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**AN INTEGRATED PACKAGE FOR MINE
MANAGEMENT, ORE RESERVE ESTIMATION
AND ORE BLENDING OPERATION**

Mostafa E. Mostafa
Nuclear Materials Authority,
P.O. Box 530, Maadi,
Cairo, Egypt
e-mail Mostafamem@hotmail.com

ABSTRACT

The present package includes a rich set of integrated modules for managing mine data like borehole location, down hole metric samples and assays. Checklists are prepared for selected boreholes and intermediate files are used for constructing semivariograms and contour maps and for visualizing the ore body, surface topography, overburden and chemical assays.

At a selected block size, kriging and reserve estimation of the ore are conducted in addition to the cumulative and grade/tonnage/cut off curves. For the sake of completeness, the estimation of deposits by triangle as one of the classical method is provided.

The blending operation aims to mix ores collected from different mine faces so that the final mixture assays fulfill some metallurgical constraints. Linear Programming is applied for solving such problems. The optimal solution (if present) is that which minimizes the objective function (cost of ore extraction) under the given constraints.

The package is illustrated through a hypothetical ore deposit simulated under controlled condition, therefore, it can be used for optimizing study as well as educational purposes.

I. INTRODUCTION

The present system was primarily written in Fortran77 for a PC-XT computer. The system actually started in 1987 by installing El Gedida Mineral Information System (GMIS) which was analyzed and designed by this author during his delegation from the Nuclear Materials Authority, Cairo, Egypt to HADISOLB as a consultant in ore reserve estimation (Mostafa, 1989). This was funded by part of a loan submitted from the International Bank for the automation of El Gedida mine under the supervision of the British Steel Corporation at Tesside Laboratory, UK (BSC, 1985). Later on, the system was rewritten in QBASIC to make use of the graphic library. Finally, it was found easy to convert the code to Visual Basic in the form of an integrated package running under Windows environment. The package can be customized to accommodate different types of ores other than strata bound deposits. Data can be communicated with programs such as ROCK WARE, SURFER and EXCEL for contour mapping, graphical presentation, tabulation and spread sheet calculations. The package is provided with utility programs for data input and output, slicing thick ores, calculating averages of assays, preparing semivariograms (SVGs), kriging, reserve estimation and ore blending operation.

It is worth mentioning that the package has been used several times in estimating different deposits. The estimation of the coal seam of Gebel Maghara coal mine, North Sinai (Morsy et al., 1992) and the iron ore deposit of El-Gedida Mines, Bahariya Oasis (Mostafa, 1989 and Hasan, 2001) are two prominent examples.

For privacy reason, the present package will be illustrated through a hypothetical ore body. In general, simulation of a phenomenon requires mastering the physical/mathematical laws so that the phenomenon can be repeated under specific conditions with successive approach to the real world. Simulation in geology dates back to the basic work of Harbaugh and Carter (1970) and requires basic knowledge in the field of study. For example, the lineaments detected from satellite images and aerial photos are mostly related to brittle failure of rocks. Therefore, the simulation of these lineaments leads to the exact counting and proper analysis and formulation of a general rock failure criterion (Mostafa and Qari, 1995, Mostafa and Zakir, 1996, Zakir et al., 1999 and Mostafa 2003). Moreover, the simulation of an ore body and related objects will eventually increase our knowledge about the development of such a process in nature. Thus, the simulation allows us to control the conditions under which ore bodies are formed. These predefined conditions help applying different optimization studies. The package can also be used for educational purposes.

The main objective of the present work is to apply systematic geostatistical techniques to estimate the hypothetical ore deposit and to prepare Grade/Tonnage/Cutoff curves characterizing the deposit. The evaluation was carried out to fulfill the following steps:

- Study the spatial variability of the ore body thickness, C1 component and other associated elements.
- Estimate the reserve of the deposit at different mining block sizes.
- Demonstrate the balance between tonnage, grade cut-off and average grade.

II. DATA HANDLING

The simulated data pertains to six zones. Each zone has two master files, namely, BHTOPO holds borehole and ore body data and BHSAMPLES holds down hole metric samples and assays. The following requirements have to be fulfilled to get the data into the master files:

- Borehole name, zone/campaign number.
- BH location (X-Easting, Y-Northing and Z-Elevation) and ore body top and base elevation.

- Assays of down hole metric samples for C1, C2, C3, C4 and C5. Figure (1) shows the interactive Borehole Manager window (BH-Manager). Boreholes are plotted in different colors according to the zone number. The package is provided with functions for performing the following tasks:
- Selection and display of a single or a group of boreholes within a rectangle or polygon.
- Borehole checklist.
- Borehole averages in the form of row-column format.
- Bench-slicing information for a whole zone or a block of it (full ore thickness was considered in the present study).

It is essential that all data be checked after entry and that any necessary amendments are carried out as soon as possible. The editing routines allow one to alter individual sample value, delete or insert individual sample lines and delete an entire borehole.

A full description of BHN#7, Zn#0 is given as a checklist (Fig.2), which can be used for checking data entry and reporting. Table (1) shows some selected boreholes from zone#0 including:

- Borehole location and ore bed topography.
- Down hole sample assays.
- Borehole average assays.

III. THE ORE BODY

Visualizing an ore body aims to investigate its morphological and geological setting. Figure (3) shows a 3-D surface topography of the mined area. Figure (4) shows three borehole logs selected from different zones; their locations are shown as small circles in Fig. (1). The logs are correlated relative to the sandstone bed representing the ore body which is underlined by a conglomeritic bed and overlaid by a thick succession of shale. The contour maps in figure (5) show (a) the surface topography of the mined area, (b) thickness of the overburden rocks, (c) thickness of the ore body and (d) average assays of C1 component. The interpretation of such maps completely defines the spatial setting of the ore body. Table (1) and Fig. (2) show that the ore bed thickness is calculated from the difference between the two topographic levels T2 and T4.

IV. GEOSTATISTICS

The knowledge of grades and reserves of an ore as well as error estimation of these values are fundamental for mining engineers and mining geologist. Old methods estimate the ore as if the grade variance were constant everywhere leading to what is called a “deterministic model of estimation” which is characterized by an estimation error equals to zero. However, grade distribution in space can be accounted for by SVG constants. On the basis of variogram, geostatistics not only estimates the grade and reserve, but also the error estimation of these two values. This leads to the “undeterministic model of estimation”.

David (1977) states that kriging is in fact a word which has been coined to cover both “best linear unbiased estimator“ (BLUE) of a point and the best linear weighted moving average of a block. This particular nature of estimating problems in mine planning is such that it most probably deserves the use of a spatial name. This name appeared around 1960 to designate a technique created in France by Matheron (1963a,b) after the name of D.C. Krige; a South African mining geologist who was probably the first to make use of spatial correlation and BLUE in the field of mineral resources evaluation in 1951.

