Fig. 4.43a: Actual, and Estimated Error Distribution, S1.1, SLKGM

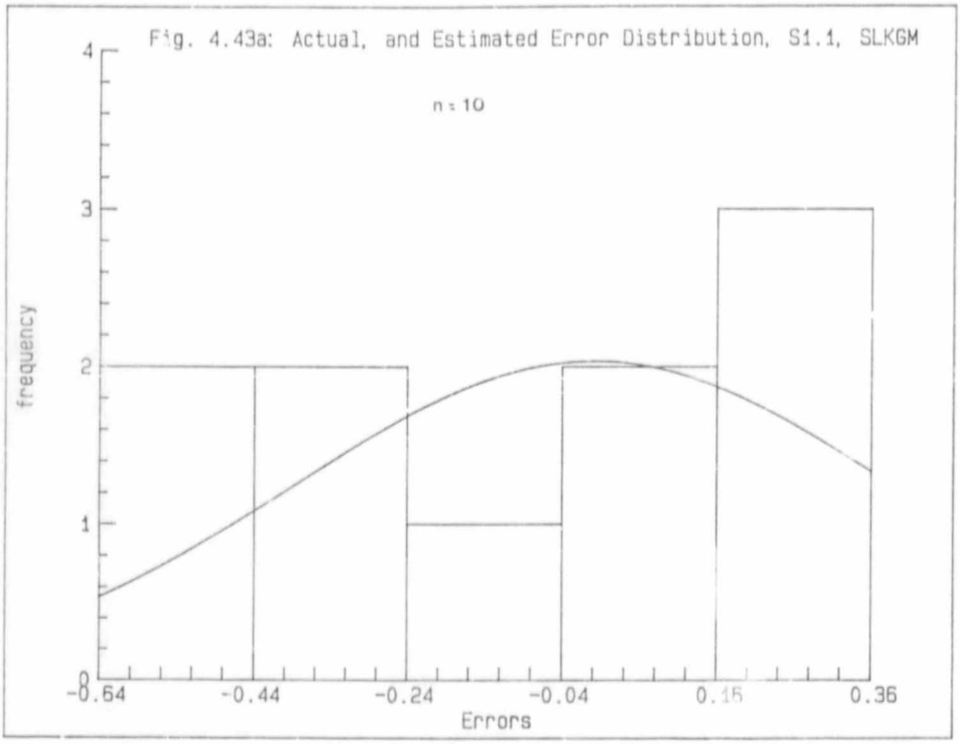
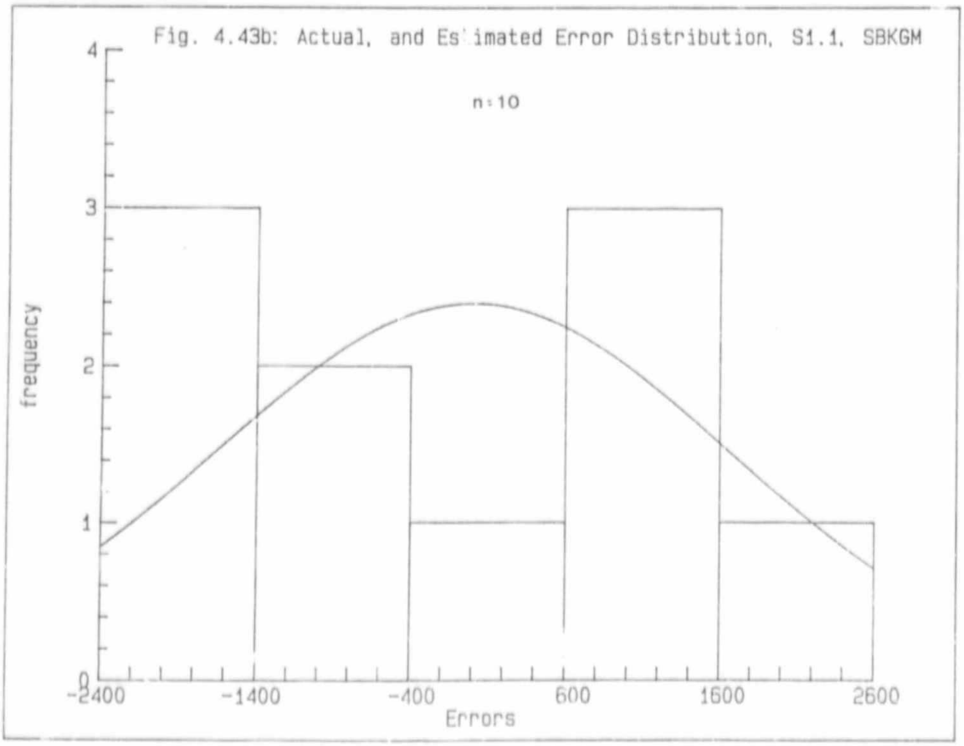


Fig. 4.43b: Actual, and Estimated Error Distribution, S1.1, SBKGM



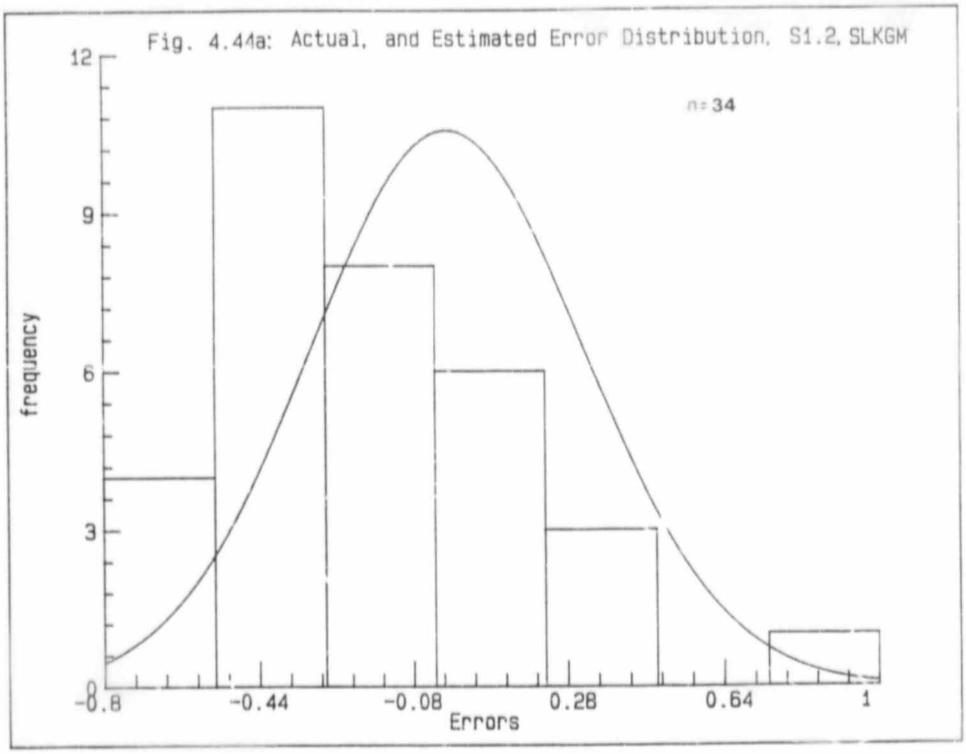


Fig. 4.44a: Actual, and Estimated Error Distribution, S1.2, SLKGM

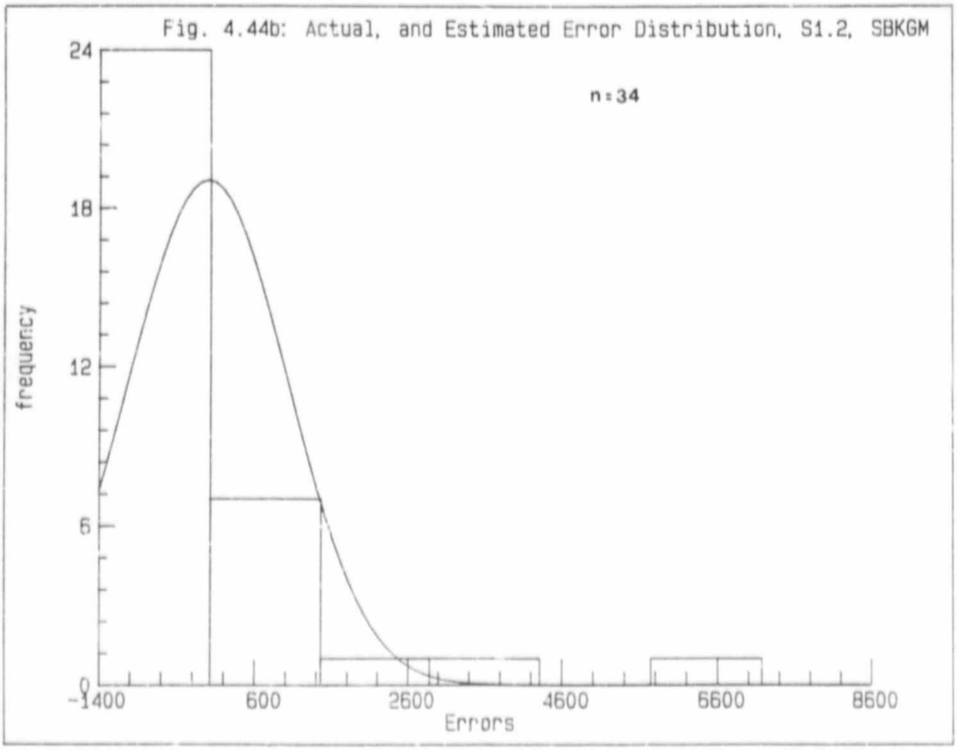


Fig. 4.44b: Actual, and Estimated Error Distribution, S1.2, SBKGM

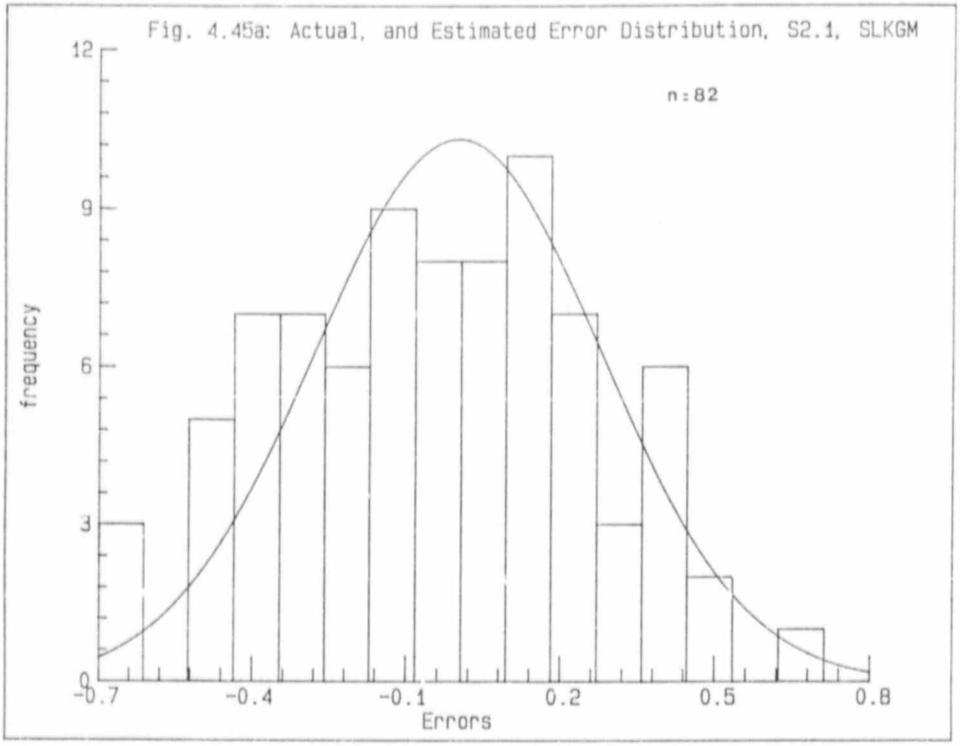


Fig. 4.45a: Actual, and Estimated Error Distribution, S2.1, SLKGM
n=82
frequency
Errors
12
9
6
3
0
-0.7
-0.4
-0.1
0.2
0.5
0.8

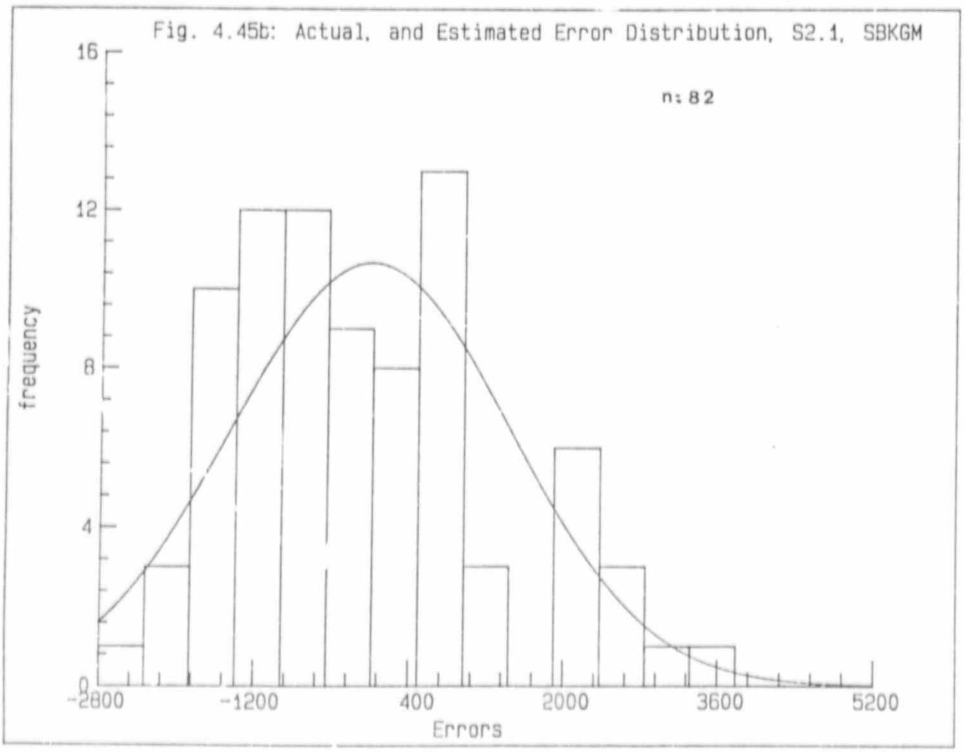


Fig. 4.45b: Actual, and Estimated Error Distribution, S2.1, SBKGM
n=82
frequency
Errors
16
12
8
4
0
-2800
-1200
400
2000
3600
5200

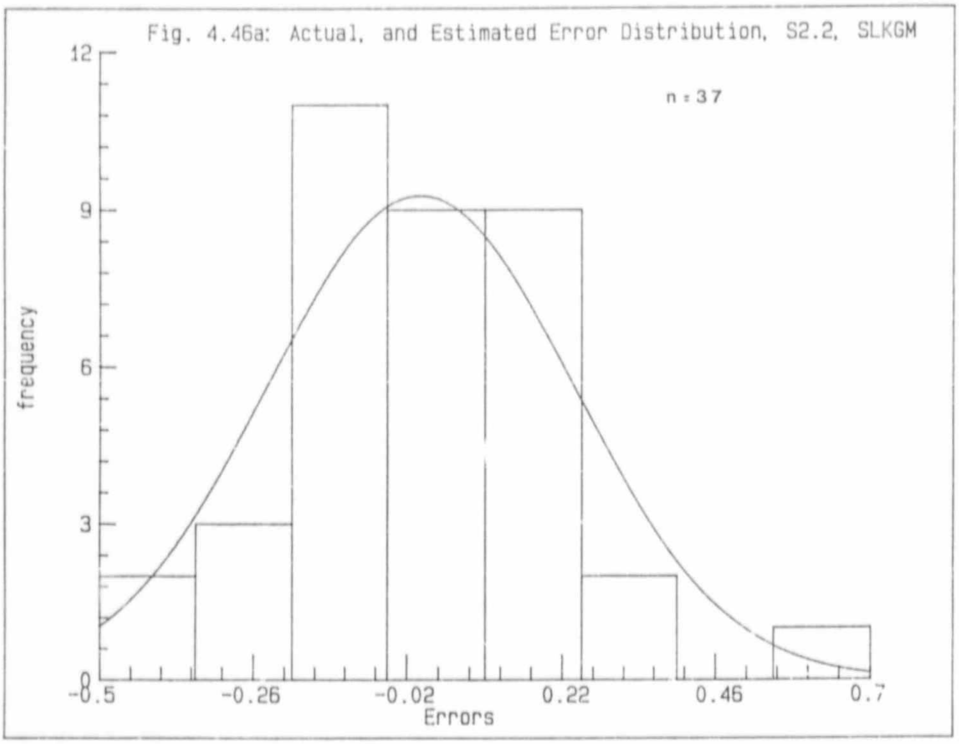


Fig. 4.46a: Actual, and Estimated Error Distribution, S2.2, SLKGM
n = 37
frequency
Errors

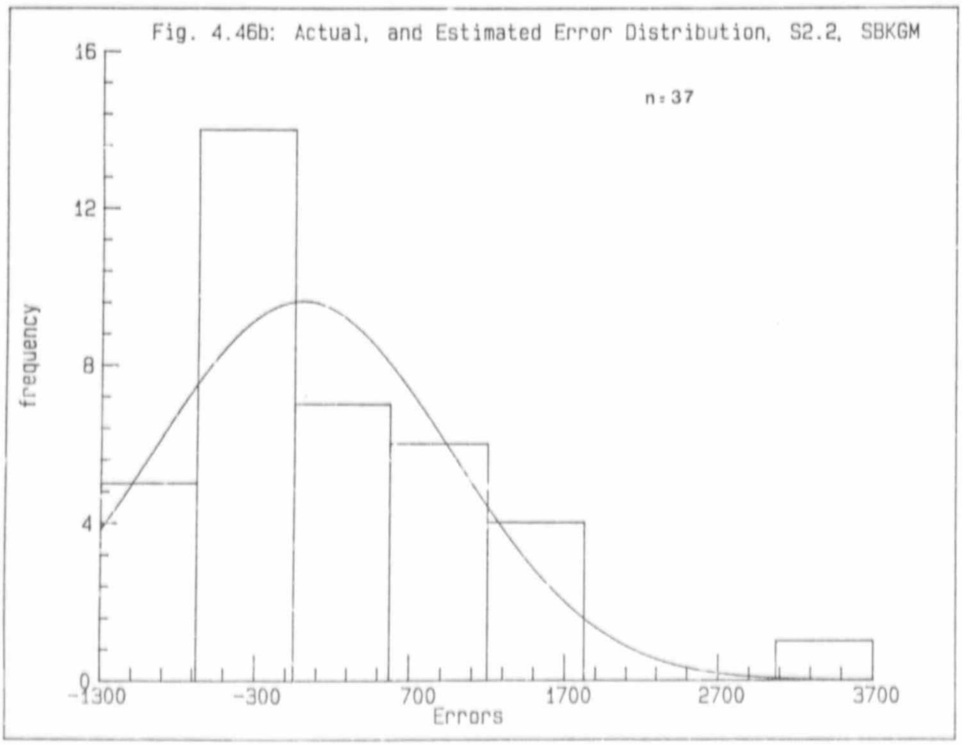


Fig. 4.46b: Actual, and Estimated Error Distribution, S2.2, SBKGM
n = 37
frequency
Errors

and the average estimation variance, for the four subareas S1.1, S1.2, S2.1 and S2.2. No test of the significance of the fit of the model to the data is given, as about half of the models fail this test. However, for practical estimation purposes, the main interest lies not in the statistical significance, but in the practical application. Keeping the limited number of data in mind, the plots show that the kriging estimation error - for practical purposes - closely enough approaches the actual error.

