Chapter - R

GEOSTATISTICS

JEOİSTATİSTİK

Enjeksiyon ile Kaya Kütlesi Özelliklerinin Kıyaslandığı Bir Su Aktarma Tünelinin Jeoistatistiksel Değerlendirmesi

Geostatistical Evaluation of a Water Conduit Tunnel Due to Grout & Rock Mass Properties

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ÖZET Taşınan suyun sağlıklı ve güvenli bir şekilde iletilmesi için zayıf kaya ortamlarında açılan su tünelleri gerektiği gibi sağlamlaştırılmalıdır. Bir nehir ya da göl kaynağından çok yüksek miktarda kullanılabilir su ileten nakil tünelleri büyük şehirlerin su gereksinimini karşılarlar. Bu yüzdendir ki bu tünellerin zeminden etkili bir şekilde yalıtılmış olması ve duraylı olarak uzunca bir süre hizmet vermesi gerekir. Bu hedefe katkıda bulunmak için daha büyük çapta açılan tünellere özel borular yerleştirilir ve borularla zemin arasında kalan boşluklar çimento şerbeti ile doldurulur. Bu çimento şerbeti donarak priz aldıktan sonra ikincil enjeksiyon işine geçilir. Bu da katılaşmış malzemenin hemen arkasındaki destekleme sistemlerini geçip zayıf kaya ortamına ulaşan deliklerin içinden yapılır. Bu çalışmada DSİ tarafından İstanbul'da açtırılan bir su taşıma tünelinde yapılan enjeksiyon miktarları ile ortamın jeolojisi incelenmiştir. Enjeksiyon koordinatlarına ve ayna profillerinden çıkarılan Q kaya sınıflaması değerlerine bağlı olarak enjeksiyon miktarları ve jeoloji, jeoistatistiksel yaklaştırma yoluyla çözümlenmiş ve karşılaştırılmıştır. Veriler daha düşük Q değerlerinde daha fazla şerbet enjekte edildiği sonucunu desteklemektedir.

ABSTRACT Tunnels excavated in weak rocks need to be properly reinforced for transmitting fluids on safe and healthy. Water conduit tunnels that transfer huge amount of usable water from a source lake or river meet water requirements of big cities. That is why they should be remained stable and isolated from ground effectively. In order to contribute this goal, special steel pipes are inserted into tunnels and spaces between ground and pipes are filled out by cement grouts. After allowing to cure these cement injections, secondary grout application is performed by using holes previously opened on the pipe circumferences by drilling holes enriching weak rock formation behind preliminary grout and support elements . In current study, a conduit tunnel excavated in İstanbul by DSİ, are investigated with respect to geology and amount of the grout application. Geostatistical approximation of Q rock classification of tunnel and grout measures due to face and grout coordinates are performed and compared to each other. Results point out that the less the Q values, the more the amount of the grouts.

1 INTRODUCTION

Nowadays, big cities' population has been increasing day by day. Due to this fact that, people necessities have multiplied by a few times more than that population growth. One of the important requirements of people is to have enough usable and drinking water, especially in arid seasons. In order to eliminate this kind of risks, city managements employ consultant companies for constructing dam reservoirs in which huge amount of water could be collected and saved. These storages are fed by various watercourses such as rivers, streams and creeks. On the other hand, in case a barren weather condition comes up, other options should be put into practice such as water transmission with special underground tunnels from long distance water resources like Melen creek that is one of them to transfer water for İstanbul city people.

The second phase of Istanbul potable water project consists of 11 different contracts. Among them, "the Ayazağa tunnel with about 2.5 km" being in the stage of conveyance line between Cumhuriyet advancement station and Kağıthane is investigated for the current study. During the excavation process of tunnel diameter with 5m, conventional drilling and blast method and hydraulic hammers were used for penetrating tunnel in the sedimentary rock formations. Rock classification practices were simultaneously performed by a geologist and the values of Q tunneling quality index evaluated from face measures during the tunnel progress were recorded. After the tunnel is excavated and supported by rock bolts, wiremesh and shotcrete as well as girder steel sets, the steel pipes having a diameter with 4m were inserted into tunnel space. These pipes will transfer drinking water coming from Melen reservoir with a high pressure flow.

Current study also covers the consolidation injection activities on the tunnel kilometers between 18 +256 and 20 +584. According to drill hole coordinates, amount of the injection volumes (AIV) were recorded and compared with Q classification values. Indeed, the coordinates of Q values and AIV rarely fit into one another. That is why the number of AIV values was multiplied by using 2D geostatistical approximation method. In order to reflect those activities how to be performed, information in detail first needs to be given about grout works carried out in the tunnel.

2 GROUTING WORKS

The grout is an intensive and viscose mixture having fine aggregates, cement, water and additives harmonizing the properties of it. Tunnel grouting for filling empty space and reinforcing rock of tunnel perimeter is generally divided into two main processes: contact and consolidation injections. The first one is used for filling spaces between pipes and preliminary support works previously completed in the tunnel circumference. After that part of process is concluded and grout is cured, the second one is initiated. Aim of consolidation grout is to enhance formation properties and assist overall tunnel construction for working together with formation. Moreover, this dual grout application provide impermeable zone in which any flow in or outside of tunnel is not permitted.

2.1 General Grouting Process

Grout process is mainly separated into three processes: drilling holes with an essential distance, preparing the grout mix with respect to geological properties of formation and, pumping the grout into drilled holes by special techniques (Balkıs, 2009). Important points that should be strictly followed are the grout type, the grout pressure, the application distance between holes with reference to tunnel axis, and the amount of the injection.

Grout mix flow is inversely affected by the grout pressure. The more filled the hole, the higher the grout pressure. After drilling hole is full, the grout pressure is in the maximum level.

3 AYAZAĞA TUNNEL

3.1 Information

Ayazağa tunnel is a water conduit tunnel with a length of 2557m and a steel pipe diameter of 4m. Tunnel move upward by an inclination of 0.08% since it has 2m elevation between starting and ending points of tunnel. Overburden thickness of tunnel is

90m in average. Tunnel is projected by a final flow capacity with 32 m^3/sec (DSİ, 2010).

Tunnel geologic units covered Devonian Kartal, Carboniferaus Thracian, Neogen Belgrad, Kuaterner Aluvional Formations as well as diabase and andesite dykes (Emiroğlu, 2010).

New Austrian Tunnelling Method (NATM) was used for tunnel excavation and supports. Conventional drilling and blast method is selected for loosening and excavating rock. If the blasting method is not allowed, a hydraulic breaker is practiced for extracting weak formation.

The Q classification is performed for the determination of rock classes, face penetration length in every cycle, maximum stand up times and, support type selection. Q values were found out by recording data collected from face profiles by geologists. A cross sectional view of face is drawn in a paper sheet. Then, all parameters related to calculate Q were filled out and evaluated on the paper as a report. Up to them, 5 different support types were fixed into tunnel perimeter and beyond (Kap, 2013).

3.2 Grout Application in Ayazağa Tunnel

After excavation and support works, with a diameter of 4m, steel pipes (Figure 1) that will transfer water, were inserted into tunnel inside.

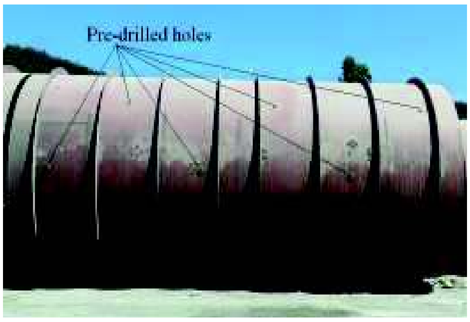

Figure 1. Steel pipes with pre-drilled holes

Due to construction diameter of tunnel with 5m, a contact grout application is performed for filling empty spaces out between primary support and pipe as seen in the Figure 2.

Figure 2. Contact grout between pipe and support

After a curing time is allowed passing for the first sequence, consolidation grout application is secondly initiated. For this sequence, first of all, a hole which reaches the contact grout zone, then primary support zone and at last weak rock formation, was drilled by a rotary type drilling machine that may be telemetrically rotated as seen in the Figure 3. After drilling phase, packers (Figure 4) are fixed for tightly sealing the drill hole output.

Figure 3. SMS rotary type drilling machine

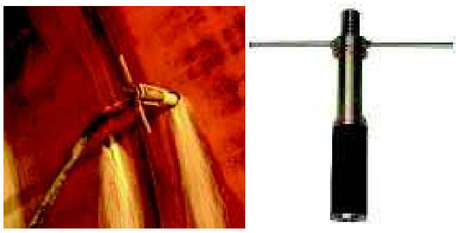

Figure 4. Packers

Then, grout pumps are connected to packers in order to send the grout from pipe inside to outwards trough the pipe perimeter and beyond by drill holes eccentrically enumerated as seen in the Figure 5 (Bahçıvan, 2011).

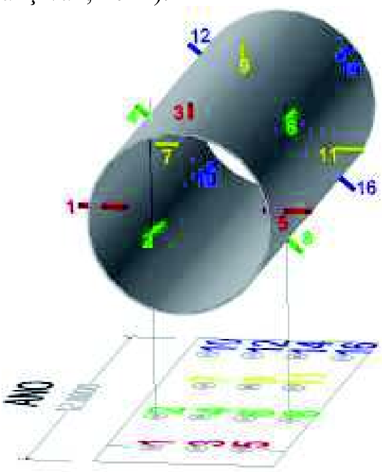

Figure 5. Pre-drilled holes in pipe (12m)

The pipe consists of 14 pre-drilled holes whose coordinates is very well known. More or less than 12m, the grout is consecutively injected from the same numbered holes for latter pipes. Grout mix for lower drill holes contains a cement-water ratio of 1 to 1. On the other hand, those of upper drill holes are 1 to 3. Grout pump is first started from the lowest level hole (2, 8, 10, 16) and then it is pumped from the second higher level (1, 5, 7, 11). At last, drilled holes numbered by 3 and 9 are grouted. For a pipe, all drill holes are connected to a packer in which its vane is in the open position. During pump activity from a drill hole, if it is thrown out from other hole, then packer vane is closed and that hole is noted for further investigations. Before pumping, drill hole should be controlled if it is stuck or not. If so, it is re-drilled. After pumping is finished, packers should be removed 24 hours later just after controlling by its vane whether grout is cured or not.

Important point that should be paid attention is to figure out the case whether grout hole is fully filled or not. If a hole continues accepting the grout mix less then 0.7lt per minute through 20 minutes in total with a constant pumping pressure (about 4 kg/cm^2 for Ayazağa tunnels), then it can be said that the present hole is filled. In such a case, one may agree in that the grout pump should be stopped. If the drill hole accepts grout rates higher than 0.7lt per minute and pumping pressure is still in the low levels, then pumping activity cannot be interrupted until they reach tolerable limits. This type of practitioner behavior ensures that the pipe perimeter and beyond is fully filled.

3.3 Amount of the Grouts

Ayazağa tunnel covers 1710 grout injections from the numbered holes seen in Figure 5. 589 of them run into Q classification values (between 0.25 and 0.027) evaluated from face profiles closed to the numbered drill hole coordinates during the excavation process. A list of the number of grout injections is tabulated in the Table 1 according to total amount of injection volume and the name of relevant hole.

Table 1. The list of grout injections

L	Po	DHN	ToG	TAIV
Pe	F	3	49	95548
	BW	9	38	81360
LT	F	4	39	38058
	BW	12	44	36462
RT	F	6	39	31506
	BW	14	43	31678
LM	F	1	48	18988
	BW	7	38	22768
RM	F	5	48	26232
	BW	11	38	23310
LL	F	2	38	8850
	BW	10	45	11790
RL	F	8	36	6720
	BW	16	46	13420
T	F-BW	1-16	589	446690

L: level, Pe: Peak, LT: Left top, RT: Right top, LM: Left middle, RM: Right middle, LL: Left low, RL: Right low, Po: Position, DHN: Drill hole name from Figure 5, F:Forward BW: Backward, ToG: Times of Grout, T: Total, H: Hole, TAIV: Total amount of Injection Vol. (lt)

As seen from Table 1, Figure 5 and 6 below, The highest level holes (3 and 9) are grouted by the highest amount of the grout volume in total (95548 lt and 81548lt) and they are decreased while their level is getting lower.

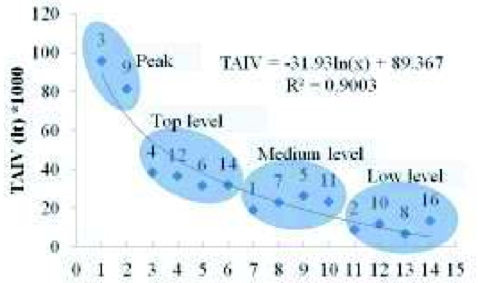

Figure 6. Variation of TAIV

4 GEOSTATISTICAL EVALUATION

4.1 Geostatistical Process

Geostatistical process was first proposed by Matheron in 1971 based on regionalized variable theory. It uses global x and y coordinates pointing out the areal variables. In these coordinates, whatever it is, a z parameter map is plotted by using known values coordinates.

Geostatistical calculation actually focuses on evaluating values of points where its values are unknown from those where its values are known in a coordinated area. It is generally used for reserve estimation of underground mines from well logs. Besides, it was used in tunneling activities such as determination of settlement map of a sewerage tunnel in Istanbul (Tuncdemir et al. 2011). Numerous studies have emerged in various sciences so far.

Based on an ongoing graduate thesis (Kap, 2013) performed on Ayazağa conduit tunnel, 2D geostatistical analysis is performed for every drilling holes with respect to their x and y coordinates and their AIV Values by using Tercan (2005) concept because Q values and hole coordinates are randomly intersected. By using 2D geostatistical estimation technique the number of AIV values can be multiplied and these new estimated values coordinates definitely overlap on Q values coordinates. Hence, one can compare and interpret same coordinated values.

Since it manually takes a long evaluation process, AIV values of the drill hole named as "3" (3#) is only evaluated for the current study to present calculation steps. One can use same process for other holes and / or another project values as well.

The tunnel kilometers between 20+581 – 20+282.6 consist of 17 AIV values for 3# as seen in the Table 2. If a cross sectional side view of tunnel is plotted, a graph in Figure 7 can be plotted.

Table 2. AIV and coordinates of 3#

Point	X (m)	Y (m)	AIV (lt)
1	20581	2.5	600
2	20553.1	2.5	1000
3	20539.6	2.5	900
4	20525.7	2.5	1700
5	20512.4	2.5	1450
6	20498.6	2.5	1200
7	20471.2	2.5	1700
8	20458.1	2.5	1450
9	20445.4	2.5	1700
10	20431.4	2.5	1450
11	20417.8	2.5	900
12	20404.5	2.5	800
13	20390.5	2.5	900
14	20363.2	2.5	300
15	20323.2	2.5	600
16	20296.7	2.5	2700
17	20282.6	2.5	2700
T	20400	2.5	Unknown

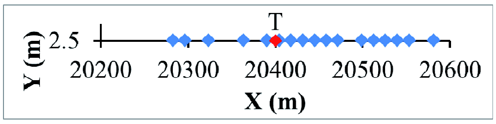

Figure 7. 2D coordinates of drill hole 3#

In Figure 7, Letter "T" is the point (X=20400, Y=2.5) whose AIV value is unknown. AIV value of this point can be estimated by using other known AIV values of 17 points according to their coordinate values. Indeed, for this data set, 299 new points (20581-20282) in which "T" is one of them can be generated by an interval with 1m. As it can be seen from example below, it is exhaustive to estimate it manually. That is why geostatistical calculation software should be used (eg. GS+, Ecosse, GsWin…).

For figuring the concept out, AIV value of "T" point is manually estimated below. In order to do that, first of all, semi variance values according to separation distance should be estimated by using Equation (1).

$$\gamma(\boldsymbol{h}) = \frac{1}{2\boldsymbol{N}} \sum_{\boldsymbol{i}=1}^{\boldsymbol{N}} \left[\boldsymbol{Z}(\boldsymbol{x}+\boldsymbol{h}) - \boldsymbol{Z}(\boldsymbol{x})\right]^2 \qquad (1)$$

γ(h): Semivariance value in distance of "h".

N: The number of couples

Z(x+h): z value of point where it is in the distance of "h" from x coordinate.

Z(x): z value in the x coordinate

The Active Lag Distance (ALD) specifies the range over which semivariance will be calculated. The active lag distance is divided into a number of different lag intervals classes (LCI) for analysis (Robertson, 2008). Since spherical model correlation coefficient is so high, ALD equals "160" for current example. LCI changes between "0 and 40" (Table 3), "40 and 80" (Table 4), "120 and 160" (Table 5) and "80 and 120" (Table 6).

