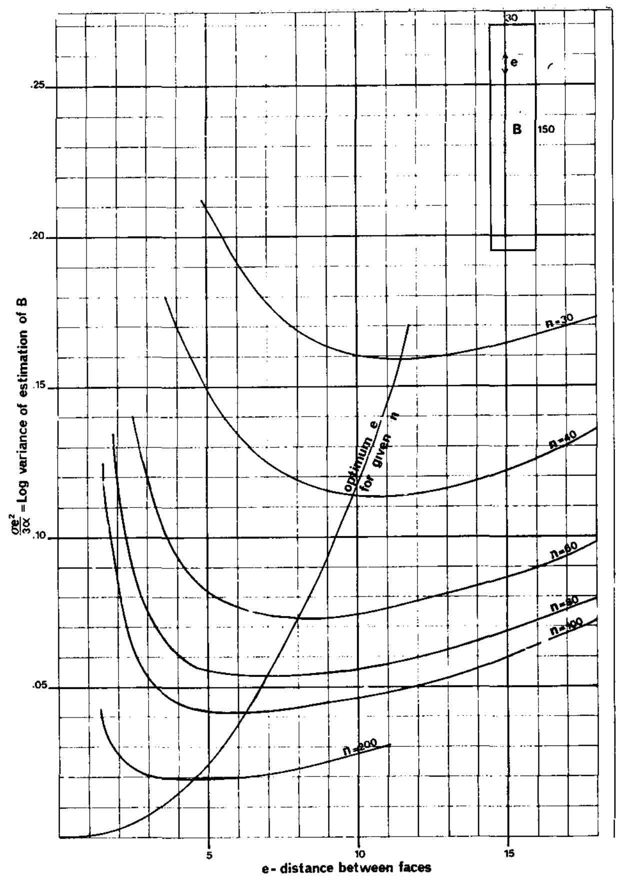


Fig. 13—Influence of spacing between faces of a 30 m by 150 m block

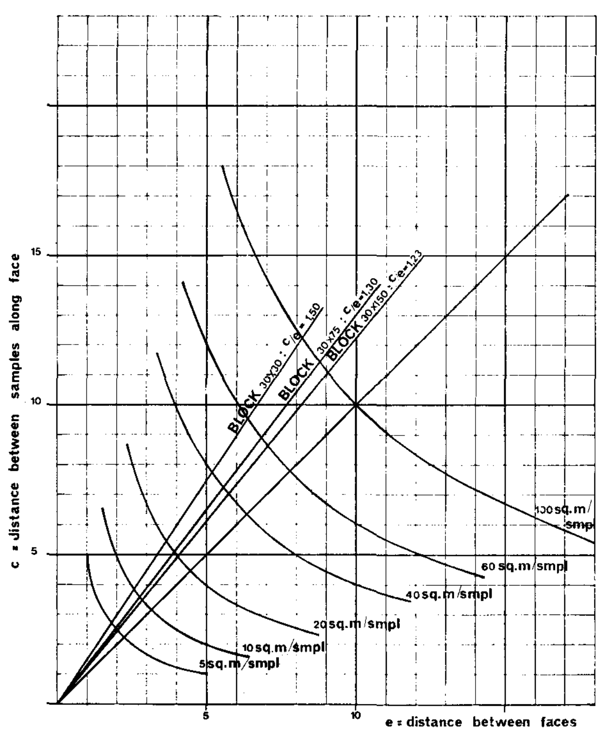


Fig. 14—Influence of block size and sample density on optimum grid spacing

ACKNOWLEDGEMENT

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NIM reports

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Report no. 1247

The pelletizing of chromite fines. A preliminary investigation of the preparation of cold-bonded pellets. (17th Mar., 1971; re-issued Dec. 1975).

After a brief review of various techniques of agglomeration, an account is given of tests done on chromite pellets bonded with dicalcium silicate slag and hardened by autoclaving. The results show that satisfactory pellets can be made of chromite (90 per cent) bonded with activated slag (10 per cent), the chromite having been milled to 75 per cent minus 200 mesh or finer. Calcium hydroxide (2 per cent) is used for the activation of the slag. Pellets of chromite having the same grading but bonded with Portland cement by the Grangcold process can also be successfully made.

Report no. 1374

Flotation tests on copper–zinc ore from the Letaba district. (12th Oct., 1971; re-issued Nov. 1975).

The bulk-float method for the separation of copper and zinc, as used on Prieska ore, was tried on copper–zinc ore from the Letaba district. A salable zinc concentrate could readily be produced, but the zinc content of the copper concentrate was about 20 per cent, which could not be reduced by normal flotation techniques. A report on a mineralogical examination of the ore is included.

Report no. 1388

Flotation tests on samples of copper–zinc ores from Finland. (12th Nov., 1971; re-issued Nov. 1975).

The results are given of laboratory tests on two samples of copper–zinc ores supplied by the Outokompu Oy, Finland: one from the Pyhäsalmi Mine, and the other from the Vihanti Mine.