Details of the SVG and kriging techniques have been recently issued by Carr (1995) who provides the code in Fortran-77 along with a lot of illustrations and explanations. He

used the covariance data to construct the kriging system of equations and for calculating the kriging variance instead of using Clark's auxiliary functions. These auxiliary functions adapted to the spherical model are provided as subroutines (Clark, 1976) or tables (Clark, 1982) and have markedly improved the kriging method and made it amenable for computer programming

Semivariogram

The experimental SVG can be constructed along a specified direction or for all directions (global). The data from all zones has been considered in constructing the global SVGs for ore thickness and C1 target assay. The calculation of SVGs helps determine the structure of a deposit and the behavior of grade variations and also serves as a base for the next kriging step.

The SVG curve of each parameter is adjusted to a spherical model to get the variograms constants including nugget (C0), sill (C1) and range (a). The SVGs are given in figure (6) along with their constants. The latter are grouped into a file for subsequent kriging. Details of the SVG calculation are given in Append. A.

Kriging

Figure (7) shows the interactive window for the geostatistical calculations including kriging, reserve estimation and ore characterizing curves. The following data are necessary for kriging:

- Boundary of the kriged area (polygon /rectangle)
- Indices of the SW corner
- Block size.
- Number of blocks to be kriged east and northward.

The calculations will only be carried out on the blocks inside the selected polygon. For each block, the thirty surrounding boreholes (at most) are stored and the averages of the nearest six holes are considered in kriging.

In the kriging operation, the user should define the parameter to be kriged (thickness and other assays). SVG constants corresponding to each selected parameter are read from the SVG file which contains two rows, one for each parameter (thickness and C1-component) and three columns for their corresponding constants of the spherical model (range, sill and nugget). This procedure aims to estimate the ore at different block sizes; 50x50, 100x100 and 150x150m. Block size 100x100m is chosen for all presentations and reserve calculations and consequently for cumulative and grade/tonnage curves. At block size 100x100m, 104 kriged blocks cover the polygon-shaped area as shown in Figure (7) and the result of kriging is given in Table(2).

In the present work, the full thickness of the ore bed is prepared for kriging. There are some considerations about the cut off thickness or assays and the critical thickness of gangue layers to be excluded/included with the ore. These mining constraints can be considered in the calculation if needed.

V. RESERVE ESTIMATION

The quality of estimation depends essentially on the number of samples available, the size of these samples with respect to that of the sample space, the spatial distribution of these samples and the intrinsic dispersion of the mineralization.

Estimated values for ore thickness and C1 component are calculated at different block sizes: 50, 100, 150 meters. Only, the result of estimation by blocks 100x100m is discussed. Estimates of kriged thickness of the same area differ according to the size of block. The total reserve of the polygon shaped area (Figs.7, Table 2) is summarized as follow:

Block dimension = 100x100m
 Total number of estimating blocks = 104
 Estimated reserve = 9,278,200 ton
 Weighted block average (C1- Component)=14.06%
 Average block thickness = 4.46 m
 Average Block grade (C1- Component) = 13.99%
 Standard deviation (C1- Component) = 1.224%

VI. GRADE/TONNAGE/CUT OFF CURVES

The histogram and cumulative curve for the values of the grade/tonnage distribution estimated by 100mx100m block size are prepared and given in Figure (8a,b). The cumulative curve gives the proportion of the ore above any given cut off.

The preparation of the cut off grade/average grade above cut off curves is of primary importance for mining geologists in reserve estimation and grade control. Fig.(8c) enables us to get the average ore grade above any given cut off. and it follows the ore reserve estimation at the optimal block size. Data points of the curve are calculated in terms of the cut off grade (c), average grade (\bar{g}) and standard deviation (s) of the ore to cover the normal distribution behavior (Z-table). Details of the method is given in Append. C.

VII. CLASSICAL RESERVE ESTIMATION

Classical methods of reserve estimation are frequently used in case of gypsum, bentonite and similar deposits characterized by stationary grade/chemical assays. Estimation by triangle is one of these methods which involves constructing a triangle with drill holes or pits/trenches at the vertices. The volume of a triangular prism is calculated, the sum of the total volume of prisms is multiplied by the density to give the estimated tonnage along with the weighted average grade. It is common practice to weight the average grade of the deposit in each borehole by its respective distance from the point of intersection of the median lines at the center of the triangle.

Figure (9) shows the interactive window for triangular prism estimation. The screen displays map dimension and the working locations (borehole, trench, pit) annotated with one of the following items according to the marked check box:

- Borehole name.
- Ore thickness.
- Ore base elevation.
- Ore Top elevation.
- Chemical assay.

Moreover, a tool-tip displays the above five pieces of information in the moment the mouse is close to a working.

The selection frame selects one of three different jobs:

- Draw Boundary: Workings defining ore boundaries are clicked in sequence to automatically create a group of labeled triangles radiating from a common vertex.
- Drawing Triangles: Mouse movement automatically illuminates the near by triangle which is labeled upon clicking.
- Estimate triangles: Clicking the labeled triangles leads to current and cumulative reserve calculation.

At any stage of processing, the above-mentioned jobs can be edited, saved and resumed. Table (3) shows a summary of reserve calculation of the labeled triangles in figure (9).

VIII. ORE BLENDING OPERATION

Proper blending is necessary to achieve optimal depletion plan and mining recovery. The blending operation aims to mix ores which are collected from several working faces each of which has its own chemical assays, estimated reserve and cost of extraction. The final assay of the mixed ore should fulfill some constraints necessary for metallurgical purposes.

Linear Programming (LP) is an algorithm for solving problems under a given set of constraints in the presence of given objective function(s). The objective function is either a cost to be minimized or a profit to be maximized. The algorithm is part of the field of Operations Research (Nedler and Mead,1965). The art of scientific computing is given by Press et al. (1981) who provide a miscellaneous collection of mathematical and statistical libraries and subroutines.

Linear Programming (LP) can be applied for solving the problem of ore blending. The optimal solution (if present) is that which minimizes the objective function (cost of ore extraction) under the given constraints.

Figure (10) shows the interactive window for the blending operation. The operation starts by displaying a number of working faces along with their chemical assays and reserves in ton. The user marks the check boxes of the selected faces needed for blending. If the trial is successful, removed tonnage is displayed under the contributing faces. The assay of the current blending trial is also shown. If the trial fails, other combined faces are selected as long as there are more promising faces. Check boxes of the exhausted faces are automatically disabled.

The input file is updateable and contains data pertaining to different working faces (face number, reserves in ton, chemical assays) as well as the metallurgic constraints.

There are many paths the program can go through. Finding the path that achieves the highest depletion depends to some extent upon the user's skill. At any stage, the program can be reset for repeating the operation. Parallel with each trial of the blending operation, the following calculations are performed:

- Chemical assays of the blended ore.
- Removed/remaining ore for each face.
- Overall extracted ore from the selected faces.
- Final extracted ore tonnage.
- Final assays.

The program keeps track of the trials for following up and printing of the final report (Fig. 11).