As seen from Table 3, there are 24 couples having an interval between the first and second point coordinates less than 40 and higher than 0. For example, in the Table 2, Interval (I) among x coordinates of the first (1#) and second point (2#) is 27.9m (20581-20553,1); (It is between 0 and 40). At the same time, SVV of this couple is 80000 ($(1000\text{-}600)^2/2$). All values given in the Table 2 should be scanned and calculated in the same manner and all couples between 0 and 40 interval needs to be added into Table 3 as one of the couple of lag class 1. Average values of "SVV" and "I" will be a point in the graph of semivariance versus distance (Figure 8). Same procedures should be followed for Lag class 2, 3 and 4, eventually Table 4, 5 and 6 may be derived.

When semivariogram average values (AV) with respect to average separation distance (Interval in Table 3, 4, 5 and 6) are plotted, it is understood that a spherical model can be fitted to the graph by a very high correlation coefficient (r^2=0.996). It means that from the 17 known AIV values, other unknown AIV values (299 points) with an increment 1 by 1m can be strongly estimated.

Table 3. Lag class 1 (0-40 m)

C	SVV	I (m)	PC1	PC2
1	80000	27.9	1	2
2	5000	13.5	2	3
3	245000	27.4	2	4
4	320000	13.9	3	4
5	151250	27.2	3	5
6	31250	13.3	4	5
7	125000	27.1	4	6
8	31250	13.8	5	6
9	125000	27.4	6	7
10	31250	13.1	7	8
11	0	25.8	7	9
12	31250	39.8	7	10
13	31250	12.7	8	9
14	0	26.7	8	10
15	31250	14	9	10
16	320000	27.6	9	11
17	151250	13.6	10	11
18	211250	26.9	10	12
19	5000	13.3	11	12
20	0	27.3	11	13
21	5000	14	12	13
22	180000	27.3	13	14
23	2205000	26.5	15	16
24	0	14.1	16	17
AV	**179843.75**	**21.425**	**24 couples**	

C: The couple number, SVV: Semivariance value, I:interval, PC1-2: 1st and 2nd point couple, AV: Average

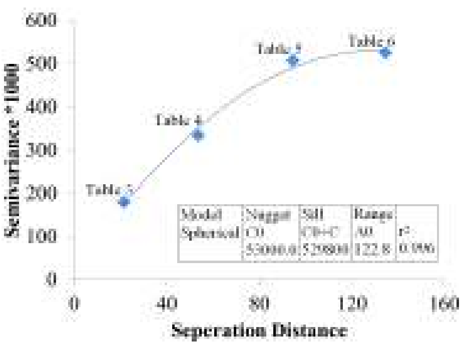

Figure 8. Spherical model of drill hole 3#

Table 4. Lag class 2 (40-80 m)

C	SVV	I (m)	PC1	PC2
1	45000	41.4	1	3
2	605000	55.3	1	4
3	361250	68.6	1	5
4	101250	40.7	2	5
5	20000	54.5	2	6
6	45000	41	3	6
7	320000	68.4	3	7
8	0	54.5	4	7
9	31250	67.6	4	8
10	31250	41.2	5	7
11	0	54.3	5	8
12	31250	67	5	9
13	31250	40.5	6	8
14	125000	53.2	6	9
15	31250	67.2	6	10
16	320000	53.4	7	11
17	405000	66.7	7	12
18	151250	40.3	8	11
19	211250	53.6	8	12
20	151250	67.6	8	13
21	405000	40.9	9	12
22	320000	54.9	9	13
23	151250	40.9	10	13
24	661250	68.2	10	14
25	180000	54.6	11	14
26	125000	41.3	12	14
27	45000	67.3	13	15
28	45000	40	14	15
29	2880000	66.5	14	16
30	2205000	40.6	15	17
AV	**334500**	**53.74**	**30 couples**	

Table 5. Lag class 4 (120-160 m)

C	SVV	I (m)	PC1	PC2
1	361250	122.9	1	8
2	605000	135.6	1	9
3	361250	149.6	1	10
4	101250	121.7	2	10
5	5000	135.3	2	11
6	20000	148.6	2	12
7	0	121.8	3	11
8	5000	135.1	3	12
9	0	149.1	3	13
10	405000	121.2	4	12
11	320000	135.2	4	13
12	151250	121.9	5	13
13	661250	149.2	5	14
14	405000	135.4	6	14
15	605000	148	7	15
16	361250	134.9	8	15
17	605000	122.2	9	15
18	500000	148.7	9	16
19	781250	134.7	10	16
20	781250	148.8	10	17
21	1620000	121.1	11	16
22	1620000	135.2	11	17
23	1805000	121.9	12	17
AV	**525217.39**	**134.7**	**23 couples**	

Table 6. Lag class 3 (80-120 m)

C	SVV	I (m)	PC1	PC2
1	180000	82.4	1	6
2	605000	109.8	1	7
3	245000	81.9	2	7
4	101250	95	2	8
5	245000	107.7	2	9
6	151250	81.5	3	8
7	320000	94.2	3	9
8	151250	108.2	3	10
9	0	80.3	4	9
10	31250	94.3	4	10
11	320000	107.9	4	11
12	0	81	5	10
13	151250	94.6	5	11
14	211250	107.9	5	12
15	45000	80.8	6	11
16	80000	94.1	6	12
17	45000	108.1	6	13
18	320000	80.7	7	13
19	980000	108	7	14
20	661250	94.9	8	14
21	980000	82.2	9	14
22	361250	108.2	10	15
23	45000	94.6	11	15
24	20000	81.3	12	15
25	1805000	107.8	12	16
26	1620000	93.8	13	16
27	1620000	107.9	13	17
28	2880000	80.6	14	17
AV	**506250**	**94.632**	**28 couples**	

From this point and on, a kriging application may be performed since semivariogram model is plotted. In the kriging application for current data set, unknown AIV of T point in the Figure 7 can be estimated from the other 17 values of known AIVs. Estimation may be performed by weighted average of known values since "λ_i" sum up "1". Equation 2 and 3 is the mathematical definition of previous sentence.

$$z(x_0) = \sum_{i=1}^{n} \left[\lambda_i * z(x_i) \right] \qquad (2)$$

$$\sum_{i=1}^{n} \lambda_i = 1 \qquad (3)$$

where

x_0= estimated value in the X_0 point

$z(x_i)$=data used for estimation of X_0

λ_i=weights

For current data set, "x_i" values are the known AIV values of every point seen in the Table 2. In order to find out unknown AIV value of "T" point, individual λ_i weights should be found out. In geostatistical estimation, these weights are calculated from average estimation error which equals zero with a minimum variance value. Therefore, when equation 2 is reorganized, a new one is typed like equation 3

$$z(x_0) = \lambda_1 * z(x_1) + \cdots \lambda_{17} * z(x_{17})$$
$$z(x_0) = \lambda_1 * 600 + \lambda_2 * 1000 \cdots \lambda_{17} * 2700 \quad (3)$$

A variogram function value calculated from the distance between the first point and unknown T equals cumulative product of individual weight and relevant variogram function value as seen in a matrix in Table 7

Table 7. Equation Matrix

$$\begin{array}{l}
\lambda_1 * \gamma(x_1, x_1) + \cdots + \lambda_{17} * \gamma(x_1, x_{17}) + \varepsilon = \gamma(x_1, x_0) \\
\lambda_1 * \gamma(x_2, x_1) + \cdots + \lambda_{17} * \gamma(x_2, x_{17}) + \varepsilon = \gamma(x_2, x_0) \\
\vdots \quad \vdots \quad \vdots \quad \vdots \quad \vdots \quad \vdots \quad \vdots \quad \vdots \quad \vdots \\
\lambda_1 * \gamma(x_{17}, x_1) + \cdots + \lambda_{17} * \gamma(x_{17}, x_{17}) + \varepsilon = \gamma(x_{17}, x_0) \\
\lambda_1 + \lambda_2 + \cdots\cdots \lambda_{16} + \lambda_{17} = 1
\end{array}$$

Table 7 means that 17 different equations can be constructed. In addition, 18th equation is the understanding that total value of λ_i weights is "1". 18 unknown weight values and 18 equations are on the hand which points out that these unknown weight values can be solved. In order to do that, a distance matrix should be built as a sum up list in the Table 8. For example, the distance between x1 and x0 (T) points can be found by subtraction of x1=20581 and x0=20400 coordinates; 181.

Table 8. Distance Matrix

	x1	x2	x3	….	….	x17	x0
x1	0	27.9	41.4	….	….	298.4	181
x2	27.9	0	13.5	….	….	270.5	153
x3	41.4	13.5	0	….	….	257	140
….	….	….	….	….	….	….	….
….	….	….	….	….	….	….	….
x17	298	270.5	257	….	….	0	117

Spherical variogram values (γ(h)) from relevant distance in Table 8 may be found from equation 4 (spherical variogram model) by using nugget effect (c0), sill (c0+c), range of influence (A0) come from the Figure 8.

$$\gamma(h) = 0 \Leftrightarrow h = 0$$
$$\gamma(h) = C_0 + C\left[\frac{3}{2}\frac{h}{A_0} - \frac{1}{2}\frac{h^3}{A_0^3}\right] \Leftrightarrow h \le A_0 \quad (4)$$
$$\gamma(h) = C_0 + C \Leftrightarrow h > A_0$$

Spherical variogram values of every couple are summed up and listed in the Table 9.

Table 9. Values' Matrix

γ(xi,xi)	x1	x2	x3	…	x17	x0
x1	0	212697	284983	…	529800	529800
x2	212697	0	131309	…	529800	529800
x3	284983	131309	0	…	529800	529800
….	….	….	….	0	….	….
x17	529800	529800	529800	…	0	528437

For example distance between x1 and x2 is "h=27.9m" in the Table 8. When C0, C, h and A0 values substitute on equation 4 since h less than A0, spherical variogram value can be found as "212697".

$$\gamma(h) = 53000 + 476800\left[\frac{3}{2}\frac{27.9}{122.8} - \frac{1}{2}\frac{27.9^3}{122.8^3}\right]$$
$$\gamma(h) = 212697$$

If Table 7 equation matrix is rearranged by using values' matrix (Table 9), a new equation matrix can be drawn by Table 10. By using numerical analysis technique, weight matrix (B in Equation 5 and 7) which denotes (λ_1, λ_2, λ_3, … λ_{17}), can be evaluated by coefficient matrix (A in equation 5 and 6)) that covers $\gamma(x_1,x_1)$, $\gamma(x_1,x_2)$, $\gamma(x_1,x_3)$,…. $\gamma(x_{17},x_{17})$ and result matrix (C in Equation 5 and 7) that refers $\gamma(x_1,x_0)$, $\gamma(x_2,x_0)$, $\gamma(x_3,x_0)$,…. $\gamma(x_{17},x_0)$.

Table 10. New equation matrix

$$\begin{array}{l}
\lambda_1 * 0 + \cdots + \lambda_{17} * 529800 + \varepsilon = 529800 \\
\lambda_1 * 212697 + \cdots + \lambda_{17} * 529800 + \varepsilon = 529800 \\
\vdots \quad \vdots \quad \vdots \quad \vdots \quad \vdots \quad \vdots \quad \vdots \quad \vdots \quad \vdots \\
\lambda_1 * 529800 + \cdots + \lambda_{17} * 0 + \varepsilon = 528437 \\
\lambda_1 + \lambda_2 + \cdots\cdots + \lambda_{16} + \lambda_{17} = 1
\end{array}$$

In order to find out weight matrix unknown values; Inverse of A matrix should be evaluated and then a product of A and C matrix should be taken. Results will give the B matrix (equation (8) and Table 11)).

$$\boldsymbol{AB} = \boldsymbol{C} \tag{5}$$

$$\boldsymbol{A} = \begin{vmatrix} \gamma(x_1,x_1) & \gamma(x_1,x_2) & \dots & \dots & \gamma(x_1,x_{17}) & 1 \\ \gamma(x_2,x_1) & \gamma(x_2,x_2) & \dots & \dots & \gamma(x_2,x_{17}) & 1 \\ \vdots & \vdots & \vdots & \vdots & \vdots & \vdots \\ \gamma(x_{17},x_1) & \gamma(x_{17},x_2) & \dots & \dots & \gamma(x_{17},x_{17}) & 1 \\ 1 & 1 & 1 & 1 & 1 & 0 \end{vmatrix} \tag{6}$$

$$\boldsymbol{B} = \begin{vmatrix} \lambda_1 \\ \lambda_2 \\ \vdots \\ \lambda_{17} \\ \varepsilon \end{vmatrix} \quad \boldsymbol{C} = \begin{vmatrix} \gamma(x_1,x_0) \\ \gamma(x_2,x_0) \\ \vdots \\ \gamma(x_{17},x_0) \\ 1 \end{vmatrix} \tag{7}$$

$$\boldsymbol{B} = \boldsymbol{C} * \boldsymbol{A}' \tag{8}$$

Table 11. Weight matrix results

B	λ_1=	0.002529	λ_7=	-0,00067	**λ_{13}=**	**0,32981**
	λ_2=	0,002911	λ_8=	0,001062	λ_{14}=	0,043761
	λ_3=	0,007424	λ_9=	0,004796	λ_{15}=	0,00278
	λ_4=	0,006355	λ_{10}=	0,02256	λ_{16}=	-0,00548
	λ_5=	-0,01132	λ_{11}=	0,105759	λ_{17}=	-0,00071
	λ_6=	-0,00639	**λ_{12}=**	**0,494821**	ε=	3411,219

As seen in Table 11, the highest impact on the T point approximation is the weight coefficients of λ_{12}, λ_{13} being the closest points as seen in the Table 2. This phenomenon shows that 2D geostatistical approximation is viable for the calculation steps of the AIV values of 3#.

Equation 3 can be solved by equation 9 at the moment since all unknown coefficients are predicted.

$$z(x_0) = \lambda_1 * z(x_1) + \lambda_2 * z(x_2) + \cdots \lambda_{17} * z(x_{17})$$
$$z(x_0) = 0{,}002529 * 600 + 0{,}002911 * 1000 \cdots \quad (9)$$
$$\cdots \text{-} 0{,}00071 * 2700 = 825.0497$$

It should be here noted that AIV estimation for the T point is the 825.0497 lt which is pretty close values to 12th and 13th AIV values in Table 2. This phenomenon suits the geostatistical principles that mean the closer the predicted value to a known value, the closer the results of predicted value to that known value.

Estimation error (Ee) is another definition of the acceptability of results. It can be calculated by equation 10. The lower the Ee, the more acceptable the results become.

$$\sigma_k^2(x_0) = \lambda_1 * \gamma(x_1,x_0) + \lambda_2 * \gamma(x_2,x_0) \cdots \\ \cdots\cdots\cdots\cdots \lambda_{17} * \gamma(x_{17},x_0) + \varepsilon \tag{10}$$

$$\sigma_k^2(x_0) = 0.002529 * 529800 + 0.002911 * 529800 \cdots \\ \cdots\cdots\cdots - 0.00071 * 528437 + 3411.219 = 112069.8$$

When same procedure is pursued for the 298 points remained, one have 299 approximate values to show AIV values in the tunnel routes in an increment 1 by 1. At the moment, if one has Q values running into tunnel routes and overlap on these increments of AIV values, then a relationship graph may be plotted as seen in the Figure 9.

29 Q values were recorded between 20+581 and 20+282.6km. 299 AIVpre (predicted values from kriging) values were created by 2D geostatistical approximation. 17 AIVreal values were recorded by site engineers.

As seen from the Figure 10, the data shown by triangles are real AIV values. The data drawn by diamonds are Q values and finally, the thick line point out that squares with an increment one by one are the predicted AIV values. Trend lines given with the corresponding data are the same colored lines which refer to 5th order polynomial functions.

The view of this type of trend line proves that real values get closer the average values by krigging approximation.

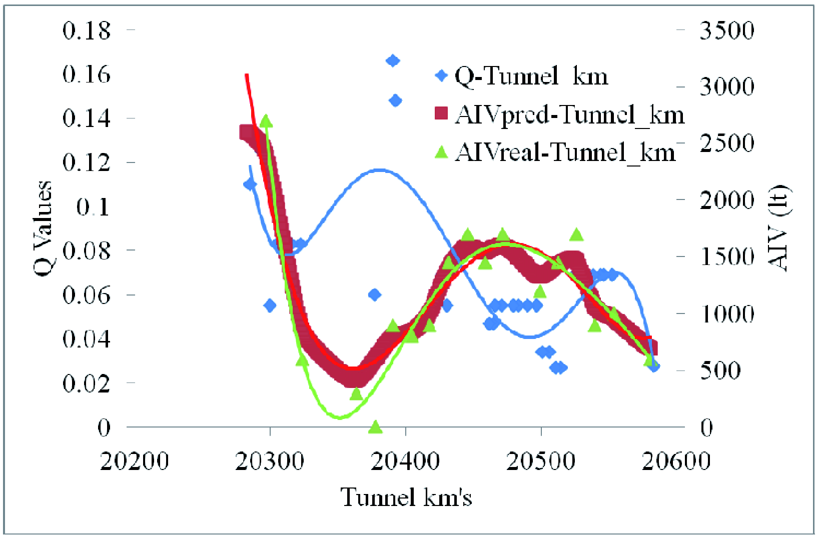

Figure 10. Relationships between tunnel kms, Q values and AIV

Another important judgment is that AIV values are low when Q values are high which can be understood from blue colored anticline and red or green colored syncline between 20300 and 20450 and also 20540 and 20580 approximately. Inversely, AIV values are high when Q values are low which can be understood from blue colored syncline and red or green colored anticline between 20450 and 20540 approximately.