It is shown that, with slight modifications, the separation process developed at the National Institute for Metallurgy is applicable to these ores.

The copper and zinc recoveries obtained are compared with those reported by the two Finnish mines for 1970.

This work has confirmed the finding from earlier work on zinc–copper ores that there are considerable differences between deposits, particularly in the 'activity' and floatability of the sulphide minerals.

Report no. 1482

Reactions occurring during the smelting of high-carbon ferrochromium from Transvaal chromite ores. 9th Apr., 1973; re-issued Nov. 1975).

A new technique — a variation of the SCICE (stationary charge in controlled environment) technique — was used in the study. A large sample (4 kg) was employed, and the rates of reduction of both the iron and the chromium from the chromite ore were measured.

It was shown that, at temperatures up to 1500 °C, reduction of the chromite ore is controlled by the diffusion of iron and chromium ions in the chromite spinel. At 1500 °C, the reduction rate of chromium from the chromite spinel is increased by the fluxing reaction. In the presence of slag, at temperatures higher than 1500 °C, the reduction of chromite ore by carbon is rapid. It was found that these mechanisms of reduction are not independent of one another but have to occur in the order described.

Report no. 1566

Comparative tests on coke, char, and anthracite. (17th Aug., 1973; re-issued Nov. 1975).

In an attempt to determine whether the anthracite from the Balgray Collieries in Natal was suitable as a fuel in low-shaft blast furnaces, comparative tests were conducted on two Balgray anthracites, anthracite from the Natal Anthracite Collieries, coke from Iscor, and char from the Ferrometals plant in Witbank.

The results, especially for shatter index and M_{10}, varied considerably owing to the different breaking properties and non-representative size distributions of the samples. However, the tests did show that the Balgray dull anthracite was the best of the three anthracites.

It is concluded that comparative tests of this type do not give suitable results because of the different characteristics of the samples being compared.

Report no. 1764

Correlations of the free-energy and enthalpy changes of complex formation of the d^{10} metal ions Ag^+, Hg^{2+}, and Cd^{2+} with a variety of monodentate ligands.

The standard changes in free energy, enthalpy, and entropy at 298,15 ±0,05 K for the reactions involved in the formation of complexes between the metal ions silver, mercury, and cadmium, and a variety of charged and uncharged ligands involving the donor atoms phosphorus, arsenic, sulphur, nitrogen, oxygen, and the cyanide ion in aqueous solution, were measured by silver-, mercury-, and glass-electrode potentiometry and titration calorimetry.

The standard state, denoted by the superscript ° in $\Delta G°$, $\Delta H°$, and $T\Delta S°$, refers, in most instances, to a hypothetical 1 M solution in which the ionic strength, μ, is 0,5 M.

The results show that, if Ag(I) is regarded as the reference metal ion, the free-energy changes form at least three distinct linear free-energy relations for each pair of metal ions containing the 'soft'-, the N-, and the O-donor ligands respectively. The free-energy changes for metal–ligand (ML) and metal–double ligand (ML_2) complexes lie on the same linear free-energy relation.

The enthalpy changes in these enthalpy-stabilized complexes also form distinct linear enthalpy relations, which closely resemble the linear free-energy relations, showing that the separate linear free-energy relations obtained for the differing donor atoms of the ligands can be attributed primarily to bonding factors. The enthalpy changes for the ML and ML_2 complexes form separate linear enthalpy relations.

The entropy changes for neutral ligands are shown to deviate fairly systematically from certain restrictions imposed by the Powell, Latimer, and Cobble theory. The deviations, which are larger for Ag(I)–Cd(II) than for Ag(I)–Hg(II), and larger for ML than for ML_2 complexes, can be attributed to partial desolvation of the central metal ion on complex formation.

Report no. 1777

Ferro-alloys in Brazil.

When the National Institute for Metallurgy was approached to contribute a paper to the first Latin American Seminar on Ferro-alloys in Salvador, Bahia, Brazil (23rd to 27th June, 1975), it was decided to contribute a paper and to use the opportunity to assess the current state of development of the ferro-alloy industry in Brazil, with particular reference to ferrochromium. This report summarizes the seminar very briefly and discusses in detail the past, present, and future status of the Brazilian ferro-alloy industry. A summary of current practice at a number of ferro-alloy plants is included.

By 1980, the Brazilian ferro-alloy industry could have an installed transformer capacity of 821 MVA and could be exporting a total of 497 000 t of 'tonnage' alloys per year. The major competition for export markets with South Africa will be in manganese-based and silicon-based alloys, and the growth of the Brazilian ferro-alloy industry should be watched carefully to see whether it achieves the forecast export volume. The development of the Brazilian steel industry should also be monitored closely till 1980 so that possible variations in the amount of ferro-alloys available for export can be forecast.

