CINOLA GOLD PROJECT

GEOSTATISTICAL STUDY

PART 1: GLOBAL RECOVERABLE RESERVES

PART 2: LOCAL RECOVERABLE RESERVES

825945

MARCH 1988

CINOLA GOLD PROJECT

GEOSTATISTICAL STUDY

PART 1: GLOBAL RECOVERABLE RESERVES

NOVEMBER 1987

1.0 Introduction

This report presents the results of the first of three steps of the in-house geostatistical study of the gold reserves of the Cinola Gold Project.

1st Step:	global recoverable reserves - preliminary reserve statement
2nd Step:	local recoverable reserves - final reserve statements
3rd Step:	optimized mineable reserves

The reserves estimated in this first work are global which means that they can be properly considered only for large portions of the deposit, like the entire deposit (geological in-situ reserves) or, say, the material contained in a final pit outline.

They are recoverable in that the resultn derived through the individual estimation of 30 m x 30 m x 6 m blocks of material (the block-model) have then been modified to reflect a selectivity level corresponding to a realistic size of selective mining units, around 2 m x 2 m x 6 m, (non-recoverable reserves, calculated on big blocks, are valuable indicators of the attractiveness of a project but, if used as is in cashflow analyses at a given cutoff grade, they would systematically underestimate the economic value of the project).

The present block model is a refinement of the previous PAH (Pincock, Allen & Holt) block model. The methodology is roughly the same. The differences are in the amount and improved quality of the data now available, in the detailed reinterpretation of the geological controls, and in the fact that the PAH report concluded only with non-recoverable reserves over 20 m x 20 m x 6 m blocks.

The present block model, which will be the basis for the second step, or calculation of local recoverable reserves, has been derived independently from other in-house reserve works, and using a methodology totally different from previous existing geostatistical works performed on the deposit in the times of Cinola Consolidated.

2.0 Database Acquisition

Prior to starting the geostatistical work, all data available, including the most recent drilling results were put together in a database and carefully checked for errors.

Major errors were found even in the database used by PAH in the previous geostatistical model (and obtained by them directly from some other consultants previously involved). These errors were corrected.

That database was also fed all the available geological information, including the relogging of the existing drillholes.

Finally, a data file was extracted from that database for use in the geostatistical study of the gold grades. This file, which contains gold fire assays, survey data and lithological types, consists of 312 reverse circulation and diamond drillholes assayed for gold and other metals, on average, every two mstres.

3.0 Geological Controls of the Mineralization

As is very often the case, the lithelogic units throughout the deposit are not directly relevant to the determination of grade-modelling parameters. In the other hand, it is important to ensure that different sets of parameters can be used, should there be different homogeneous units of mineralization.

In order to identify such units, the lithologies were studied, as well as the gold grades, the features observed underground and the geological theory of the deposit. Cross sections already correlated for lithologies by the geologists were reexamined and carefully discussed, downhole gold grade profiles for some typical drillholes studied in detail and the results of these observations checked against the geological interpretation of the deposit, until a clear understanding of the various populations of gold grades that coexist at Cinola could be reached.

During that work, most of the anisotropies datected in previous variogram analyses by PAH were found very consistent with the new understanding of the geology. However, because it is based on many more data and on a new geological interpretation, the present modelling geological rock-model does not match the rougher one previously built and used by PAH.

Finally, the results of the statistical and variographical analyses in the areas separated in the rock-model were examined and checked for consistency with the theory of the deposit, and minor corrections brought wherever needed to the rock-model and to the general modelling methodology.

In conclusion, two major zones of mineralization were separated, each one hosting two distinct populations of gold grades:

<u>Zone 1</u> includes most parts of the breccia, the rhyolite and the conglomerate. Gold is found in a pervasive continuous background mineralization which averages 1.62 g/t and includes some stockwork mineralization. Gold is also found in dipping, occasional quartz veins, at an average grade of 12.64 g/t. representing 3.7% of the assays in that zone.

<u>Zone 2</u> consists mainly of the sediments. Again gold is found in two populations: a pervasive continuous background grade later cut by higher, less continuous grades found in subvertical quartz veins. The eastern part of zone 2 has been subjected to argilization. As a consequence, gold grades decrease toward east. In the argilized part, as well as in the sediments directly above the argilized portion, the grades are lower and the mineralized quartz veins less numerous. Zone 2 was thus further split into two sub-zones, namely:

<u>Zone 2 West</u> where the pervasive grade averages 0.95 g/t, with a significant amount of quartz vein related high grades: 17.6% of the assays, averaging 5.6 g/t.

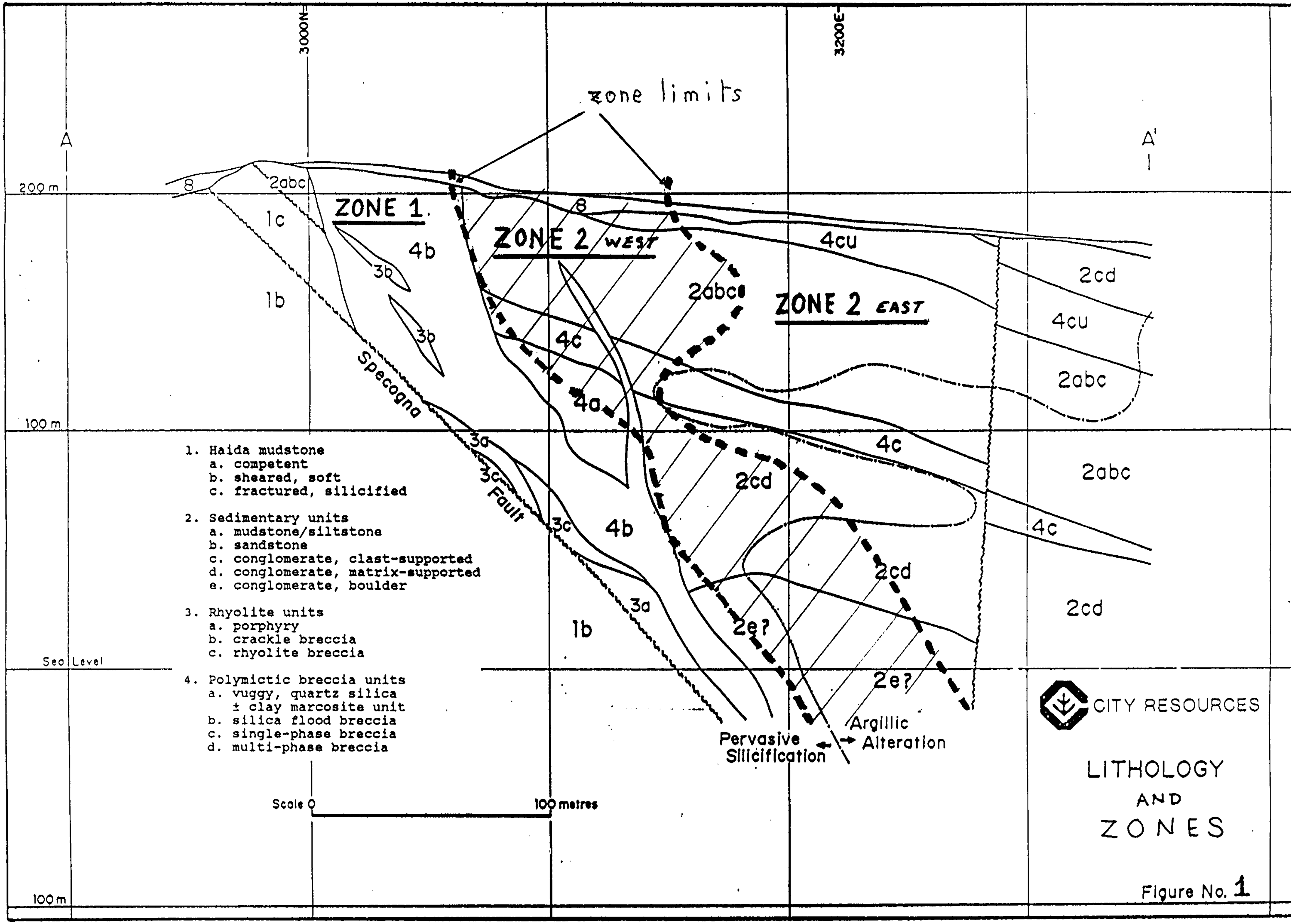
<u>Zone 2 East</u> where the background grade averages only 0.46 g/t, and the quartz veins at 4.2 g/t account for only 3.9% of the data.