5 CONCLUSIONS

Grout injection activities and Q rock classification with respect to tunnel kilometers were performed in the Ayazağa water conduit tunnel.

Amount of the grout injections and their coordinates due to tunnel axis were recorded. Besides, Q values are determined from the face readings but their coordinates are fairly overlap with the grout coordinates. In order to have an intersection on them, data multiplication is performed by kriging approximation manually for the drill hole of 3# for only the data collected from the tunnel km's between 20+581 and 20+282.6km. After the number of data for AIV real values are increased by data reproduction, it can be compared with the Q values. Results derived from Figure 10 by an eye examination point out that AIV values are inversely proportional with Q values in general even though a small portion on the left of graph does not cover the concept.

A practitioner may predict the total amount of the grout volume for the drill hole 3# if the tunnel in which he worked on has similar Q classification values which vary between 0.027 and 0.167, namely very weak and weak rock formation. This will clarify the costs of grout for the preliminary project planning.

From the figures and tables given in the section of geostatistical process, it can be said that 2D geostatistical approximation seems viable for predicting and multiplication of data of the drill hole of 3# since the correlation coefficient of spherical model is exceptionally high (r^2=0.996). This can be controlled by estimation error as well. Nonetheless, same procedure needs to be carried out for other drill holes to see the derivation of similar results.

ACKNOWLEDGEMENTS

Authors deeply express sincere appreciation to DSİ İstanbul Region Management, Ayazağa tunnel engineers, especially Mining Engineer Ural Tolga Seyhan and Dr. Serkan Dağlıoğlu who kindly assist and share data.

SUPPLEMENTARY DATA

Spreadsheet files of geostatistical analysis and other kriging, Q classification values and amount of grout injection volumes as well as tables viewed in a short form text ahead may be found from current internet address:
http://www.web.itu.edu.tr/~tdemir/imc13.rar

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Feasibility Study of a Mining Project under Uncertainty: Can the Real Options Approach be a Viable Solution?

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ABSTRACT Nowadays, making investment decision in the mine sectors is faced with huge challenge of uncertainties particularly when this investing is highly capital intensive and long term. These uncertainties are originated from environmental, market and political changes. In such condition mining companies cannot have a stable vision of their enterprises. Therefore, they should consider the investments which encompass flexibility by adapting market conditions. Traditional discounted cash flows (DCF) or net present value (NPV) methods constitute most investment decision making structures. In recent years, the Real Options Approach (ROA) is presented based on the advanced options theory technique of DCF and NPV. In this paper the both techniques are compared and evaluate the role of real options valuation in a mining project (Kahnoj titan plant as a case study) feasibility studies. The management team's flexibility to change the project's course is embedded in the form of options to abandon, expand and contract of the project. Monte Carlo simulation-based method using MATLAB software command-line functions is used for valuation of the options values. Based on the results, the values of the project's abandonment, contraction and expansion options are estimated 18,241,730 and 13,407,795 and 219,909,955 dollars respectively. The added values could be substantial for a non-economic project and it is close to break point estimated by DCF analysis (with the net present value of -1,695,397 dollars). This research capable to be implemented into the similar projects that needed managerial decisions making using real options.

Key words: Feasibility Study, uncertainty, Monte Carlo simulation, real options approach

1 INTRODUCTION

Investments in mining projects and related industries are quite different from the other projects. All mining projects are associated with uncertainties in the future profitability. This uncertainty affects on the ore prices, operating cost, and so on. Investment in the mining projects is encountered a delay between decision making steps and investment procedures. A successful investor is the one who has the managerial flexibility and ability to delay, expand, contradict, or abandon the project. There are varieties of factors that assist the investor to have a right investment decision making.

Valuation of the mining project is depending on the different factors such as the world ore prices, operating costs, and so on. DCF techniques are commonly used for economic evaluation of mining projects. However these techniques, due to the lack of consideration of the managerial flexibility, may underestimate some projects. In this paper, the real options theory is applied to meet theses flexibilities.

2 LITERATURE REVIEW

ROA has considered recently among some researchers and economists. They have mentioned ROA as means of better assessing projects under uncertainty.

Under uncertainty conditions, the ROA presents a better result comparison with the traditional DCF analyses such as NPV and the internal rate of return (IRR) methods. ROA is capable to embed the value of managerial flexibility to change or revise the project course based on the new market conditions. Therefore, using the same discounting rate, the value of a project calculated by the ROA is always higher than that valuated by the traditional NPV method.

Moyen et al. and Miller and Park explained uncertainty and profitability of the project as the cause of discrepancy between the ROA and the DCF estimates (Moyen et al., (1996)), (Miller et al., 2002). The first and likely most enforceable model for assessing options is Black and Scholes model (Black, 1973). However, as explained by Berridge and Schumacher (2004), the Black–Scholes model is just applicable for valuing European options not for American options (Berridge, 2002).

The lattice method presented is another common technique for valuing options (Cox et al., 1979). Barraquand and Martineau (1995) criticized the application of lattice method by offering limitations of which. They mentioned that the method becomes inapplicable when projects facing multiple uncertainty. (Barraquand et al., 1995) developed the finite difference method for options valuation, however this method have the same liabilities as the lattice method (Brennan et al., 1978).

The first time proposed by Boyle the Monte Carlo method based on simulation is used for valuing European options (Bpyle 1977). Afterward an improved version of the Monte Carlo method was developed to value American options. Despite the previous techniques, the simulation-based techniques have better performance facing multiple uncertainties. This means that both product volatility over time and cash flows variability can be embedded in the valuation (Longstaff et al., 2001).

The ROA is not a new technique for natural resource investment applications. Since the first paper applying the ROA to value a simple copper mine was published by Brennan and Schwartz, considerable work has been done employing the same concepts of uncertainty and operating flexibility. (Brennan et al., 1985) applied the ROA to value management flexibility to develop petroleum leases (Paddock et al., 1988).

Trigeorgis addressed the consequence of embedding the distinct real options on the value of a natural resource extraction project (Trigeorgis, 1993). Using data from Canadian copper mines illustrated an analogy between the NPV method and the option valuation method. This comparison shows that the NPV method underestimates mining projects (Moyen et al., 1996). McCarthy and Monkhouse offered a trinomial lattice method for valuing a copper mine have both wait and abandon options (McCarthy, 2003).

Abdel Sabour and Poulin (2006) and Roussos G. Dimitrakopoulos, Sabry A. Abdel Sabour (2007) are other significant researchers who conducted assessing mine plans under uncertainty(Sabour et la., 2007). Graham A. Davis and Alexandra M. Newman presented a Modern Strategic Mine Planning using ROA technique (Graham et al., 2008). S Shafiee1, E Topal and M Nehring used an adjusted Real Option Valuation to maximize mining project value using Century Mine case study (Shafiee et al., 2009). Luis Martinez using real options presented a project valuation for open pit mining risks and merged econometric techniques (Martinez, 2011).

3 TRADITIONAL EVALUATION METHOD

DCF and the associated NPV techniques have traditionally provided the major tools for project evaluation. The discounted cash flow formula, equation (1), is derived from the future value formula for calculating the time value of money and compounding returns.

$$NPV = C_0 + \frac{C_1}{(1+r)^1} + \frac{C_2}{(1+r)^2} + \cdots + \frac{C_T}{(1+r)^T} \qquad (1)$$

Where NPV is the net present value, C_0 to C_T are the cash flows expected through the project's life (T) and r is discounting risk-adjusted rate.

NPV analysis evaluates the cash flows forecasted to be delivered by a project by discounting them back to the present using the time span of the project and the firm's weighted average cost of capital. If the result is positive, then the firm should invest in the project. If negative, the firm should not invest in the project. In addition Internal Rate of Return (IRR) cloud be calculated by giving the equation equal zero.

DCF is used by analysts to evaluate projects. However, this tool has certain limitations. For example, if the project's cash flows and decisions are likely faced to many uncertainties (where managers have the flexibility to change the course of the project), part of the value related to these options could be not incorporated in the valuation process.

4 REAL OPTION APPROACH

Nowadays management is looking for a way to reduce uncertainty and to assess the impact of managerial uncertainty on project. ROA has been developed to promote this goal. This approach has provided a basis for the development in financial and decision making analyses. ROA is one of the modern valuation methods that provide a tool to adapt and revise mining projects under uncertainty and future variable movements.

A lot of progresses that have been done in the Real Option literature have changed the way of thinking about an investment opportunity. During project management, managers may do several choices about project characteristics every time new information from market is available. ROA is the way to respond to market changes. This possibility that managers have to adapt their decisions to the change of market has value that must be considered during the decision making process. Therefore, this flexibility creates "options" that increase the value of the project and determines the failure of the traditional technique (such as NPV).

ROA accounts for a range of possible outcomes over the life of a project using stochastic processes and calculates a "composite" options value for a project, considering only those outcomes that are favorable (i.e., options are exercised) and ignoring those that are not (letting the options expire). This assumes that the decision makers will always take the value-maximizing decision at each decision point in the project life cycle. Whereas DCF accounts for the downside of a project by using a risk-adjusted discount rate, ROA captures the value of the project for its upside potential by accounting for proper managerial decisions that would presumably be taken to limit the downside risk. Table 1 summarizes the major differences between DCF and ROA.

4.1 Options

As discussed if there is large uncertainty related to the project cash flows and contingent decisions are involved, where mangers have flexibility to change the course of the project, ROA can be applied for project valuation using different options. In this research several real options embedded virtually in every project are investigated and used to calculate the generated additional value of managerial flexibility. Following options are the focus of this paper:

- Option to expand
- Option to contract
- Option to wait
- Option to abandon

For every mentioned options, this research focuses on one or two aspects (such as practical issues, input parameter variability, etc.) that are most relevant to that option type. For example, the option to expand issue involves how the option size influences the option value. Using the option to contract of the scale, the impact of the volatility factor on the option value is highlighted.

Table 1: the major differences between DCF and ROA

Real Options Analysis	Discounted Cash Flow
Recognizes the value in managerial flexibility to alter the course of a project	All or nothing strategy. Does not capture the value of managerial flexibility during the project life cycle.
Uncertainty is a key factor that drives the option's value.	Uncertainty with future project outcomes not considered.
The long-term strategic value of the project is considered because of the flexibility with decision making.	Undervalues the asset that currently (or in the near term) produces little or no cash flow.
Payoff itself is adjusted for risk and then discounted at a risk-free rate. Risk is expressed in the probability distribution of the payoff.	Expected payoff is discounted at a rate adjusted for risk. Risk is expressed as a discount premium.
Investment cost is discounted at the same rate as the payoff, that is, at a risk-free rate.	Investment cost is typically discounted at the same rate as the payoff, that is, at a risk-adjusted rate.

The analysis of the option to wait discusses how leakage of asset value can be accounted for in the options calculations. And finally, in the case of the option to abandon how to solve the options problem for various strike prices is illustrated. Indeed How to calculate the probability of exercising the option of abandoning the project is presented

4.1.1 Option to expand

Option to expand is usual in any project. In some cases, the initial NPV can be marginal or even negative, but when growth opportunities with high uncertainty exist, the option to expand can provide a considerable value. Without regarding to an expansion option, great opportunities may be overlooked. Investment for expansion is the strike price that will be acquired as a result of exercising the option. The option would be activated if the expected payoff is greater than the strike price.

4.1.2 Option to contract

Once a project has been developed, management may have the option to decelerate the production rate or change the scale of production. In a project, there might be the option to decrease production by contemplating scaling down its operations by either selling or outsourcing one or more plants to gain efficiencies through stabilization.

The option to contract is significant in today's competitive marketplace, where companies need to downsize or outsource swiftly as external conditions change. Organizations can hedge themselves through strategically created options to contract. The option to contract has the same characteristics as a put option, because the option value increases as the value of the underlying asset decreases.

4.1.3 Option to wait

Investing in a mining project has much in common with exercising a financial option. First, both are at least partially irreversible. Second, timing is crucial. Indeed, taking an irreversible action means forfeiting the option to wait for new information concerning market conditions. (Margaret E. Slade, 2000). Option to defer investment, an opportunity to invest at some point in the future, may be more valuable than an opportunity to invest immediately.

A deferral option gives an investor the chance to wait until conditions become more favorable, or to abandon a project if conditions deteriorate. Such options allow a firm to delay an investment until it's sure about other relevant issues.

4.1.4 Option to abandon

Management may decide to abandon the project and sell any accumulated capital equipment in the open market. Alternatively, it may sell the project, or its share in the project, to another company whose strategic plans make the project more attractive. Selling for salvage value would be similar to exercising an American put option. If the value of the project falls below its liquidation value, the company can exercise its put option.

To clarify the mechanism of this valuation process, assume that the only operating flexibility available to the mine manager is to close the mine early. This option is irreversible. At any time the decision to abandon the mine is made by comparing the expected value of all future cash flows and the abandonment cost at that time.

5 MONTE CARLO SIMULATION

To asses real options value partial differential equations, dynamic programming, and simulation methods can be applied. In this paper referred to Monte Carlo simulation method. The simulation method for solving real options problems is similar to the Monte Carlo technique for DCF analysis. Traditional Monte Carlo simulation has been considered a powerful and flexible tool for capital budgeting for a very long time. It involves simulation of thousands of paths the underlying asset value may take during the option life given the boundaries of uncertainty defined by the volatility of the asset value.

6 PRACTICAL APPLICATION: FEASIBILITY STUDY OF A MINING PROJECT UNDER UNCERTAINTY

The final goal of mine managers and decision makers is to make decisions that seem to be the optimum based on the information available at the designing time.

In this study, the applicability and usefulness of the simulation-based ROA method is investigated. Proposed method is applied to choose the best mine plan among different feasible options.

Thought five-year life of options, Iranian Kahnoj Titan is selected to perform the reality examination. Kahnoj Titan faces a myriad of uncertainties and decision-making flexibilities. Production of this industry plant consists of titan pigment, titan magnetite concentrate, and sorrel iron. By using the traditional DCF technique this project was evaluated (Table 2). As it shown in the table, the NPV is negative and the IRR is lower than the risk rate. This situation provides a suitable platform to conduct the real options valuation method on this project.

Table 2: DCF technique analysis of the project

Year	Cash Flows ($)	Ye ar	Cash Flows ($)
-3	-7,975,775	9	6,425,675
-2	-8,222,920	10	6,447,380
-1	-712,316	11	5,289,546
0	1,858,005	12	5,280,964
1	5,136,277	13	5,273,607
2	5,501,333	14	5,267,270
3	6,277,526	15	5,261,785
4	6,425,675	16	5,257,013
5	6,425,675	17	5,252,909
6	6,425,675	18	5,249,475
7	6,425,675	19	5,247,448
8	6,425,675	20	9,760,454
Risk-Adjusted Rate			25%
Net Present Value (NPV)			-1,695,397 $
Internal Rate of Return (IRR)			22%

7 SOLVING THE PROBLEM USING THE REAL OPTIONS VALUATION MONTE CARLO SIMULATION-BASED

First the input parameters required to conduct the simulations are defined:

- Current value of the underlying asset (So)
- Volatility of the asset value (σ)
- Strike price (X)
- Option life (T)
- Risk-free rate corresponding to the option life (r)

- Incremental time step (δt)

7.1 Estimation of current value of the asset

With the real options, the current value of the underlying asset value is estimated from the cash flows that asset is expected to generate over the project life. In the other hand, the present value of the expected free cash flows based on the DCF calculation is regarded the value of the underlying asset. First, the future revenues are calculated based on the number of units expected to be sold, price per unit, operation cost, and then it discounted to the present time by an appropriate risk adjusted discount rate.

7.2 Calculating volatility factor

Volatility refers to the variability of the asset value and as an important input variable; it can have a significant impact on the option value. The volatility factor (σ) used in the options models. It is the volatility of the product prices, which have a significant impact on the cash flow returns.

In this reach the volatility factor for each product is measured as the standard deviation of the natural logarithm of product prices. And then according to the weighted proportion of each product in cash flows returns, the volatility factor is calculated.

7.3 Exercise or strike price

In the real options world, exercising an option typically involves development of a product, construction of a new facility, launching a large marketing campaign, etc., which does not happen in an instant but in fact takes a long time. The strike price or the investment cost directly impacts the option value.