- a further method for checking the validity of the semivariogram, and therefore the estimation variance, is described by David (David, in print), and has been applied originally by Verly. It is based on the assumption of a normal distribution of the errors, and does not require a regular or pseudoregular sample grid.

If the errors are normally distributed, the true error falls, twice out of three times, in the interval of the estimated grade plus/minus one standard deviation. So the absolute value of the true error will, twice out of three times, fall between zero and one standard deviation. This means that, if the absolute true error is plotted on the vertical axes, against the standard error of the estimation on the horizontal axes, one third of the points will fall above the line X=Y, and two thirds below this line, if the semivariogram (and therefore the error) are estimated correctly.

Figures 4.47 to 4.50 show these plots for the four sub-areas S1.1, S1.2, S2.1 and S2.2.

A simple visual check shows that about one third of all points plot above the line X=Y on each graph.

A binomial test shows that in all cases, the number of errors larger than one standard deviation, falls within the 95% central co[illegible]nterval (Table 4.11).

Table 4.11: Binomial statistics of actual errors / estimated errors

Sub-area	method	n	probability of $n > V_e$	$n > V_e$ theor.	actual	95% central interval
S1.1	SLKGM	10	.33	3	2	0 - 6
	SBKGM	10	.33	3	2	0 - 6
S1.2	SLKGM	34	.33	11	16	6 - 16
	SBKGM	34	.33	11	10	6 - 16
S2.1	SLKGM	82	.33	27	33	19 - 35
	SBKGM	82	.33	27	23	19 - 35
S2.2	SLKGM	37	.33	12	10	7 - 17
	SBKGM	37	.33	12	11	7 - 17

At the same time, the Figures 4.47 to 4.50 show the very low variation of the estimated errors. This proves that the average estimation variance is, because of the pseudoregular data grid, a close estimate of the actual error variance.