Figure 1 shows a schematic of lithology and zones in cross-section. The boundary between Zones 1 and 2 is most likely a geological boundary, and was determined based on the limit of the breccia and, in a few places, using grade contours at a 1.71 g/t cutoff grade.

The boundary between the sub-zones of Zone 2 is much less precise because the transition is not as sharp.

It is worth noting that a halo effect can be observed around the vein related grades: the background grade tends to increase progressively when approaching the mineralized quartz veins.

Finally, these zones and sub-zones were outlined in 23 cross-sections, then re-interpreted in plan views before being digitized in and used to flag the data and the blocks of the block model.


3000N
3200E
zone limits
A
A'
200 m
100 m
Sea Level
100 m
8
2abc
1c
1b
ZONE 1
4b
3b
3b
ZONE 2 WEST
8
4cu
2cd
4cu
2abc
ZONE 2 EAST
4c
4a
2abc
4c
2abc
Specogna
3a
3c
Fault
3c
4b
2cd
2cd
4c
2cd
3a
1b
2e?
2e?
Pervasive Silicification
Argillic Alteration
1. Haida mudstone
a. competent
b. sheared, soft
c. fractured, silicified
2. Sedimentary units
a. mudstone/siltstone
b. sandstone
c. conglomerate, clast-supported
d. conglomerate, matrix-supported
e. conglomerate, boulder
3. Rhyolite units
a. porphyry
b. crackle breccia
c. rhyolite breccia
4. Polymictic breccia units
a. vuggy, quartz silica
± clay marcosite unit
b. silica flood breccia
c. single-phase breccia
d. multi-phase breccia
Scale 0
100 metres
CITY RESOURCES
LITHOLOGY
AND
ZONES
Figure No. 1

The high grade, vein related populations could not be outlined in cross-sections due to their complexity and their small dimensions compared to the spacing of the available data. It is however, very important to characterize them since the corresponding grades obviously cannot be projected using the same modelling parameters as for the continuous, background mineralization. It is out of the question to ignore them, since they host a proportion of the total ounces of gold large enough to make the difference between an economic and an uneconomic project. It would not be admissible either to scale them down to a given conventional grade, because, with no production history in the deposit that value would be completely arbitrary.

As a consequence, it was decided to use a two-step geostatistical estimation process, as explained further, in which the high grade populations are treated separately. In order to do so, these populations need to be characterized statistically.

4.0 Statistics

Basic statistics were calculated for each population in each of one of the three rock-zones described in the previous section. The gold histograms are all lognormal-like, and the coefficients of variation, ranging from 0.4 to 1.5 show that the rock-zones as defined are quite homogeneous statistically.

Cumulative frequency plots of the gold grades were studied for further definition of the various grade populations in each rock-zone. On a log-probability scale, these plots are similar in shape. The plot corresponding to Zone 1 is shown as an example on Figure 2. As seen on this plot, the high grade populations do show, but how many distinct populations there are, and which grades define them, cannot be determined from the plots alone. For instance, on Figure 2., the plot suggests 4 g/t, 6 g/t and 11 g/t as possible starting grades for the highest grade population. In fact, there are possibly three high grade populations, but the visual examination of some downhole grade histograms show that only the highest one correspond to the discontinuous character of the mineralized quartz veins. Between 4 g/t and 11 g/t, the one or two populations of grades that seem to exist are most likely linked to the stockworking and to the halo effect already mentioned earlier.

Finally, the study of the plots complemented with visual observation of the assay logs allowed to determine that the following grade thresholds would characterize quite well the high grade, quartz vein related populations:

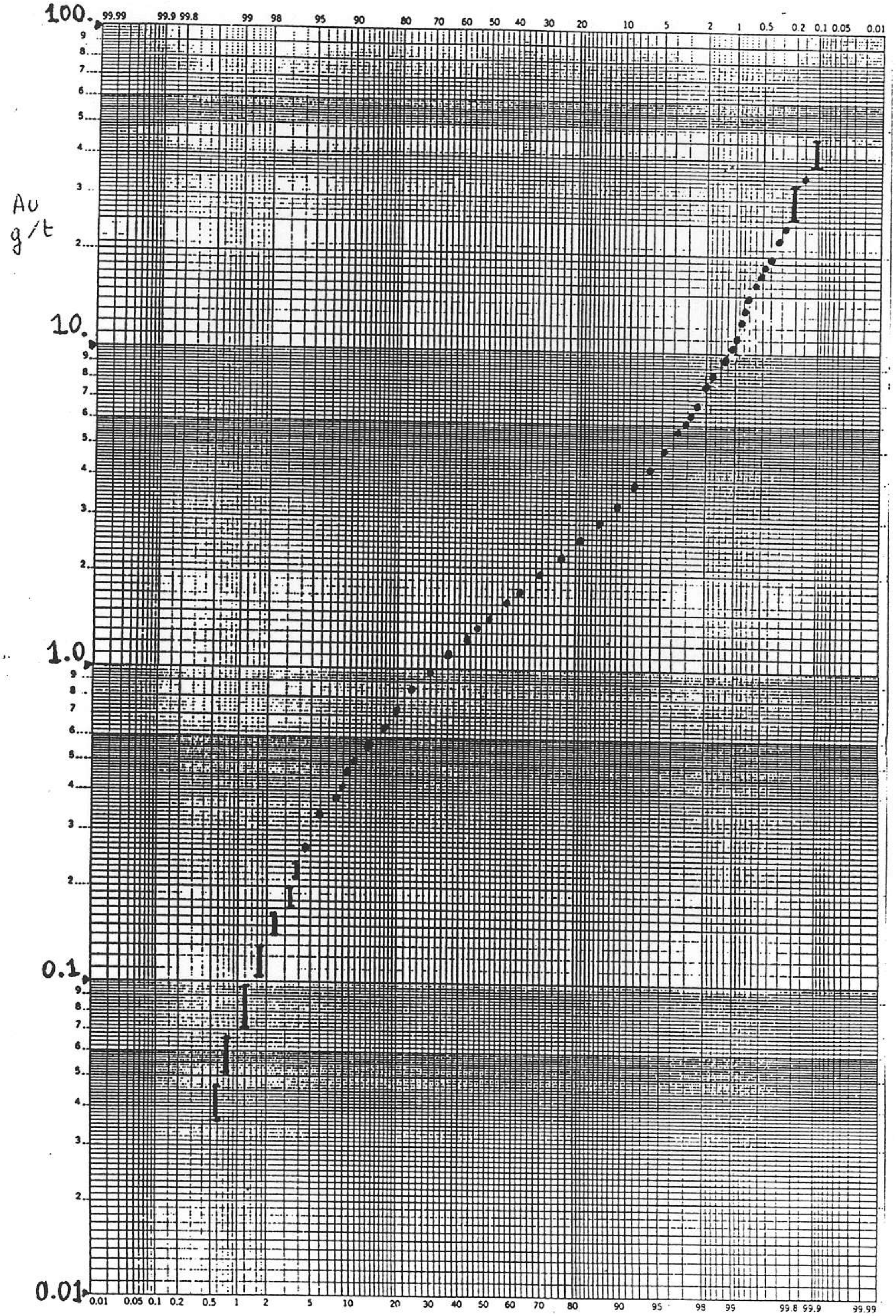


FIGURE 2: CUMULATIVE FREQUENCY PLOT OF GOLD GRADES IN ZONE 1

- <u>Zone 1:</u> 11.0 g/t and above
- <u>Zone 2, West:</u> 2.3 g/t and above
- <u>Zone 2, East:</u> 2.0 g/t and above

5.0 Grade Modelling Methodology

In order to properly handle the higher grade population in each rock-type, we have used a two-step estimation procedure which allows to project the high grade assays separately, with the proper interpolation parameters that account for their lack of continuity perpendicularly to the quartz veins.

The procedure itself is a simplified "indicator kriging" procedure in which each block of the block-model is considered a mixing of background grade material plus a certain proportion possibly equal to zero of high grade material. The variables to be estimated in such conditions are thus, for each 30 m x 30 m x 6 m block:

- the grade of the background material
- the grade of the high grade material
- the proportion of high grade material in the block.

The two grades are interpolated using standard kriging and the relevant, separate variograms and data search neighborhoods, and using only the data of the rock type and population (background or high-grade) being estimated.

The proportion of high-grade material is interpolated by kriging of the indicator variable of that high-grade material (i.e. a variable worth 1 for each high-grade assay and 0 for each background assay).

It is noteworthy that this simple two-step grade projection geostatistical method seems to have been independently found best lately by many a geostatistician working in major gold mining companies in North America. The method seems to be the only one to be at the same time: simple, optimal, and proper to handle the high grade populations so typical of epithermal gold deposits.

6.0 Variograms

6.1 Background Gold Grades

Variograms of background gold grade were calculated with no unexpected difficulty. Table 1 shows the main directions of anisotropy and how the variograms were modelled using spherical variograms. As can be seen, the anisotropy ellipsoid is tilted in Zone 1, so that none of its main axes is vertical: the anisotropies

Rock Type	Azimuth	Dip	Nugget/ Total Sill Ratio	Range
Zone 1	N7° E E7° S E7° S	0° (strike) +35° -55° (dip)	1/2 1/2 1/2	90 m 32.5 m 55 m
Zone 2 West	N55° E N35° W -	0° (strike) 0° -90° (vertical dip)	1/2 1/2 1/2	140 m 115 m 90 m
Zone 2 East	N55° E N35° W -	0° (strike) 0° -90° (vertical)	1/2 1/2 1/2	140 m 115 m 55 m

TABLE 1

VARIOGRAMS OF THE BACKGROUND GRADES

detected by the variograms correspond to the strike and dip directions of the veins in each rock-zone.

The main difference between Zone 2 West and East is in the vertical range, which is much shorter in the argilized zone.

6.2 High Grades and Proportions

Variograms are difficult to derive for the gold grades of the high grade populations, because these data are not numerous. However, it was possible to calculate them vertically and along strike in Zone 2 West, and similar variograms were found there for high grades and proportions. Missing information had to be gnessed with the help of the geological interpretation. In particular, the variograms for Zone 1 were assumed to be same as for background grades in that zone, flattened perpendicularly to the vein plans and with the same ratio nugget/total sill as in Zone 2.

Table 2 shows the resulting spherical models.

7.0 Reserve Calculations

7.1 Block Kriging

30m x 30m x 6m blocks were kriged using the methodology and modelling parameters described there above.

7.2 Selectivity Correction

As mentioned earlier, the grades and high grade proportions for 30 m x 30 m x 6 m blocks are not directly usable as projected production figures, so that they need to be adjusted for selectivity, before a preliminary global reserve statement can be prepared for in-situ, geological reserves.

The method used here is known as "affine correction of variance". It consists of modifying the frequency distribution histogram of the block grades, while preserving its shape and mean, to bring its variance up to the variance of the real grades of smaller mining units so that the obtained histogram is a good representation of the histogram of the mining units. The latter variance can easily be calculated from the variograms.

The method was applied to the background grade and the transformed histogram used to calculate global reserves at various cut-off grades. The results were then

Rock Type	Direction	Nugget/ Total Sill Ratio	Range
Zone 1	Horizontal along strike	0.65	90 m
	down dip	0.65	55 m
	perpendicular to veins	0.65	2 m
Zone 2 West and East	Horizontal along strike	0.65	90 m
	vertical	0.65	90 m
	perpendicular to veins	0.65	2 m

TABLE 2

VARIOGRAMS FOR HIGH GRADES AND HIGH-GRADE PROPORTIONS corrected to incorporate the high grade material contained in the block, under the reasonable assumption that up to a cut-off grade of 2.06 g/t, all the high grade material contained in a mineable big block will be mined as ore.

7.3 Preliminary In-situ Geological Global Recoverable Reserve Statement

Table 3 shows the end result of that process. Should existing open pit outlines be entered into the computer model, a similar statement could be calculated for mineable reserves.