Dealing with expansion or contraction options in this research Approximate costs for exercising the options can be obtained through the rule of six-tenths (equation 2). As shown in table, the cost of a similar item of different size or capacity (double or half capacity) for mentioned options is developed.

$$C_B = \left(\frac{S_B}{S_A}\right)^{0.6} \times C_A \qquad (2)$$

Where C_B is the approximate cost ($) of equipment having size S_B (cfm, Hp, ft^2, or whatever), C_A is the known cost ($) of equipment having corresponding size S_A (same units as S_B), S_B/S_A is known as the size factor (dimensionless).

In relation to the abandon option if the project payoff is not attractive, the option to abandon the project is executable. Exercising the project abandon could minimize the losses by either selling off the project's salable equipment or reducing production costs. This option has the characteristics of a put option. The strike prices of these options are presented in Table 3.

7.4 Simulation

In this study, The Monte Carlo simulation is used to simulate the thousands paths of the uncertain asset values, defined by equation (3) through randomly and changing values. To simulating every time increment asset value, the underlying asset value is again calculated for the next time increment using the same equation. In this fashion, asset values for each time step are calculated until the end of the option life. Finally, by applying the decision rule, maximization of the value, the value at the end of the fifth year is compared with strike price of each option.

If this value was more than the cost of exercising the option, the option value for that simulation would be the difference between the asset value and its strike price. Otherwise, that option would be worthless and thus, zero value is allocated for it. The Option values for each simulation are discounted to their current values using a risk-free rate and ultimately the mean of which is considered the real option value.

Input parameters at table (5) are determined for each real world option. The asset value, which may has fluctuations over the option life, is defined by the following equation.

$$St = S_{t-1} + S_{t-1}(r\times\delta+\sigma\times\varepsilon\times\sqrt{\delta_t}) \qquad (3)$$

Where S_t and S_{t-1} are the underlying asset values at time t and time t–1, respectively; σ is the volatility of the underlying asset value; and ε is the simulated value from a standard normal distribution with mean of zero and a variance of 1.

In this research a command-based MATLAB simulation was conducted using the volatility factor. A Path is created for each simulated underlying asset value over the option life. Running 100,000 trials develops millions simulated value paths. The average of the option values from such numbers of trials in the case study is the value of the option at the end of five years.

Table 6 shows the results of the simulation using the first 20 trials as a sample. The probability of exercising the options, expansion, contraction and abandonment of the project, are 0.32%, 78.02%, 99.97% of the time respectively. And the estimated average values for each mentioned option are 13,407, 221,605,352 and 19,937,127dollars respectively. The added values could be substantial for a non-economic project close to break point estimated by DCF analysis (whit the net present value of -1,695,397 dollars).

8 CONCLUSION

This paper proposed an approach to investigate the role of the real options valuation in a particular mine, Titan Kahnoj plant. Using the ROA provides incorporating managerial flexibility to make investment decisions.

To examine the performances of the ROA versus the traditional NPV method and its advantage, a command-based Monte Carlo simulation thought MATLAB software was carried out. Both methods were applied for valuation of the case study. The managerial flexibility to change the project course was considered in the form of the main options (expansion, contraction and abandonment options).

This research can be suitable to discuss of the implementation and management decisions using real options to go out from the current situation in similar projects.

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APPENDIX

Table 3: Calculation of the proportion of each product in cash flow returns and asset value.

Year	Income of Sorel Iron ($)	Income of Titan Pigment ($)	Income of Titan magnetite concentrate ($)	Operational Cost ($)	Cash flows ($)
0	14,400,000	21,000,000	1,130,000	2,481,296.90	33,048,703
1	2,880,000	42,000,000	2,260,000	3,481,269.90	42,177,410
2	3,240,000	47,250,000	2,542,500	4,962,589.72	47,449,586
3	3,600,000	52,500,000	2,825,000	5,582,913.54	52,721,763
4	3,600,000	52,500,000	2,825,000	6,203,237.36	52,721,763
5	3,600,000	52,500,000	2,825,000	6,203,237.36	52,721,763
6	3,600,000	52,500,000	2,825,000	6,203,237.36	52,721,763
7	3,600,000	52,500,000	2,825,000	6,203,237.36	52,721,763
8	3,600,000	52,500,000	2,825,000	6,203,237.36	52,721,763
9	3,600,000	52,500,000	2,825,000	6,203,237.36	52,721,763
10	3,600,000	52,500,000	2,825,000	6,203,237.36	52,721,763
11	3,600,000	52,500,000	2,825,000	6,203,237.36	52,721,763
12	3,600,000	52,500,000	2,825,000	6,203,237.36	52,721,763
13	3,600,000	52,500,000	2,825,000	6,203,237.36	52,721,763
14	3,600,000	52,500,000	2,825,000	6,203,237.36	52,721,763
15	3,600,000	52,500,000	2,825,000	6,203,237.36	52,721,763
Asset value (S_0)					179,764,920.45

Table 4: calculation of the volatility factor.

Year	P_t ($/ton)	Ratio(P_t/P_{t-1})	ln Ratio	Deviation	Deviation^2	Volatility
Titan Magnetite Concentrate (T.M.C.)						
2006	53.9					
2007	59.7	1.108	0.044	-0.017	0.000	
2008	70.5	1.181	0.722	0.011	0.000	
2009	93	1.319	0.120	0.059	0.003	
2010	90	0.968	-0.014	-0.075	0.006	
2011	109	1.211	0.083	0.022	0.000	
Mean			0.061		0.010	0.045
Titan Pigment (T.P.)						
1992	2010					
1993	1920	0.955	-0.020	-0.110	0.012	
1994	1850	0.964	-0.016	-0.106	0.011	
1995	1910	1.032	0.014	-0.076	0.006	
1996	1890	0.990	-0.005	-0.095	0.009	
1997	1760	0.931	-0.031	-0.121	0.015	
1998	1840	1.045	0.019	-0.071	0.005	
1999	1890	1.027	0.012	-0.079	0.006	
2000	1880	0.995	-0.002	-0.093	0.009	
2001	1890	1.005	0.002	-0.088	0.008	
2002	1800	0.952	-0.021	-0.111	0.012	
2003	1830	1.017	0.007	-0.083	0.007	
2004	1780	0.973	-0.012	-0.102	0.010	
2005	2180	1.225	0.088	-0.002	0.000	
2006	2150	0.986	-0.006	-0.096	0.009	
2007	1870	0.870	-0.061	-0.151	0.023	
2008	2210	1.182	0.073	-0.018	0.000	
2009	2210	1.000	0.000	-0.090	0.008	
2010	2320	1.050	0.021	-0.069	0.005	
2011	2856	1.231	0.090	0.082	0.007	
Mean			0.008		0.162	0.092
Sorrel Iron (S.I.)						
2001	103					
2002	107	1.039	0.017	-0.052	0.003	
2003	127	1.187	0.074	0.006	0.000	
2004	184	1.449	0.161	0.093	0.009	
2005	257	1.397	0.145	0.077	0.006	
2006	259	1.008	0.003	-0.065	0.004	
2007	313	1.208	0.082	0.014	0.000	
2008	498	1.591	0.202	0.134	0.018	
2009	343	0.689	-0.162	-0.230	0.053	
2010	422	1.230	0.090	0.022	0.000	
Mean			0.068		0.093	0.102
The exercised volatility		25.39 %				

Table 5: Monte Carlo simulation input parameters and results.

Input Parameters		Results of simulation	
Present value of future cash flows	179764920	DCF-based Net present value	-1695397 $
Volatility	25.39%	Option values	values added by Options
Risk-free rate of return	18%	Option to expand	13407 $
Time to expiration	5 years	Option to contract	219909955 $
Time step	1 year	Option to abandon	18241730 $
Total no. of simulation trials	100000		
Strike price			
Option to expand	374579711 $	Probability of exercising the option	
Option to contract	163045289 $	Option to expand	0.32 %
Option to abandon	151160266 $	Option to contract	99.97 %
		Option to abandon	78.02 %

Table 6: The First 20 Simulation Trials for abandonment option.

	Time Increment No.							
N0.	Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Option value at year 5	Present value of the option
1	179764920	172913321	223982577	209399269	195131858	186063520	0	0
2	179764920	299286843	387680012	362438508	337743775	322047851	133097519	54113413
3	179764920	293019886	379562134	354849178	330671543	315304287	126353955	51371684
4	179764920	229181480	296869310	277540411	258630206	246610917	57660586	23443045
5	179764920	237125038	307158967	287160117	267594474	255158590	66208258	26918269
6	179764920	159619636	206762656	193300521	180129998	171758839	0	0
7	179764920	274988539	356205301	333013088	310323254	295901641	106951309	43483157
8	179764920	222654780	288414977	269636532	251264857	239587857	50637526	20587682
9	179764920	162315660	210254939	196565425	183172448	174659898	0	0
10	179764920	192571405	249446597	233205349	217315912	207216617	18266285	7426517
11	179764920	131220870	169976428	158909412	148082126	141200324	0	0
12	179764920	236593133	306469967	286515977	266994222	254586234	65635902	26685566
13	179764920	234670086	303978955	284187153	264824073	252516937	63566605	25844253
14	179764920	209330635	271155598	253500896	236228623	225250400	36300068	14758506
15	179764920	175360807	227152918	212363191	197893834	188697139	0	0
16	179764920	159994033	207247629	193753919	180552503	172161710	0	0
17	179764920	144037283	186578117	174430180	162545387	154991436	0	0
18	179764920	186972036	242193477	226424473	210997050	201191411	12241080	4976852
19	179764920	244112481	316210124	295621961	275479771	262677434	73727103	29975203
20	179764920	178939949	231789145	216697558	201932879	192548477	3598146	1462897
0	0	0	0	0	0	0	0	0
0	0	0	0	0	0	0	0	0
0	0	0	0	0	0	0	0	0

A Fuzzy VIKOR Technique to Selection of Optimum Underground Mining Method for Jajarm Bauxite Mine, Iran

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ABSTRACT Underground mining method (UMM) selection is the first and one of the most crucial decisions that should be made by mining engineers. In this regard some of the parameters such as geological and geotechnical properties, economic parameters and geographical factors are involved. Choosing a suitable underground mining method to extract mineral deposits is very important in terms of the economics, safety and the productivity of mining operations. This paper attempts to demonstrate the calculation of the weighting factors for each selected underground mining method. In practice, underground mining method could be selected using multiple criteria decision making (MCDM) techniqes and decision makers have always some difficulties in making the right decision in the multiple criteria environment. Most multi-criteria methods focus on ranking and selecting from a set of alternatives. In this research Jajarm bauxite mine was selected as a case study and optimal method of mining for this mine was proposed using fuzzy VIKOR technique. The the fuzzy VIKOR technique was developed to solve MCDM problems with conflicting and non-commensurable criteria assuming that compromising is acceptable to resolve conflicts. In this technique also importance weights of decision makers' opinions have considered different. Finally according to this technique the most appropriate mining methods for this mine were ranked.

Keywords: Multi-criteria decision making, VIKOR, Fuzzy logic, mining method selection

1 INTRODUCTION

Relaible selection of UMM is necessary to optimal design of mine (Alpay & Yavuz, 2009).

To make a suitable decision on underground mining method selection, all known criteria related to the problem should be analyzed. Although an increasing in the number of related criteria makes the problem more complicated, this may also increase the correctness of the decision. Due to the arising complexity in the decision process, many conventional methods are able to consider limited criteria and may be generally deficient. Therefore, it is clearly seen that assessing all of the known criteria connected to the mining method selection by combining the decision making process is extremely significant (Hartman & Mutmansky, 2002).

Once selected a mining method, it is nearly impossible to change it owing to the rising costs and mining losses, it is very important to re-analyze the decision made before carrying it out (Naghadehi & Ataei, 2009).

In this sensitively analysis was generally used on the final decision (Alpay & Yavuz, 2009).

The aim of this paper is to compare the many different geological, geotechnical, economical and technical aspects in the selection of the most appropriate underground mining method for Jajarm

Bauxite Mine in Iran, with reference to some different extraction methods. The comparison has been performed with the combination of the VIKOR method and fuzzy logic (Fuzzy VIKOR Method).

In the multiple criteria decision making (MCDM) problems, since that the valuation of criteria leads to diverse opinions and meanings, each attribute should be imported with a specific importance weight (Chen, Tzeng & Ding, 2003). A question rises up here and that is ‘‘how this importance weight could be calculated’’? In literature, most of the typical MCDM methods leave this part to decision makers, while sometimes it would be useful to engage end-users into the decision making process. To obtain a better weighting system, weighting methods are usually divided into two categories: subjective methods and objective methods (Wang & Lee, 2009). While subjective methods determine weights solely based on the preference or judgments of decision makers, objective methods utilize mathematical models, such as entropy method or multiple objective programming, automatically without considering the decision makers' preferences. The approach with objective weighting is particularly applicable for situations where reliable subjective weights cannot be obtained (Deng, Yeh & Willis, 2000). On the other hand, new researches entail new MCDM approaches such as VIKOR.

VIKOR is a helpful tool in multi-criteria decision making (MCDM), the obtained compromise solution could be accepted by the decision makers because it provides a maximum group utility (represented by min S,) of the majority, and a minimum of the individual regret (represented by min R) of the opponent.

2 VIKOR TECHNIQUE

Vlsekriterijumska Optimizacija I Kompromisno Resenje (i.e. VIKOR) method was developed by Opricovic in 1998 for multi-criteria optimization of complex systems (Opricovic & Tzeng, 2002). VIKOR focuses on ranking and sorting a set of alternatives against various, or possibly conflicting and non-commensurable, decision criteria assuming that compromising is acceptable to resolve conflicts. Similar to TOPSIS as a MCDM method, VIKOR relies on an aggregating function that represents closeness to the ideal, but unlike TOPSIS, introduces the ranking index based on the particular measure of closeness to the ideal solution. This method uses linear normalization to eliminate units of criterion functions (Opricovic & Tzeng, 2004).

The VIKOR method was developed for the multi-criteria optimization of complex systems. It determines the compromise ranking list and the compromise solution. The weight stability intervals for the preferred stability of the compromise solution can be obtained from the initial weights given by the AHP in the traditional method. This traditional method focuses on ranking and selection from a set of alternatives in cases of conflicting criteria. It introduces a multi-criteria ranking index based on the particular measure of ‘‘closeness’’ to the ‘‘ideal’’ solution (Chiu & Tzeng, 2012). The VIKOR method began with the form of L_p-*metric*, which was used as an aggregating function in a compromise programming method and developed into the multi-criteria measure for compromise ranking. We assume the alternatives are denoted as $A_1, A_2, \ldots, A_i, \ldots, A_m$. w_j is the weight of the jth criterion, expressing the relative importance of the criteria, where $j = 1, 2, \ldots, n$, and n is the number of criteria. The rating of the jth criterion is denoted by f_{ij} for alternative A_i. The form of L_p-*metric* is formulated as follows:

$$L_i^p = \left\{ \sum_{j=i}^{n} \left[\omega_j (f_j^+ - f_{ij}) / (f_j^+ - f_j^-) \right]^p \right\}^{1/p} \quad (1)$$

$1 \le p \le \infty$; $i=1,2,\ldots,m$

The VIKOR method is not only generated with the above form of L_p-*metric*, but also uses $L_i^{p=1}$ (as S_i in Eq. (2)) and $L_i^{p=\infty}$ (as R_i in Eq. (3)) to formulate the ranking measure. (chen et al.,2011; Chiu & Tzeng, 2012).

$$L_i^{p=1} = S_i = \sum_{j=1}^{n} \left[\omega_j (f_j^+ - f_{ij}) / (f_j^+ - f_j^-) \right] \quad (2)$$

$$L_i^{p=\infty} = R_i = \max_j \left\{ \omega_j (f_j^+ - f_{ij}) / (f_j^+ - f_j^-) \right\} \quad (3)$$

When p is small, the group utility is emphasized (such as p=1) and as p increases, the individual regrets/gaps receive more weight(Chiu & Tzeng,2012). In addition, the compromise solution $\min_i L_i^p$ will be chosen because its value is closest to the ideal/aspired level. Therefore, $\min_i S_i$ expresses the minimization of the average sum of the individual regrets/gaps and $\min_i R_i$ expresses the minimization of the maximum individual regret/gaps for prioritizing the improvement. In other words, $\min_i S_i$ emphasizes the maximum group utility, whereas $\min_i R_i$ emphasizes selecting minimum among the maximum individual regrets. Based on the above concepts, the compromise-ranking algorithm VIKOR consists of the following steps.

Step 1: Determine the best f_j^+ , and the worst f_j^- values of all criterion functions, j = 1, 2, . . ., n. If we assume the jth function represents a benefit, then $f_j^+ = \max_i f_{ij}$ (or setting an aspired level) and $f_j^- = \min_i f_{ij}$ (or setting a tolerable level). Alternatively, if we assume the jth function represents a cost/risk, then $f_j^+ = \min_i f_{ij}$ (or setting an aspired level) and $f_j^- = \max_i f_{ij}$ (or setting a tolerable level).

Step 2: Compute the values S_i and R_i, $i = 1, 2, \ldots, m$, using the relations Eq.(4)&(5).