Fig. 4.47a: Estimated Error vs Absolute Actual Error S1.1 SLKGM

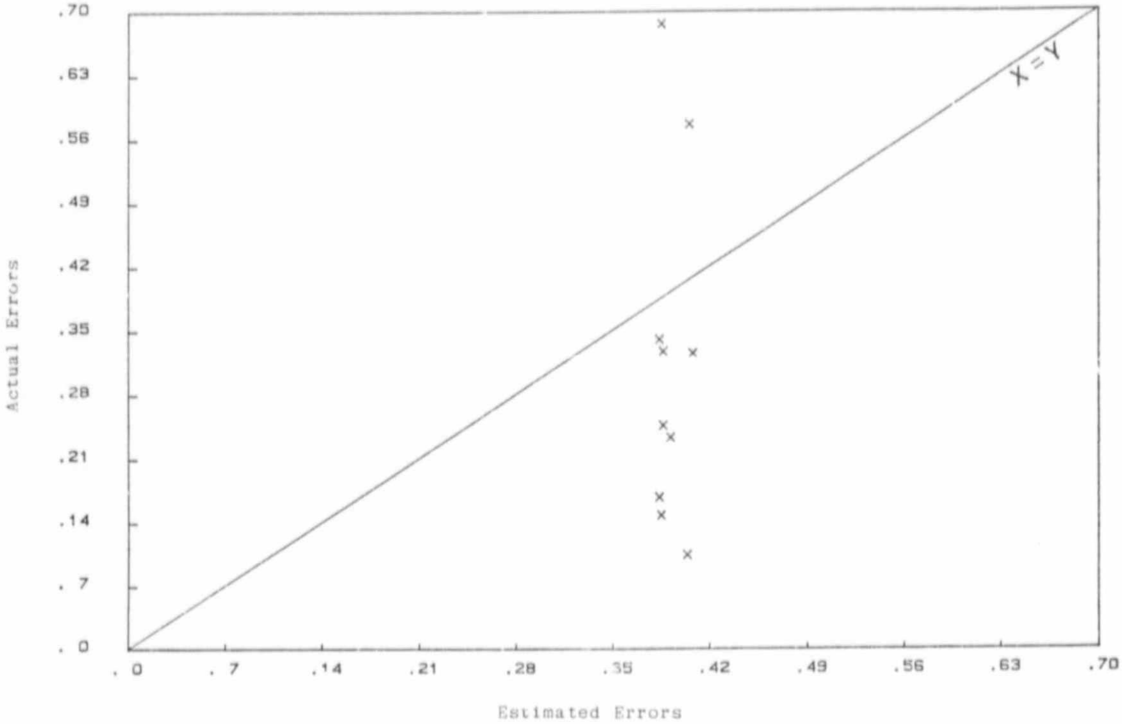


Fig.4.47b: Estimated Error vs Absolute Actual Error S1.1 SBKGM

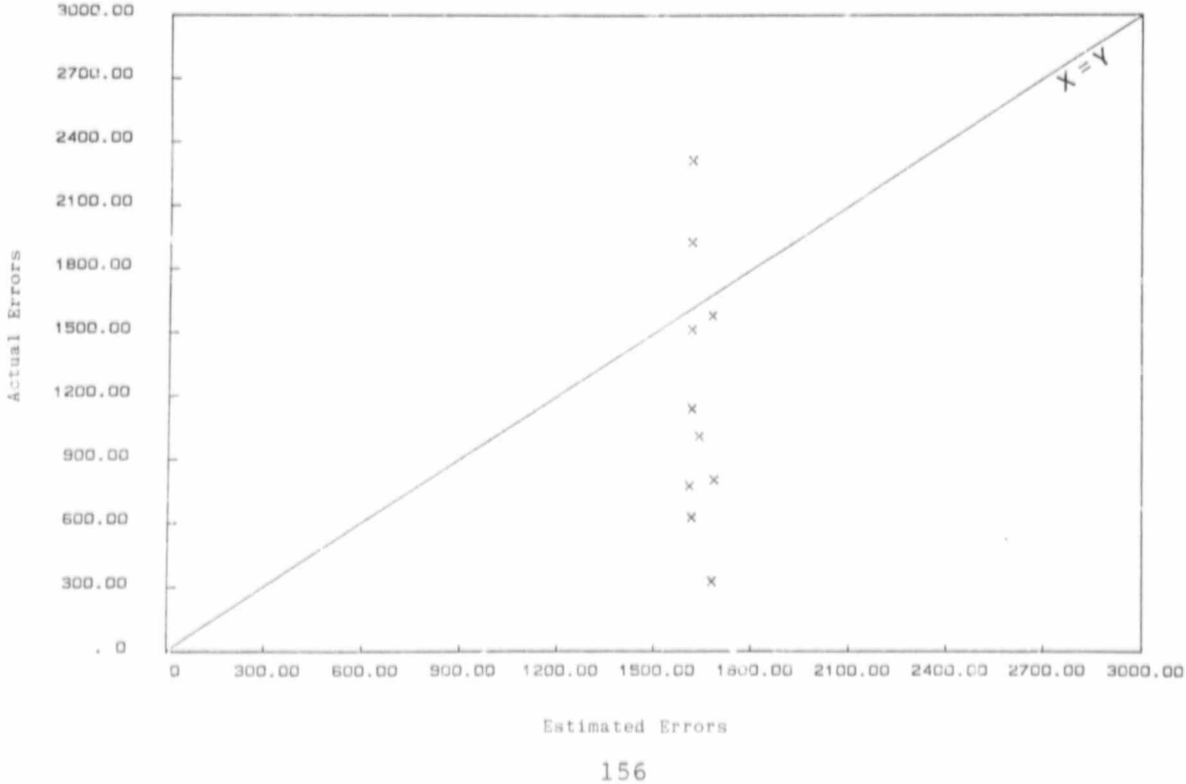


Fig. 4.48a: Estimated Error vs Absolute Actual Error S1.2 SLKGM

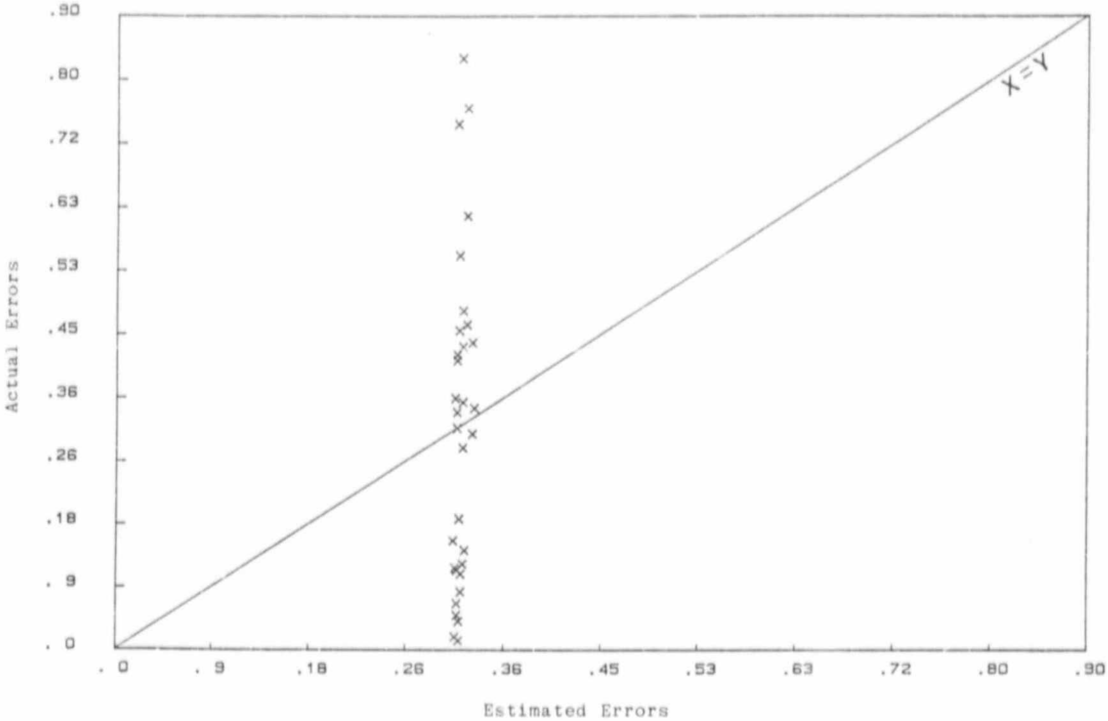


Fig.4.48b: Estimated Error vs Absolute Actual Error S1.2 SBKGM

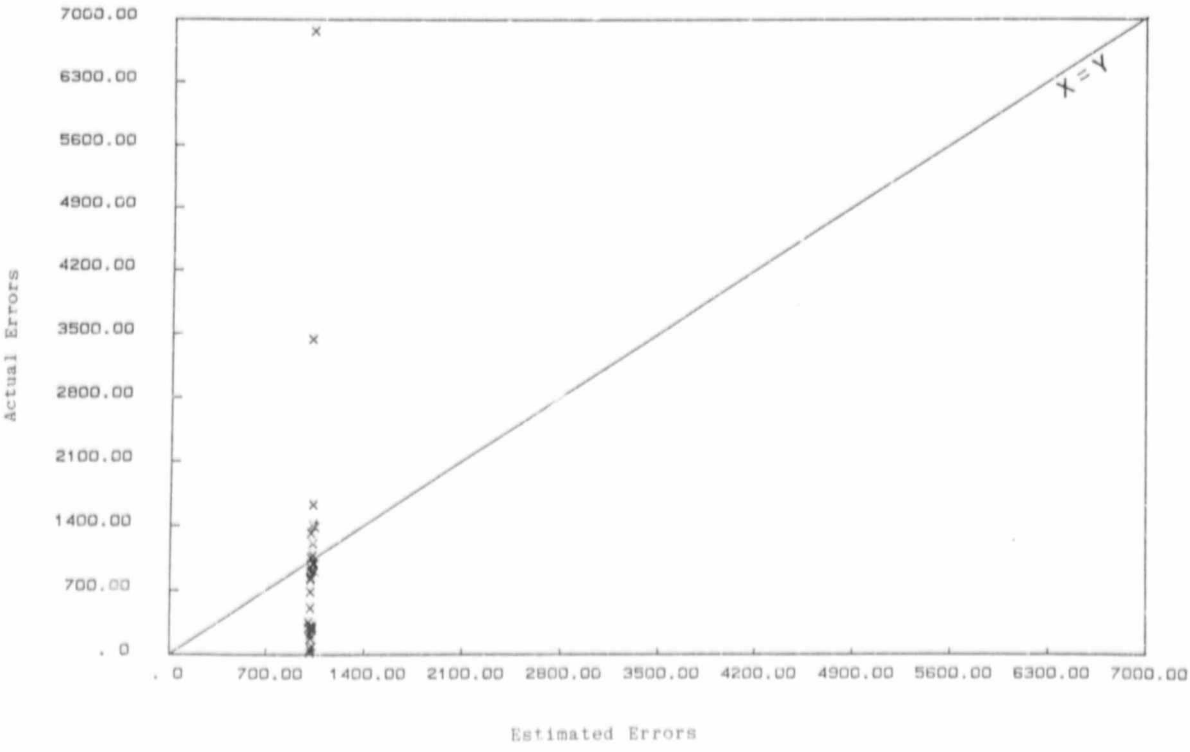


Fig. 4.49a: Estimated Error vs Absolute Actual Error S2.1 SLKGM

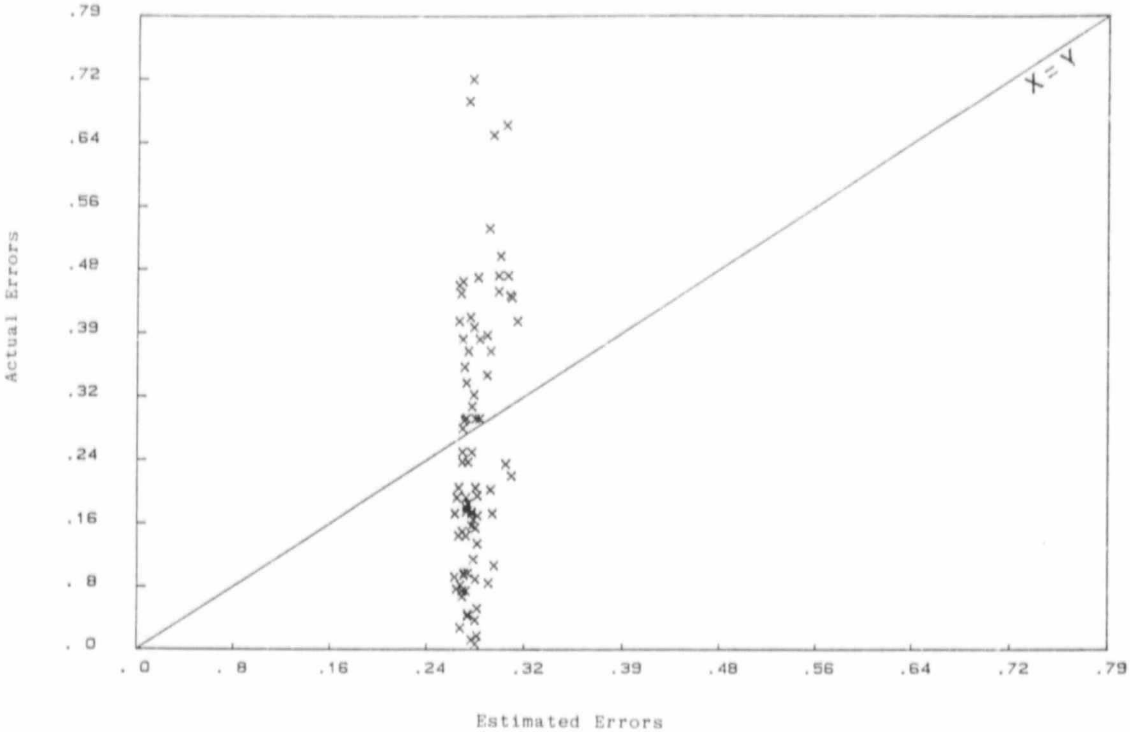


Fig.4.49b: Estimated Err r vs Absolute Actual Error S2.1 SBKGM

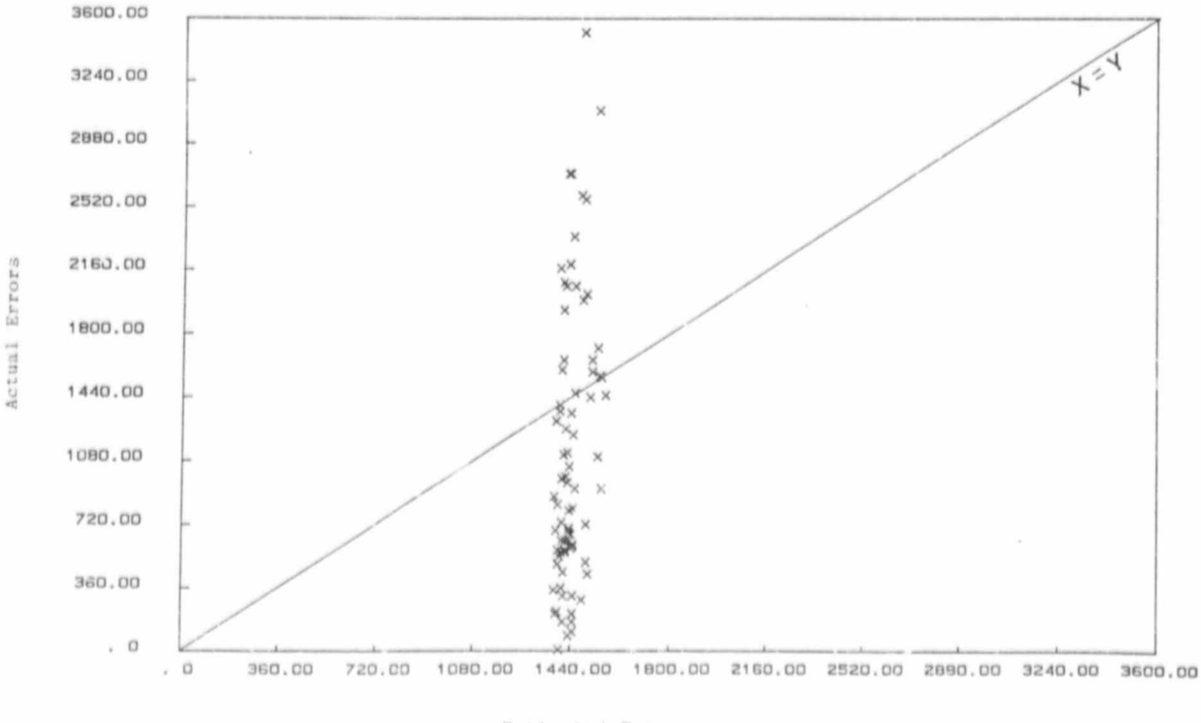


Fig. 4.50a: Estimated Error vs Absolute Actual Error S2.2 SLKGM

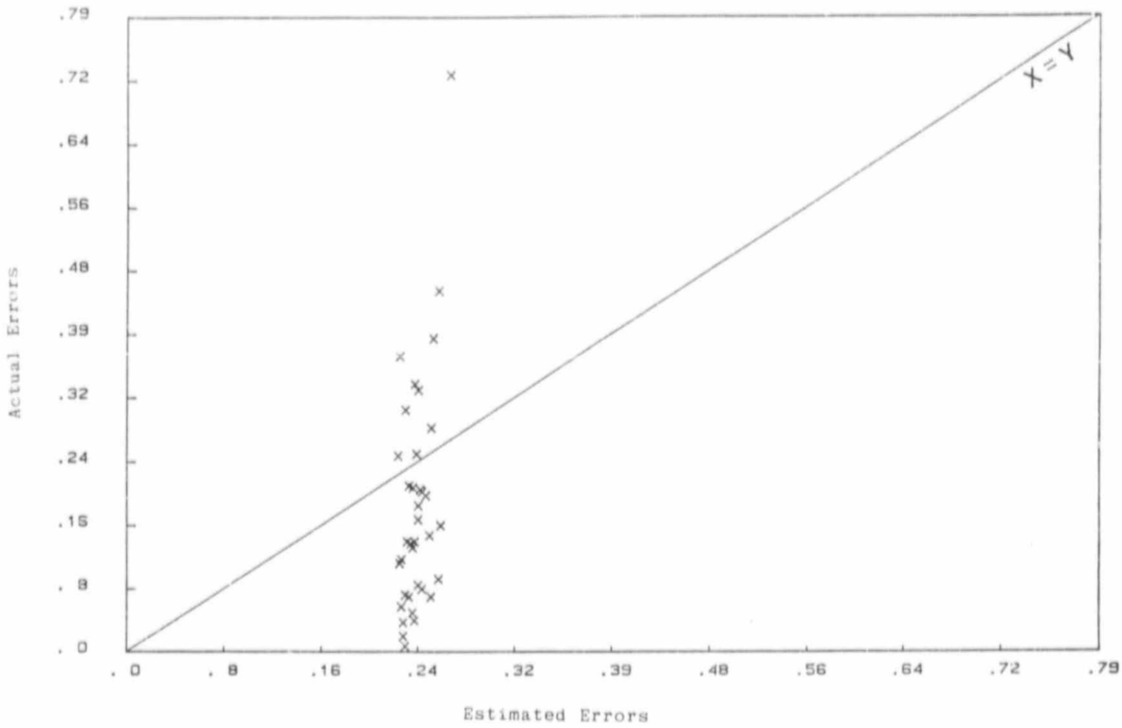
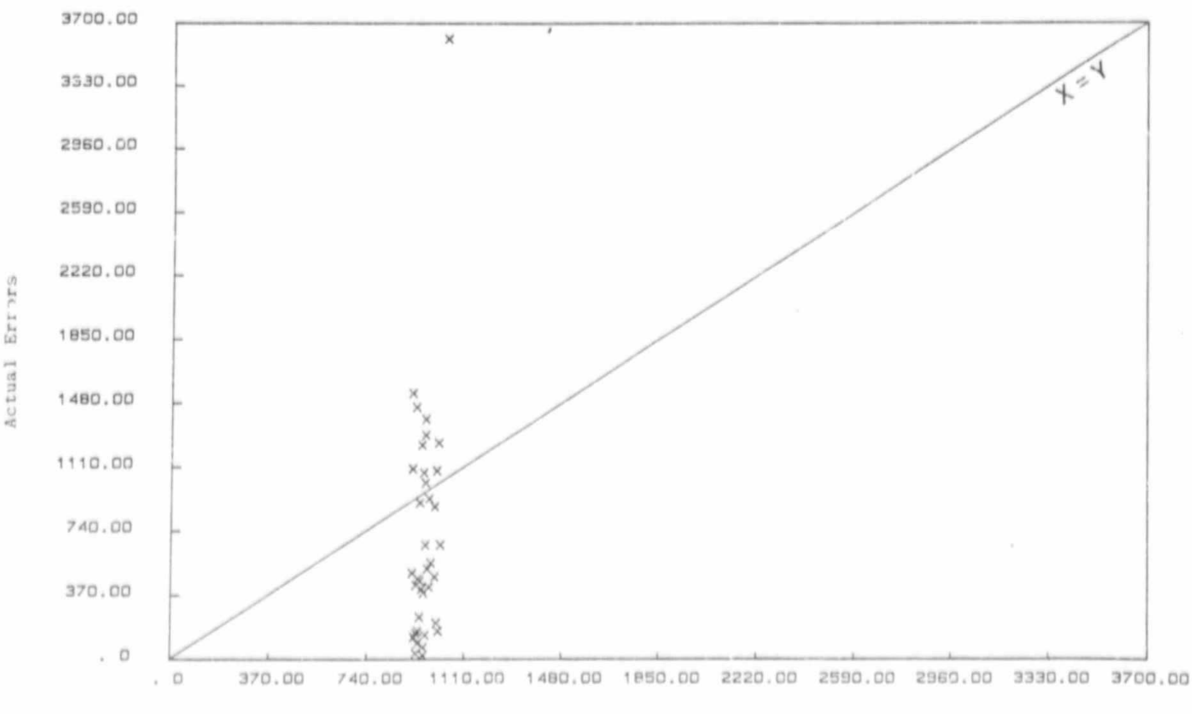


Fig. 4.50b: Estimated Error vs Absolute Actual Error S2.2 SBKGM



5. POSSIBLE IMPROVEMENTS BASED ON THEORETICAL CALCULATIONS

5.1 Ore block size and shape

5.1.1 Theoretical considerations towards the optimization of ore block size and shape.

Because of the longwall mining situation, the estimation process is based of the extrapolation of the known gold grades to ore blocks ahead of the existing face. It is obvious that the larger the distance over which extrapolation takes place, the higher the estimation err will be.

To express this numerically, the $(100m)^2$ ore block is subdivided into four sub-blocks, as shown in Figure 5.1. The logarithmic estimation variances are then calculated for each of the four sub-blocks, using local semivari grams. The results are given in Table 5.1. In order to relate the different logarithmic estimation variances, these are also expressed in percent (%) of the 'true' block variance of a block the size of (100m*25m).

It is easily seen that the estimation variance rises very quickly with increasing distance from the face, and the sub-block that is centered 37.5m ahead of the

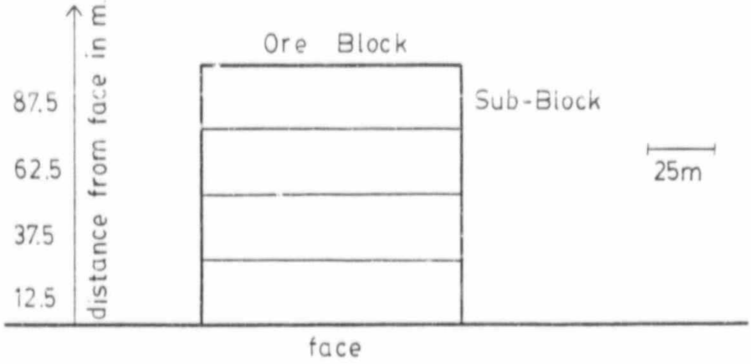


Figure 5.1: Subdivision of ore block into sub-blocks.

Table 5.1: Log. Estimation Variance of (100m * 25m) sub-blocks, estimated from face samples.

Distance from face	S1.1 Var.	%	S1.2 Var.	%	S2.1 Var.	%	S2.2 Var.	%	S2.3 Var.	%
12.5m	0.122	53	0.112	36	0.051	36	0.034	34	0.043	46
37.5m	0.221	95	0.259	84	0.100	71	0.070	70	0.075	81
62.5m	0.230	99	0.297	97	0.123	88	0.088	88	0.086	93
87.5m	0.231	100	0.305	99	0.134	95	0.095	95	0.091	98
True block variance:	0.231	100	0.307	100	0.140	100	0.100	100	0.092	100

face does already have estimation variances in the order of the 'true' block variances. This is especially true for the Subfacies 1 subareas, which, because the longwalls run subparallel to the palaeochannels, require most of the extrapolation to be done in the direction of the maximal spatial variability.

Table 5.1 expresses numerically what common sense tells: the distance of the ore blocks valued ahead of the face must be kept at minimum; and quantifies this to a maximum of two subblocks, or even less.

Western Deep Levels gold mine is planning a mining layout of 200m longwalls, separated by 44m wide pillars. Because of the rigidity of the longwall mining method at this great depth, the minimal size of a minable block parallel to the longwall is 200m, and it seems useless to estimate ore blocks smaller than that in this direction.

As mentioned earlier, the ore block size is limited into the face by the quickly rising estimation error, and a size of 50m is taken as the maximal distance in order to get reliable results. This is equivalent to an average stope advance of six months at Western Deep Levels gold mine.

The recommended new ore block size, under the given condition considered as optimal, is 200m*50m, shaped so as to be with the long side parallel to the longwall. This of course implies a block revaluation at 6 monthly intervals rather than annual imtervals as for $(100m)^2$ blocks.

5.1.2 Practical verification

In the same way as described in chapter 4.5, but now for ore blocks of (200m*50m) - with the longer side running parallel to the longwall - an oreblock estimation is simulated in the mined out subareas (see Figure 5.2). The estimation methods used are simple kriging with mean, both on an untransformed and a logtransformed basis, as these methods give the best results. Local semivariograms are used.

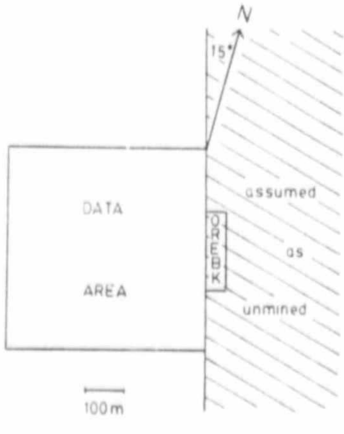


Figure 5.2: Estimation set-up.

The regression results for the individual subareas and the combined analysis are listed in Table 5.2. The actual error variances are lower than for $(100m)^2$ ore blocks estimated with local semivariograms. For the combined subareas the reduction in the error variances against $(100m)^2$ ore blocks is 29% for simple lognormal kriging with a global mean, and 41% for simple linear kriging with a global mean.

Figures 5.3a and 5.3b show the regression diagrams based on the combined data for simple linear and lognormal kriging with a global mean for an oreblock size of (200m*50m). The 90% confidence interval is reduced against the $(100m)^2$ ore blocks (see Figures 4.41, 4.42).

The correlation coefficient is 0.76 for simple lognormal kriging, and 0.74 for simple linear kriging.

The theoretical advantages of the modified ore block shape is thus obvious in practice. Overall and conditional unbiasedness are maintained, and the accuracy of the estimates is improved.