The major problem for the expansion of the Brazilian industry appears to be the assurance of adequate ore supplies in the future, and further information is required on this point. However, in view of the large unexplored areas in Brazil, it is difficult to determine the prospects for future discoveries of ore. In any case, a major problem in the exploitation of present and future deposits is the development of a satisfactory infrastructure, and more information is required on government planning for the development of this infrastructure.

Report no. 1779

A summary of work done on the concentration of andalusite from deposits in the western Transvaal.

Most of the andalusite-bearing materials examined contained andalusite assaying more than 53,0 per cent Al_2O_3 and less than 2,0 per cent Fe_2O_3. The andalusite content varied from 4,0 per cent to over 50 per cent.

The efficiency of various methods of concentration was investigated. Heavy-medium separation (HMS) was found to be the only practical method of concentration. A standard method, which includes the use of washability curves and the Tromp efficiency curve, was developed for assessment of the ores and their amenability to HMS. Results showed that the procedure is reliable in the estimation of the results of HMS in practice.

It was found that, owing to the effects of weathering and alteration, the fractions of lower specific gravity in an andalusite concentrate, when separated by the use of heavy liquids, were of a much lower quality than were the fractions of higher specific gravity. It was concluded that separations at higher specific gravities than are normally used may be necessary if high grades of concentrate are to be obtained, and that it is possible to produce concentrates assaying more than 55 per cent Al_2O_3 by separation at higher densities.

Several ores were tested by HMS, and, for most of them, concentrates assaying more than 53,0 per cent Al_2O_3 and less than 2,0 per cent Fe_2O_3 were produced. Two- or three-stage separations produced concentrates assaying more than 55,0 per cent Al_2O_3, and it was found that the grade increased with an increase in the specific gravity of separation.

In an examination of several operating plants, it was found that the heavy-medium separators were operating efficiently, but that concentrate grades were generally below market specifications. Estimates were made of the results that can be obtained when separation is done at higher specific gravities, and it was shown that concentrate grades of more than 54 per cent Al_2O_3 can be produced if the specific gravity of separation is increased.

Report no. 1784

A single-standard calibration method for use in analysis by X-ray-fluorescence spectrometry.

A simple, effective, and quick method for correction of matrix effects in X-ray-fluorescence spectrometry is described. The 'single-standard calibration method' involves measurement of three net intensities, namely, those of the standard, a mixture of the standard and the sample, and the sample. These three net intensities and the mass fraction of the sample in the mixture of the standard and sample are substituted into a simple equation, the solution of which yields the concentration of the element measured. This method was applied to samples of anode sludge by fusion of the sample with Na_2O_2 in a zirconium crucible, leaching of the melt in hydrochloric acid and diethylenetriamine (DETA), and making it up to volume. The three net intensities required for the single-standard calibration method are measured, and the data are processed on-line with the use of a desk-top computer. The laboratory method is given in an appendix.

Iron and steel making

An international meeting, 'Automation in Iron and Steel Making' is to be held in Brussels on 17th and 18th May, 1976, and in Düsseldorf on 20th and 21st May, 1976. The programme is as follows: Automation of Burden Preparation Plants, Coke Oven Plants, Blast Furnaces and Steel Making Plants (in Brussels) and Automation in the Field of Hot Rolling Mills, Cold Rolling Mills, and Laboratories (Düsseldorf).

Further information is available from Centre de Recherches Metallurgiques, B-4000 Liege, Abbaye du Val-Benoit, and Verein Deutscher Eisenhuttenleute, D-4000 Düsseldorf, Postfach 8209.

Continuous casting

An international conference on continuous casting will be held in Biarritz, France, from 30th May to 2nd June, 1976. It is being organized jointly by The Metals Society and IRSID.

The following topics will be discussed:

Liquid Metal Preparation and Refractories
Machine Maintenance Related to Maximum Availability
Quality
Non-conventional and New Techniques
Plant Experience: Blooms and Billets
Plant Experience: Slabs

The official languages will be English, French, and German, with simultaneous translation.

Enquiries should be directed to Conference Secretariat (Continuous Casting), The Metals Society, 1 Carlton House Terrace, London SW1Y 5DB.

Company Affiliates

The following members have been admitted to the Institute as Company Affiliates.

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S.A. Land Exploration Co. Limited.
Stilfontein G.M. Co. Limited.
The Griqualand Exploration and Finance Co. Limited.
The Messina (Transvaal) Development Co. Limited.
The Steel Engineering Co. Ltd.
Trans-Natal Coal Corporation Limited.
Tvl Cons. Land & Exploration Co.
Tsumeb Corporation Limited.
Union Corporation Limited.
Vaal Reefs Exploration & Mining Co. Limited.
Venterspost G.M. Co. Limited.
Vergenoeg MiningCo. (Pty) Limited.
Vlakfontein G.M. Co. Limited.
Welkom Gold Mining Co. Limited.
West Driefontein G.M. Co. Limited.
Western Deep Levels Limited.
Western Holdings Limited.
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