Step 3: Compute the Q_j values for $i=1, 2, \ldots, m$ with the relation Eq.(6).

$$S_i = \sum_{j=1}^{n} \left[\omega_j (f_j^+ - f_{ij}) / (f_j^+ - f_j^-) \right] \quad (4)$$

$$R_i = \max_j \left\{ \omega_j (f_j^+ - f_{ij}) / (f_j^+ - f_j^+) \right\} \quad (5)$$

$$Q_i = v \left[\frac{S_i - S^*}{S^- - S^*} \right] + (1 - v) \left[\frac{R_i - R^*}{R^- - R^*} \right] \quad (6)$$

Where, $S^* = Min_i S_i$, $S^- = Max_i S_i$, $R^* = Min_i R_i$, $R^- = Max_i R_i$ and $0 \le v \le 1$, where v is introduced as a weight for the strategy of maximum group utility, whereas $1-v$ is the weight of the individual regret. In other words, when $v > 0.5$, this represents a decision-making process that could use the strategy of maximum group utility (i.e. if v is big, group utility is emphasized), or by consensus when $v \approx 0.5$, or with veto when $v < 0.5$ (Opricovic,1998 & Kackar 1985).

Step 4: Rank the alternatives, sorting by the value of $\{S_i, R_i, \text{ and } Q_i | i = 1, 2, \ldots, m\}$, in decreasing order. Propose as a compromise the alternative ($A_{(1)}$) which is ranked first by the measure $\min\{Q_i | i = 1, 2, \ldots, m\}$ if the following two conditions are satisfied (Huang et al., 2009):

C_1. Acceptable advantage: $Q(A_{(2)}) - Q(A_{(1)}) \ge 1/(m-1)$, where $A_{(2)}$ is the alternative with second position in the ranking list by Q; m is the number of alternatives.

C2. Acceptable stability in decision making: Alternative $A_{(1)}$ must also be the bestranked by $\{S_i \text{ or/and } R_i | i = 1, 2, \ldots, m\}$.

If one of the conditions is not satisfied, then a set of compromise solutions is proposed, which consists of:

• Alternatives $A_{(1)}$ and $A_{(2)}$ if only condition C_2 is not satisfied.

• Alternatives $A_{(1)}, A_{(2)}, \ldots, A_{(M)}$ if condition C_1 is not satisfied. $A_{(M)}$ is determined by the relation $Q(A_{(M)}) - Q(A_{(1)}) < 1/(m-1)$ for maximum M (the positions of these alternatives are close.

The compromise solution is determined by the compromise-ranking method; the obtained compromise solution could be accepted by the decision makers because it provides maximum group utility of the majority (represented by min S, Eq. (4)), and minimum individual regret of the opponent (represented by min R, Eq. (5)).

The VIKOR algorithm determines the weight stability intervals for the obtained compromise solution with the input weights given by the experts.(Opricovic, 1998)

3 FUZZY LOGIC

A linguistic variable is defined as a variable whose values are not numbers, but words or sentences in natural or artificial languages. The concept of a linguistic variable appears as useful means for providing approximate characterization of phenomena that are too complex or ill-defined to be described in conventional quantitative terms (Zadeh, 1965).

The use of linguistic variables enables Decision Makers (DMs) to specify both the importance associated with each of a set of criteria, and the preference with respect to a number of strategic criteria which impact the selection and justification of several alternatives. The value of a linguistic variable can be quantified and extended to mathematical operations using fuzzy set theory (Zadeh, 1975).

A fuzzy number is a special fuzzy set $F = \{x \in R \mid f_\lambda(x)\}$, where x takes its values on the real line $\vartheta^1 : -\infty < x < +\infty$

and $f_\lambda(x)$ is a continuous mapping from ϑ^1 to the close interval [0,1]. A triangular fuzzy number can be denoted as $A=[a_1,a_2,a_3]$ (where $-\infty < a_1 \le a_2 \le a_3 < +\infty$ and $a_1,a_2,a_3 \in R$) and its membership function $f_A(x) : \vartheta^1 \to [0, 1]$ can be given as:

$$f_A(x) = \begin{cases} (x-a_1)/(a_2-a_1) \\ (x-a_3)/(a_2-a_3) \\ 0 \qquad Otherwise \end{cases}$$

where $b_1 \le b_2 \le b_3$, b_1 and b_3 stand for the lower and upper value of the support of A , respectively, and b is the mid-value of A.

The main operational laws for two triangular fuzzy numbers $A=[a_1,a_2,a_3]$ and $B=[b_1,b_2,b_3]$ and one no fuzzy number $n=[n_1,n_2,n_3]$ are as follows (Kaufmann & Gupta, 1991):

$$(A \oplus B) = [a_1+b_1, a_2+b_2, a_3+b_3]$$
$$(A\,!\ B) = [a_1+b_1, a_2+b_2, a_3+b_3]$$
$$(A \otimes B) = [a_1b_1, a_2b_2, a_3b_3]$$
$$(A \otimes n) = [a_1n_1, a_2n_2, a_3n_3]$$

3.1 Defuzzification

Fuzzy numbers can be regarded as systems with numerical input and numerical output. Internally these systems work with fuzzy values, which have to be mapped to non-fuzzy (crisp) values after processing. This conversion is called defuzzification. In this paper the mean value method is used for defuzzification.

A fuzzy number $A=[a_{1i},a_{2i},a_{3i}]$ can always be given by its corresponding left and right representation of each degree of membership:

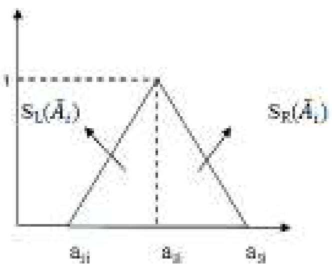

Figure 1.*A* triangular fuzzy number A_i

Mean value method for defuzzification:

$$S\left(\tilde{A}_i\right)=\frac{1}{2}\left(S_L\left(\tilde{A}_i\right)+S_R\left(\tilde{A}_i\right)\right) \tag{7}$$

$$\left(S\left(\tilde{A}_i\right)=\frac{1}{2}\left(\left(a_{2i}-\int_{a_{1i}}^{a_{2i}} f_{\lambda i}(x)\right)+\left(a_{2i}+\int_{a_{2i}}^{a_{3i}} f_{\lambda i}(x)\right)\right) \tag{8}$$

4 FUZZY VIKOR TECHNIQUE

Assumptions and method steps are as follows:

K = Number of decision makers, where *K=1, 2,…,k.*

i= Number of alternatives, where *i=1, 2,…,m.*

j= Number of criteria, where *j=1, 2,…,n.*

Step 1: Making matrix of criteria- decision makers.

Form a group of decision makers, determine the evaluation criteria and feasible alternatives. *k* decision makers use the linguistic variables, such as very low, low, medium, high and very high (the corresponding fuzzy numbers of linguistic terms are shown in Table 1) to assess the importance weight of criteria. Triangular fuzzy numbers for importance weight of the criteria are shown in Figure 2.

Hence the matrix of criteria- decision makers can be written as Figure 3.

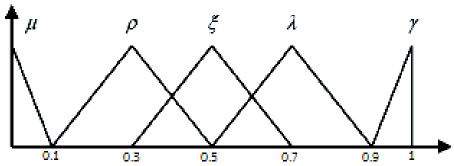

Figure 2. Triangular fuzzy numbers for importance weight of the criteria

Table 1. Linguistic variables for the importance weight of criteria

Symbol	Linguistic terms	Triangular fuzzy number
γ	very high	$\gamma=(\gamma^L,\gamma^M,\gamma^R)$
λ	high	$\lambda=(\lambda^L,\lambda^M,\lambda^R)$
ξ	medium	$\xi=(\xi^L,\xi^M,\xi^R)$
ρ	low	$\rho=(\rho^L,\rho^M,\rho^R)$
μ	very low	$\mu=(\mu^L,\mu^M,\mu^R)$

$$\begin{array}{c c} & \begin{array}{ccccc} D_1 & D_2 & \cdots & D_k & \omega_j \end{array} \\ \begin{array}{c} C_1 \\ C_2 \\ \vdots \\ C_n \end{array} & \begin{bmatrix} x_{11} & x_{12} & \cdots & x_{1k} & \omega_1 \\ x_{21} & x_{22} & \cdots & x_{2k} & \omega_2 \\ \vdots & \vdots & \cdots & \vdots & \vdots \\ x_{n1} & x_{n2} & \cdots & x_{nk} & \omega_n \end{bmatrix} \end{array}$$

Figure 3. Matrix of criteria- decision makers

x_{jk} = The rating of the criteria C_j with respect to decision maker D_k.

ω_j is the importance weight of the *j*th criterion holds.

w_k is the importance weight of decision makers opinions, where $w_k=[0,1]$.

$$\tilde{\omega}_j = \begin{pmatrix} min\left\{x_{j1}^L, x_{j2}^L, \cdots, x_{jk}^L\right\}, \\ w_1 x_{j1}^M + w_2 x_{j2}^M + \cdots + w_k x_{jk}^M, \\ max\left\{x_{j1}^R, x_{j2}^R, \cdots, x_{jk}^R\right\} \end{pmatrix} \quad (9)$$

Step 2: Making matrix of decision makers-alternatives- criteria.

Identify the appropriate linguistic variables for evaluating the importance weight of criteria, and the rating of alternatives.

K decision-makers use linguistic variables: very poor, poor, medium, good and very good (the corresponding fuzzy numbers of linguistic terms are shown in Table 2) to evaluate the rating of *m* candidates in *n* criteria. Triangular fuzzy numbers for the rating of alternative are shown in Figure 4.

Table 2. Linguistic variables for the rating of alternative

Symbol	Linguistic terms	Triangular fuzzy number
σ	very good	$\sigma = (\sigma^L, \sigma^M, \sigma^R)$
τ	good	$\tilde{\tau} = (\tau^L, \tau^M, \tau^R)$
η	medium	$\eta = (\eta^L, \eta^M, \eta^R)$
ψ	poor	$\psi = (\psi^L, \psi^M, \psi^R)$
x	very poor	$x = (x^L, x^M, x^R)$

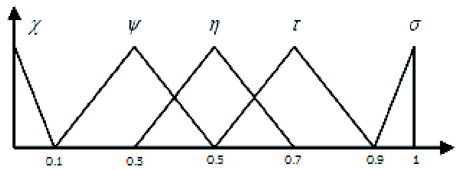

Figure 4. Triangular fuzzy numbers for the rating of alternative

Hence the matrix of decision makers-alternatives- criteria and the fuzzy decision matrix can be written as Table 3.

Step 3: combination of matrix of decision makers- criteria and decision makers-alternatives- criteria.
with the relation Eq.(10).

$$Z_{ij} = (\min\left\{y_{1ij}^L, \ldots, y_{kij}^L\right\}, w_1 x_{1ij}^m + \ldots + w_k x_{kij}^m, \max\left\{y_{1ij}^R, \ldots, y_{kij}^R\right\}) \quad (10)$$

Z_{ij} : A Fuzzy Variable of the *i*th alternative according to *j*th criteria.

Table 3. Matrix of decision makers-alternatives- criteria

		C_1	C_2	…	C_n
D_1	A_1	γ_{111}	γ_{112}	…	γ_{11n}
	A_2	γ_{121}	γ_{122}	…	γ_{12n}
	⋮	⋮	⋮	…	⋮
	A_m	γ_{1m1}	γ_{1m2}	…	γ_{1mn}
D_2	A_1	γ_{211}	γ_{212}	…	γ_{21n}
	A_2	γ_{221}	γ_{222}	…	γ_{22n}
	⋮	⋮	⋮	…	⋮
	A_m	γ_{2m1}	γ_{2m2}	…	γ_{2mn}
	⋮	⋮	⋮	…	⋮
D_k	A_1	γ_{k11}	γ_{k12}	…	γ_{k1n}
	A_2	γ_{k21}	γ_{k22}	…	γ_{k2n}
	⋮	⋮	⋮	…	⋮
	A_m	γ_{km1}	γ_{km2}	…	γ_{kmn}

Table 4. Aggregated triangular fuzzy number decision matrix

	C_1	C_2	...	C_n
	ω_1	ω_2	...	ω_n
A_1	Z_{11}	Z_{12}	...	Z_{1n}
A_2	Z_{21}	Z_{22}	...	Z_{2n}
⋮	⋮	⋮	...	⋮
A_m	Z_{m1}	Z_{m2}	...	Z_{mn}

Step 4: Defuzzification

Convert fuzzy number to non- fuzzy number (with using the relations Eq.(7 & 8)). The result of this step is given in Table 5.

Step 5: Determine the best f_j^+ , and the worst f_j^- values of all criterion functions, j=1, 2, . . ., n. If we assume the jth function represents a benefit, then f_j^+=max$_i$ f_{ij} (or setting an aspired level) and f_j^- = min$_i$ f_{ij} (or setting a tolerable level). Alternatively, if we assume the jth function represents a cost/risk, then f_j^+=min$_i$ f_{ij} (or setting an aspired level) and f_j^- = max$_i$ f_{ij} (or setting a tolerable level).

Table 5. Non fuzzy number decision matrix

	C_1	C_2	...	C_n
	ω_1	ω_2	⋯	ω_n
A_i	f_{11}	f_{12}		f_{1n}
A_2	f_{21}	f_{22}		f_{2n}
⋮	⋮	⋮		⋮
A_m	f_{m1}	f_{m2}		f_{mn}

Step 6: Compute the values S_i and R_i, i = 1, 2, . . .,m, using the relations Eq.(4) & (5).

Step 7: Compute the Q_j values for i=1,2,…,m with the relation Eq.(6).

Step 8: Rank the alternatives, sorting by the value of $\{S_i, R_i,$ and $Q_i | i$ =1, 2, . . .,$m\}$, in decreasing order.

Step 9: Propose as a compromise the alternative ($A_{(1)}$) which is ranked first by the measure min$\{Q_i | i$ = 1, 2, . . .,$m\}$ if the following two conditions are satisfied (Huang et al., 2009).

C_1. Acceptable advantage: $Q(A_{(2)})-Q(A_{(1)}) \geq 1/(m-1)$, where $A_{(2)}$ is the alternative with second position in the ranking list by Q; m is the number of alternatives.

C_2. Acceptable stability in decision making: Alternative $A_{(1)}$ must also be the bestranked by $\{S_i$ or/and $R_i | i$ = 1, 2, . . .,$m\}$.

If one of the conditions is not satisfied, then a set of compromise solutions is proposed, which consists of:

- Alternatives $A_{(1)}$ and $A_{(2)}$ if only condition C_2 is not satisfied.
- Alternatives $A_{(1)}$,$A_{(2)}$, . . . , $A_{(M)}$ if condition C_1 is not satisfied. $A_{(M)}$ is determined by the relation $Q(A_{(M)}) - Q(A_{(1)}) < 1/(m - 1)$ for maximum M (the positions of these alternatives are close).

5 CASE STUDY

The purpose of this paper is to selection of the optimum underground mining method for Jajarm Bauxite Mine, using data obtained from the mine site.

5.1 Selection of Criteria

There are too many factors affecting mining method selection such as spatial characteristics of the deposit, geologic and hydrologic conditions, geotechnical properties, economic considerations, technological factors and environmental concerns. Main criteria and their sub-criteria are mentioned follows (Hartman and Mutmansky, 2002):

(a) Spatial characteristics of the deposit such as general shape, plunge, dip, depth, ore thickness existence of previous mining.

(b) Geologic and hydrologic conditions such as mineralogy and petrography, chemical composition, deposit structure, uniformity of grade, alteration and weathered zones, and existence of strata gases.

(c) Geotechnical properties such as elastic properties, plastic or viscoelastic behavior, state of stress, rock mass rating and other physical properties affecting competence.

(d) Economic considerations such as reserves, production rate, mine life, productivity, comparative mining costs and comparative capital costs.

(e) Technological factors such as recovery, dilution, flexibility of the method to changing conditions, selectivity of the method, concentration or dispersion of workings, ability to mechanize and automate and capital and labor intensities.

(f) Environmental concerns such as ground control to maintain integrity of openings, subsidence or caving effects at the surface, atmospheric control, availability of suitable waste disposal areas, workforce and comparative safety conditions of the suitable mining methods.

According to this criterion, 12 criteria having the most important are selected in Jajarm mine which are shown in Table 6.

Table 6. Important creteria

Symbol	Criteria
C_1	Deposite thikness
C_2	Deposite dip
C_3	Deposite shape
C_4	RMR of hangingwall
C_5	RMR of ore
C_6	RMR of footwall
C_7	Depth
C_8	Recovery
C_9	Production
C_{10}	Ore grade
C_{11}	Ore uniformity
C_{12}	Dilution

5.2 Candidate Mining Methods

According to the mine and ore body conditions, five mining methods that are possible and appropriate to this mine, considered. These mining methods are given in Table 7.