Table 5.2: Regression results of simple linear and lognormal kriging of (200m*50m) ore blocks for individual subareas, and combined subareas. The error variances of the subareas are adjusted for global bias. The global bias of the combined analysis is already zero for linear and lognormal simple kriging.

subarea	method	n	regression formula $\ln(Z + b) / Z =$	standard error of slope	corrected/ transformed error variance actual	corrected/ transformed error variance estimated
S1.1	SLKGM	14	$-5.51+1.6625\ln(Z^{*}+211)$	.4094	.13490	.14014
	SBKGM	14	$-5265+2.2610\ Z^{*}$	.5310	.39074	.38682
S1.2	SLKGM	27	$-0.04+1.0031\ln(Z^{*}+112)$	.3976	.07073	.10915
	SBKGM	27	$728+0.6600\ Z^{*}$	.5070	.05021	.08031
S2.1	SLKGM	74	$-3.10+1.3688\ln(Z^{*}+171)$	.2733	.07301	.06909
	SBKGM	74	$-2741+1.6593\ Z^{*}$	.3406	.06974	.09104
S2.2	SLKGM	35	$-2.77+1.2037\ln(Z^{*}+230)$	.3151	.04294	.05335
	SBKGM	35	$-2726+1.3377\ Z^{*}$	.3405	.05045	.06689
Combined	SLKGM	150	$-0.82+1.0986\ln(Z^{*}+b)$	.0782	.06790	
	SBKGM	150	$-883+1.2379\ Z^{*}$	.0939	.07037	

Fig.5.3a: SLKGM - With Subdivision (200m*50m)

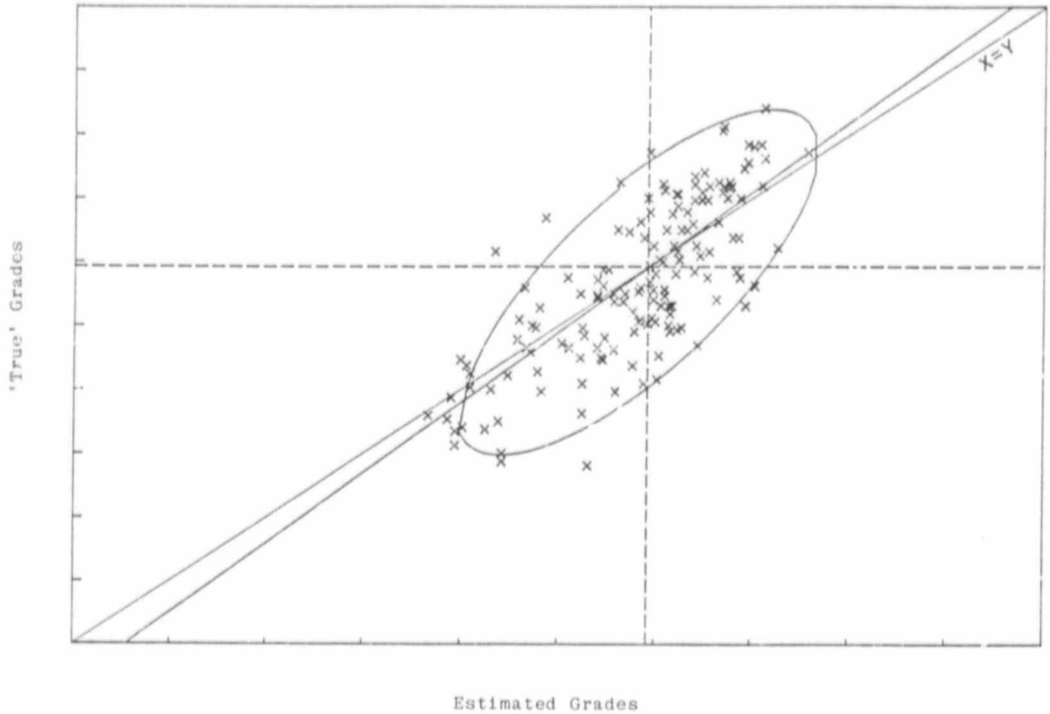
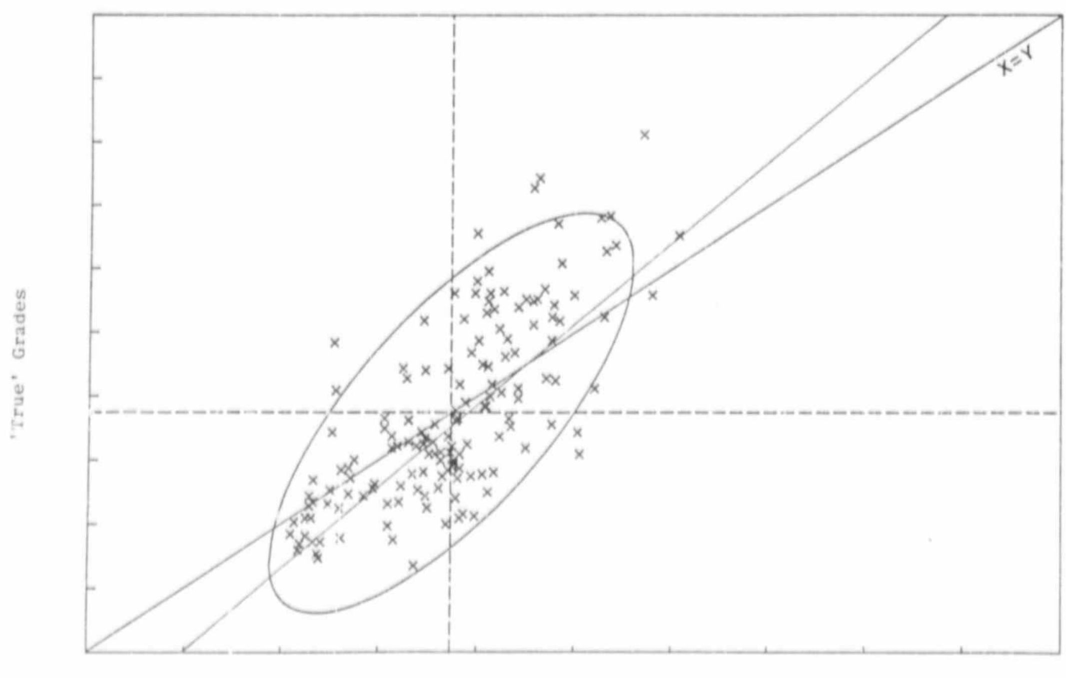


Fig.5.3b: SBKGM - With Subdivision (200m*50m)



5.2 Theoretical considerations towards the optimization of the sampling grid

This theoretical study on the effect different sample grids have on the estimation variance of ore blocks the size of (200m*50m) is based on local semivariograms. Obviously practical considerations such as ongoing grade control could also have an influence on the sampling spacing to be recommended.

The sample grid is idealized. The spacing between the samples along the face, and the spacing between the sampled faces are varied. Each face is of the length of the ore block (i.e. 200m). The first row of samples is located along the side of the ore block.

Extending the sampling spacing from the present 5m*10m grid, the estimation variance is calculated for the ore block. The increment chosen is 5m, both along each face and between faces, and the maximum intervals are 30m in both directions. The estimation procedures used are simple linear kriging with a global mean and simple lognormal kriging with a global mean. Backtransformed semivariograms are used for linear kriging.

Figures 5.4a to 5.4h give the results in the form of contour plots. The sample spacing along each face is read off the horizontal axis, and the spacing between sampled faces on the vertical axis. The theoretical estimation variance for the ore block is contoured as the variable dependent on these two sample intervals.

The estimation variance of the present sample grid (5m*10m) is therefore read off in the lower left corner. Because of the idealized setup, and because the estimation variances are not corrected for any additive constant, these estimation variances are lower than the average theoretical estimation variances listed in Tables 4.5 to 4.8.

The estimation variances for linear kriging are not transformed to logarithmic variances. Unlike logarithmic estimation variances, the untransformed estimation variances give error limits that do not depend on the estimated grade. These estimation variances are a direct measurement of the accuracy of the estimated grades in $(cmg/t)^2$, if a normal error distribution is accepted.

The standard errors of the estimates (square root of estimation variance) are generally quite high, and vary from between 700cmg/t and 1600cmg/t in the four subareas, if the present 5m*10m sampling grid is used (see Figs. 5.4a to 5.4h).