2m x 2m x 6m mining units were considered here because it corresponds best to what is believed to be the implicit selection level at which the in-house cross-sectional hand-calculated reserves were derived. These reserves are shown in Table 4. It is noteworthy that both these statements match each other surprisingly well, although they were derived independently by totally different methodologies. (The difference in tonnage at the lowest cut-off grade is only due to a different overall mineralization envelope). However, it must be stressed that the care put in the hand-calculated reserves to account, almost deterministically, for the geometrical shape of the high grade bearing veins is the geologist equivalent of the stochastic way these high grade populations have been handled in the geostatistical model.

8.0 Next Step

This concludes the first step of the geostatistical study of the Cinola deposit. As already mentioned, this was a preliminary step that had two objectives:

- provide a preliminary geological reserve model that could be compared with other existing estimates in order to gain confidence in the reserves of the deposit and check the model before using it in the next step;

- prepare the basis for the final model in which each 30m x 30m x 6m block will be informed with selected ore tonnage and grade at any cut-off grade, for a given expected blasthole spacing and a given bench height (i.e. a given mining selectivity level). This work is presently under way.

TABLE 3

CINOLA PROJECT

In-Situ Geological Geostatistical Reserves Globally Adjusted for Mining Selectivity on 2m x 2m x 6m Mining Units

Cut-Off Grade oz/t	g/t	Tonnage (tonnes) x 10^6	Grade g/t	oz/t
0.02	0.69	56.44	1.64	0.047
0.03	1.03	30.80	2.27	0.066
0.035	1.20	21.27	2.79	0.081
0.04	1.37	16.83	3.19	0.090
0.05	1.71	12.97	3.68	0.107
0.06	2.06	10.05	4.20	0.129

TABLE 4

HAND CALCULATED CROSS-SECTIONAL GEOLOGIC RESERVE ESTIMATE

Cut-Off Grade oz/t	g/t	Tonnage (tonnes) x 10^6	Grade g/t	oz/t
0.02	0.69	47.04	1.69	0.049
0.03	1.03	30.15	2.15	0.062
0.035	1.20	20.35	2.66	0.077
0.04	1.37	16.33	3.00	0.087
0.05	1.71	12.8	3.41	0.099
0.06	2.06	10.75	3.71	0.108

CINOLA GOLD PROJECT

GEOSTATISTICAL STUDY

PART 2: LOCAL RECOVERABLE RESERVES

MARCH 1988

1.0 INTRODUCTION

This purely technical report describes the calculations leading to the final block model on the Cinola gold project. The reserves calculated here are recoverable and local, in that they give for each 30 m x 30 m x 6 m block of the block model the reserves that are expected to be mined out of that block if the cutoff grade applies to the selection of individual 5 m x 5 m x 6 m mining units. They can be made mineable when constrained to the inside of a given open pit outline. They are suitable for open pit optimization, mining sequence optimization, and more generally any calculation or optimization relying on local reserves.

Section 4.0, of this report, also presents the reserve comparisons that were used to:

a. validate the previous global block model (Part 1 Report) which served as the basis of the present block model, and,

b. validate the final reserves derived from the present block model.

2.0 Method Used in Calculating Local Recoverable Reserves

2.1 Equations

The geostatistical method known as "lognormal shortcut" was applied to the background grade as estimated in the previous global model (see Part 1 report) and the high grade material from the gold veins then incorporated according to certain mining assumptions.

The rationale and the methodology for using the lognormal shortcut are fully described in Appendix 1 to the present report, especially in Section 2.2.2.2 of that Appendix.

The classical lognormal equations used for the calculation of the tonnage, quantity of metal and grade above the cutoff grade COG in each block for the background mineralization, once the local dispersion variance V^2 of its mining units (designed by σ_v^2 in Appendix 1) has been calculated, are:

Tonnage: $$T(COG) = To.G\left(\frac{1}{s}.\ln\left[\frac{COG}{m}\right] + \frac{s}{2}\right)$$

Quantity: $$Q(COG) = To.m.G\left(\frac{1}{s}.\ln\left[\frac{COG}{m}\right] - \frac{s}{2}\right)$$

Grade: $$g(COG) = \frac{Q(COG)}{T(COG)}$$

Where:

To is the tonnage of the block

m its average background grade, estimated in Part 1 of the study

s^2 is the logarithmic dispersion variance of the mining units given by:

$$s^2 = \ln(1 + \frac{V^2}{m^2})$$

and G(u) is the gaussian posterior cumulative frequency distribution function, approximated by:

$$G(u) = 0.5[1 - \text{sign}(u).\sqrt{1-\exp(-2u^2/\pi)}\ ]$$

2.2 Variance Calculations

The formula for the calculation of the dispersion variance of the mining units in a block of average grade m, as per section 2.2.2.2.3 of Appendix 1, is:

$$V^2 = R.F(m).[f.D^2(v|B) + \sigma_e^2]$$

In which:

* $D^2(v|B)$ was derived from the variograms in each zone of the deposit, and the Kriging variance σ_e^2 automatically calculated by the Kriging program. Both calculations were performed using variograms with a

total sill equal to 1.00. $D^2(v|B)$ is shown on Table 2.1 for each zone of the deposit.

* Factor f was adjusted from the overall distributions of background grades in each zone of the deposit. f is also tabulated in Table 2.1.

* Coefficient R was calculated for each single block as explained in Section 2.2.2.2.3.3 of Appendix 1.

* The proportional effect relation F(m) was calculated in each zone in the same step as for the calculation of coefficient R, using a standard proportional effect study procedure. Figure 2.1 shows the relation local-variance/local-mean in Zone 1 and the fitted proportional effect relation. The scatter diagrams in both parts of Zone 2 are very similar to this one. On Figure 2.1 the scattering of the diagram illustrates well the need for using coefficient R (Correction for local departure from the average proportional effect). The fitted relations F(m) are tabulated in Table 2.1 for each zone.