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Append. A

The experimental semivariogram is calculated using the formula::

$$2\gamma(h) = \frac{1}{N(h)} \sum_{i=1}^N [Z(i) - Z(j)]^2$$

where:

h = distance between i-th and j-th samples

N(h) = number of sampled pairs at h-distance.

z(i) = grade at sample i

z(j) = grade at sample j

For a certain direction, the calculated $2\gamma(h)$ is plotted against values of h to give the experimental SVG. The spherical SVG model:

$$\gamma(h) = c_0 + c_1 \left[\frac{3h}{2a} - \frac{1}{2} \cdot \frac{h^3}{a^3} \right] \quad h < a$$

$$\gamma(h) = c_0 + c_1 \quad h > a$$

is used for fitting the experimental SVG to get the constants c_0, c_1, a

Append. B

According to Clark (1982), the estimation variance for the general "unbiased linear" estimator as a function of the weights (w) is given by:

$$\sigma_\varepsilon^2 = 2 \sum w_i \bar{\gamma}(S, A) - \sum_{i=1}^n \sum_{j=1}^n w_i w_j \bar{\gamma}(S_i, S_j) - \bar{\gamma}(A, A)$$

where:

$\bar{\gamma}(S_i, S_j)$ = average semivariogram value between i-th and j-th samples
(i, j=1,2,3,...n = total number of samples)

$\bar{\gamma}(S, A)$ = average semivariogram value between sample point and every point within the block A.

$\bar{\gamma}(A, A)$ = average semivariogram value between points within the block A

Using LaGrange's multiplier, the variance and the constraint:

$$\sum w_i - 1 = 0$$

are combined as follows:

$$\sigma_\varepsilon^2 - \lambda(\sum w_i - 1)$$

Differentiating the variance with respect to the weights and equating to zero: $\frac{\partial \sigma_\varepsilon^2}{\partial w_i} = 0$

gives the normal system of (n+1) equations in (n+1) unknowns (w's and λ) known as kriging system of equations:

$$w_1 \bar{\gamma}(S_1, S_1) + w_2 \bar{\gamma}(S_1, S_2) + w_3 \bar{\gamma}(S_1, S_3) + \dots w_n \bar{\gamma}(S_1, S_n) + \lambda = \bar{\gamma}(S_1, A)$$

$$w_1 \bar{\gamma}(S_2, S_1) + w_2 \bar{\gamma}(S_2, S_2) + w_3 \bar{\gamma}(S_2, S_3) + \dots w_n \bar{\gamma}(S_2, S_n) + \lambda = \bar{\gamma}(S_2, A)$$

$$w_1 \bar{\gamma}(S_3, S_1) + w_2 \bar{\gamma}(S_3, S_2) + w_3 \bar{\gamma}(S_3, S_3) + \dots w_n \bar{\gamma}(S_3, S_n) + \lambda = \bar{\gamma}(S_3, A)$$

$$w_1 \bar{\gamma}(S_n, S_1) + w_2 \bar{\gamma}(S_n, S_2) + w_3 \bar{\gamma}(S_n, S_3) + \dots w_n \bar{\gamma}(S_n, S_n) + \lambda = \bar{\gamma}(S_n, A)$$

$$w_1 + \dots w_2 + \dots w_3 + \dots w_n \dots = \dots 1$$

The weights minimize the estimation variance (kriging variance) are calculated as follows:

$$\sigma_k^2 = \sum w_i \bar{\gamma}(S_i, A) + \lambda - \bar{\gamma}(A, A)$$

$\bar{\gamma}(S_i, S_j)$ are evaluated from the semivariogram.

$\bar{\gamma}(A, A)$ and $\bar{\gamma}(S, A)$ are evaluated from the H and F auxiliary functions respectively and they are given as subroutines or tables (Clark, 1976, 1982).

Append. C

Clark (1982) states that the proportion (p) of an ore above certain cut off is determined using the normal distribution curve (Z-table) as follows:

$$p = \text{pr } \{g > c\}$$

the z-score
$$z = \frac{c - \bar{g}}{s}$$

where:

\bar{g} = average grade

s = standard deviation

c = cut off grade

From the normal table $\Phi(z)$ measures the probability of lying below the cut off (c) and $1 - \Phi(z)$ measures the probability above the cut off.

The average grade (\bar{g}_c) of the ore above the cut off is calculated as follows:

$$\bar{g}_c = \bar{g} + \frac{s}{p} \phi(z)$$

$\phi(z)$ measures the height of the standard normal curve at the value z:

$$\phi(z) = \frac{1}{\sqrt{2\pi}} \exp(-z^2 / 2)$$

By calculating z-value for the different cut off grades, the proportion of the distribution and average grade above the cut off are calculated to construct the cut off grade/average grade above cut off curves characterizing the ore deposit.

Append. D

Press et al. (1981) formulate the Linear Programming problem as follows:

Under the set of N-independent variables x_1, x_2, \dots, x_N , the cost function:

$$z = a_{01}x_1 + a_{02}x_2 + \dots + a_{0N}x_N$$

needs to be minimized subject to the primary constraints:

$$x_1 \geq 0, \dots, x_2 \geq 0, \dots, x_N \geq 0$$

and simultaneously subject to $M = m_1 + m_2 + m_3$ additional constraints, m_1 of them has the form:

$$a_{i1}x_1 + a_{i2}x_2 + \dots + a_{iN}x_N \leq b_i, \quad (b_i \geq 0) \dots \dots i = 1, \dots, m_1$$

other m' s can be similarly formulated according to the problem in hand.

TABLES AND FIGURES

Table (1) A selected sample of Borehole Data, Zone #0

(a) Topographic Data

BHN	X	Y	Zone	N	Topographic Levels, m				
					T1	T2	T3	T4	T5
1	569	2343	0	7	113.88	74.04	71.06	67.32	66.11
2	1492	2499	0	7	110.10	70.25	67.40	63.54	59.54
3	3133	2572	0	7	103.28	69.36	66.14	62.40	61.19
4	1132	2560	0	7	112.13	71.64	68.66	65.05	63.71
5	2429	2599	0	7	106.30	68.49	66.14	62.28	57.53
6	3339	2497	0	7	102.28	69.88	66.76	63.15	61.81
7	456	2265	0	7	113.88	74.55	71.31	67.57	66.36
8	2808	2571	0	7	104.80	68.86	65.89	62.03	60.94
9	3027	2410	0	8	102.78	69.24	66.01	62.15	61.06
10	2792	2508	0	7	104.28	68.74	65.50	61.76	60.55
11	1129	2512	0	7	111.85	72.01	69.04	65.30	64.09
12	743	2480	0	7	113.63	73.29	70.18	66.44	65.22

(b) Down hole Assays, BHN #7

SN	Depth, m		Assays, %				
	Fr	To	C1	C2	C3	C4	C5
1	74.55	73.55	15.24	7.82	11.05	14.57	9.15
2	73.55	72.55	17.17	11.69	9.66	9.26	13.02
3	72.55	71.55	19.67	16.69	6.47	18.01	15.63
4	71.55	70.55	17.23	11.81	11.39	15.26	13.52
5	70.55	69.55	20.2	17.74	8.02	25.57	4.81
6	69.55	68.55	20.4	18.15	6.34	15.75	1.2
7	68.55	67.57	15.92	9.19	3.21	21.6	5.64
average			17.98	13.30	8.02	17.15	9.00