Table 7. Candidate mining methods

Symbol	Method
SHS	Shrinkage stoping
CFS	Cut & fill stoping
SS	Stull stoping
SLS	Sublevel stoping
BM	Bench mining

5.3 Fuzzy VIKOR technique for selection of optimum underground mining method

Step 1: Form a group of decision makers; determine the evaluation criteria and feasible alternatives.

Step 2: Identify the appropriate linguistic variables for evaluating the importance weight of criteria, and the rating of alternatives.

Step 3: Aggregated triangular fuzzy number decision matrix. (Table 8)

Step 4: Defuzzification

Convert fuzzy number to non-fuzzy number.

(Table 9)

Step 5: Determine the best f_j^+ , and the worst f_j^- values of all criterion functions, $j = 1, 2, \ldots, n$ (Table 10).

Table 8. Aggregated triangular fuzzy number decision matrix

	ω_j	SHS	CFS	SS	SLS	BM
C_1	(0.9,1,1)	(0.1,0.44,1)	(0.9,1,1)	(0,0.44,1)	(0.1,0.34,0.7)	(0,0,0.1)
C_2	(0.5,0.75,1)	(0.1,0.54,1)	(0.3,0.66,1)	(0,0.41,1)	(0.1,0.54,1)	(0.1,0.34,0.7)
C_3	(0.5,0.85,1)	(0.3,0.5,0.7)	(0.3,0.6,1)	(0,0.41,1)	(0,0.06,0.5)	(0.1,0.4,0.7)
C_4	(0.1,0.52,0.9)	(0.1,0.3,0.5)	(0.1,0.38,0.9)	(0.1,0.3,0.5)	(0.1,0.34,0.7)	(0.1,0.3,0.5)
C_5	(0,0.15,0.7)	(0.1,0.3,0.5)	(0,0.15,0.5)	(0,0,0.1)	(0.1,0.65,1)	(0,0,0.1)
C_6	(0.9,1,1)	(0,0.24,0.5)	(0.1,0.52,0.9)	(0,0.24,0.5)	(0,0.3,0.7)	(0.1,0.3,0.5)
C_7	(0.3,0.56,0.9)	(0,0.25,0.7)	(0.1,0.56,0.9)	(0,0.15,0.5)	(0.1,0.46,0.9)	(0,0,0.1)
C_8	(0.3,0.64,0.0.9)	(0.1,0.3,0.5)	(0.5,0.91,1)	(0,0,0.1)	(0.1,0.3,0.5)	(0,0,0.1)
C_9	(0.3,0.85,1)	(0,0,0.1)	(0.5,0.7,0.9)	(0,0.21,0.5)	(0.1,0.15,0.5)	(0,0,0.1)
C_{10}	(0.3,0.66,1)	(0.1,0.4,0.7)	(0.5,0.7,0.9)	(0,0.31,0.9)	(0,0.35,0.9)	(0.1,0.3,0.5)
C_{11}	(0.5,0.85,1)	(0.1,0.3,0.5)	(0.1,0.3,0.5)	(0.1,0.3,0.5)	(0.1,0.3,0.5)	(0.1,0.21,0.5)
C_{12}	(0.1,0.44,1)	(0.1,0.36,0.7)	(0.5,0.7,0.9)	(0,0.09,0.5)	(0.1,0.3,0.5)	(0,0,0.1)

Table 9. Non fuzzy number decision matrix

	ω_j	SHS	CFS	SS	SLS	BM
C_1	0.975	0.495	0.975	0.47	0.37	0.025
C_2	0.75	0.545	0.655	0.455	0.545	0.37
C_3	0.8	0.5	0.625	0.455	0.155	0.4
C_4	0.51	0.3	0.44	0.3	0.37	0.3
C_5	0.25	0.3	0.2	0.025	0.6	0.025
C_6	0.975	0.245	0.6	0.245	0.325	0.3
C_7	0.58	0.3	0.53	0.2	0.48	0.025
C_8	0.62	0.3	0.83	0.025	0.3	0.025
C_9	0.75	0.025	0.525	0.23	0.1875	0.025
C_{10}	0.655	0.4	0.525	0.38	0.4	0.3
C_{11}	0.8	0.3	0.3	0.3	0.3	0.255
C_{12}	0.495	0.38	0.525	0.17	0.3	0.025

Table 10. Determine the best f_j^+ , and the worst f_j^+ values of all criterion functions, j = 1, 2, . . ., n.

	C_1	C_2	C_3	C_4	C_5	C_6	C_7	C_8	C_9	C_{10}	C_{11}	C_{12}
f_j^+	0.975	0.655	0.625	0.44	0.6	0.6	0.53	0.83	0.525	0.525	0.3	0.525
f_j^+	0.025	0.37	0.155	0.3	0.025	0.245	0.025	0.025	0.025	0.3	0.255	0.025

Step 6: Compute the values S_i and R_i, i = 1, 2, . . .,m (Table 11).

Step 7: Compute the Q_j values for i=1,2,…,m (Table 12)

Step 8: Rank the alternatives, sorting by the value of $\{S_i, R_i,$ and $Q_i|i =1, 2, . . ., m\}$, in decreasing order (Table 13).

Table 11. Index S_i and R_i

	SHS	CFS	SS	SLS	BM
S_i	4.5401	0.1739	5.284	4.0789	7.2717
R_i	0.975	0.1739	0.975	0.8	0.975

Table 12 . Index Q_i

	SHS	CFS	SS	SLS	BM
Q_i	0.8076	0	0.858	0.6659	1

Table 13 . Rank the alternatives

	1	2	3	4	5
S_i	CFS	SLS	SHS	SS	BM
R_i	CFS	SLS	SHS	SS	BM
Q_i	CFS	SLS	SHS	SS	BM

Step 9: conditions

- Since the $Q(A_{(2)})-Q(A_{(1)}) \geq \frac{1}{5-1}$

  (0.6659-0 ≥ 0.25,
- Alternative $A_{(1)}$ also be the bestranked by $\{S_i$ or/and $R_i | i = 1, 2, \ldots, m\}$.

Hence alternative $A_{(1)}$ or cut and fill stoping method is optimum underground mining method for the mine under question.

6 CONCLUSION

There is no single appropriate mining method for a deposit. Usually, two or more feasible methods are possible and each method entails some inherent problems. Consequently, the optimal method is one that offers the least problems. Selection of an appropriate mining method is a complex task that requires consideration of many technical, economical, political, social, and historical factors. The appropriate mining method is that which is technically feasible for the ore geometry and ground conditions, while being a low-cost operation.

In this paper using fuzzy VIKOR method, the degree of importance of the effective factors on the model was investigated.

As a result, using this approach the cut and fill stoping method was selected as optimum underground mining method in Jajarm Bauxite Mine.

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Maden Sahalarındaki Mikrosismik Olayların Kökeni, Önemi, İzlenmesi ve Analizi

Origin, Importance, Monitoring and Analysis of Microseismic Events in Mine Sites

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ÖZET Yeraltı ve yerüstü madencilik faaliyetleri nedeniyle maden sahası dolayında küçük ölçekli (M<3.0) mikrodeprem etkinliğinin oluştuğu bilinen bir gerçektir. Bunlar, genellikle insan kaynaklı nedenlere bağlı olarak kontrolsüz bir şekilde meydana gelen, uyarılmış mikrosismik olaylardır. Bu olayların zaman ve uzay boyutu içerisinde izlenerek, ortama ait statik ve dinamik parametrelerin belirlenmesi amacıyla kullanılan yöntemlere ise Pasif Mikrosismik İzleme Yöntemleri denilmektedir. Bu yöntemler, maden sahasına ilişkin fiziksel parametrelerin belirlenmesinin yanısıra maden sahalarındaki sismik etkinliğin artışına bağlı olarak gelişen göçük, heyelan, kaya patlaması gibi risklerin belirlenmesi ve tehlikelerin önceden kestirilmesi gibi çalışmalar için de kullanılmaktadır. Bu nedenle, Pasif Mikrosismik İzleme Yöntemleri, maden işletmelerine ekonomik bir getiri sağlamaktan çok ekonomik kayıpların önlenmesi veya en aza indirilmesi amacı ile katkılarda bulunmaktadır. Ayrıca bu yöntemlerle, bölgesel depremlerin de takibi yapılarak maden sahalarında yaratacakları olası etkiler de incelenebilmektedir. Genellikle yüksek frekanslı algılayıcıların kullanıldığı bu tür yöntemlerde maden sahası dolayına yerleştirilmiş en az 8-10 istasyon ile gerçek zamanlı ve sürekli izlemeler yapılabilmektedir. Maden sahalarında meydana gelen mikrosismik olayların kaydedilmesinden analizine kadar birçok aşamasında sismoloji (deprembilim) kapsamında kullanılan yöntem ve teknikler uygulanmaktadır. Bu bildiride maaliyetleri açısından oldukça ekonomik, sonuçları açısından hayli önemli olan Pasif Mikrosismik İzleme Yöntemlerinin maden sahalarında uygulama teknikleri ve elde edilen sonuçlar anlatılmaktadır.
Anahtar Kelimeler: Mikrosismik, Uyarılmış Sismik Olaylar, Pasif Mikrosismik İzleme Yöntemleri

ABSTRACT It is a known fact that, due to underground and surface mining activities, small scale (M<3.0) microearthquakes occur around mine sites. These are induced microseismic events that usually develop in uncontrolled way depending on the cause of human activities.
The methods used to determine the static and dynamic parameters of the environment by monitoring the microseismic events in time and space domain are called Passive Microseismic Monitoring Methods. These methods not only used for determination of physical parameters on the mine site, but also they are used for determination of the risks of collapse, landslides, rock burst etc. and prediction of hazard which are developed due to increase in seismic activity in mine sites. For this reason, Passive Microseismic Monitoring

Methods contribute to prevent or minimize economic losses in mines rather than providing an economic return. Moreover, regional earthquakes and their impacts on mine sites may be followed by these methods. In such methods that are used usually high-frequency sensors, a real-time and continuous monitoring can be done by at least 8-10 stations installed around the mine site. Seismological methods and techniques are used in monitoring and analyzing of microseismic events in mine sites. In this paper, implementation techniques and obtained results of Passive Microseismic Monitoring Methods, which are cost-affordable and very important in terms of the results, are discussed.
Keywords : Microseismic, Induced Seismic Events, Passive Microseismic Monitoring Methods

1 GİRİŞ

Durağan (statik) ve/veya hareketli (dinamik) yüklerin etkisiyle elastik bir ortamda meydana gelen gerilme birikimleri, ortam içerisinde çatlak, kırık veya göçük oluşumlarına yol açabilmektedir. Bu olaylarla birlikte aynı zamanda ortam içerisinde elastik dalga yayılımı da söz konusu olur. Sismik dalga yayılımı olarak da bilinen bu yayılım, çatlak, kırık veya göçüğün meydana geldiği noktadan yayılarak bütün yönlere doğru ilerler. Çatlak, kırık veya göçük boyutları küçüldükçe, ortam içerisinde daha düşük enerjili sismik dalgaların yayılımı gerçekleşir. Doğal veya insan kaynaklı etkilere bağlı olarak herhangi bir jeolojik ortam veya mühendislik yapı ve malzemeleri içerisinde kontrolsüz bir şekilde gelişen bu tür küçük ölçekli (Richter ölçeğine göre $-3.0<M<3.0$) sismik olaylara "mikrosismik" olaylar denir. Oluşum ortamı ve kaynağı tamamen doğal ise bunlar "mikrodeprem" olarak adlandırılmaktadır. Ancak, insan etkisi ile bu tür mikrosismik olaylar oluşuyorsa bunlar "uyarılmış" veya "zorlanmış" sismik olaylar (induced seismic events) sınıfında yer alır. Ayrıca, bu olayların oluşumu doğrudan insan tarafından kontrol edilebilir bir kaynağa bağlı olmadığından, yani aktif bir kaynak kullanılmadığından, meydana gelişlerinde bir rastgelelik vardır. Bu nedenledir ki bu tür olaylar kendiliklerinden oluşamadıklarından daha çok "pasif mikrosismik" olarak adlandırılır. Doğal/yarı-doğal yapılar veya insan yapımı malzemeler içerisinde oluşan benzer küçük ölçekli pasif mikrosismik olayların kaydedilmesi ve analiz edilmesini kapsayan yöntemlere ise "pasif mikrosismik izleme yöntemleri" denilmektedir.

Maden sahaları, pasif mikrosismik olayların meydana geldiği ve gözlendiği yarı-doğal jeolojik ortamlardan biridir. Bu tür bölgelerdeki gerilme farklılığının oluşumu, madencilik faaliyetleri içerisindeki insan kaynaklı etkilere bağlı olduğundan, maden ocaklarının yer aldığı alanlar yarı-doğal ortamlar olarak nitelendirilir. Bu bölgelerde gözlenen mikrosismik olaylar uyarılmış sismik olaylardır, yani oluşumlarında bir zorlanma vardır. Maden sahalarında mikrosismik olayların meydana gelmesine yol açan gerilme farklılıkları, çoğunlukla rezerv niteliğindeki ekonomik değere sahip cevherin veya üzerindeki örtü katmanının alınması ile yer içerisindeki statik denge durumunun bozulması ile açıklanmaktadır. Mikrosismik olayların oluşumunda madencilik faaliyetlerinin yanısıra yerel jeolojik ve tektonik yapı, maden geometrisi, derinlik, yeraltı suyu, basınç ve sıcaklık gibi diğer faktörler de önemli roller oynamaktadır.

Farklı tekniklerle üretim yapılan yeraltı, yerüstü ve özel amaçlı maden işletme sahaların tamamında bu tür mikrosismik olaylar meydana gelmektedir. Maden sahalarında meydana gelen bu tür mikrosismik etkinlikler maden içerisinde göçüklere, kaya patlamalarına, ani gaz gelişi gibi bir takım olaylara yol açabilmektedir. Sismik etkinliğe bağlı olarak gelişen bu olaylar maden ocaklarında hem can hem de ekonomik kayıplar açısından büyük tehlikeler oluşturmaktadır. Bu tehlikelerin risk değerlendirmesi açısından maden işletmeleri tarafından önceden belirlenmesi ise son derece önemlidir. Potansiyel tehlike bölgeleri, çoğunlukla mikrosismik olayların meydana geldiği lokasyonlardır. Sismik risk çalışmaları için bu lokasyonların olabildiğince doğru bir şekilde belirlenmesi gerekmektedir.

Maden sahalarındaki bu tür mikrosismik etkinliğin izlenerek analiz edilmesi Pasif Mikrosismik İzleme Yöntemleri ile yapılabilmektedir. Başta mikrosismik olayların konumları olmak üzere birçok analiz yöntemi ile maden sahasına ilişkin önemli bilgiler elde edilebilmektedir. Bu yöntemlerin başarı ile uygulandığı dünya ölçeğinde birçok saha ve çalışma bulunmaktadır.

Bu çalışmada, maden sahalarında gözlenen mikrosismik olayların oluşum nedenleri, izleme ve analiz yöntemleri çeşitli örneklerle verilerek, bunların maden işletmeleri açısından önemi anlatılmaktadır.

2 FARKLI MADEN İŞLETME YÖNTEMLERİNE GÖRE MİKROSİSMİK OLAY GELİŞİMİ, İZLENMESİ VE ANALİZİ

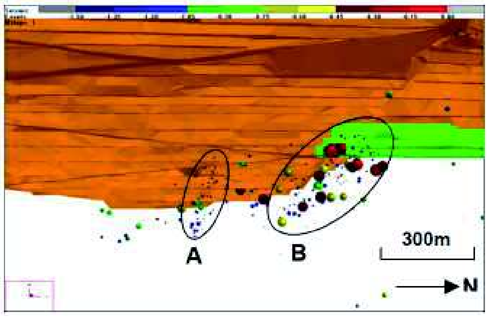
Maden işletme yöntemleri rezerv, saha ve maliyet özelliklerine göre değişiklik gösterse de mikrosismik olayların oluşumu hemen hemen birçok sahada gözlenen ortak bir durumdur. Ancak, maden çıkarma tekniğine göre konum, zaman ve sinyal biçimleri değişkenlik gösterebilmektedir. İzleyen paragraflarda, özellikleri birbirinden farklı maden ocaklarında meydana gelen bu mikrosismik olayların kökeni, izleme ve analiz yöntemleri verilmektedir.

2.1 Maden İşletme Yöntemleri ve Mikrosismik Olay Gelişimi

2.1.1 Açık Ocaklar

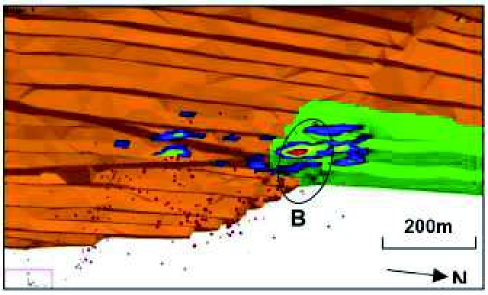
Yüzeye yakın cevherin ekomik olarak çıkarılması için örtü katmanının kaldırılması ile maden işletiminin yapıldığı ocak türleridir. Yeraltı maden ocaklarındaki kadar çok sayıda ve belirgin ölçüde olmasa da açık ocaklarda da mikrosismik olaylar gözlenmektedir. Bunların konumlarının ve büyüklüklerinin belirlenmesi, maden sahasının neresinde ve ne tür bir kaya kütlesi hareketinin olacağının önceden saptanması bakımından önemlidir.