FIGURE 2.1

CINOLA GOLD PROJECT

RELATION VARIANCE/MEAN IN ZONE 1

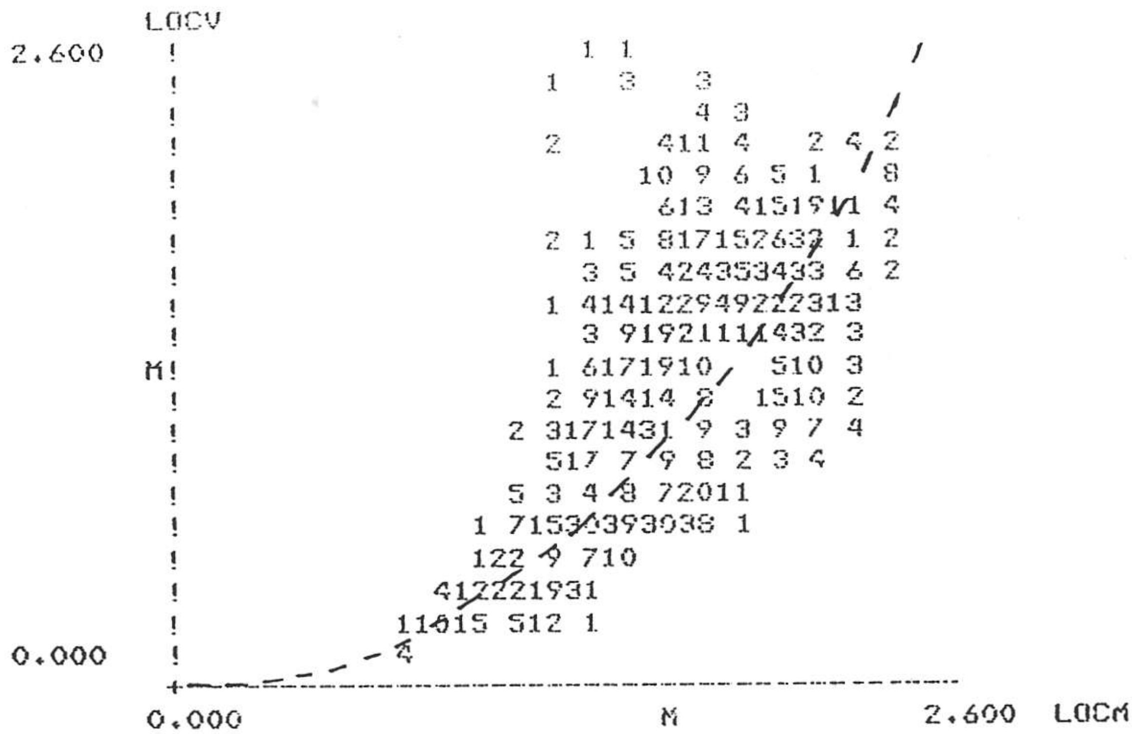


TABLE 2.1

CINOLA GOLD PROJECT - VARIANCE CALCULATION PARAMETERS
(5 m x 5 m x 6 m mining units in 30 m x 30 m x 6 m blocks)

	Zone 1	Zone 2 West	Zone 2 East
f	1.000	1.570	0.895
D^2 (v B)	0.200	0.100	0.100
F(m)	$0.27 * m^{2.47}$	$0.14 * m^{1.49}$	$0.40 * m^{1.94}$

3.0 Merging the high grade vein material into the local block model

There are two reasons why we have been able to keep the high grade values from the gold veins separate until this point:

a. they could actually be separated among the assay data using a simple threshold on the assay gold values in each zone of the deposit (see Part 1), and,

b. everywhere, the grade of that vein material is high enough to be above any realistic cutoff grade that will be actually used when mining.

We now need to merge them into the block model for each new cutoff grade considered. In order to do so, some assumptions have to be made regarding the way mining will take place, namely:

* in Zone 1 and Zone 2 West, the high grade material from the veins is surrounded by a halo of high background grades so that it is justified to assume that all the vein material will be mined within the ore detected by the blastholes, whether these holes hit the veins or not.

* <u>in Zone 2 East</u>, the background grade is lower than in the rest of the deposit, the mineralized veins are subvertical, and the halo effect cannot be observed. As a consequence, at the mining stage, the veins are most likely to be missed by the vertical blastholes, at a 5 m spacing, so they will be mined out only wherever the "background grade" itself will be considered above the cutoff grade as per the blasthole assays. As a consequence, <u>for Zone 2 East, the vein material should be included in the reserves only to the prorata of the proportion of "background ore" in each block.</u>

In both cases however, it is easy to modify the previously calculated "background" tonnage and grade above cutoff grade for each block, by using the proportion and grade of the vein material as calculated in the global model (Part I), in order to account for the corresponding high grade material as per the above assumptions in each zone. The final reserve results are fully described in the next section.

4.0 <u>Comparison of available reserve estimates. Final reserves.</u>

At the present stage, the ore reserves of the deposit have only been estimated from the exploratory drillhole data inhouse as well as externally. Thanks to the high density of drillhole assays throughout the deposit, and the detailed understanding of the geology, this estimation was performed quite precisely. However, it is well known that ore reserve estimates produced independently from the same set of data using different and proper methods are not expected to agree perfectly due to possible differences in: the interpolation methods; the semi-subjective choice of the parameters of the estimation; the appreciation of the geological controls (the human factor); the degree of selectivity (size of the mining units and bench height) and internal or mining dilution built in the estimates; and more generally any other technical parameters or assumptions. The present report as well as the Part 1 report present numerous examples of such parameters and factors.

It is therefore critical to compare several independent estimates at several cutoff grades at both the level of "in situ" (the entire orebody) and mineable reserves grades, and study and explain their differences and similarities. Only by doing so, can one form a judgement on how consistent and valid each of these models is, and hence, derive a large measure of confidence in the reserves within the orebody. The end product of these comparisons is judgement on whether the present detailed grade block model can safely be used in further computerized engineering works.

4.1 Grade Models Available

There are four grade models available at the present time in the Company:

#1. William Hill Mining Consultants (H.M.C.) Ltd's hand calculated model of mineable reserves at five cutoff grades within a preliminary ultimate pit outline. This is a cross sectional model subjected to blocking over 20 m x 20 m x 20 m blocks. (H.M.C.'s)

#2. City Resources' in situ reserves at five relevant cutoff grades, hand calculated using grade contours and no blocking, thus more selective than H.M.C.'s model. (cross sectional model)

#3. City Resources' geostatistical in situ recoverable reserves at the same five relevant cutoff grades (a grade block model of 30 m x 30 m x 6 m panels modified to account for mining selectivity on 2 m x 2 m x 6 m units to make it directly comparable to estimate #2 at the level of the entire deposit). (See Part 1)

#4. City Resources' present detailed geostatistical block model of mineable reserves. As explained above, this model is derived on the basis of the previous geostatistical block model (#3) using an improved version of the geostatistical "lognormal shortcut" method, along with a stochastic model of the gold vein material. As has been seen, it offers for each 30 m x 30 m x 6 m block an estimate of the tonnage and grade that will be recovered when the selection against a given cutoff grade is performed on 5 m x 5 m x 6 m mining units within the block. This model was then constrained to the above mentioned pit outline, for comparison to H.M.C.'s model.

These models have been described in detail in internal and external reports, including the present one. As seen in this report and in the Part 1 report, special care has been given in the two geostatistical models to the separate handling of higher grade assays from the mineralized gold veins, thanks to a two-step estimation process using indicator kriging for the vein material.