All these estimation variance contour plots show that

FIGURE 5.4a: Error variance (200m*50m) S1.1 SLKGM

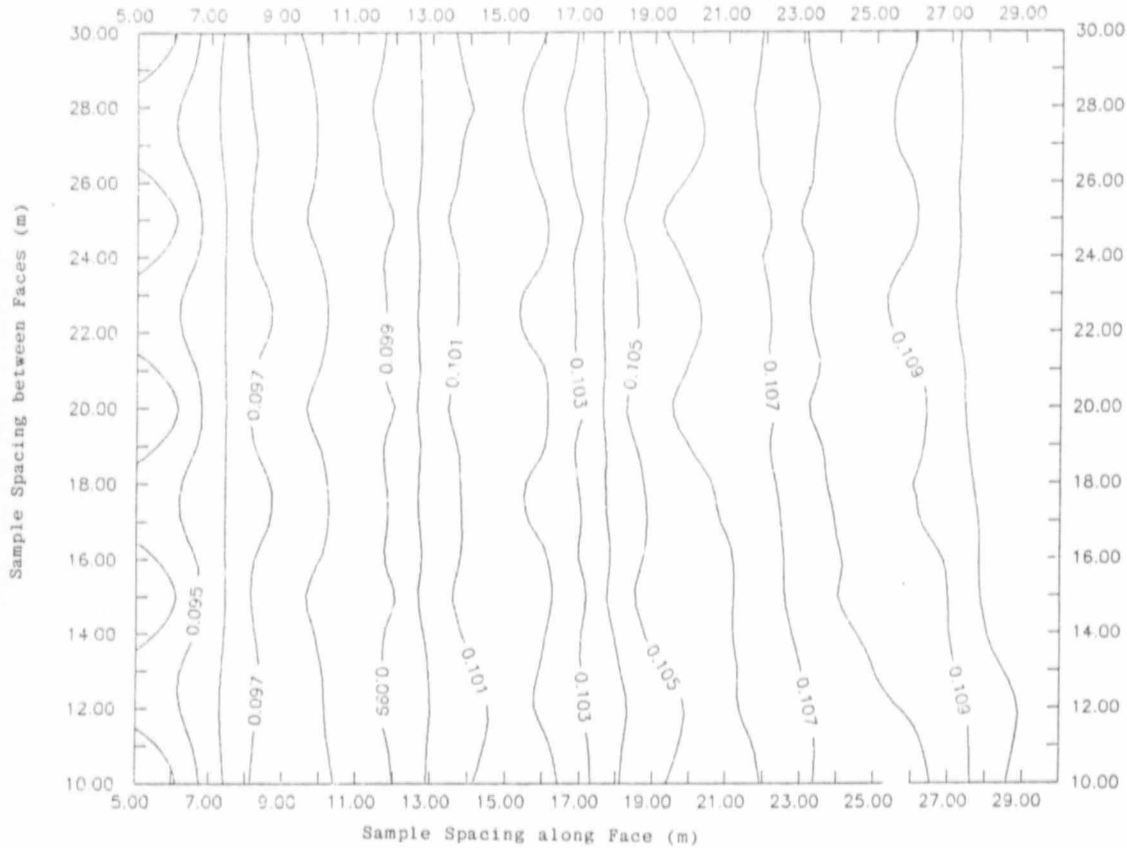


FIGURE 5.4b: Error variance (200m*50m) S1.1 SBKGM

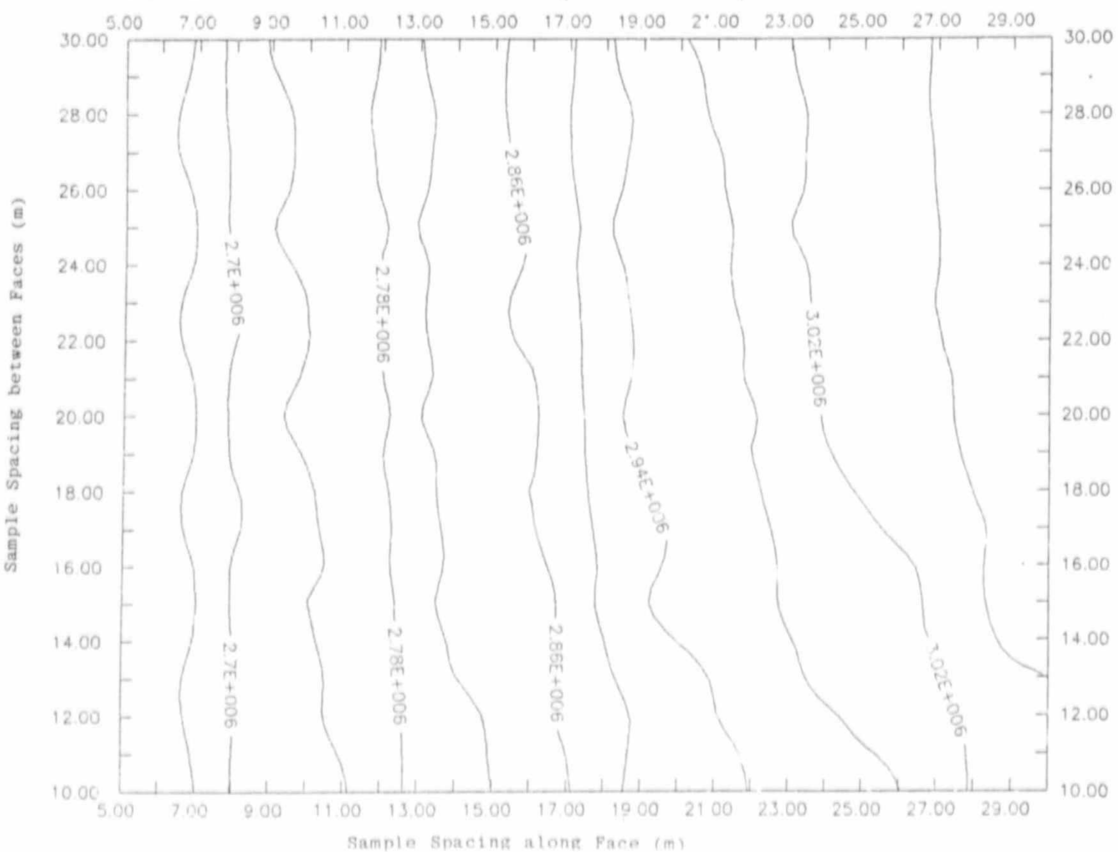


FIGURE 5.4c: Error variance (200m*50m) S1.2 SLKGM

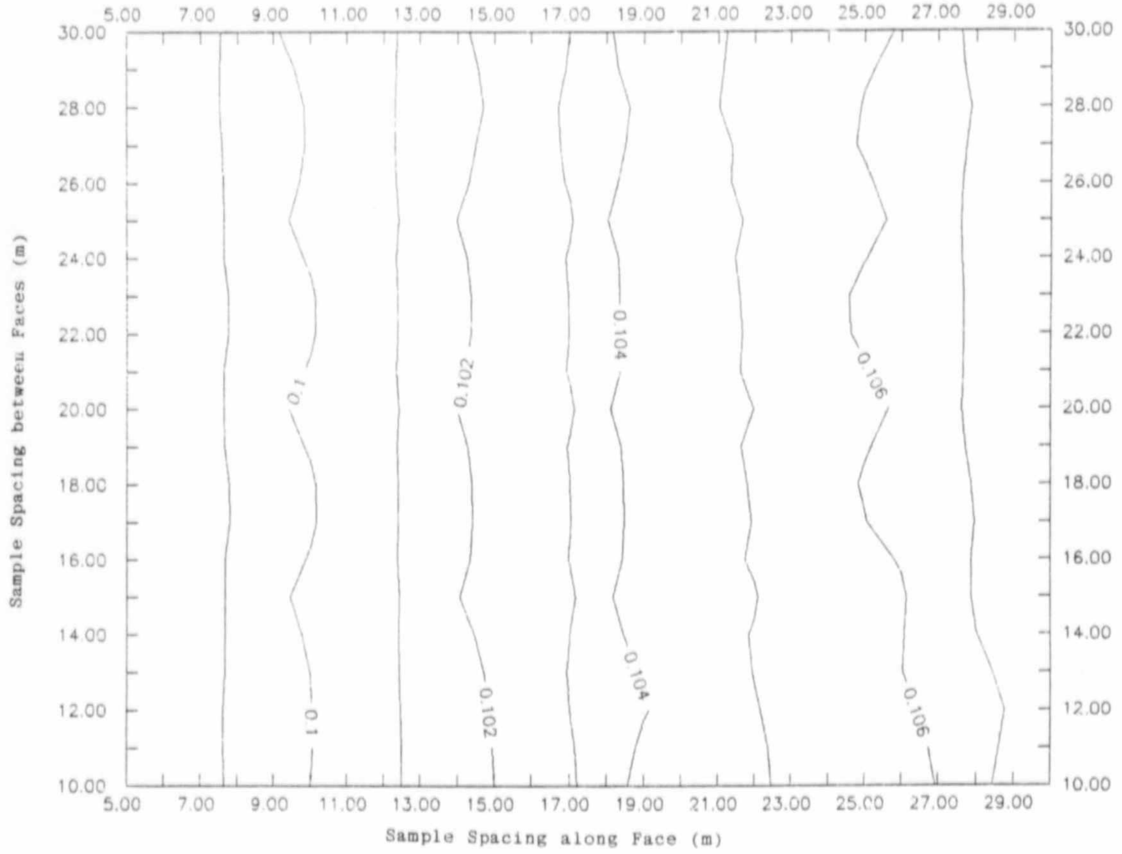


FIGURE 5.4d: Error variance (200m*50m) S1.2 SBKGM

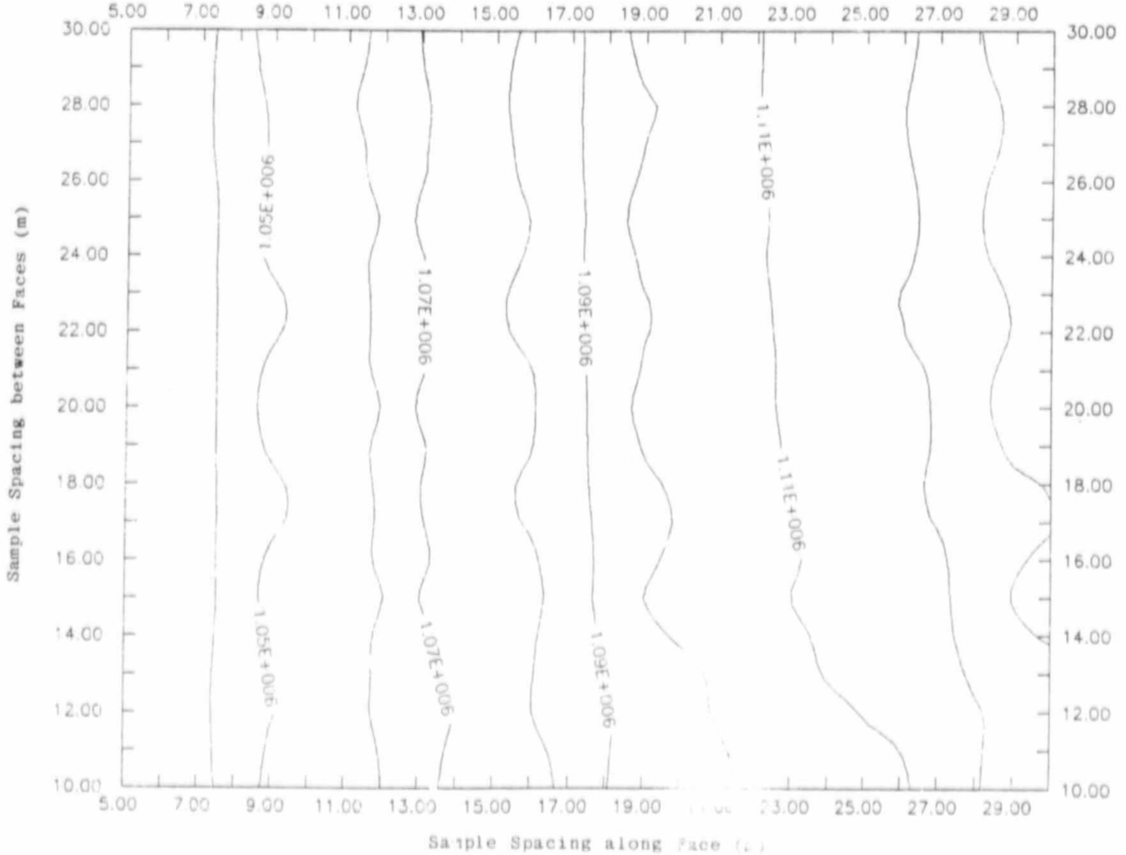


FIGURE 5.4e: Error variance (200m*50m) S2.1 SLKGM

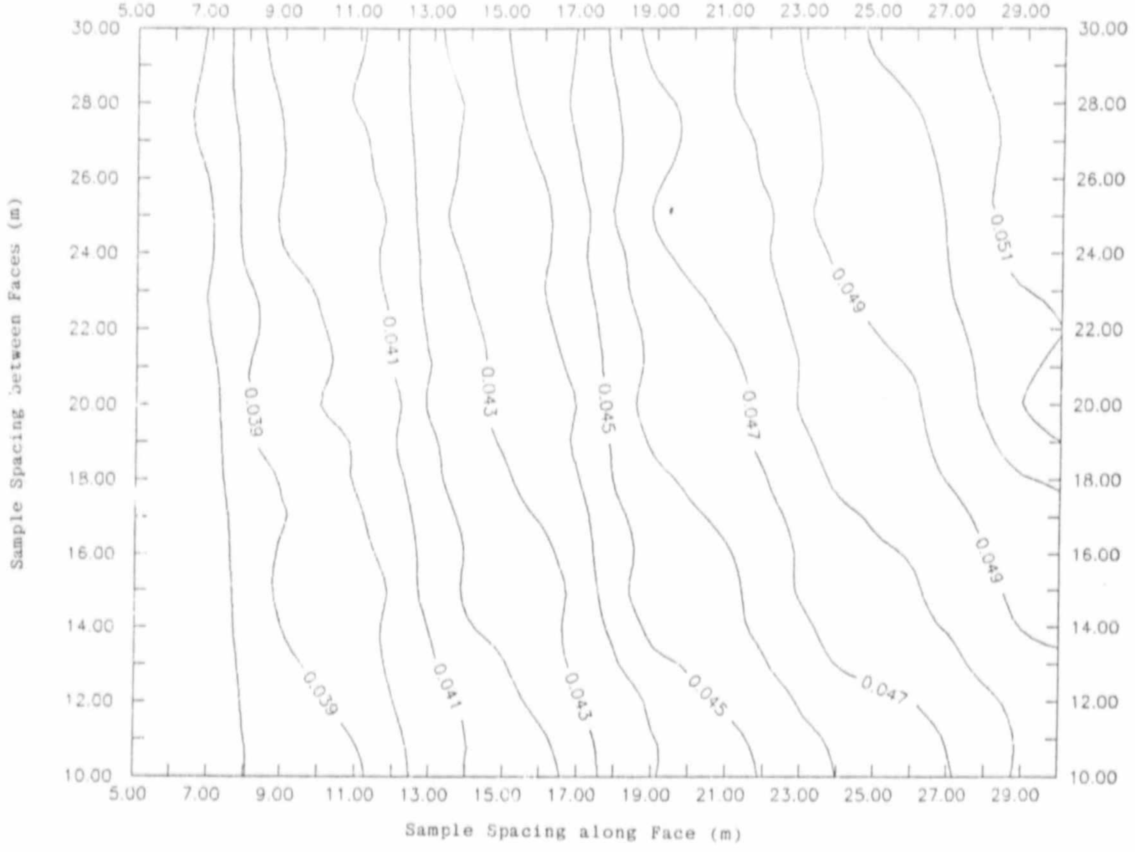


FIGURE 5.4f: Error variance (200m*50m) S2.1 SBKGM

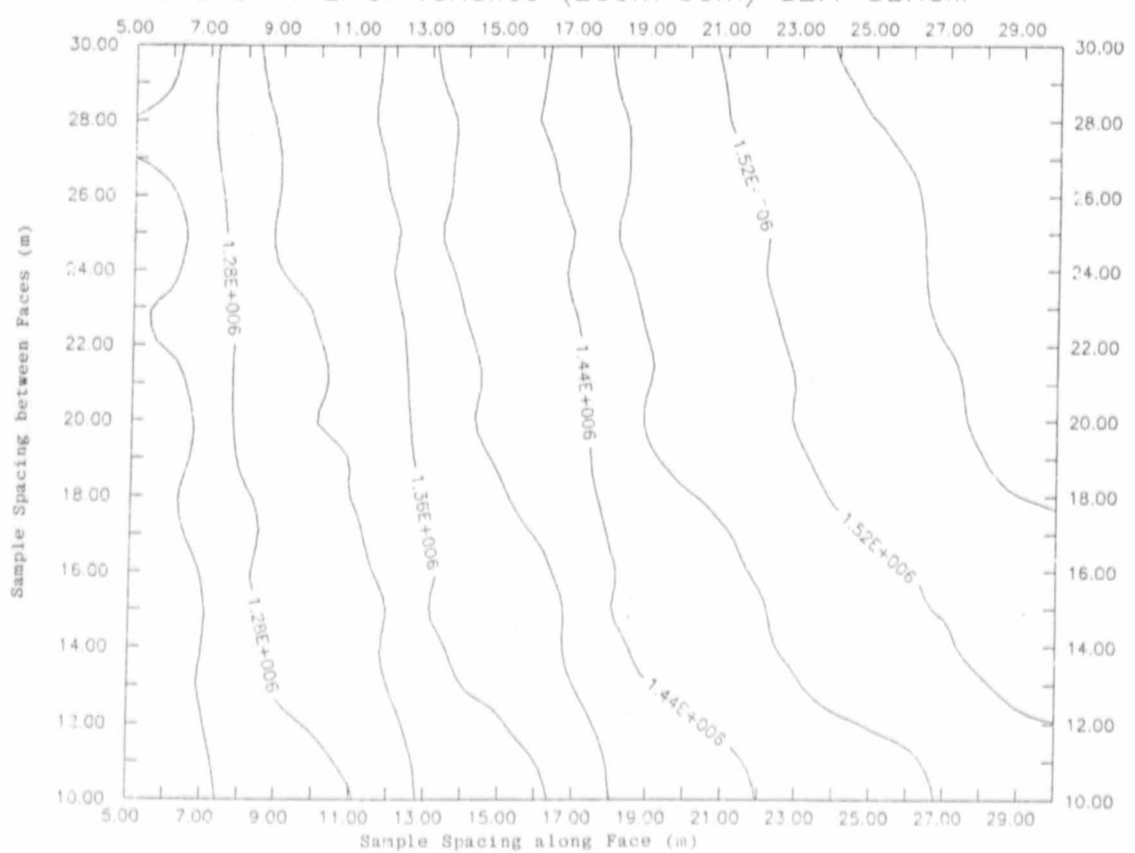


FIGURE 5.4g: Error variance (200m*50m) S2.2 SLKGM

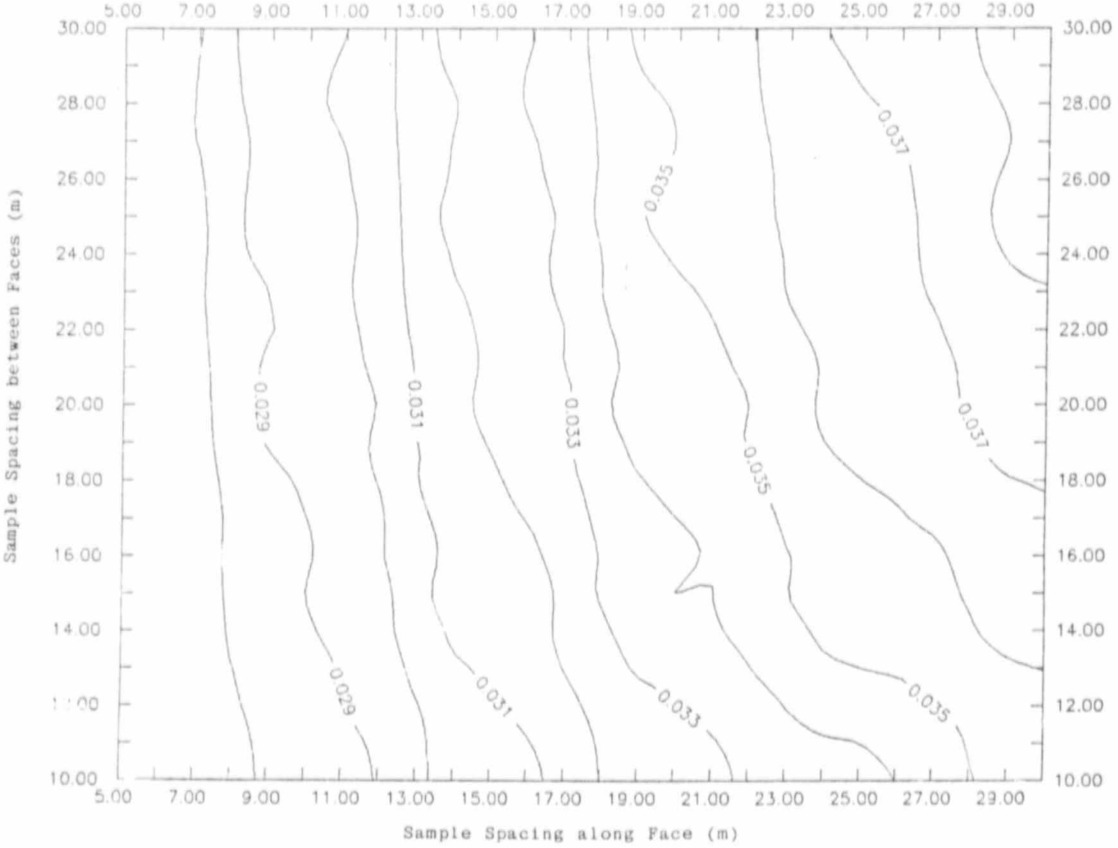


FIGURE 5.4h: Error variance (200m*50m) S2.2 SBKGM

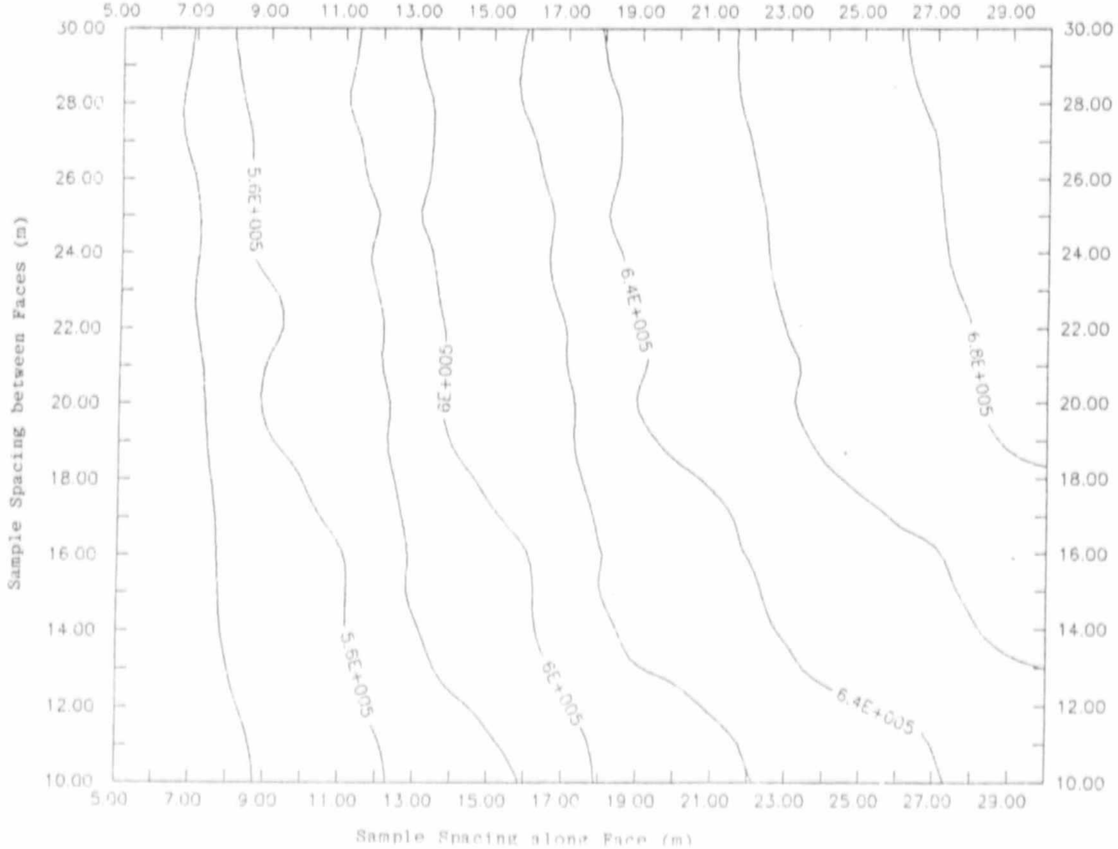


FIGURE 5.5:Number of samples required per block

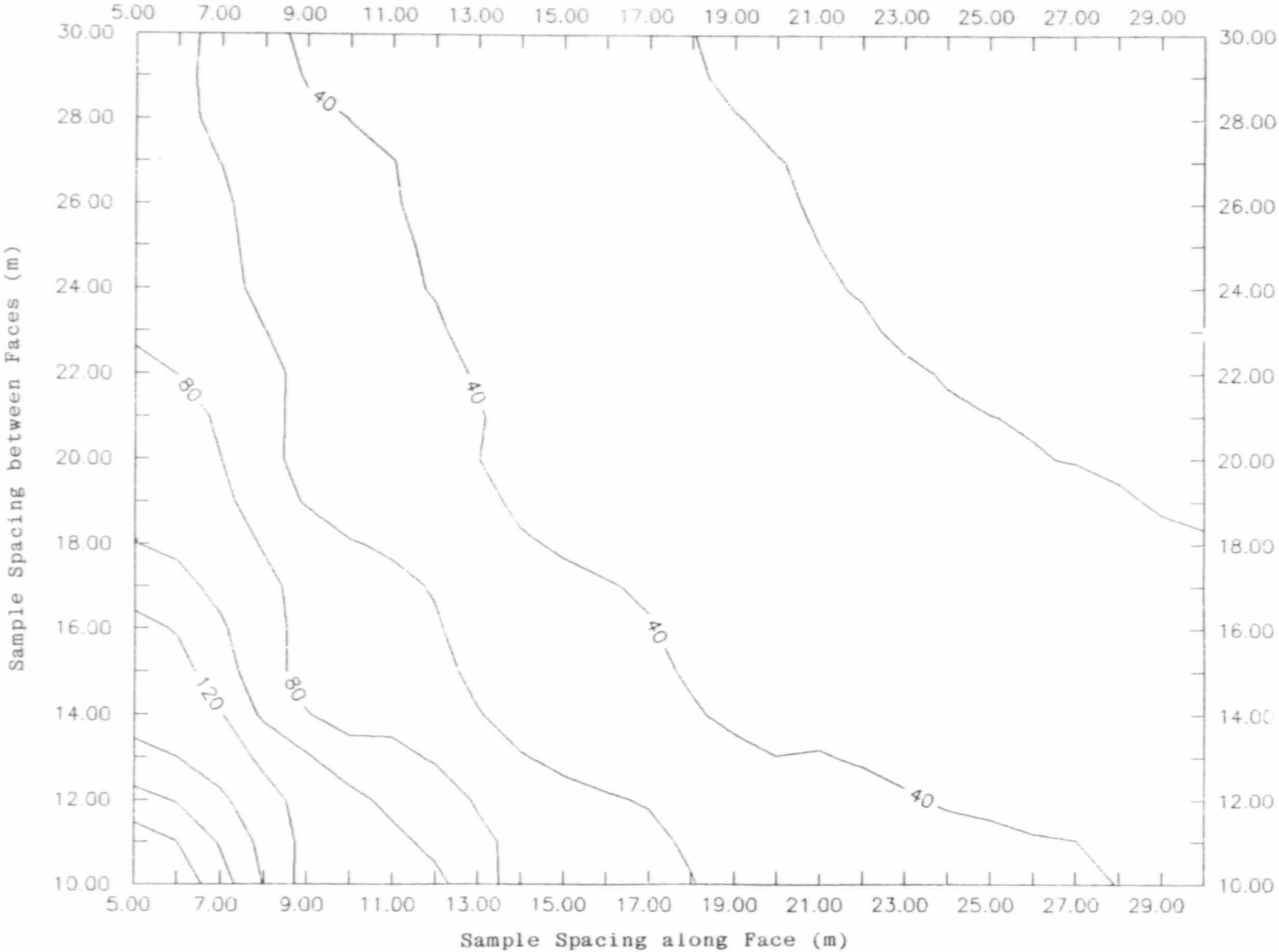
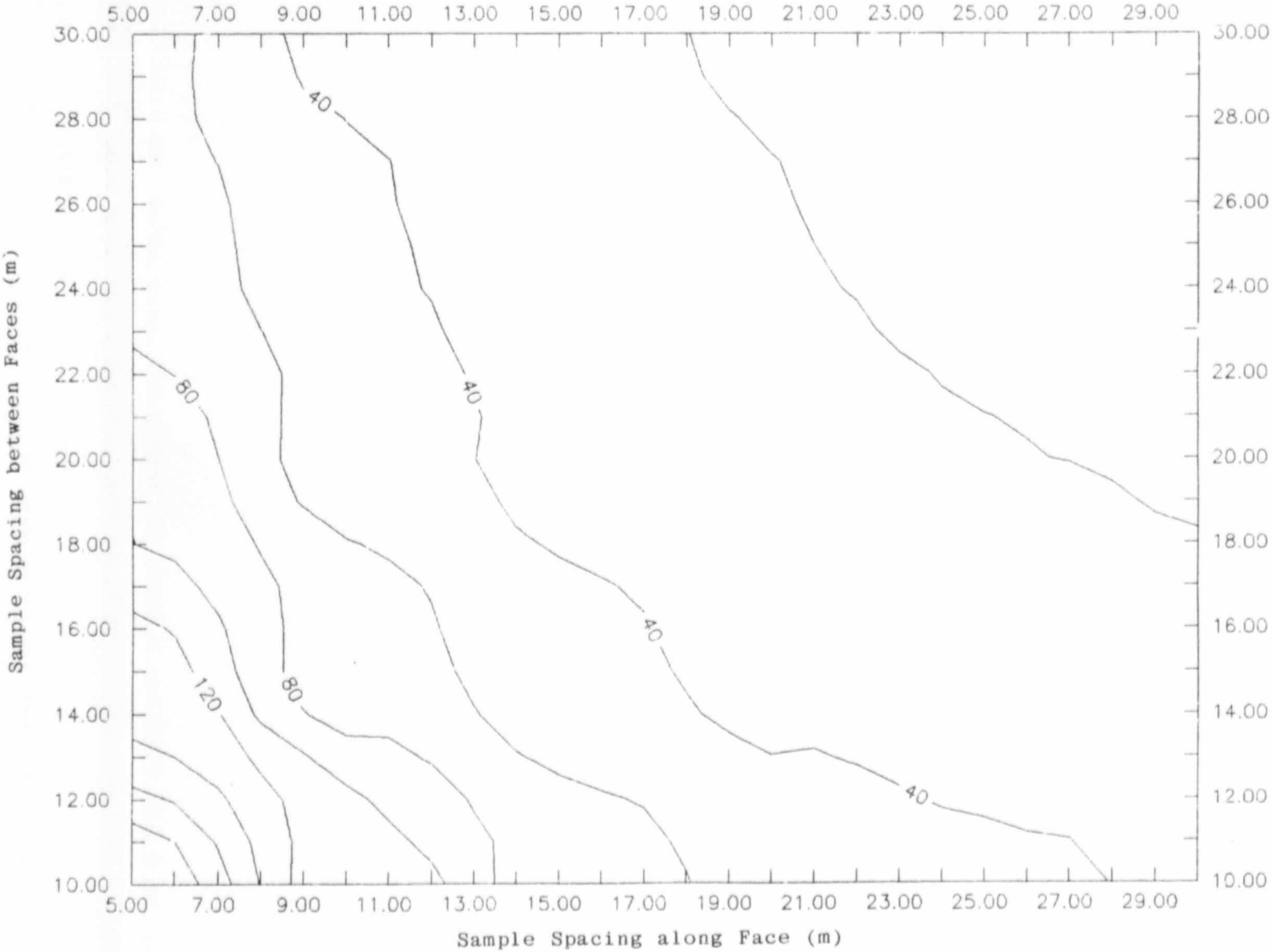


FIGURE 5.5:Number of samples required per block



the estimation variance is not very sensitive against the distance between the sampled faces. This is due to the screening of samples further away from the ore block by the row of samples nearest to the ore block. This screening is almost perfect for small sample intervals along the face, as then the estimation variance does almost not increase for wider spacing between sampled faces. For larger sample intervals along the face, the estimation variance increases faster with increasing distance between sampled faces. This shows that the screening becomes less effective as the sampling interval along the face is extended.

For the subfacies 1 subareas S1.1 and S1.2, this increase in the estimation variance is less pronounced. Here the range of the semivariograms in the main direction of the estimation (E-W) is so short that the influence of a second sampled face 10m away from the ore block side is already negligible.

This theoretical exercise shows that the sample spacing between faces can be extended considerably without much loss in the accuracy of the estimates.

In Figure 5.5, the number of samples required for a unit area (here the size and shape of an ore block of 200m*50m is used) is shown, based on different sampling grids. If a grid of 5m spacing along the face, and 30m between sampled faces were chosen, the number of samples per ore block is 67. This is a reduction in the number of samples by 66%, if compared with the 200 samples required by the present 5m*10m sample grid for the same area. The increase in the estimation variance by this reduction of the sample density is less than 1%.

The sample grids chosen so far all have their first row of samples located on one side of the ore block, so that their distance to this side of the block is zero. As there is a time delay between sampling and the actual availability of the data for the estimation, during which production proceeds, this distance of the samples to the ore block will in practice be larger than zero.

As the distance of this important nearest sampled face to the ore block increases, so does, however, the estimation variance of the ore block. Figures 5.6.a and 5.6.b show this relationship for the four subareas, both for linear and logtransformed simple kriging with a global mean. The sample grid is 5m*30m. The estimation variance increases by between 10% and 15%, if the nearest sampled face is moved only 5m away from the ore block side. This shows the sensitivity of the estimation variance to this distance.

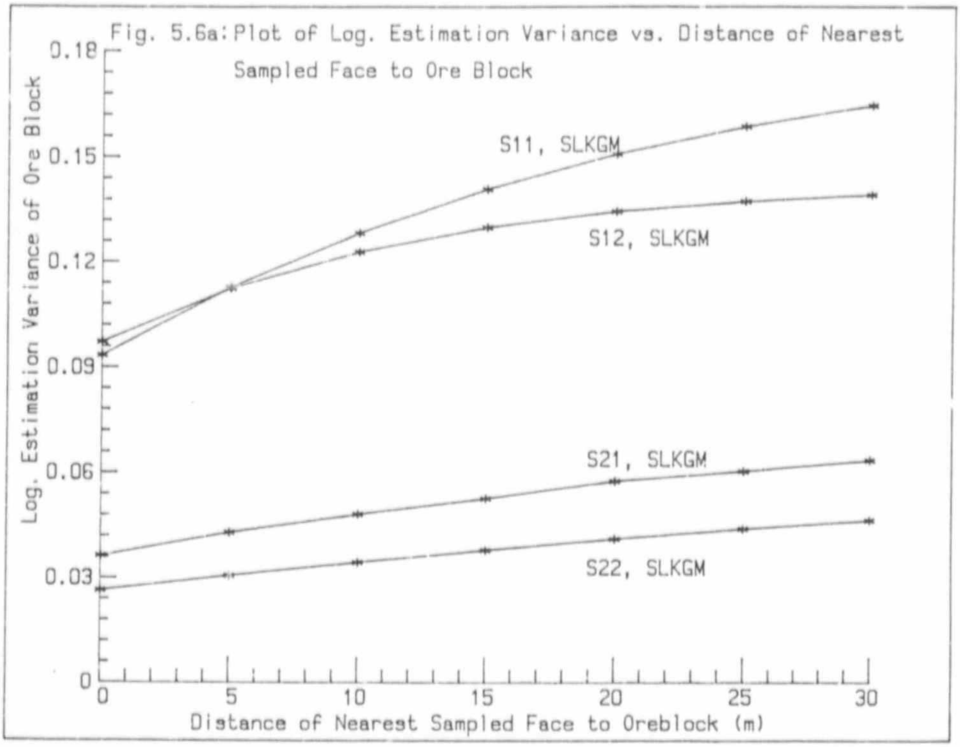


Fig. 5.6a: Plot of Log. Estimation Variance vs. Distance of Nearest
Sampled Face to Ore Block
Log. Estimation Variance of Ore Block
0.18
0.15
0.12
0.09
0.06
0.03
0
S11, SLKGM
S12, SLKGM
S21, SLKGM
S22, SLKGM
0
5
10
15
20
25
30
Distance of Nearest Sampled Face to Oreblock (m)

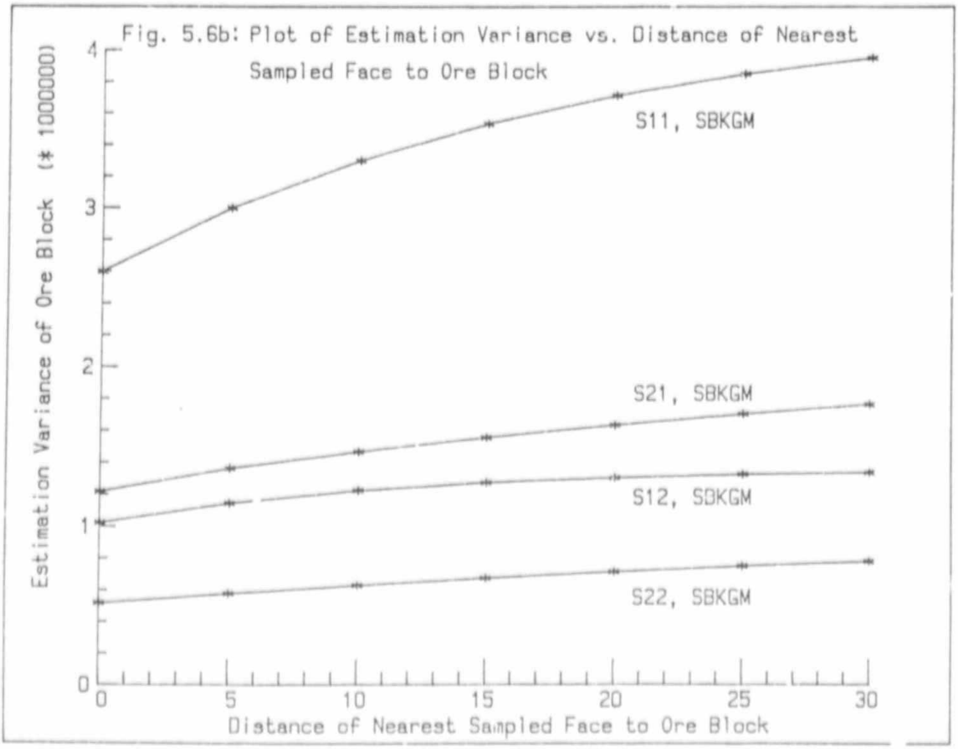


Fig. 5.6b: Plot of Estimation Variance vs. Distance of Nearest
Sampled Face to Ore Block
Estimation Variance of Ore Block (* 1000000)
4
3
2
1
0
S11, SBKGM
S21, SBKGM
S12, SBKGM
S22, SBKGM
0
5
10
15
20
25
30
Distance of Nearest Sampled Face to Ore Block

The overall conclusion is that the sample interval between faces can be increased without a significant loss in accuracy. It is neccessary, however, that whenever an ore block is to be estimated, a row of samples as close to this ore block as possible be taken and included in the estimation.

Thus if ore reserve estimation is to be done say twice a year, sampling must be as up to date as possible immediately before ore reserves are estimated.

6. CONCLUSIONS

This study quantifies the improvements which subdivision on a sedimentological basis have on the longwall ore reserve estimation of $(100m)^2$ ore blocks in the Carbon Leader reef as follows:

- if simple kriging is used, the block estimates are conditionally unbiased, no matter whether the study area is subdivided or not.

- untransformed and logtransformed kriging in general give very similar results, if a backtransformed semivariogram is used for untransformed kriging.