Bu tür ocaklardaki mikrosismik olayların kökeni, daha çok madencilik faaliyetlerden kaynaklanmaktadır. Yüzey örtüsünün kaldırılması ile ayna gerisinde yaratılan gerilme alanındaki değişimler mikrosismik olaylara yol açmaktadır. Sweby vd., (2006), Avustralya'nın batısında bulunan ve açık işletme yöntemi ile yapılan Keith Dağı nikel yatağında yaptıkları çalışma ile ocağın belirli bölgelerinde mikrosismik aktivitenin meydana geldiğini gözlemlemişlerdir (Şek. 1).



Şekil 1. Açık ocak dolayında oluşan mikrosismik olaylar (daire boyutları moment büyüklüğüne göre çizdirilmiştir). A ve B bölgeleri kümelenmeyi göstermektedir (Sweby vd., 2006)

Sweby vd., (2006), maden sahasının belirli bölgelerindeki üç kuyuya her birinde ikişer adet olmak üzere toplam altı adet alıcı yerleştirerek altı ay boyunca izleme yapmışlar ve 270 adet mikrosismik olayın konumlarını 5 m hata ile ve büyüklüklerini de M=-2'ye kadar hesaplayabilmişlerdir. Yaptıkları analiz sonucunda, mikrosismik olayların önemli bir bölümünün örtü katmanın alınması sırasında oluştuğu gözlenmiştir (Şek. 2).



Şekil 2. Son dönem yapılan madencilik faaliyeti (yeşil) ile daha önce yapılan bölgelerle (kahverengi) hesaplanmış yığınsal moment değerleri arasındaki ilişki (Sweby vd., 2006)

2.1.2 Yeraltı Ocakları

Başta Avustralya, Kanada ve Güney Afrika olmak üzere, neredeyse 100 yıldan beri, yeraltı metal madenlerindeki sismik olaylarla ve beraberinde gelen hasarlarla ilgili raporlar tutulmaktadır. Sismik olaylar ve kaya patlamaları ile ilgili ilk rapor, 1930'lu yıllarda 700 m derinliğe sahip Creighton maden sahası için hazırlanmıştır (Trifu ve Suorineni, 2009). Dolayısıyla, yeni bir problem değildir ancak modern madencilik teknolojileri ile daha da derinlerde madencilik yapabilme işinin paralelinde, derinleştikçe artan yerinde arazi gerilmeleri ve maden kaynaklı gerilmeler nedeniyle uyarılmış sismik olaylar da dramatik bir şekilde artmıştır. Bu sebeple, özellikle derin madencilik yapılan yeraltı metal madenlerinde uyarılmış sismik aktivitelerin (mine-induced seismicity) incelenmesi konusu son yıllarda oldukça önem kazanmıştır. Avustralya'da en derin madecilik faaliyetleri 1000-1650 m civarında yapılmaktadır. Kanada'da 1500-2500 m, Güney Afrika'da ise 3000-3800 m'lerde madencilik faaliyetleri yapılmaktadır. (Potvin vd., 2007). Güney Afrika'daki 73 yeraltı metal madeninin %20'sinde, Richter ölçeğine göre M=2 büyüklüğündeki depreme eş değer sismik olaylar gözlendiği ve yine madenlerin %15'inde de en az bir kere büyük hasara sebep veren kaya patlamasının olduğu Hudyma (2005)'nın çalışmasında dikkat çekilmiştir.

Yeraltı madenlerinde gözlenen uyarılmış mikrosismik etkinlik, mevcut jeolojik yapı içerisinden cevherin alınması ile birlikte duraysız hale geçen kaya kütlesinin deformasyona uğraması sonucu oluşmaktadır. Bu deformasyonlar sadece çatlak ve kırık şeklinde olabileceği gibi bunlarla birlikte gelişen kaya patlaması, göçük gibi felaket boyutlarına da çıkabilmektedir. Yeraltı madenciliğinde mikrosismik olayları tetikleyen diğer iki etken ise kullanılan kazı teknikleri ve bölgesel tektonik olaylardır.

Yeraltı madenciliğinde mikrosismik olaylara bağlı olarak gelişen tehlike ve risk oldukça fazladır. Bu amaçla Hudyma ve Potvin (2009), yeraltı metal madenlerinde sismik risk yönetimi için mühendislik yaklaşımı geliştirmişlerdir. Bu yaklaşım, risk önleme ölçülerinin geliştirilmesi için gerekli olan, hasarın anlaşılıp tanımlanması prensibine dayanan, kabul görmüş risk yönetim tekniklerini esas almaktadır. Bu çalışmada araştırmacılar, sismik risk yönetiminin ilk ve en önemli adımının, hasarı anlamak ve tanımlama olduğunu söylemektedirler. Sismik izleme sistemlerinin kurulmasının sismik kaynakları belirlemek, sismik tehlikeyi anlamak için gerekli araçlar olduğunu belirtmektedirler. Birkaç on yıldır Güney Afrika ve Kanada'da uygulanan bu tekniklerin Avustralya'da da hızlı bir şekilde uygulanmaya başladığını, yine çalışmalarında belirtmişlerdir.

Bu çalışmada önerilen sismik risk yönetim filozofisi, mühendislik yaklaşımı olarak tanımlanmıştır. Bu yaklaşım,

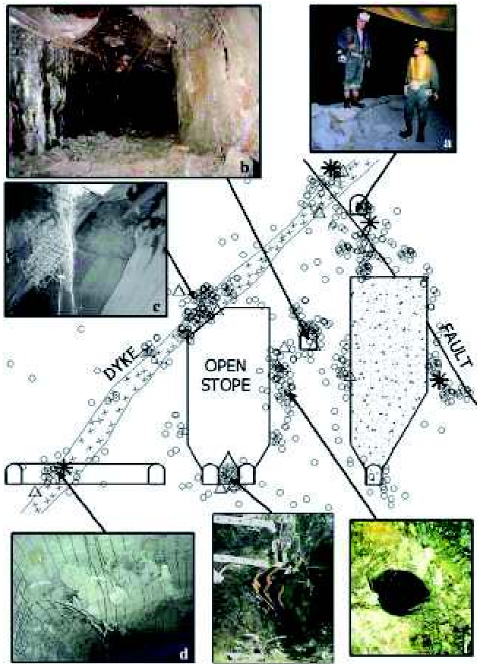
i. madendeki aktif sismik kaynaklarını, her kaynağı oluşturacağı hasar ölçeğinde belirleme (ya da her kaynağın oluşturacağı en büyük hasarı tanımlama),
ii. en büyük sismik olayın yaratacağı en büyük tehlikeyi kestirme,
iii. bu hasarların oluşabilme durumlarında hasarı en aza indirebilecek önlemleri ortaya koyma şeklindedir.

Fay, dayk, kesme bölgeleri ya da jeolojik kontaktlar, kaya kütlelerinde sismisiteyi arttıran jeolojik oluşumlardır.

Sismik kaynaklar, yeterli doğrulukta sismik olay üretebilme potansiyeline bağlı olarak, mikrosismik izleme sistemleri ile belirlenebilirler. Mikrosismik olayların yerleri doğru belirlenebilirse, aktif sismik kaynakların nerelerde kümelendiği de bulunabilir. Şekil 3, gerilme değişimi veya farklı jeolojik koşullar nedeniyle oluşmuş mikrosismik kaynaklar ve yerel kütle hareketlerini göstermektedir.

2.1.3 Çözelti Madenciliği

Özellikle kaya tuzu gibi suda eriyen minerallerin kazanılmasında uygulanan bir yöntemdir. Bu yöntemde yeraltında su ile temasa geçen kaya tuzu yataklarındaki tuz kısmı erir ve anhidrid, kalsit gibi maddelerden ayrılır. Bu eriyik yeryüzüne ya sondaj delikleri yardımıyla ya da yer altı madenciliği ile çıkarılarak tuz elde edilir. Yöntem, trona için de benzeri şekilde uygulanmaktadır. Çözelti madenciliğindeki en büyük sıkıntı, yeraltında oluşturulan erime boşluklarının durumu ile ilgili bilgi belirsizliğidir. Tasman ve kuyuların çökmesi gibi problemlerin oluşmaması için, yeraltında oluşturulan boşlukların durumu takip edilmelidir.



Şekil 3. Gerilme değişimi veya farklı jeolojik koşullar nedeniyle oluşmuş mikrosismik kaynaklar ve yerel kütle hareketleri. Daireler, mikrosismik olayların konumlarını göstermektedir (Hudyma, 2008)

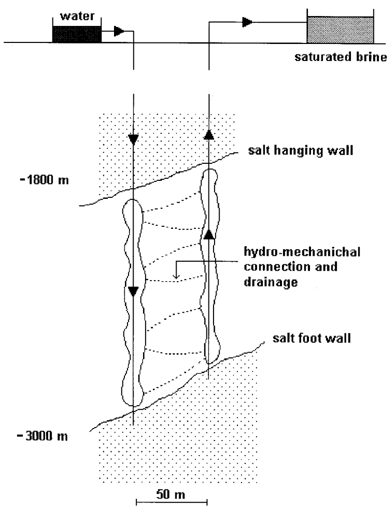
Çözelti madenciliğinde, mikrosismik olaylar genellikle erime boşluklarının sınırlarında meydana gelmektedir. Enjeksiyon kuyularından cevherli zona gönderilen basınçlı sıcak su, bir yandan cevheri eritirken bir yandan da jeolojik yapı içerisinde gerilme alanının değişimine yol açmaktadır. En büyük deformasyonlar, çoğunlukla boşluğun tavanında oluşmaktadır. Bu kesimler gravite çekim kuvvetinin etkisiyle de daha duraysız hale gelerek boşluk içerisine çökmektedir. Çözelti madenciliğinde erime boşlukları dolayında meydana gelen bu tür yapısal deformasyonlar mikrosismik olayların oluşumuna yol açmaktadır.

Mikrosismik izleme yöntemleri, jeoteknik yöntemlerin yanında son yıllarda kullanılmaya başlanmıştır. Mikrosismik izleme yöntemlerinin çözelti madenciliğindeki ilk uygulamaları

1980'lerin sonları, ve 1990'ların başlarında olmakla beraber (Guarascio, 1987, Maisons vd., 1997), son 10 yıl içinde birkaç önemli çalışma ile çözelti madenciliği sırasında yeraltında oluşturulan boşlukların takibi yapılmıştır (Mercerat vd., 2009, Trifu ve Sgumila, 2009).

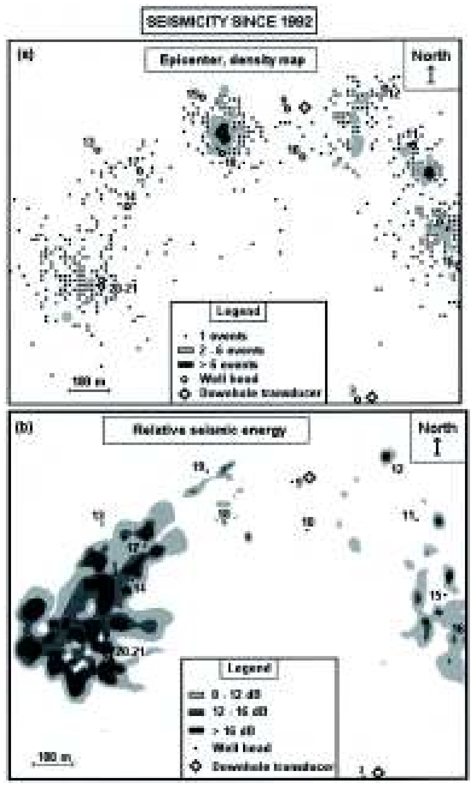
Guarascio (1987), çalışmasında kaya tuzunun eriyik halde yeryüzüne çıkarken oluşturduğu boşluklar ve bunların etrafındaki arazi gerilmelerindeki değişiklikler sonucu meydana gelen kırılma ve kopmaların yarattığı sismik olayları dinlemek için yeryüzünde bir sismik ağ kurmuş; odak noktası konumlarının belirlenmesi için modellemeler yapmıştır. Sismik kaynağın yerinin bulunmasıyla ilgili, o dönemki veri işlem tekniklerine göre iyi sonuçlar elde edilmiştir.

Maisons vd. (1997) ise benzer çalışmayı, kuyu içerisinde mikrosismik izleme yaparak gerçekleştirmişlerdir. Çalışma, 2000 m derinde ve 100 $^{\circ}$C sıcaklıkta da mikrosismik dinleme yapılabildiğini göstermektedir (Şek. 4). Çalışmada bir adet üç eksenli kuyu içi jeofon 1800 m derine yerleştirilmiş, bir adet de yine üç eksenli jeofon yeryüzünden 25 m derine yerleştirilmiştir. Böylece, sismik aktivite sınırları belirlenmeye çalışılmıştır.



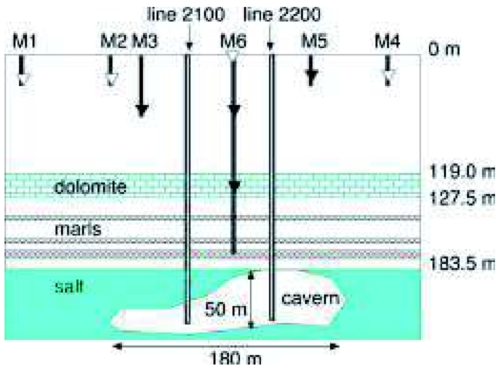
Şekil 4. Tuz eriyiği elde etmenin şematik gösterimi (Maisons ve diğ., 1997)

1992-1994 yılları arasındaki ölçümlerden elde edilen 2223 sismik olay, Şekil 5'de yoğunluk ve enerji haritaları olarak gösterilmiştir.



Şekil 5. a) Sismik yoğunluk haritası b) Sismik enerji haritası (Maisons ve diğ., 1997)

Fransa'da kaya tuzu elde etmek amacıyla yapılan madencilik işlemi sırasında yeraltında boşluk oluşumunu izlemek için Mercerat vd. (2009) de yerel bir sismik ağ kurmuşlardır. Çalışma alanında derinlikleri 30-125 m arasında değişen yerlere dört adet üç bileşenli ve dört adet tek bileşenli jeofonlar yerleştirilmiştir. Bu jeofonların Frekans bantları 28 Hz ile 1.5 kHz arasındadır. Tuz tabakasında açılan ve yerden 180 m derinde yer alan erime boşluklarının düzensiz bir şekli vardır. Yaklaşık 50 m yükseklikte, 180 m çaptadır (Şekil 6).



Şekil 6. Ana litolojik birimler, üç bileşenli jeofon (içi dolu üçgenler), tek bileşenli jeofonlar (içi boş üçgenler) (Mercerat vd., 2009)

2005-2007 yılları arasında 2000'den fazla sismik olay kaydedilmiştir. Ölçümlerin yapıldığı süre boyunca, çözelti ile dolu olan boşluk, duvarlarına 2 MPa'lık basınç uygulamıştır. Sismik ağdan izlenen olayların büyük çoğunluğu bu boşlukların tavanındaki çökmelerden ve tuz tabakasının üstündeki killi marndaki dökülmelerden kaynaklıdır. Daha trajik bir tasman ihtimali görülmemiştir. Bu sebeple, mikrosismik izlem neticesinde, erime boşluklarının üstündeki örtü tabakasının duraylı olduğu sonucuna varılmıştır.

Trifu ve Sgumila (2010)'nın çalışması ise, mikrosismik izlemenin bir başka uygulamasına örnektir. Bu çalışmada konu edilen işletme, Romanya'da 25 yıldan beri eriyik halde kaya tuzu çıkaran bir işletmedir. İşletmecilik hatası nedeniyle (kuyuların acele bir şekilde açılması, hızlı pompalama gibi) 2001 ve 2004 yıllarında, yüzeydeki yerleşim yerlerinde hasara neden olan göçmeler meydana gelmiştir. Bu sebeple, boşluk içerisindeki eriyiğin kontrollü bir şekilde boşaltılarak sterille doldurulması planlanmıştır. Çalışma, 1km^2'lik alanı kapsayan mikrosismik izleme yardımıyla kontrollü göçertmenin nasıl yapıldığına iyi bir örnektir.

2.1.4 Diğer Yeraltı Uygulamaları

Pasif mikrosismik yöntemlerin bir diğer uygulama alanı da tünel, metro, hidroelektrik santral inşaatlarının yapıldığı sahalardır. Bu tür yerlerde de tıpkı maden ocaklarında olduğu gibi yer içerisinden veya yüzeyden malzeme alınmasıyla gerilme farklılıklarının yaratılması söz konusudur. Böylece mikrosismik etkinliğin oluşumuna yol açılmaktadır.