4.2 Comparison of In Situ Reserves Estimates:

The results from the in-house cross sectional and the geostatistical in situ models at five relevant cutoff grades ranging from 0.03 oz/t to 0.06 oz/t are shown again on Table 4.1 for the entire orebody (from the Part 1 report). It shows that the two models are quite consistent with each other, which gives

TABLE 4.1

CINOLA PROJECT - COMPARISON OF IN SITU RESERVES

TABLE 4.1-1

MODEL #2: Hand Calculated Cross Sectional In Situ Geologic Reserve Estimate

Cut-Off Grade oz/t	Cut-Off Grade g/t	Tonnage (tonnes) x 10^6	Grade g/t	Grade oz/t
0.030	1.03	30.15	2.15	0.062
0.035	1.20	20.35	2.66	0.077
0.040	1.37	16.33	3.00	0.087
0.050	1.71	12.80	3.41	0.099
0.060	2.06	10.75	3.71	0.108

TABLE 4.1-2

MODEL #3: In-Situ Geological Geostatistical Reserves Globally Adjusted for Mining Selectivity on Small Mining Units

Cut-Off Grade oz/t	Cut-Off Grade g/t	Tonnage (tonnes) x 10^6	Grade g/t	Grade oz/t
0.030	1.03	30.80	2.27	0.066
0.035	1.20	21.27	2.79	0.081
0.040	1.37	16.83	3.19	0.090
0.050	1.71	12.97	3.68	0.107
0.060	2.06	10.05	4.20	0.129

us not only confidence in the results, but also in the adequacy of the first geostatistical block grade model to form the basis of the more sophisticated geostatistical model of mineable reserves.

4.3 Comparison of Mineable Reserves Within the Ultimate Pit

Table 4.2 shows the results of the H.M.C.'s and the geostatistical mineable reserves models at three cutoff grades, around and including the base case of 1.10 g/t within a preliminary ultimate open pit outline designed by H.M.C. In addition to these and for additional reference, the results of the in situ cross sectional model (#2) after being constrained to the same ultimate pit outline are shown on Table 4.2 as well.

The comparison of all these results leads to the following observations and conclusions:

- The geostatistical model finally appears as selective as the very selective cross-sectional model. This indicates to us that performing the mining selection on units smaller that 5 m x 5 m x 6 m would not bring any significant improvement to the gold grades.

- The cross sectional and detailed, mineable, geostatistical models (#2 and #4) are in good agreement, but are

TABLE 4.2

CINOLA PROJECT - COMPARISON OF MINEABLE RESERVES

CUTOFF GRADE	H.M.C.'S (#1)				CROSS SECTIONAL (#2)				GEOSTATISTICAL (#4)			
g/t (oz/t)	Tonnes x 10^6	g/t	oz/t	Tonnes of Au	Tonnes x 10^6	g/t	oz/t	Tonnes of Au	Tonnes x 10^6	g/t	oz/t	Tonnes of Au
0.94 (0.0275)	27.75	1.98	0.058	54.94	31.38	2.01	0.059	63.07	31.06	2.01	0.059	62.43
1.10 (0.032)	24.80	2.11	0.062	52.33	23.44	2.35	0.069	55.08	22.63	2.36	0.069	53.40
1.37 (0.040)	19.02	2.33	0.068	44.32	15.52	2.94	0.086	45.62	14.63	2.97	0.087	43.45
									16.83	3.19	0.090	(#3)

2 Moz Au

much more selective than H.M.C.'s model: 5 m square mining units over a 6 m bench height in the first case, versus at least 20 m square mining units over 20 m vertically in the other case. The results clearly reflect this difference in selectivity.

- H.M.C.'s model of the total mineable reserves appears conservative at low cutoff grades on tonnes of contained metal and very consistent with other estimates at medium (base case) and high cutoff grades, given the differences in selectivity and interpolation methods.

Given all the above, and as a selection process on 5 m square mining units over a 6 m bench height is believed to be achievable, a realistic estimate of the mineable reserves at all cutoff grades is best represented <u>by the detailed geostatistical model of mineable reserves (#4) as per Table 4.2.</u>

4.4 <u>Comparison of Mineable Reserves Within Stage One Pit (Years 1 to 3)</u>

Stage One pit is the preliminary pit outline designed by H.M.C. for the first three years of the life of the mine. The mineable reserves contained in that pit are of great importance as they cover the "payback period". Of importance also, for the same reason, are the tonnage and grade of high grade material to be mined during these first three years.

Table 4.3 shows a comparison of the three reserve estimates in the base case (cutoff grade 1.10 g/t) and the amount and quality of the ore above 2.17 g/t which, in H.M.C.'s mine schedule is to be processed immediately, while the ore below 2.17 g/t is to be stockpiled and later reclaimed.

Also included in Table 4.3 are some additional figures aimed to ease the detailed comparison of the geostatistical model with the two other models and show how it relates to the economics of H.M.C.'s reports.

These comparisons confirm the consistency between the three models, especially between the inhouse cross sectional and geostatistical models. There is no surprise in the fact that same tonnages or grades are not obtained at the same cutoff grades in the three models, once one realizes in addition to the differences in methods, that in the H.M.C.'s and inhouse cross sectional models, the polygons have NOT been redesigned for each new cutoff grade being considered.

It is also noteworthy that H.M.C.'s model seems to be more selective in Stage One pit than for the rest of the deposit.

TABLE 4.3

CINOLA PROJECT - YEARS 1 to 3 OF MINEABLE RESERVES - STAGE ONE PIT

CUTOFF GRADE	H.M.C.'S (#1)				CROSS SECTIONAL (#2)				GEOSTATISTICAL (#4)			
g/t (oz/t)	Tonnes x 10^6	g/t	oz/t	Tonnes of Au	Tonnes x 10^6	g/t	oz/t	Tonnes of Au	Tonnes x 10^6	g/t	oz/t	Tonnes of Au
1.10 (0.032) Base Case	7.54	2.42	0.071	18.24	7.49	2.36	0.067	17.67	8.28	2.26	0.066	18.72
2.17 (0.063)	4.20	3.03	0.088	12.73	3.83	3.21	0.094	12.29	2.93	3.62	0.106	10.61
Misc.					8.28	2.23			7.16 7.49 4.42	2.42 2.37 3.03		

In conclusion, and in addition to the large degree of confidence in the reserves that can be derived from these comparisons, it appears most appropriate to use the geostatistical block model of mineable reserves, as it stands, for further detailed reserves and mining engineering works.

APPENDIX 1

EXCERPT FROM:

"EXPLORATION AND PRODUCTION BLOCK MODELS AND THEIR RECONCILIATION"

APPENDIX 1

EXCERPT FROM:

"EXPLORATION AND PRODUCTION BLOCK MODELS AND THEIR RECONCILIATION"

An internal report by D. Francois-Bongarcon (including the "log-normal shortcut" method)

2. Support of the Model - the Estimation of Recoverable Reserves.

A realistic block model is based on blocks with lateral dimensions comparable to the average spacing of the data. It is well known that using much smaller blocks does not give an accurate, usable picture of what mining selection on small mining units will produce as a result. It is thus illusory to do so, and when small blocks do happen to be used, it is usually not to the effect of forecasting selective production figures.

2.1 Exploration Block Model: Economic "Index" or Forecast of Future Production?

What then can be done with an exploration block model?

In many cases, if not most, reserves are calculated from the block model, with no correction for selectivity. These reserves and tonnage-grade curves usually show more tons and less grade than can be expected and always underestimate the economic value of the project if used as is in cash-flow analyses.