(c) Average assays

BHN	X	Y	Thick, m	N	Average Assays, %				
					C1	C2	C3	C4	C5
1	569	2343	6.72	7	18.382	13.33	7.54	16.73	13.57
2	1492	2499	6.71	7	18.005	11.02	9.01	11.44	8.54
3	3133	2572	6.96	7	20.593	15.47	8.08	16.37	7.98
4	1132	2560	6.59	7	19.712	13.83	9.11	17.30	8.67
5	2429	2599	6.21	7	17.197	8.40	6.05	11.76	8.14
6	3339	2497	6.73	7	19.559	14.15	8.44	13.78	6.94
7	456	2265	6.98	7	17.981	13.31	8.03	17.13	9.01
8	2808	2571	6.83	7	20.037	14.36	7.55	16.91	11.72
9	3027	2410	7.09	8	17.928	11.75	7.99	13.89	11.01
10	2792	2508	6.98	7	18.953	12.83	6.27	13.43	10.42
11	1129	2512	6.71	7	18.501	11.88	6.98	14.99	7.68
12	743	2480	6.85	7	18.611	12.42	8.46	13.99	10.52

Table (2) A sample of kriged block thickness and grade estimates, polygon shaped area (Fig. 7) (100x100m block dimension)

Block center, m		Ore body		Grade	
x	y	thick, m	kriging var.	C1, %	kriging var.
1570	970	4.75	0.46	12.37	0.73
1670	970	4.59	0.31	12.51	0.54
1770	970	4.43	0.46	12.58	0.74
1870	970	4.26	0.49	12.18	0.78
1970	970	4.05	0.51	11.72	0.81
2070	970	3.78	0.52	11.97	0.83
2170	970	3.7	0.31	12.42	0.54
1270	1070	5.69	0.36	13.12	0.6
1370	1070	5.27	0.32	13.08	0.55
1470	1070	5.04	0.41	12.99	0.66
1570	1070	4.87	0.43	12.82	0.7
1670	1070	4.67	0.36	12.81	0.6
1770	1070	4.48	0.42	13	0.68
1870	1070	4.3	0.4	12.85	0.66
1970	1070	4.07	0.4	12.24	0.67
2070	1070	3.86	0.4	12.22	0.66
2170	1070	3.76	0.31	12.38	0.52
2270	1070	3.68	0.33	12.05	0.56
2370	1070	3.41	0.32	11.55	0.53
1170	1170	5.94	0.37	13.82	0.61
1270	1170	5.63	0.36	13.69	0.6
1370	1170	5.28	0.36	13.78	0.6
1470	1170	5.1	0.3	13.68	0.52
1570	1170	5.01	0.37	13.1	0.6
1670	1170	4.79	0.35	12.9	0.58
1770	1170	4.56	0.35	13.49	0.59
1870	1170	4.4	0.37	13.7	0.62
1970	1170	4.13	0.41	13.08	0.66
2070	1170	3.92	0.39	12.89	0.64
2170	1170	3.76	0.37	12.93	0.63
2270	1170	3.66	0.34	12.43	0.57
2370	1170	3.54	0.31	12.04	0.53

Table (3) Summary of reserve estimation using the triangle method (Fig. 9)

Triangle number	Vertex number			Ton, million	Weighted average%
	V1	V2	V3		
1	42	29	12	73.6	29.24
2	42	12	18	98.4	26.50
3	42	18	7	166.8	27.80
4	42	7	5	223.9	26.95
5	42	5	46	204.2	30.69
6	42	46	29	89.9	32.55
Overall tons				856.8	
Grand average%					28.96

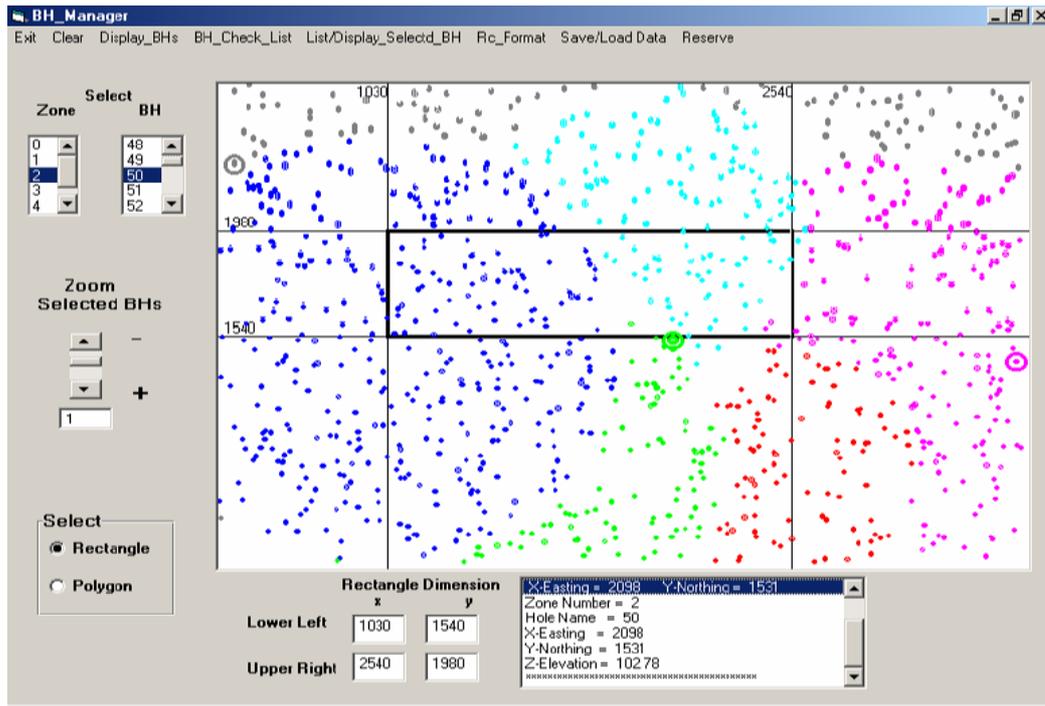


Fig. (1) Interactive window for managing borehole data

Zone Number = 0
 Bore Hole Name = 7
 X-Easting = 456
 Y-Northing = 2265
 Z-Elevation = 113.88
 Over_Burden = 39.33
 Bed_Thicknes = 6.98
 Bed Top Elevation = 74.55
 Bed_Base_Elevation = 67.57
 Total Number of Samples = 7
 Start Sample = 43
 End Sample = 49
 Sample Assays

Fr	To	C1	C2	C3	C4	C5	Length
074.55	073.55	015.24	007.82	011.05	014.57	009.15	01.00
073.55	072.55	017.17	011.69	009.66	009.26	013.02	01.00
072.55	071.55	019.67	016.69	006.47	018.01	015.63	01.00
071.55	070.55	017.23	011.81	011.39	015.26	013.52	01.00
070.55	069.55	020.20	017.74	008.02	025.57	004.81	01.00
069.55	068.55	020.40	018.15	006.34	015.75	001.20	01.00
068.55	067.57	015.92	009.19	003.21	021.60	005.64	00.98
Average		17.98	13.31	8.03	17.13	9.00	