- simple kriging gives the lowest actual error variances. These actual error variances are reduced by maximal 28% (simple lognormal kriging), if the study area is subdivided sedimentologically before any geostatistical analysis. The standard errors, however, still remain very high, and range e.g. for linear simple kriging from between one third and one quarter of the mean grade of the subarea.

- in the case of simple lognormal kriging, the actual error variance is reduced to 56% of the error variance of the present mine estimation method.

- calculated kriging estimation variances are a valid estimate of the actual error variance, if local subarea semivariograms are used in the calculation. The overall semivariogram does not estimate the actual errors correctly. This shows that stationarity of the semivariogram is achieved by subdivision on a sedimentological basis.

Based on the valid local semivariograms, theoretical calculations show that the maximal distance ahead of the face over which the estimation is to be done, should not exceed about one third of the range of the semivariogram under the longwall mining conditions. At the same time, the ore block size should be extended parallel to the face as much as practically possible. An empirical study over the mined out area confirms this by reducing the actual error variances for (200m * 50m) ore blocks further by about 30% as compared to $(100m)^2$ blocks (same ore block size).

The effect of different sample grids on the estimation variance was studied theoretically. This investigation shows that the estimation variance in this longwall mining operation is fairly insensitive to the sample spacing between faces because of screening by the first sample row.

REFERENCES

Armstrong, R.A.; Compston, W; Retief, E.A. and Welke, H.J. (1986): 'Ages and isotopic evolution of the Ventersdorp volcanics' Extended Abstracts, Geocongress '86, Johannesburg. Geol. Soc. S. Afr. pp 89 - 92.

Armstrong, M. (1984): 'Common problems seen in variograms', Jour. Math. Geol., Vol. 16, No.3, 1984. pp 305 - 313.

Benest, J. & Winter, P.E. (1984): 'Ore-reserve estimation using geologically controlled geostatistics', Proc. 17th APCOM Symposium, Colorado School of Mines, 1982. pp 173 - 180.

Braun, R.; Friedinger, P.; Ruber, O.; Schroer, R.; Siehl, A. & Stamm, R. (1983): 'Die kartenbezogene Auswertung geologischer Daten im Dialog - Fallstudie Loccum'. Geol. Jb., **A 70**, Hannover. pp. 179 - 210.

Buck, S.G. & Minter, W.E.L., (1985): 'Placer formation by fluvial degradation of an alluvial fan sequence; the Proterozoic Carbon Leader placer, Witwatersrand Supergroup, South Africa', J. geol. Soc. London, Vol. 142, 1985. pp 757 - 764.

Clifton, H.E. (1973): 'Pebble segregation and bed lenticularity in wave-worked versus alluvial gravel', Sedimentology, (1973), **20**. pp 173 - 187.

Cressie, N. & Hawkins, D.M. (1980): 'Robust estimation of the variogram', Jour. Math. Geol., Vol. 12, No.2. 1980. pp 115 - 125.

David, M (1977): 'Geostatistical ore reserve estimation', Elsevier Scientific Publishing Company, Amsterdam Oxford New York, 1977. 364p.

David, M (1988): 'Handbook of applied advanced geostatistical ore reserve estimation', Elsevier Scientific Publishing Company, Amsterdam Oxford New York, 1988. 232p.

Davis, J.R. (1973): 'Statistics and data analysis in geology', J.Wiley & Sons, New York - Chichester Brisbane - Toronto, 1973. 550p.

De Kock, W.P. (1964): ' The geology and economic significance of the West Wits Line'. In: Haughton, S.H.(ed.): The geology of some ore deposits in Southern Africa. Geol. Soc. S. Afr., Johannesburg, pp 323 - 391.

Dowd, P.A. (1982): 'Lognormal kriging - the general case', Jour. Math. Geol., Vol. 14, No.5, 1982. pp 475 - 499.

Graybill, F.A. (1961): 'An introduction to linear statistical models', Vol.1, McGraw-Hill Book Co., Inc., New York Toronto London, 1961.

Hallbauer, D.K. (1975): 'Plant origin of Witwatersrand Carbon'. Miner. S i. Eng., 7, pp 111 - 113.

James, C.S. (1984): 'Nu ric modelling of gold transport and deposition', Unpublished PhD-Thesis, Facu ty of Engineering, University of the Witwatersrand, Johannesburg, 1984. 257p.

Janisch, P.R. (1986): 'Gold in South Africa', Jour. S. Afr. Inst. Min. Metall., Vol 86, No.8, 1986. pp 273 - 316.

Journel, A.G. & Huijbregts, Ch.J. (1978): 'Mining Geostatistics', Academic Press Inc., London Orlando San Diego New York Toronto Montreal Sydney Tokyo,1978.600p.

Komar, P.D. & Wang, C (1984): 'Process of selective grain transport and the formation of placers on beaches'. Jour. Geol., 92, pp 637 - 655.

Krahmann, R.(1936): ' The geophysical magnetometric investigations on West Witwatersrand areas between Randfontein and Potchefstroom, Transvaal'. Trans. Geol. Soc. S. Afr., 39, pp 1 - 44.

Krige, D.G. (1951): 'A statistical approach to some basic mine valuation problems on the Witwatersrand'. J.Chem. Metall. Min. Soc. S. Afr., December 1951. pp 118 - 139.

Krige, D.G. (1960): 'On the departure of ore value distributions from the lognormal model in South African gold mines'. J.S.Afr.Inst.Min.Metall.,61. pp 231 - 244.