However, they can be (and are) used as a qualitative "index" of economic attractiveness. One is usually able with experience to compare projects based on such an "index", therefore to gain insight on the overall economic attractiveness of a given project, and appreciate the overall shape (but not the values) of the tonnage-grade curves. Such models also may give a good representation of the spatial distribution of the rich and less rich ores and, used in conjunction with the proper optimization methodology, they may be quite suitable for optimum pit contours determination, at least as a first approximation.

Although that use of an exploration block grade model is perfectly justified, it may be quite misleading if reserves are used in cash-flow analyses or taken as production forecasts. When block-model reserves are not adjusted for mining selection, misunderstandings of their correct methodology of use are often responsible for most of the difficulty of reconciliating the geostatistical block-model with traditional geologist estimates (which usually tend to reflect some degree of mining selectivity not present in the geostatistical block model). The difference in the type of estimates (e.g. Kriging versus sectional polygons) has often been unduly overblamed for the differences.

2.2 Practical Estimation of Recoverable Reserves

2.2.1 Global Level Only:

The problem here is to forecast the frequency distribution of the grades of the mining units in the deposit. Many methods exist. The simplest is probably to use the histogram of estimated block grades corrected by affine correction of variance to give it the dispersion variance of the mining units as derived from the variogram. The change of support can also be done using a generalized permanence method (Journel, p. 467-).

In both cases, it is better to take the histogram of the block grades as a starting basis instead of the histogram of the data which is usually not declusterized, and is likely to carry more unwanted features.
A model of permanence of a two parameter

distribution (e.g. lognormal) is not recommended here because of the unnecessary strength of its distributional assumptions.

2.2.2 At the Local Level:

The most desirable and useful block model that can be derived from exploration data is one that gives an estimated distribution of mining unit grades within each block. The users of the block model are then provided with locally representative tonnage-grade curves. There are two main classes of methods available depending whether these local recoverable figures are to be derived from an existing grade block model or are directly estimated at the block modelling stage, which is usually much more costly. In the latter case, of course, a mere grade block model can always be derived afterwards if needed.

2.2.2.1 **Direct Estimation**

The available methods for direct estimation of recoverable reserves are well known: Disjunctive Kriging, Multigaussian (M.G.) approach, Indicator Kriging, Probability Kriging. See relevant literature for the pros and cons of each method and their practice.

2.2.2.2 **Starting from an Existing Grade Block-Model**

From a practical point of view, here is the most usual case: a grade block-model has been built, global recoverable reserves eventually estimated, but local recoverable reserves need to be derived without getting into the complication of a direct estimation by one of the methods listed in the previous paragraph.

The problem is thus to estimate the distribution of the mining units within each given block of the block model. Again two classes of methods can be used:

- start from a local histogram of the estimated block grades in a moving neighbourhood centered on the block.

Then the methods of paragraph 2.2.1 apply to transform that histogram into our estimated histogram of mining units.

- Or use the hypothesis of permanence of a two parameter distribution, normal or lognormal. In the latter case, it is the famous "lognormal shortcut".

Whatever method will be used, three parameters will have to be evaluated for the distribution of the mining units within the block:

- its mean m_v
- its dispersion variance σ_v^2
- the shape or type of the distribution

The mean is usually chosen as the estimated mean of the block. The variance of the local distribution around the true mean can be derived from the variogram (note here that an improper nugget effect will have a definite impact on the results). As mentioned, the shape itself can be provided by a local histogram of the blocks or by a distribution model, usually lognormal.

2.2.2.2.1 The Mean

By construction, the true mean of the mining units is the true average grade of the block (unknown). The best available estimate of the grade of the block is thus used as the mean of the distribution of the mining units. This estimate does not have to be the result of lognormal kriging, nor even kriging, but in all cases, an estimation variance is needed as we will see further.

2.2.2.2.2 The Shape or Type of Distribution

If it is elected to use local experimental histograms of kriged block grades, the way to do it is to calculate them in a moving neighbourhood of the block, small enough to be considered local and big enough to contain enough blocks to build a reasonable histogram.

An affine correction of the resulting mean m and variance V^2 can then be used to customize that histogram and make it represent the mining units within the block:

$$Z' = (Z - m)\sqrt{\frac{\sigma_v^2}{V^2}} + m_B{}^*$$

It does not seem reasonable to use a generalized permanence method here. At that level of complication, it would become more rational to model the local recoverable reserves directly using indicator kriging for instance.

One may prefer to use the normal or lognormal shortcut. Let us review here its pros and cons.

<u>Pros of the Lognormal Shortcut:</u>

- **<u>Simplicity:</u> for any cut-off grade, the selected tonnage and grade are analytical functions that can be calculated easily using approximate formulae.**

- **<u>Continuous tail:</u> the most important part of the reserves is in the tail, which is not very well informed in experimental histograms. A model of distribution provides a truly continuous tail, as needed.**

- **<u>Natural factor:</u> the normal and lognormal distributions seem to be found in nature and have worked pretty good so far for all practical purposes.** The lognormal distribution of grade values has even been shown to be a consequence of the physical "law of action of mass" which rules the chemical equilibria of geochemistry.

<u>Cons of the Lognormal Shortcut:</u>

- <u>Permanence of lognormality</u> is a theoretical impossibility. This, however, is no cause for real concern for it does not mean that the distributions cannot be fitted well using lognormal models.

- Once <u>the assumption</u> is made that the missing units have a lognormal grade distribution, all subsequent results are <u>tributary of the shape of a lognormal distribution of same mean and variance.</u> The tail, in particular, may not be represented properly, although it bears most of what makes the economic value of the deposit.

2.2.2.2.3 <u>The Variance</u>

Whatever option is elected for the shape of the distribution, I propose to use a dispersion variance calculated as follows:

$$\sigma_v^2 = K.(f.D^2(v|B) + \sigma_e^2)$$

in which:

- **$D^2(v|B)$ is the theoretical dispersion variance of the mining units in the block, derived from the variogram.**
- **σ_e^2 is the estimation variance of the average grade of the block. It is used here to correct a statistical bias due to the fact the mean is only an estimate.**
- **f is a shape factor that I propose to use to improve the representation of the tail of the histogram in the lognormal shortcut.**
- **K is a local factor which accounts for local variability conditions: local "enthropy" situation, heteroscedascity (e.g. proportional effect).**

2.2.2.2.3.1 <u>Justification of the Bias Correction σ_e^2</u>

<u>The bias</u>

The true average grade of the block provides us with the mean of the distribution of the mining units. The value available to us is only the estimated grade of the block. The true grade can be assumed to be symmetrically distributed around the estimated value, with a variance equal to the estimation variance σ_e^2 of the block.

Kriging is almost conditionally unbiased (exactly in the multinormal case), so that the average true grade of those blocks that have a given estimated value, is that value. Nevertheless, the true tonnages and grades that could be read from the distributions of mining units within those blocks, should they be centered on <u>true</u> mean values, would not average to the ones actually read from their distributions centered as they are on the estimated values of the blocks. A bias is thus generated here by the fact that the mean is only estimated. It is important to correct it.