Fig. (2) Borehole check list

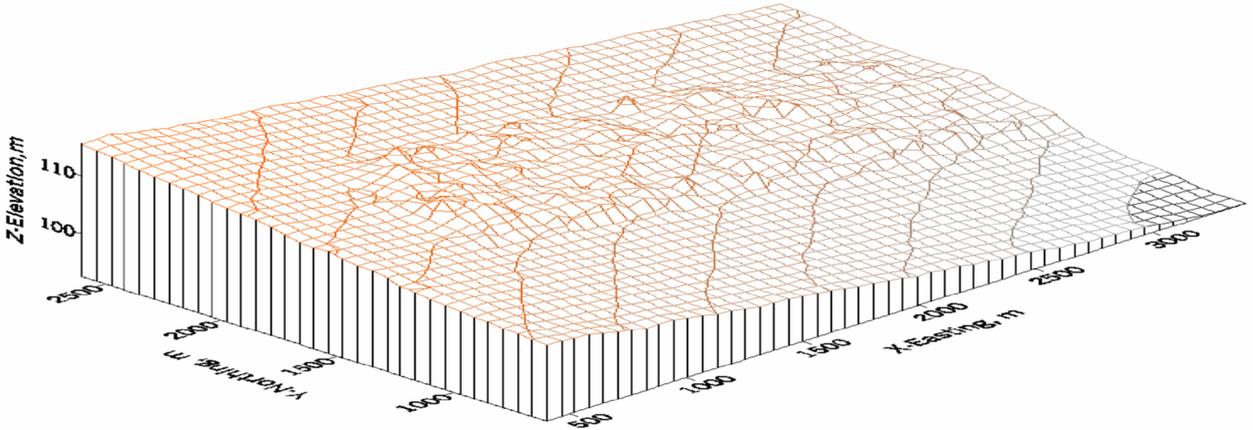


Fig. (3) Topography of the mine area

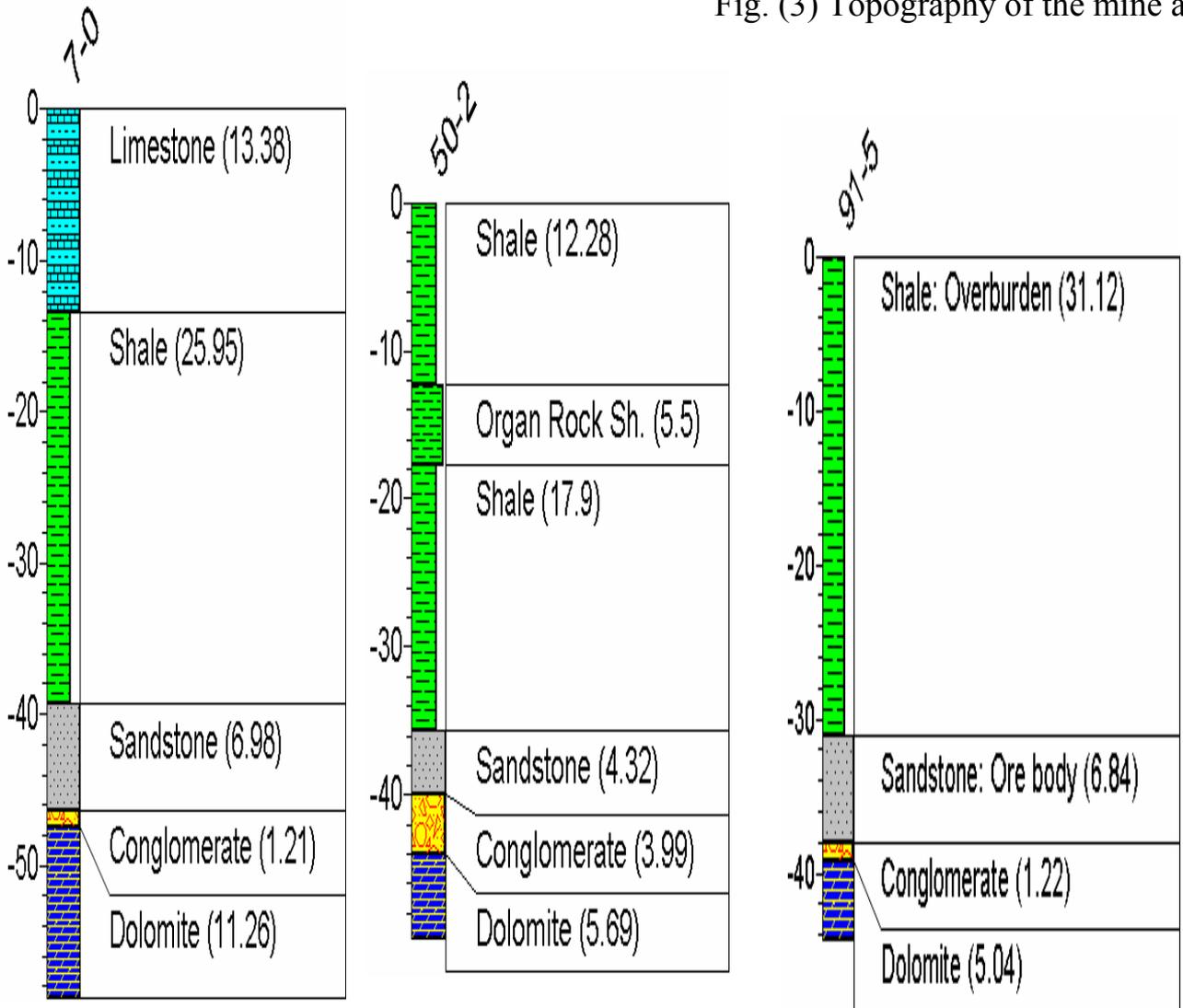


Fig. (4) Selected Borehole logs, the mining area.
 (BHN#7, Zone#0),(BHN#50, Zone#2), (BHN#91, Zone#5)