Krige, D.G., Watson, M.I., Oberholzer, W.J. & duToit, S.R.(1969): 'The use of contour surfaces as predictive models for ore values'. In: A Decade of Digital Computing in the Mineral Industry, edit. A. Weiss, AIME New York. pp 127 - 161.

Krige, D.G. (1981): 'Lognormal de-Wijsian geostatistics for ore reserve evaluation', S. Afr. Inst. Min. Metall., 1981, Monograph Series, Geostatistics 1. 51p.

Krige, D.G. & Magri, E.J. (1982): 'Studies of the effect of outliers and data transformation on variogram estimates for a base metal and a gold orebody', Jour. Math. Geol., Vol.14, No.6, 1982. pp. 557 - 564.

Lawlor, M. (1985): 'Some recent development in underground gold mining practice in South Africa', 83rd Anniversary Magazine, Western Australia School of Mines, Kalgoorlie, 1985. pp 74 - 84.

Magri, E.J. (1978): 'A comparison between geostatistical analyses and sedimentological studies at the Hartebeestfontein gold mine', S. Afr. Inst. Min. Metall., Vol 78, No.3. pp 218 - 223.

Matheron, G. (1963): 'Principles of Geostatistics', Econ. Geol., **58**, 1963. pp 1246 - 1266.

Nami, M. (1983): Gold distribution in relation to depositional processes in the Proterozoic Carbon Leader placer, Witwatersrand, South Africa', Spec. Publs. Int. Ass. Sediment. **6**, 1983. pp 563 - 575.

Oberthür, T. (1983): 'Metallogenetische Ueberlegungen zur Bildung des Carbon Leader Reef', Unpublished PhD -Thesis, Faculty of Science, University of Cologne, 1983. 214p.

Pretorius, D.A. (1981): 'Gold and uranium in quartz-pebble conglomerates'. Econ. Geol., 75th Anniversary Volume, pp 117 - 138.

Reineck & Singh (1980): 'Depositional Sedimentary Environments'. Springer Verlag, Berlin - Heidelberg - New York. 549p.

Rendu, J.M. (1978): 'An introduction to geostatistical methods of mineral evaluation', S. Afr. Inst. Min. Metall. Monograph Series, Geostatistics 2, 1978. 84p.

Rendu, J.M. (1979a): 'Kriging, lognormal kriging, and conditional expectation: comparison of theory with actual results', Proc. 16th APCOM Symposium, 1979. pp 199 - 212

Rendu, J.M. (1979b): Normal and lognormal estimation', Jour. Math. Geol., Vol. 11, No.4, 1979. pp 407 - 422.

Rendu, J.M. & Readdy, L. (1982): 'Geology and the semivariogram - a chritical relationship', Proc. 17th APCOM Symposium, Colorado School of Mines, 1982. pp 771 - 783.

Rendu, J.M. (1984): 'Geostatistical modelling and geological controls', Proc. 18th APCOM Symposium, Imperial College, London. pp. 166 - 172.

Reynolds, A.J & Stear, W.M. (1987): 'The role of computer - aided sedimentological synthesis in strategic mine planning at Blyvooruitzicht gold mining Co. Ltd.' Proc. 20th APCOM symposium, vol. 3: Geostatistics. Johannesburg, SAIMM, 1987. pp. 137 - 146.

Rivoirard, J. (1987) 'Two key parameters when choosing the kriging neighborhood'. Jour. Math. Geol., Vol. 19, No.8, 1987. pp 851 - 856.

Slingerland, R.S. & Smith, N.D. (1986): 'Occurence and formation of water-laid placers'. Ann. Rev. Earth Planet. Sci., 14, pp 113 - 147.

Storrar, C.D. (1977): 'South African Mine Valuation'. Chamber of Mines of South Africa, Johannesburg. 472p.

Verrezen, L. (1986): 'The facies technique of using sedimentology in valuation: a case study on the Vaal Reef palaeoplacer'. Geocongress '86 abstracts, Geol. Soc. S. Afr., Johannesburg. pp. 487 - 490.

Verrezen, L. (1987): 'Sedimentology of the Vaal Reef Palaeoplacer in the Western Portion of the Klerksdorp Goldfield, South Africa'. Unpubl. MSc. - thesis, Rand Afrikaans University, Johannesburg. 194p.

APPENDIX 1: METHODS AND TECHNIQUES USED

A1.1: Determination of the minimum number of samples for regularized data blocks and 'true' blocks

At Western Deep Levels Gold Mine the samples are supposed to be taken on a pseudoregular grid. The number of sample points per unit area, however, can vary considerably due to occurence of dykes and fault losses (which are not sampled), irregular stope advance, blocks on the border of mined out areas etc. In this investigation the position was exaggerated due to missing assay sheets.

The procedure of regularization is to take all data inside a block, and assign their mean grade (here the backtransformed mean based on the lognormal model is used) to the center of the block as the average grade of this block. For a given block size this estimated average grade is closer to its true grade, the more samples there are inside the block.

A way had to be found to choose the minimum number of samples blocks of a given size had to contain to allow a reasonably accurate estimation of their true grade.

This number of samples can vary from 1 to, theoretically, the area of the block divided by the area of a sample, which is almost infinite. The variance of block values for blocks containing only 1 sample is equal to the variance of the samples. At the other extreme, the variance of blocks containing an infinite number of samples is equal to the variance of the true block grades, which can be read off the variance - size of area graph (Krige, 1981).

To find the minimum number of samples a block must contain to give a reasonable estimate of its mean, the $(50m)^2$ data blocks were grouped according to the number of samples they contain, and the variance was calculated for each group. This variance was then plotted against the sample number. For higher sample numbers the class intervals were extended from one to five and ten, as the number of blocks containing many samples is limited.

The variance should lie between the variance of the samples and the variance of the true data blocks, and should decrease with increasing sample number. It was found that the decrease is fast for the lower sample numbers and then gradually approaches the variance of the true blocks. The point beyond which changes in the slope become gradual was taken as the minimum number of samples a block of that size must contain to be used as a data block in the estimation procedure as described in chapter 4.5.2.

Figure A1.1 shows the variance - number of samples graph, by which the minimum number of samples a data block had to contain to be included in the analysis as described under 4.5.2, was determined.

A steep decrease in the logarithmic block variance can be noted from n=2 up to n=6 samples per block (no variance can be calculated for n=1). For blocks containing seven and more samples, the logarithmic variance then decreases slowly, and eventually approaches the 'true' logarithmic block variance of 0.21. Only blocks containing seven and more samples are therefore used in the analysis.

An ore block of $(100m)^2$ for which a 'true' grade is required for comparison with estimates in a regression analysis, has to contain more samples than a data block. The minimum number of samples was read off in this case where the variance becomes virtually equal to the true block variance.

Figures A1.2 to A1.6 show the variance - number of samples plots for the shaft area, and for the subareas S1.1, S1.2, S2.1 and S2.2, for $(100m)^2$ ore blocks.

For the shaft area and the subfacies 2 subareas S2.1 and S2.2, blocks containing 40 or more samples are used as 'true' blocks in the regression analysis, and for the subfacies 1 subareas S1.1 and S1.2 blocks containing more than 30 samples. The lower number of samples required for subfacies 1 blocks is probably due to the higher continuity of the mineralisation in north - south direction in this subfacies.

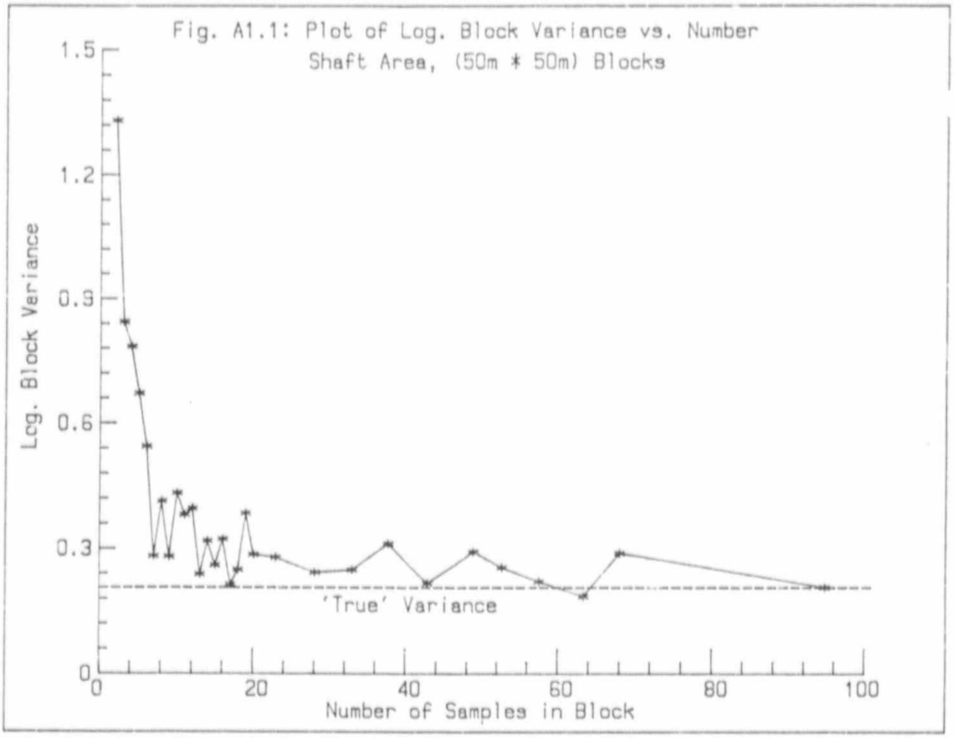


Fig. A1.1: Plot of Log. Block Variance vs. Number
Shaft Area, (50m * 50m) Blocks
Log. Block Variance
'True' Variance
Number of Samples in Block
1.5
1.2
0.9
0.6
0.3
0
20
40
60
80
100

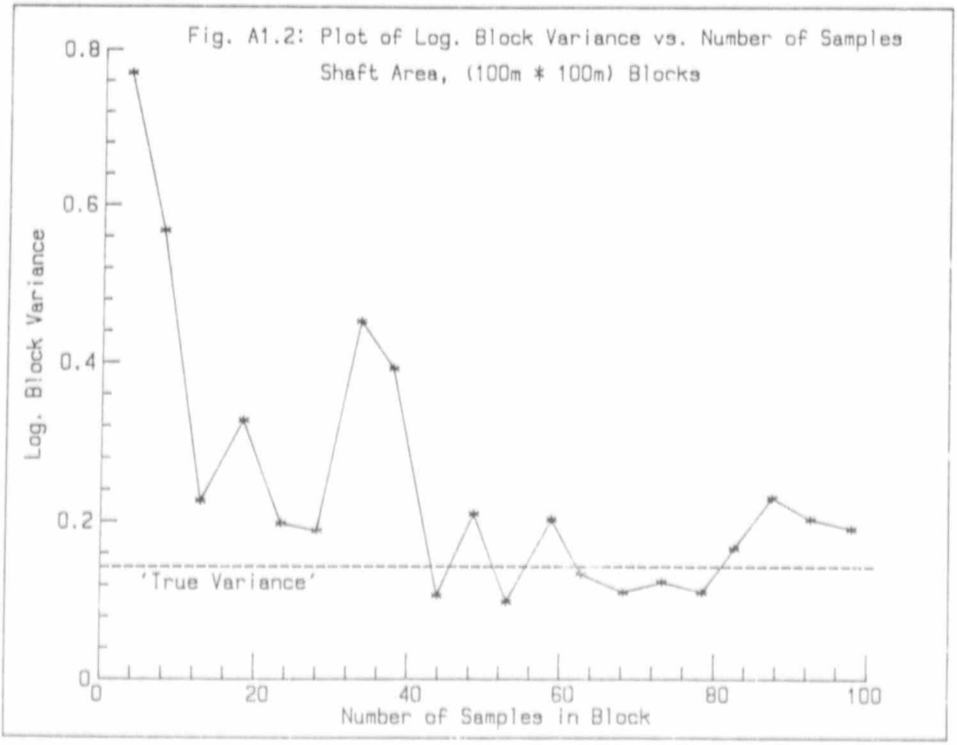


Fig. A1.2: Plot of Log. Block Variance vs. Number of Samples
Shaft Area, (100m * 100m) Blocks
Log. Block Variance
'True Variance'
Number of Samples in Block
0.8
0.6
0.4
0.2
0
20
40
60
80
100

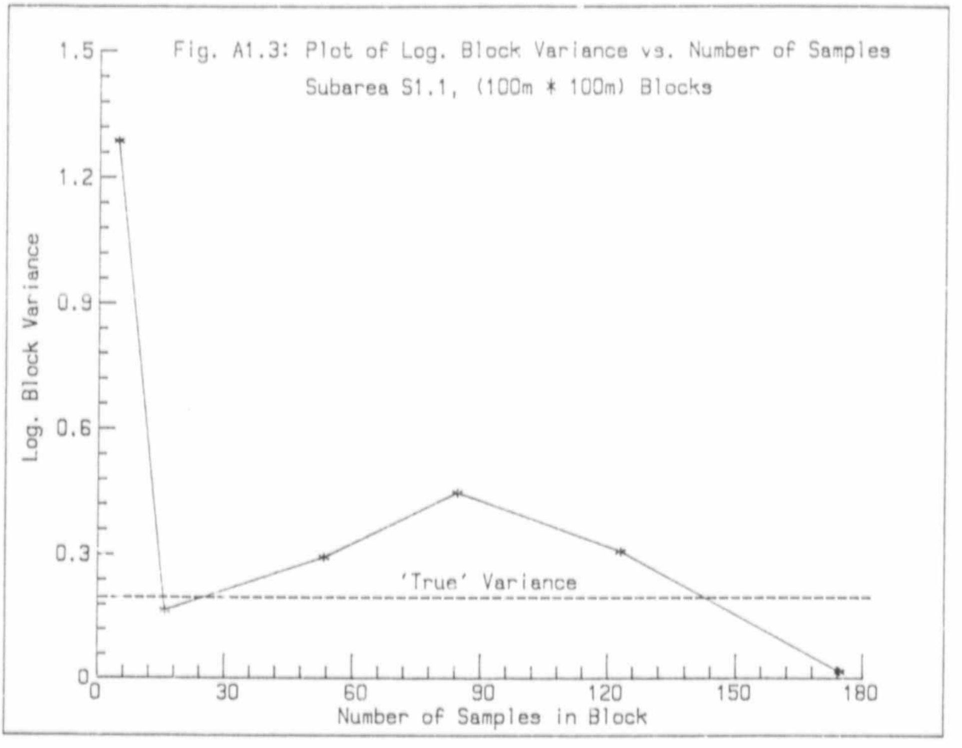


Fig. A1.3: Plot of Log. Block Variance vs. Number of Samples
Subarea S1.1, (100m * 100m) Blocks
Log. Block Variance
'True' Variance
Number of Samples in Block
1.5
1.2
0.9
0.6
0.3
0
0
30
60
90
120
150
180

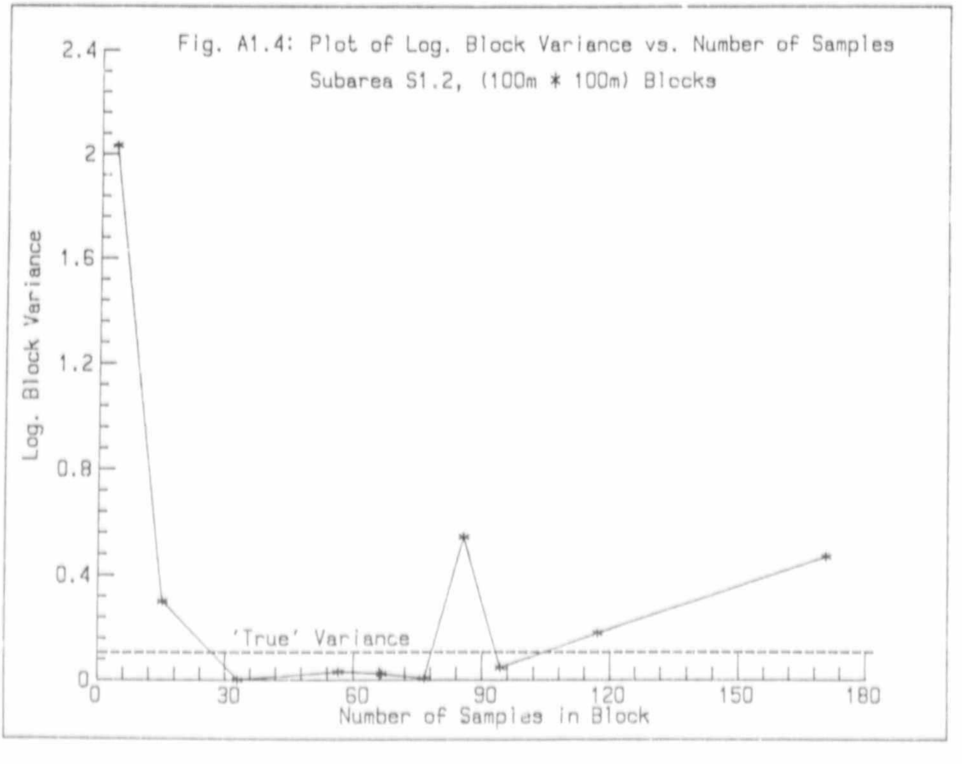


Fig. A1.4: Plot of Log. Block Variance vs. Number of Samples
Subarea S1.2, (100m * 100m) Blocks
Log. Block Variance
'True' Variance
Number of Samples in Block
2.4
2
1.6
1.2
0.8
0.4
0
0
30
60
90
120
150
180