<u>Normal Case:</u> If the true distribution of the mining units, as well as the distribution of the true mean, are considered both truly normal, the distribution of mining units to use is thus the distribution of a random variable normally distributed around a normally distributed mean! An elementary calculation (see Annex 1) shows that such a compound distribution is same as the original one but with a variance increased by the variance of the mean.

<u>General Case:</u> The previous is likely to apply well to the lognormal distribution (with the theoretical exception that the compound distribution is not any more exactly a lognormal distribution).

Moreover, by working in two different probability spaces at the same time, the following (non rigorous) demonstration confirms the result:

Let m be the true block grade, m* its estimate and σ_e^2 its estimation variance, and z the grade of a generic mining unit in the block. The dispersion variance of the mining units in the block is,

$$
\begin{aligned}
D^2(z|B) &= Var(z-m) \\
&= Var[(z-m^*) + (m^*-m)] \\
&= Var(z-m^*) + Var(m^*-m) \\
&= Var(z-m^*) + \sigma_e^2
\end{aligned}
$$

Conclusion: If the distribution used is centered on the estimate m^*, then the variance must be increased by σ_e^2 .

2.2.2.2.3.2 The Shape Factor f:

As already mentioned, in the case of the lognormal shortcut, results rely totally upon the lognormality of the mining units. Also, the economically most decisive part of the distribution, the tail is the one less likely to be properly adjusted by a lognormal distribution. In particular, the tail of the adjusted lognormal distribution is often too short.

The factor f increases the dispersion variance to provide a longer tail. f can be determined by comparing the tails of the histogram of kriged block grades and of the lognormal distribution of same mean and corrected variance.

2.2.2.2.3.3 **The Local Factor K**

The grades in a gold deposit are a typical case of heteroscedacity, i.e. of the variance being related to the level of the grade values. This has always been known to geostaticians: they study the "proportional effect" in order to use locally meaningful variance values (and variogram sill values) (David p.70, Journel p.186) and account for the higher variability of the grades in the richer areas of the deposit. Note that the variogram itself will give no warning when such a situation occurs.

Even at a given level of local average grade, the variability can still be different in different areas. One of the possible reasons, sometimes referred to as "variable enthropy", is when the intermixing of rich and poor grades at small scale presents itself in different ways in different areas. The grade of a non-punctual support then presents different variabilities in areas with the same local grade. The variance increases when enthropy (order) decreases.

Therefore, the factor K should try as much as possible to cover not only the proportional effect classically known from a classical mean-variance relation valid on average throughout the deposit, but also local departures from that average relation. These are the proposed steps involved in a thorough determination of K:

1. Study the proportional effect and modelize the average relation $V^2 = f(m)$.

2. Using the local neighbourhood mentioned in paragraph 2.2.2.2.2, calculate the local mean m_L and the local variance V_L of the data and evaluate the local departure from the average proportional effect relation as a ratio R:

   $$R = V_L^2/f(m_L)$$

3. Calculate the variance attached to the average grade M_B of the block using:

   $$V_B^2 = R.f(m_B) = V_L^2.f(m_B)/f(m_L)$$

   (See Figure 1)

4. Calculate the resulting factor K for the block, according to on what value was actually used as the sill of the model of variogram used.

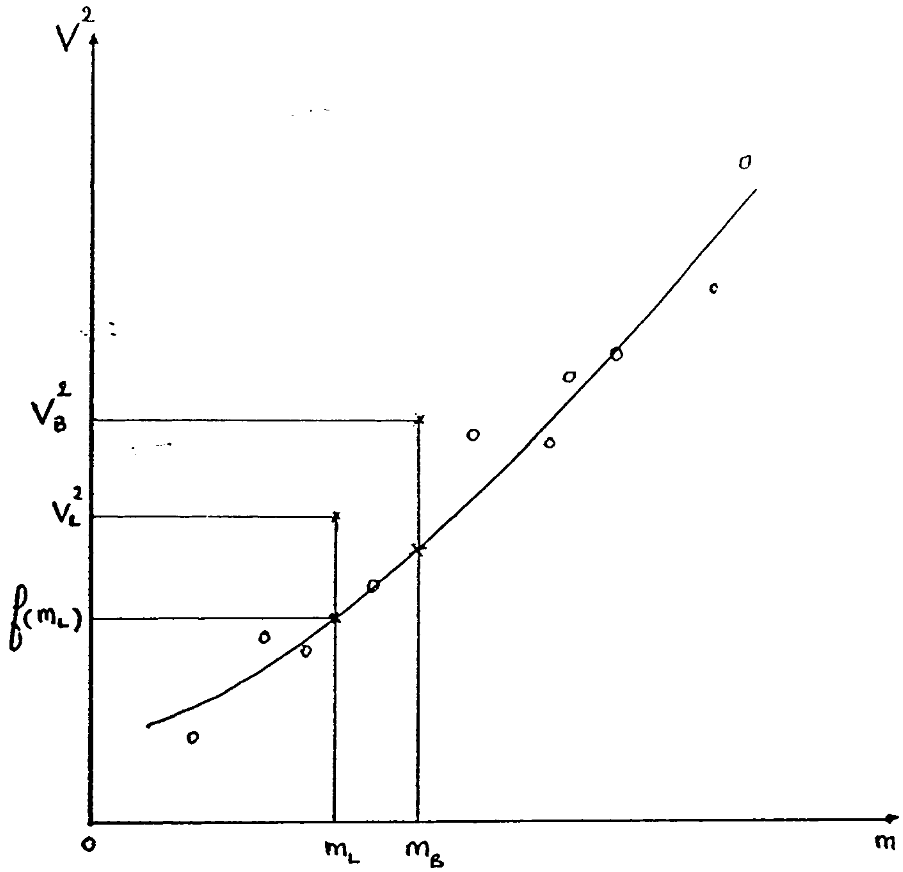


FIGURE 1

Appendix 1

Let X be a R.V. following a normal distribution (m, σ)
Φ_X its characteristic function

Let now m be a realization of a R.V. M which follows a normal distribution (m, σ_e) and Φ_M its characteristic function.

The characteristic function of X is

$$\Phi_X(u) = E(e^{iuX}) = \text{expon.}(ium + \frac{\sigma^2 . u^2)}{2}$$

The characteristic function of the compound R.V. X_M can be calculated as:

$$\Phi(u) = E_M\ [\Phi_X(u)] = E[\text{expon.}(eiuM + \frac{\sigma^2 . u^2)}{2})]$$

$$= e^{\frac{\sigma^2 u^2}{2}} . E(e^{iuM})$$

$$= e^{\frac{\sigma^2 u^2}{2}} . \Phi_M(u)$$

$$= \text{expon.}[\ ium + \frac{(\sigma^2 + \sigma_e^2).u^2}{2}\ ]$$

Same distribution as X, but with variance $\left(\sigma^2 + \sigma_e^2\right)$!