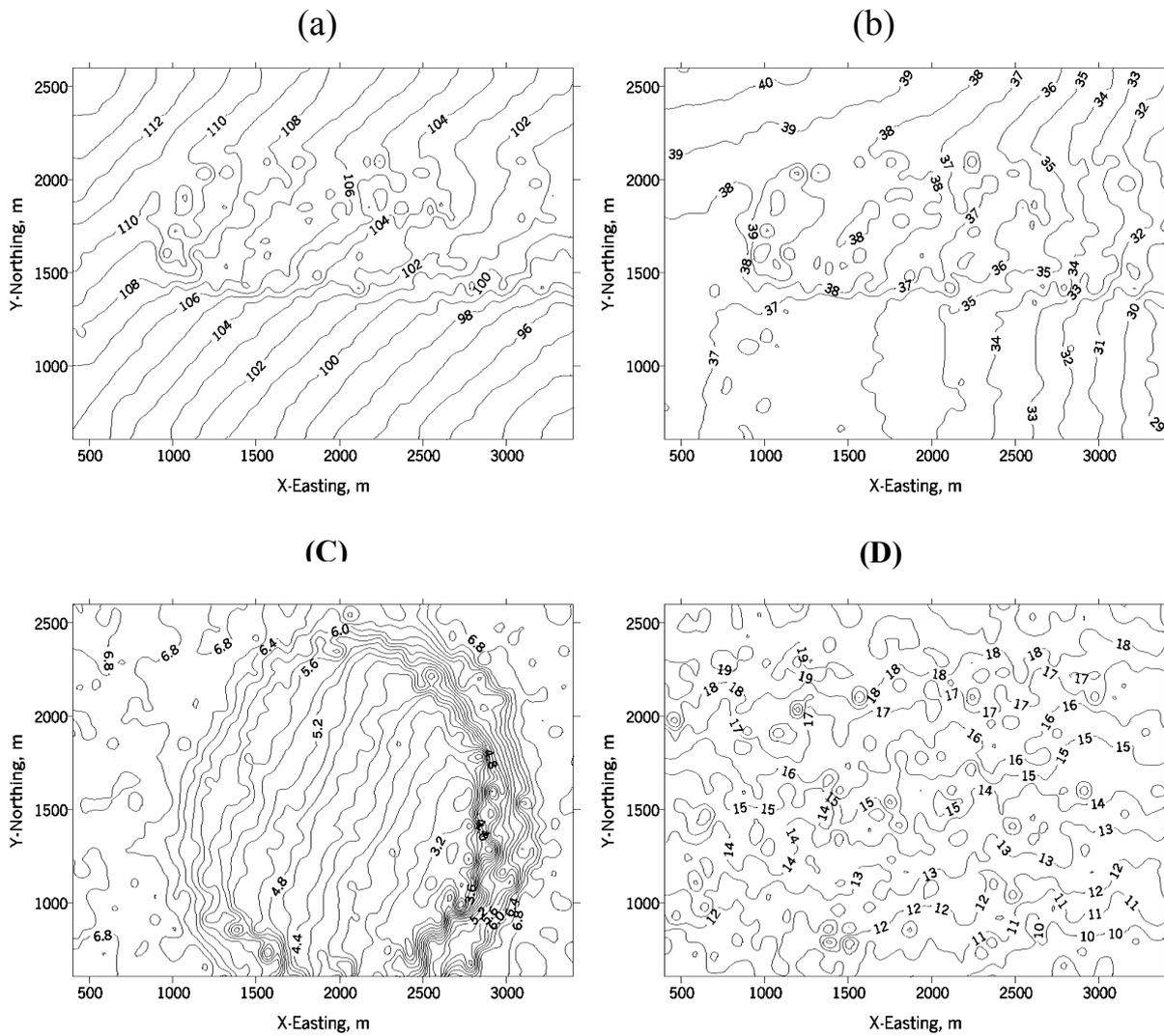


Fig. (5) Contour maps:
a- Topography
b- Overburden
c- Ore bed thickness
d- C1-Component assay

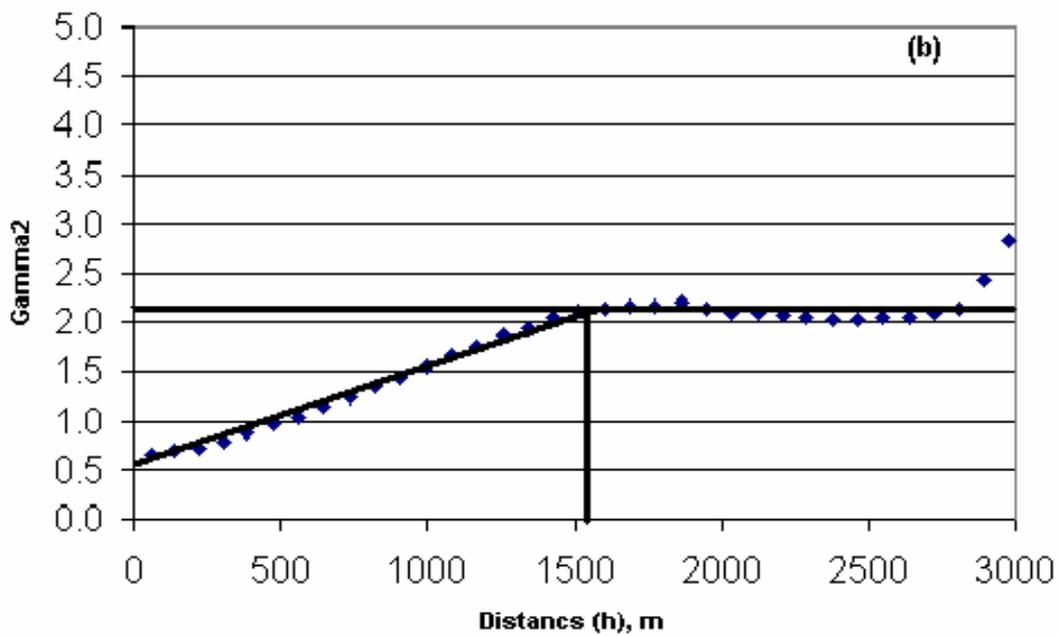
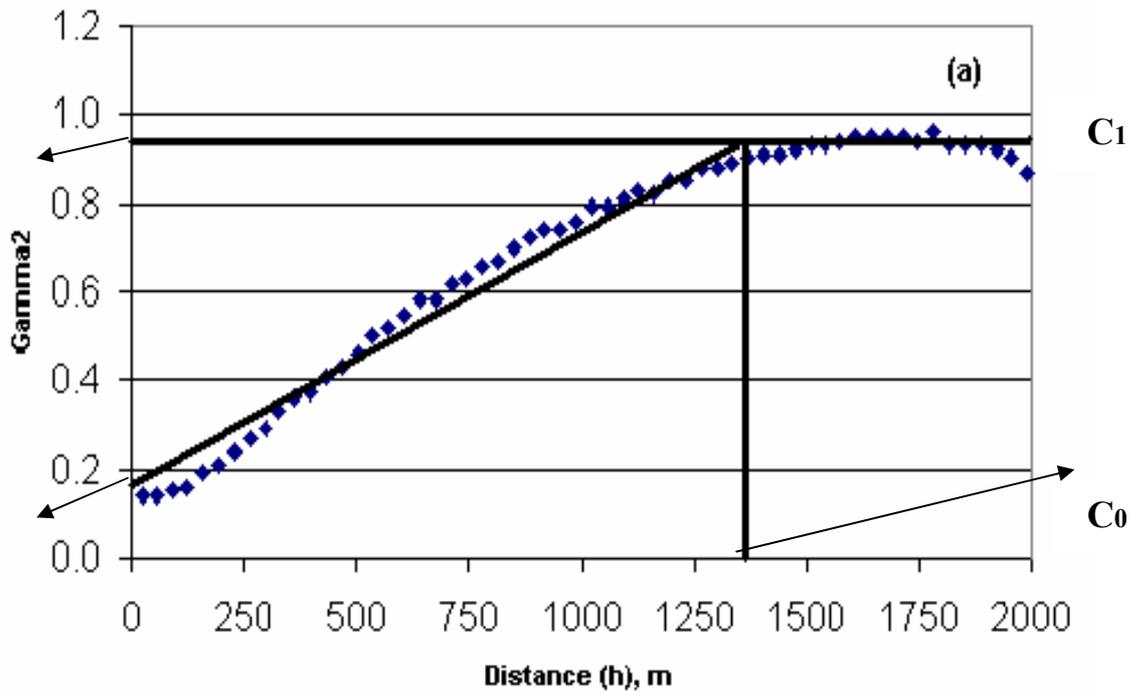