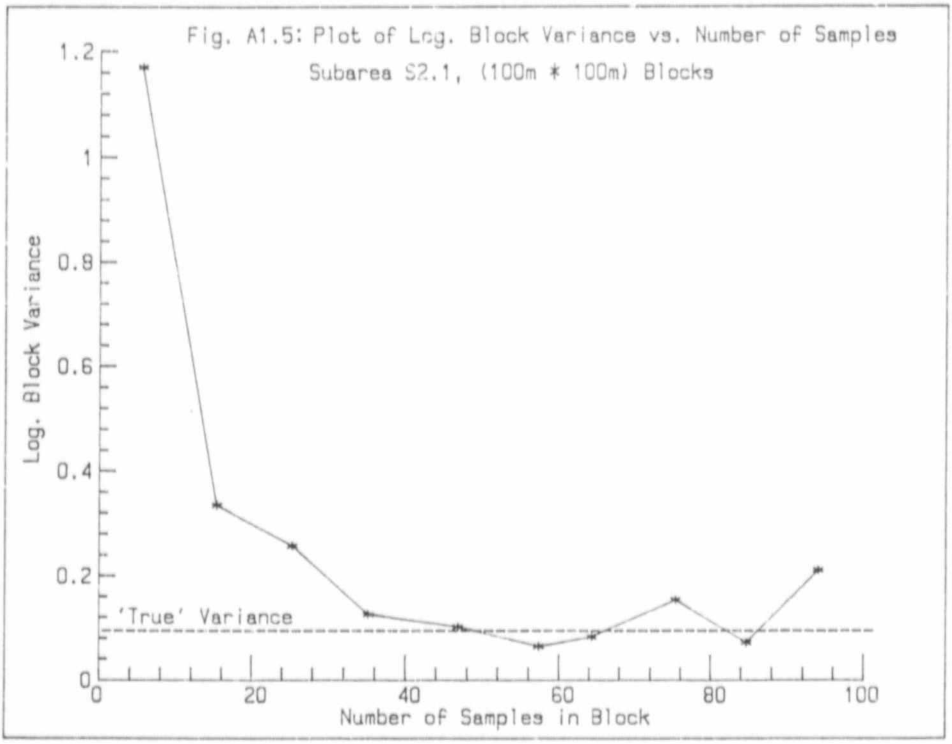


Fig. A1.5: Plot of Log. Block Variance vs. Number of Samples
Subarea S2.1, (100m * 100m) Blocks
Log. Block Variance
1.2
1
0.8
0.6
0.4
0.2
0
'True' Variance
0
20
40
60
80
100
Number of Samples in Block

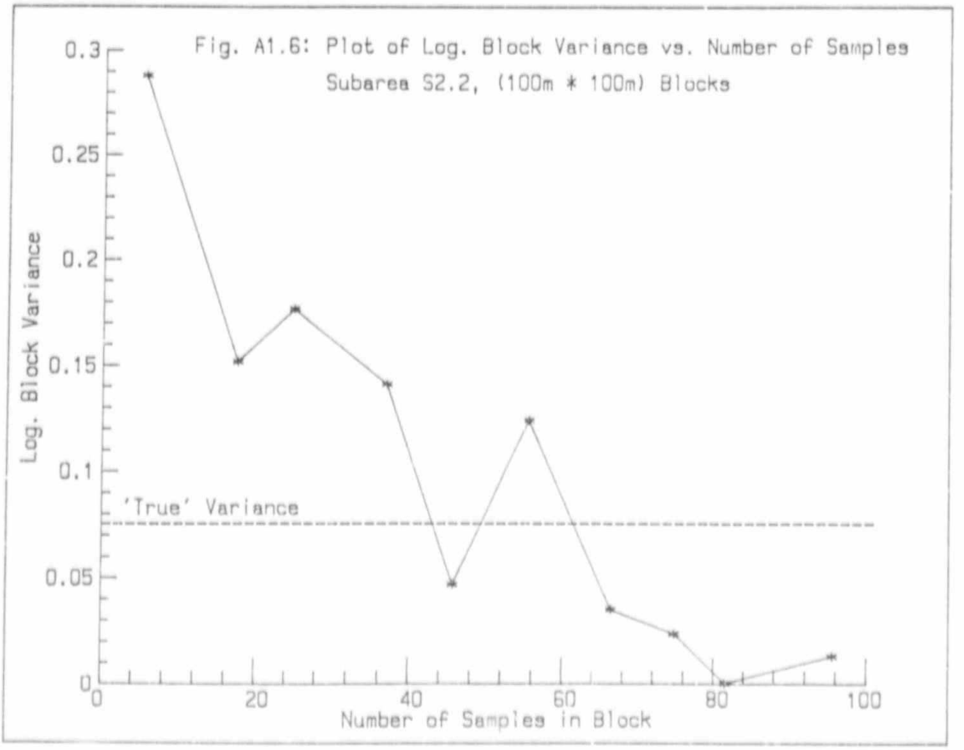


Fig. A1.6: Plot of Log. Block Variance vs. Number of Samples
Subarea S2.2, (100m * 100m) Blocks
Log. Block Variance
0.3
0.25
0.2
0.15
0.1
0.05
0
'True' Variance
0
20
40
60
80
100
Number of Samples in Block

A1.2 Semivariogram Calculation

The formula for the calculation of the standard semi-variogram estimator is (e.g. Matheron, 1963):

$$G_h = \frac{1}{2n} \sum (Z_x - Z_{x+h})^2$$

where x and x+h are two sample points located a distance h apart.

This standard semivariogram estimator is very sensitive against so called outliers (abnormally high values, e.g. Armstrong, 1984).

As outliers are common in the Witwatersrand gold deposits (Krige, 1981), and occur in the analysed data as well, other estimators were used:

- the logarithmic semivariogram is very robust against outliers (Krige & Magri, 1982), and is used in the estimation process of lognormal kriging. The formula for its estimation is (e.g. Krige, 1981):

$$LG_h = \frac{1}{2n} \sum (Y_x - Y_{x+h})^2$$

with $Y = \ln(Z+b)$.

- Hawkins & Cressie proposed another robust estimator (Hawkins & Cressie, 1980):

$$G_h = \overline{Y}^4 \ / \ (.457 + .494 * n^{-1} + .045 * n^{-2})$$

with $\overline{Y}$ being the arithmetic mean of n independent Y_x observations, and

$$Y_x = (|Z_x - Z_{x+h}|)^{0.5}$$

- As the logtransformed semivariogram generally gives stable semivariograms, the author used it to calculate a stable untransformed semivariogram, which then is utilized for untransformed, linear kriging. This approach is e.g.one of the recommendations made by Armstrong (1984) for data bases including outliers. Kriging computer programs work with covariances rather than semivariances, because of the ease of the matrix solution. Therefore the author used the logarithmic semivariogram to calculate the logarithmic covariance for each distance h, which then was backtransformed to the equivalent untransformed covariance. This untransformed covariance was then used in setting up the kriging equations.

The relation between logtransformed and untransformed variances is (e.g. Krige, 1981):

$$V^2 = \overline{Z}^2 * (\exp(LV^2)-1)$$

and for the three-parameter lognormal model (Krige, 1981):

$$V^2 = (\overline{Z}+b)^2 * (\exp(LV_b{}^2)-1)$$

For covariances, this becomes (e.g. Graybill, 1961):

$$COV = \overline{Z}^2 * (\exp(LCOV)-1)$$

and for the three-parameter lognormal model:

$$COV = (\overline{Z}+b)^2 * (\exp(LCOV_b)-1)$$

With the mean grade $\overline{Z}$ and the additive constant b known, the logarithmic covariance LCOV(h) for a certain distance can be calculated from the logtransformed semivariogram ($LCOV_h = LV^2 - LG_h$). The untransformed covariance COV_h is then calculated, using the above formula.

Semivariograms of the spatially irregularly distributed data were calculated for lag intervals of 5m, with a distance tolerance of 1m. Directional semivariograms have an angle tolerance of 22 degrees on each side.

APPENDIX 2: SOFTWARE DEVELOPMENT

Program Listings area available from

Department of Mining Engineering

University of the Witwatersrand

Johannesburg

A2.1 NIMROD - Program Description

NIMROD is a retrieval program for geological data, written in FORTRAN IV. The structure of the program is based on experience gained previously by the author with geological data bases (Braun et. al., 1983).

The information must be coded (see 3.1), and stored in a data base . This geological data base contains one layer per record. There is no restriction to the number of layers.

Additional numerical assaying results of samples are stored in a separate, sampling data base. They are related to the geological layer from which the sample was taken via the sample number.

A control card contains the specifications for the retrieval.

So the program needs the following three input files (logical file number in brackets):

- control card (10)

- geological data base (11)

- sample data base (12)

Every layer of the geological data base as well as the control card has the following fixed format:

```
****+****1****+****2****+****3****+****4****+****5****+****6****+****7****+****8
IDENTXXXXX.XXYYYYY.YYZZZZZ.ZZTHICLA  AA  BB  CC  DD  EE  FF  GG  HHSAZIDICOSAMPL
```

```
IDENT    : log number
XXXXX.XX: north-south coordinate
YYYYY.YY: east-west coordinate
ZZZZZ.ZZ: depth
THIC     : thickness (in cm)
LA       : stratigraphy (HW,RF,FW)
AA       : conact nature             !
BB       : lithology                 !
CC       : conglomerate packing      !  numerically coded
DD       : sedimentary structure     !  according to the
EE       : mineralisation            !  Chamber of Mines
FF       : pyrite content            !  logging sheet
GG       : carbon occurence          !   (Figure 3.1)
HH       : pebble size               !
S        : kind of following structure measurement ('L'
           for bedding,'F'for fault,'X'for crossbedding
AZI      : azimut
DI       : dip
CO       : colour
SAMPL    : sample number (reference to sample data set)
```

The following example shows the format of the geological log of Figure 3.1:

```
****+****1****+****2****+****3****+****4****+****5****+****6****+****7****+****8
     1XXXXX.XXYYYYYY.YYZZZZZ.ZZ 12.HW  02  05
     1XXXXX.XXYYYYYY.YYZZZZZ.ZZ  7.RT  01  04  08  13  17  19       25          1003
     1XXXXX.XXYYYYYY.YYZZZZZ.ZZ  8.RI  02  05      13
     1XXXXX.XXYYYYYY.YYZZZZZ.ZZ 21.RI  01  04  07  13  17  18       26          1002
     1XXXXX.XXYYYYYY.YYZZZZZ.ZZ 36.RI  02  05      13
     1XXXXX.XXYYYYYY.YYZZZZZ.ZZ 12.RB  01  04  07  13  17  20       26          1001
     1XXXXX.XXYYYYYY.YYZZZZZ.ZZ 14.FW      05
```

The format of the sample data base is variable except that the first five columns must contain the reference sample number (format I5, same as field 'SAMPL' of the geological data base)

On the control card the conditions are specified which each layer of the geological data base must meet to be retrieved.

The control card:

```
....+....3....+....4....+....5....
                RF  02  05
```

would retrieve the layers 3 and 5 of the example above, counted from the bottom.

A 'G' respectively a 'L' in column 30 followed by a three digit figure specifying a thickness will retrieve only layers with a greater ('G') or lower ('L') thickness than the one specified.

The control card:

```
....+....3....+....4....+....5....
         L 6.R      04
```

would retrieve only layer 2 of the example log shown above.

To sum the thickness data for each log, an 'S' is specified in column 30. The control card:

```
....+....3....+....4....+....5....
         S  R
```

would result in a data triple for each log consisting of the x- and y-coordinates and the total reef thickness (this control card was used to select the data for Enclosure 1).

The control card:

```
....+....3....+....4....+....5....
         S
```

would give data triples with the stope width as third variable.

If 'SAMPL' is specified in field SAMPL of the control card, the sample data associated with the retrieved layers are written in a separate sample data retrieval file.

The control card:

```
....3....+....4....+....5....+....6....+....7....+....8
    L10.R       04                                SAMPL
```

would extract the sample no. 1003 from the sample data file.

The retrieval analysis will be written to the files assigned to the logical file numbers 12 (geological retrieval) and 13 (sample retrieval). The formats will be the same as the input files.

File numbers 50 to 53 have to be assigned to temporary work files which the program needs during execution.

A2.2 DIGITIZE - Program Description

This program is written in HP - BASIC, as the hardware available for the data capture consists of a HP - 9816 microcomputer and a Calcpomp Series 9100 digitizer.

The program prompts the user for all input parameters.

A file name and the approximate number of data which will be digitized, is required by the program to create a file where the data are stored locally.

After placing the map on the digitizer, three points of known coordinates are digitized, to provide the program with a reference coordinate grid.

Digitizing is done by placing the keypad mark over the sample location, and pressing any button on the keypad. The four space - dependent variables (uranium, gold, channel width and stope width) are then entered, either by the keyboard of the microcomputer or the keypad, whatever was specified by the user.

As the data are digitized, they are listed over the printer.

The data will be stored on disc, one sample location per record. Because the geostatistical analysis was done on a mainframe computer (IBM 3083, University of the Witwatersrand Computer Center), the data were transfered to the mainframe.

A2.3 CONVERT - Program Desctiption

This program is written in FORTRAN IV. It converts coordinates in the plane of the reef (stope sheet coordinates, as digitized) to horizontal coordinates.

The input data file is assigned to the logical file number 10. The converted data are written fo file number 12.

The program prompts the user on the terminal for the average azimut and dip of the strata, for a benchmark position on the stope sheet, and for the distance of this stope sheet position to the true position in horizontal coordinates.

The data are then converted, using the same average azimut and dip for each sample position.

A2.4 PROBPLOT - Program Description

This program was originally intended to plot grouped data on either normal or lognormal probability paper. Later a histogram program was developed at its front end, in oder to allow the input of ungrouped data, These then will be grouped first, before being passed on to the plotting program.

Both programs are written in FORTRAN IV. The graphics language is HCBS.

The histogram program requires the data file to be assigned to the logical file number 10. A set of control cards is read from file number 11. The grouped data and some basic statistics are written to file number 12, which becomes the input file for the plotting program.

The first of the control cards contains the title.

The second card has the following format:

columns	format	parameter
1 - 2	I2	Number of classes (max. 50).
6 - 15	F10.2	Upper limit of highest class. This figure divided by the

columns	format	parameter
		number of classes becomes the class interval.
20	I1	'1' for lognormal, '0' for normal probability paper.

The third card gives the format of the data.

The fourth card contains the additive constant (if any) of a three-parameter lognormal model.

The output file (number 12) contains the upper class limits and frequencies for each class, the number of classes and the class interval used, and the arithmetic and geometric means, the unadjusted and adjusted variances of the data. This file serves as input file for the plotting program, where it is also assigned to the logical file number 12.

The plotting program reads its labels for the horizontal and vertical axes from a data file assigned to file number 11. The first two records control the labeling of the horizontal axis. On the first card the probability is given in standard deviations of a normal distribution, on the second card the corresponding probabilities in frequency percent. The format is (F5.0).

To avoid negative standard deviations below 50% frequency, the origin is shifted by adding a constant of 3. 50% thus equals three standard deviations instead of zero, and 2% equals one standard deviation instead of minus two.

The third record contains the labels for the vertical axis, and the fourth record the number of labels for the horizontal axis.

A2.5 DATABLOK - Program Description

This program superimposes a grid on irregularly spaced data, and calculates statistics on the data in each grid cell or block. These statistical parameters are then assigned to the coordinates of the block center. The following statistics are calculated: Logarithmic variance and mean, arithmetic mean, Sichel's t- estimator for the mean, and the number of data inside each block.

The irregular data are input over file number 12. A set of two control cards is read from file number 10. The first of these is of the following format:

columns	format	parameter
1 - 8	F8.2	X-coordinate of lower left corner of grid.
9 - 16	F8.2	Y-coordinate of lower left corner of grid.
17 - 24	F8.2	Angle by which grid is tilted against X,Y-coordinate system (counterclockwise)
25 - 32	F8.2	Blocksize in X-direction.
33 - 40	F8.2	Blocksize in Y-direction.

columns	format	parameter
41 - 45	I5	Number of blocks in X.
46 - 50	I5	Number of blocks in Y.
51 - 60	F10.2	Additive constant.

The second card gives the format of the input data.

The output is written to file number 11. Each block coordinates and statistics are written in one record.

A2.6 UKSIM - Program Description

This program simulates estimation in a longwall mining situation. From all the data in the mined out area only those within a defined zone (data area) on one side of the oreblock are used to estimate this oreblock (see Fig. 4.14). This pattern is moved over the whole mined out area.

The program basically is a modified version of DATABLOK, which is used to extract the data inside the data area. A two - dimensional kriging program then uses only these samples to estimate the oreblock. This kriging program is a modified version of a program donated to Wits University by Anglo-Vaal. Modifications made by the author are:

- a universal kriging option,
- a simple kriging option using local means,
- an option to use the logtransformed semivariogram, but backtransformed covariances.

The program reads two sets of control cards, and one data file. It outputs a summary of the estimation statistics, and parameters used; and a file containing center coordinates, estimates, error variances and the number of samples used in the estimation, for each oreblock.