Fig. (6) Experimental semivariograms with spherical model

Constants (C_0 , C_1 , a)

a- Thickness

b- C1-Component assay

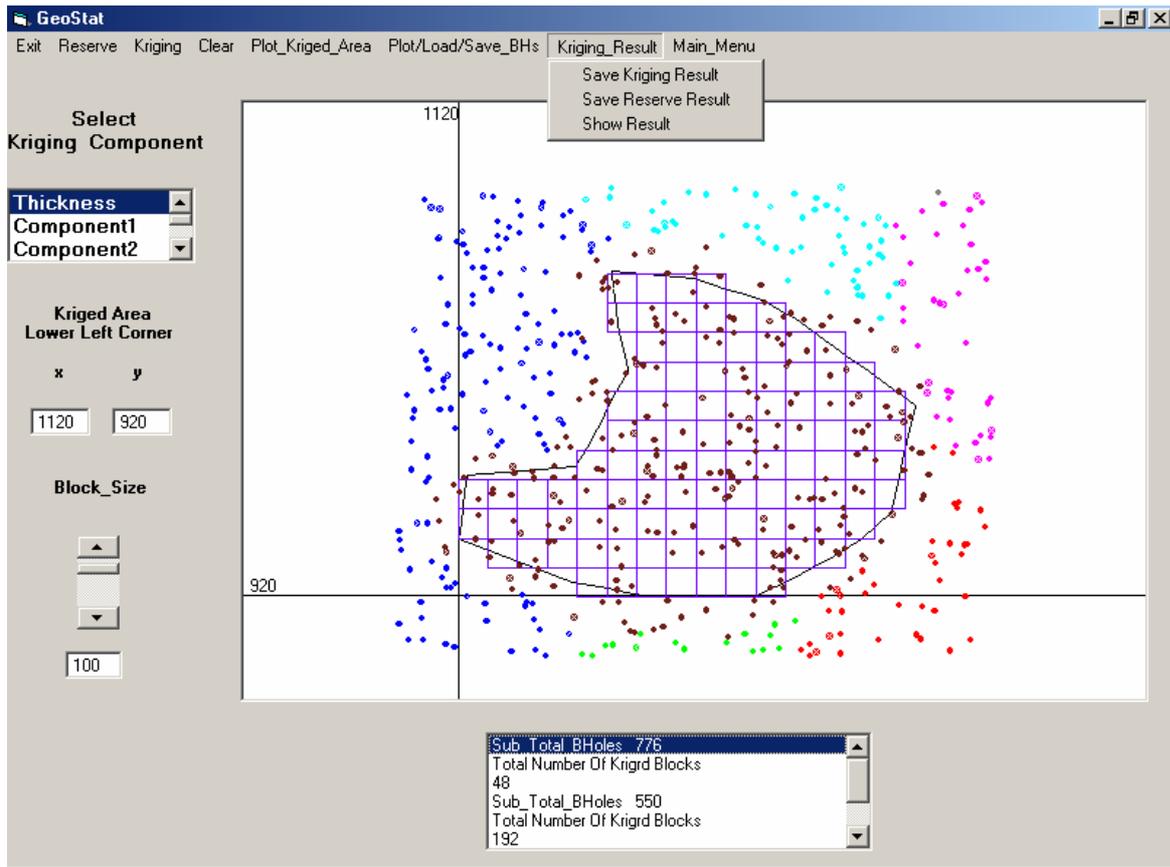


Fig. (7) Interactive window for geostatistical analysis

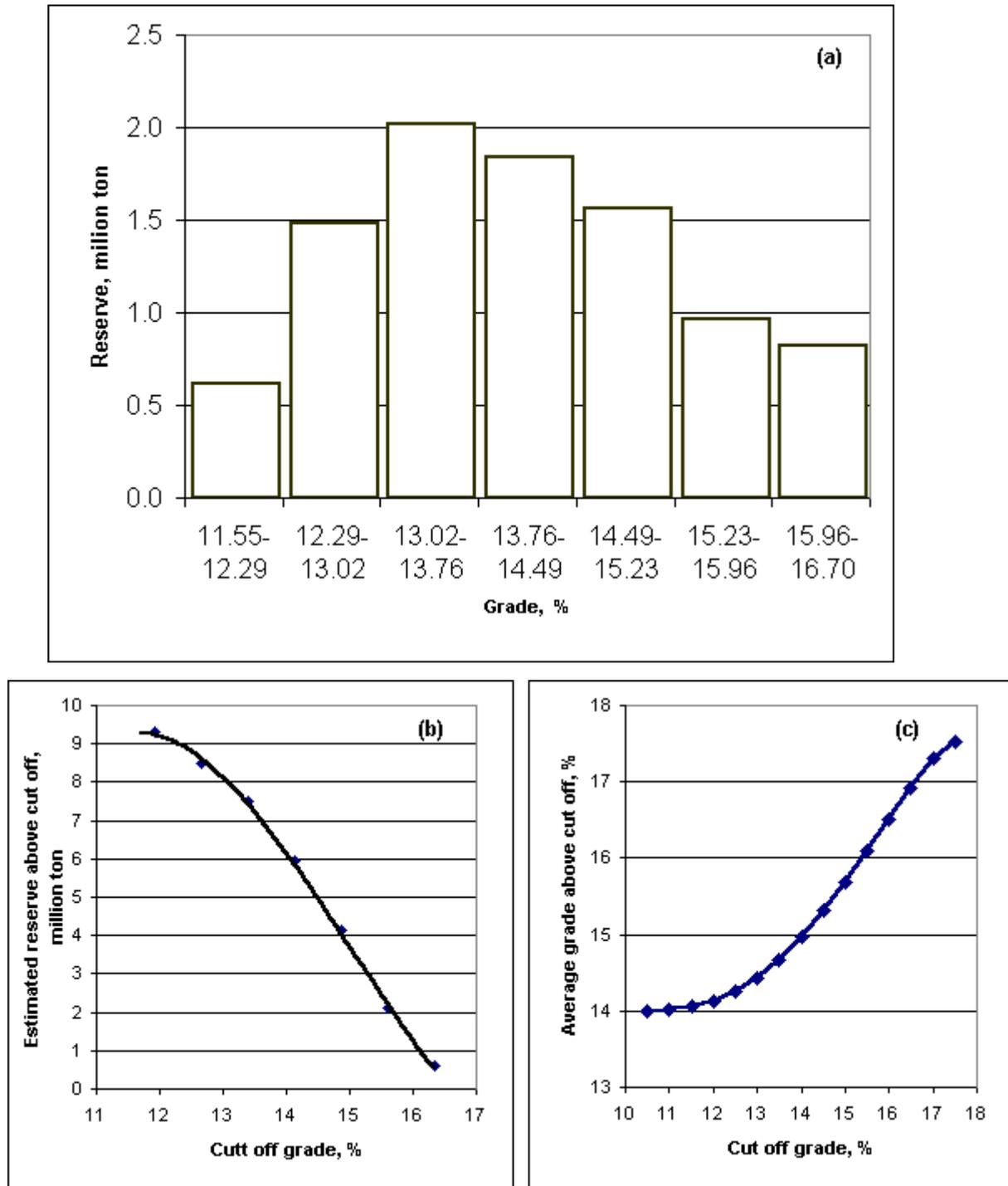


Fig. (8) Grade/tonnage/cut off distribution of C1 component
(estimated blocks = 100x100m)

a- Histogram

b- Cumulative curve

c- Cut off/average grade curve

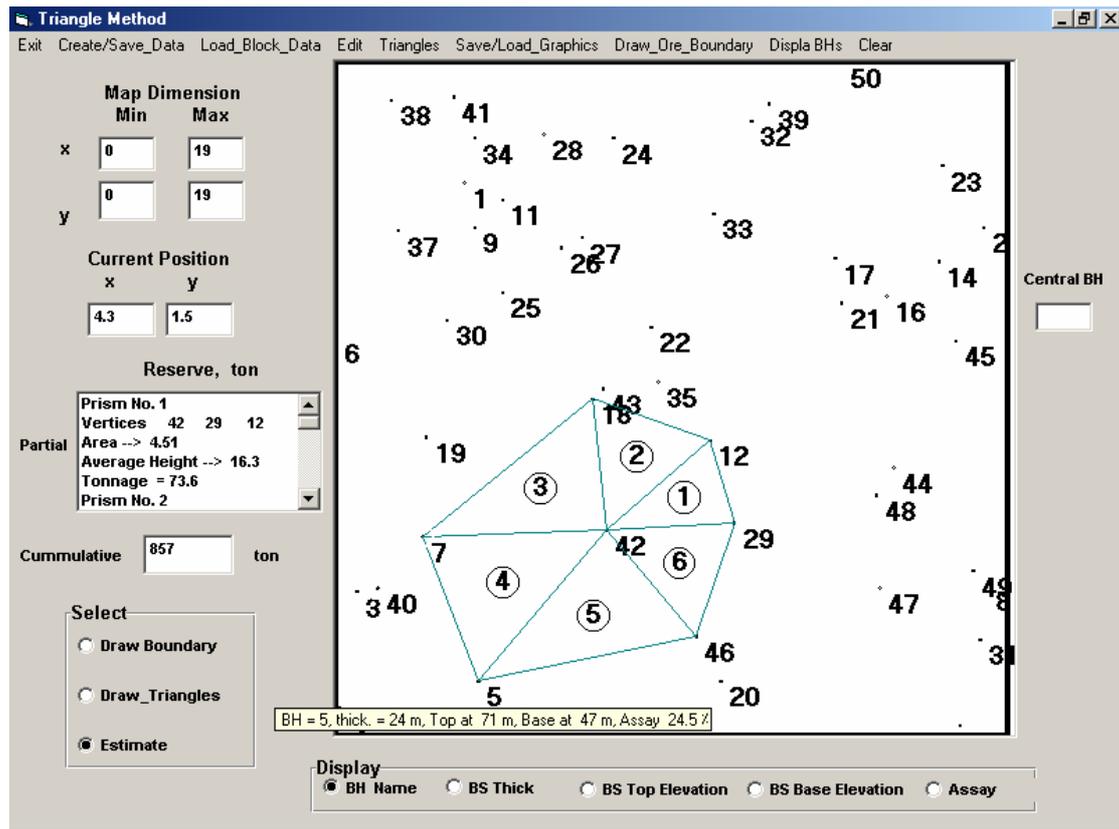


Fig. (9) An interactive window for reserve estimation by the triangle method.

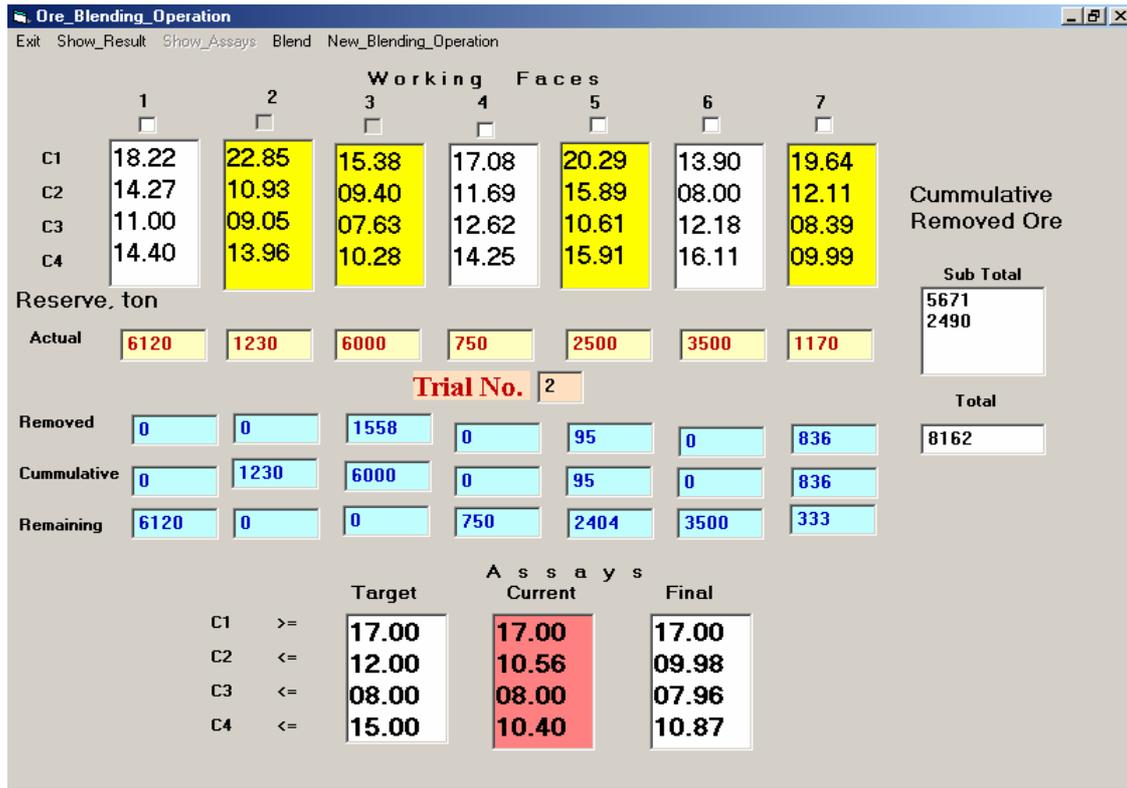


Fig. (10) Blending operation from seven mine working faces

Face	stock, ton	weight	First Trial		Second Trial		
			Ore		Ore		
			removed	remaining	removed	remaining	
1	6120	0	0	6120	0	0	6120
2	1230	0.22	1230	0	0	0	0
3	6000	0.78	4442	1558	0.63	1558	0
4	750	0	0	750	0	0	750
5	2500	0	0	2500	0.04	96	2404
6	3500	0	0	3500	0	0	3500
7	1170	0	0	1170	0.34	837	333

Average Assay%			
	Trial 1	Trial 2	
C1	17.00	C1	17.00
C2	9.73	C2	10.56
C3	7.94	C3	8.00
C4	11.08	C4	10.40

Fig. (11) Summary of two-trial blending operation