Derek vd. (1995), masif granite açılacak bir tünel etrafında gelişecek deformasyonları önceden anlamak için laboratuvar ölçeğindeki bir model tünelde, 16 adet 3 bileşenli ivmeölçerlerle kazı sırasındaki mikrosismik olayları dinlemişlerdir. Böylece, tünel açılımı sırasında karşılaşılacak ve sismik etkinlik yaratacak olayları önceden tespit edilebilmişlerdir.

Xu vd. (2011) de, hidroelektrik santral inşaası sırasında meydana gelebilecek duraylılık problemlerini önceden kestirebilmek için, çalışma alanına mikrosismik izleme ağı kurmuşlardır. Kullandıkları sismik ağ ve veri-işlem yöntemlerinin başarısıyla, meydana gelen sismik olayların konumlarını 10 m'den az bir hassasiyetle hesaplamışlardır. İki aylık süre içinde büyüklükleri -1.8 ve -0.4 arasında değişen 112 mikrosismik olay kaydetmişlerdir. Mikrosismik olayların dağılımlarının daha çok, örneğin 1081 m derinliğindeki drenaj tünelinde devam eden inşaat çalışmaları ile ilgili olduğunu saptamışlardır.

2.2 Mikrosismik Olayların İzlenmesi ve Analizi

Hangi tür metodla işletilirse işletilsin, hemen hemen bütün maden sahalarında mikrosismik etkinliği gözlemleyebilmek için bölgeye gerçek-zamanlı olarak çalışan bir izleme ağı kurulmalıdır. Bu ağ, genellikle üç bileşenli sensörlerden oluşmalı ve veri aktarımını kablolu ve kablosuz yapabilme özelliğine sahip olmalıdır. Bu sismik istasyonların yüzeye, belirli derinliklerdeki kuyulara ve/veya üretim yapılmayan boş galerilere yerleştrilmesi mümkündür. Böyle bir sismik ağın, maden sahasının alanına ve derinliğe bağlı olarak değişse de en az 10 istasyondan oluşması çözüm kalitesi açısından önemlidir. Hudyma ve Potvin, (2009), bir sahada en az kaç istasyon kullanılması gerektiğini maden sahasının geometrik boyutlarına, derinliğe, istasyon aralığına ve sensörlerin bileşen sayısına göre belirlemiştir. Günümüzde çok farklı ekipmana, tasarıma, veri toplama ve analizine sahip yerel mikrosismik ağlar maden sahalarında çalıştırılmaktadır. Mikrosismik gözlem sürelerinin genellikle, ocaklardaki faaliyetler sonlandırılana dek sürdürülmesi maden sahasının güvenliği ve verimliliği açısından önemlidir.

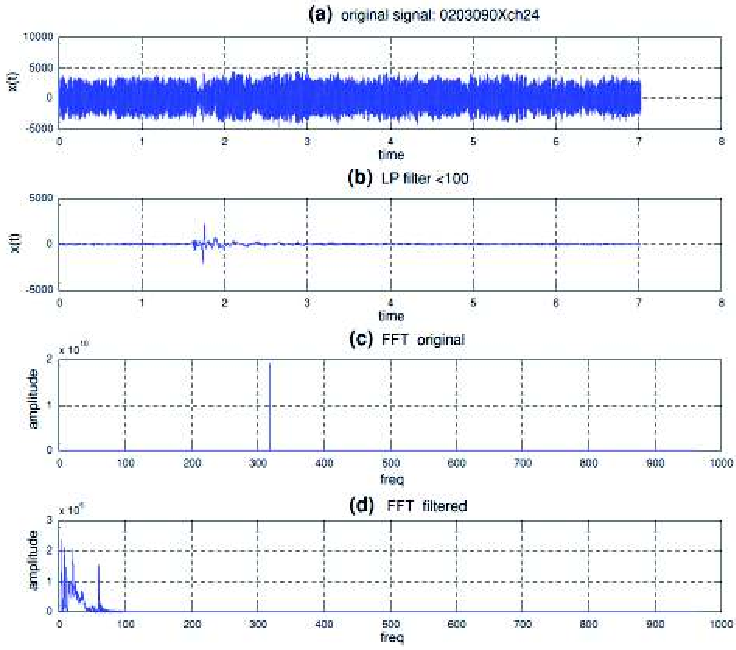
Maden sahası içerisinde gözlenen mikrosismik olayların analizi sonucu, son derece önemli bilgilere ulaşılmaktadır. Örneğin mikrodepremlerin uzamsal ortamdaki dağılımı, konumlarının zamana bağlı olarak değişimi, büyüklük ve sıklık ilişkisi (Gutenberg-Richter ilişkisi), mikrodepremlere ait odak mekanizması çözümlerinden etkin gerilme ve basıç yönlerinin tayini, spektral analiz yöntemleri ile çatlak ve kırık boyutları, maden sahası ve yakın çevresinin 3-B tomografik sismik hız yapısı ve buna bağlı olarak kayaçların litolojik ve mekanik özelliklerinin belirlenmesi vb. gibi bulgular ortaya konulmaktadır. Bu tür bilgilerin anlık gözlemleri ve parametrik değişimlerin tespiti olası risklere karşı önlemlerin alınmasında son derece önemli olmaktadır.

Bu tür çalışmaların ilk aşamasını veri toplama işlemleri oluşturur. Sismik ağdan gelen verilerin belirli bir düzen içerisinde anlık (gerçek zamanlı) olarak kaydedilmesi gerekmektedir. İstasyonlardan anlık olarak gelen veriler çoğunlukla otomatik analiz edilse de bu verinin uzman kişilerce gözden geçirilmesi her zaman önemlidir. Örneğin, Şekil 7a'da bir istasyondan gelen verinin zaman kaydı görülmektedir. Ancak, bu istasyonun bulunduğu ortam koşulları nedeni ile kayıt hayli gürültülü bir durumdadır. İçerisinde bir mikrosismik olay olduğu düşünülen kaydın süzgeçlenerek, istenen sinyalin baskın duruma getirilmesi gerekmektedir. Nitekim Şekil 7b'de kayda belirli bir süzgeç uygulanması sonucu ortaya çıkarılan sinyal biçimi görülmektedir. Şekil 7c ve 7d' de ise orijinal ve süzgeçlenmiş verilere ait frekans içerikleri verilmektedir.

Mikrosismik yöntemlerin ikinci aşamasını ise analiz çalışmaları oluşturmaktadır. Bu analiz yöntemleri deprem sismolojisi kapsamında kullanılan yöntemlerin aynısıdır. Tek farkı incelenen alanın ve sismik olayların küçüklüğüdür. Pasif mikrosismik izleme çalışmaları sonucu toplanan verilere uygulanan analiz türleri şunlardır:

a) Konum belirleme,
b) Büyüklük hesaplama,
c) Sismik sinyal ayrışımı,
d) Odak mekanizması çözümü,
e) Olasılık ve risk hesabı,
f) Gerilme analizi,
g) Kaynak parametrelerini belirleme,
h) S- dalgası ayrışımı,
i) 3-B tomografik görüntüleme

Bu analiz yöntemlerinin hepsinin bir maden sahasında uygulanması her zaman gerekmeyebilir. Maden sahasının işletme türüne, geometrisine, derinliğine, öncelikli problemine yönelik bilgilere göre uygun analiz yöntemleri seçilerek uygulanabilir.



Şekil 7. Bir mikrosismik istasyonun a) gürültülü zaman kaydı, b) süzgeçlenmiş hali, c) orijinal verinin spektrumu, d) süzgeçlenmiş verinin spektrumu

3 PASİF MİKROSİSMİK İZLEME YÖNTEMLERİNİN MADEN SAHALARI İÇİN ÖNEMİ

Yapılan birçok araştırmaya göre, gerek açık ocak gerekse yeraltı maden sahalarında karşılaşılan kaya yenilmesi olaylarının önemli bir bölümü, sahanın sismik etkinliği ile ilişkilendirilmektedir. Bir maden sahası ve yakın çevresinde gözlenen sismik etkinlik neden-sonuç ilişkisi açısından değerlendirildiğinde, kimi zaman bu olaylar bir sonuç iken kimi zaman ise bir neden olabilmektedir. Bunu daha iyi açıklayabilmek için bir maden sahasında elastik dalga yayılımına neden olan kaynaklara bakmak gerekmektedir. Bu kaynaklar,

1) Mikrosismik olaylar
2) Yerel tektonik depremler
3) Patlatmalar

şeklinde sıralanabilir. Mikrosismik olaylar, ortamın gerilme alanındaki değişimlerin veya kaya kütlesi hareketlerinin bir sonucu iken, yerel tektonik depremler ve patlatmalar ise gerilme farklılıklarına veya kütle hareketlerine yol açan bir neden olabilmektedir. Bir maden sahasında meydana gelen kaya yenilmesi hareketlerinin ne tür bir sismik kaynaktan ileri geldiğinin bilinmesi ve hatta öncede tespit edilebilmesi önemlidir. Kaynağı ne olursa olsun bir sismik etkinliğe bağlı olarak gelişen bu tür olaylar maden ocaklarında çalışanlar ve işletmeler için büyük risk oluşturmaktadır.

Risk yönetimi için ise tehlikelerin nerelerde ve nasıl geliştiğinin mutlaka tespit edilmesi gerekmektedir. Potansiyel tehlike bölgeleri, çoğunlukla mikrosismik olayların meydana geldiği lokasyonlardır. Sismik risk çalışmaları için bu lokasyonların olabildiğince doğru bir şekilde belirlenmesi gerekmektedir.

Maden sahalarındaki faaliyetlere bağlı olarak oluşan mikrosismik etkinliğin uzamsal ve zamansal olarak izlenmesi son derece önemlidir. Bu yöntem, öncelikli olarak kaya kütlesinin yerel durumu ve gerilme koşulları hakkında bilgi verir. Ayrıca, mikrosismik olayların yeraltında hangi derinliklerde ve ne boyutta oluştuğunun zamana bağlı olarak izlenmesi, tehlike kestirimlerinin yapılmasında önemli rol oynamaktadır. Böylece oluşacak tehlikelere karşı gerekli önlemlerin daha önceden alınmasını sağlayacaktır.

Maden sahalarında pasif mikrosismik izleme yöntemlerinin, farklı izleme yöntemleri ile birlikte kullanımı, maden sahasının güvenliği, personel ve ekonomik kayıpların önlenmesi açısından gerekli bir uygulamadır. Bu tür dinleme yöntemleri ile

a) Sismik olayların türleri, birbirinden ayırt edilerek neden-sonuç ilişkisi ortaya konulabilir,
b) Maden kaynaklı uyarılmış mikrosismik olayların nerede, ne zaman ve ne büyüklükte olduğu saptanabilir,
c) Kaya içerisinde meydana gelen kırık ve çatlak boyutları belirlenebilir,
d) Maden sahası ile ilgili sayısal modelleme çalışmaları yapılabilir,
e) Maden sahasındaki hangi jeolojik birimlerin sismik açıdan daha aktif olduğu konusunda bilgi verilebilir,
f) Sismik tehlike analizleri ile gelecekte olabilecek kütle hareketlerinin kestirimi ve risk analizleri yapılabilir,
g) Maden sahası içerisindeki gerilme alanları ve gerilme boşalımının meydana geldiği bölgeler tespit edilebilir,
h) Mikrosismik olayların yoğunlaştığı bölgelerdeki etkin gerilme bileşenlerinin (basınç ve çekme eksenleri) yönü ve doğrultuları saptanabilir,
i) Kayaç içerisindeki çatlak ve kırıkların türleri (kesme veya açılma) belirlenebilir.

Tüm bu çıktılar, bir maden sahası için yapılan mikrosismik izleme yöntemlerinden elde edilebilen ve maden sahasının güvenliği ve planlaması açısından hayati önem taşıyan önemli bilgilerdir.

4 TARTIŞMA

Pasif mikrosismik izleme yöntemleri yer ve malzeme bilimlerinde kullanımı giderek yaygılaşan önemli bir araç olmuştur. Petrol ve doğal gaz sahalarından baraj bölgelerine, jeotermal alanlandan, uçak gövdelerine ve hatta en küçük yapı malzemesine kadar birçok alana uygulanabilmektedir. Maden sahalarında da yaygın bir şekilde kullanılan pasif mikrosismik izleme yöntemleri ile bugüne kadar birçok başarılı sonuçlar elde edilmiştir ve hala da devam edilmektedir.

Maden sahalarının öncelikle güvenliği daha sonra ise verimliliği amacı ile bu tür izleme yöntemlerine başvurulmaktadır. Ekonomik getiriden çok ekonomik kayıpların önüne geçilmesi için tercih edilmektedir. Yerüstü ve yeraltı madencilik faaliyetlerinden dolayı yerkürenin içsel tepkisini mikrosismik olaylarla göstermesi ve bu tepkinin uygun sistemlerle izlenmesi, maden sahalarındaki tehlike ve risklerin belirlenmesi amacı ile uygulanan en iyi yollardan biri olmuştur. Nedeni ise konumu yer içerisinde olan kaynaklardan zaman boyutu içerisinde sürekli bilgi akışının sağlanmakta olmasıdır. Bu nedenledir ki pasif mikrosismik izleme yöntemlerinin etkili bir şekilde kullanılmasından beri maden sahalarında sıklıkla karşılaşılan kaya yenilmesi olaylarının zaman ve uzay boyutunda nasıl geliştiği konusunda değerli bilgiler elde edilmiştir.

Bu çalışmada, madencilik konusunda ileri bir konumda olan ülkelerde uzun bir zamandır beri kullanılmakta olan pasif mikrosismik izleme yöntemler hakkında genel bir bilgi verilmiş ve bu yöntemlerin madencilik sektörü açısından önemi vurgulanmıştır. Türkiye'de de önemli bir yere sahip olan madencilik sektörü henüz bu tür izleme yöntemleri ile tanışmamıştır. Bu nedenledir ki Pasif mikrosismik izleme yöntemlerinin maden sahaları için öneminin ve gerekliliğinin vurgulanmaya çalıştığı bu çalışma ile Türkiye madenciliğinin de dikkati çekilmeye çalışılmıştır.

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Grade/Reserve Characterisation of Feldspar Deposits: A Practical Multivariate Geostatistical Approach

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ABSTRACT This paper aims at predicting a primary variable in presence of a secondary variable on the basis of multivariate case. Main idea of the paper is to implement collocated co-kriging technique with a primary and auxiliary variable as a case study. Mostly drilling logs consist of multiple variable records and may be identical to each other. This technique enables multiple variable analyses especially where the secondary data is sparsely distributed. A case study was carried out on determining grade and reserve characteristics of a feldspar deposit in Sarpdere Region (Çine-Aydın-TURKEY). Collocated co-kriging estimation technique was applied to two main feldspar compounds (Na_2O and SiO_2). Individual experimental variograms and cross variogram of the compounds have been modelled and spatial structure compatibility of the models on the study field were tested by cross validation procedure. Also, grade/tonnage curve graphs and tabulated tonnage distributions at sequential cut-off interval have been calculated.

1 INTRODUCTION

Natural resources are main locomotives of industries and directly related to welfare and development of a country. Thus, an accurate and reliable evaluation of a natural resource is extremely important in terms of mining production planning or maintaining of resources. There is a strong balance between recoverable reserve and investments in mining. Especially, erroneous judgements of reserve characteristics may have severe results economically. Grade and reserve are two main parameters of an ore deposit. Thus, knowledge of these parameters provides a vital information for decision makers for a more successful mine planning.

Geostatistics basically build on theory of regionalised variables (Matheron, 1971), and has been advanced in recent years especially parallel to faster computers. New techniques were introduced to geostatistics family. Whole calculations have been made to get more reliable, precise and realistic results to approach the natural occurrence structure of interest by minimising estimation or simulation errors (Ersoy and Yünsel, 2004-2006; Diko et al., 2001).

Geostatistical ore reserve modelling is becoming increasingly common in recent years while traditional methods have been lost their popularity. Geostatistical methods have certain advantages over traditional techniques such as determining and minimising estimation variance etc.

Co-kriging is a traditional geostatistical technique for integrating several variables at a target point consists of a linear combination of all variables available at the neighbouring points. This method is obviously demanding than the kriging algorithm as it requires a consistent multivariate model (Webster and Olivier, 2007; Goovaerts, 1997).

The basic theory of co-kriging to implement into geostatistical analyses has been well developed in several studies (McBratney and Webster, 1983; Vauclin et al., 1983; Davis, 1986, Kitadinis, 1999, Wu and Murray, 2005). Geostatistical grade/reserve estimation techniques have absolute advantages over traditional methods such as minimising estimation variance, probabilistic analysis, as taking into account the complex nature of geology (faults, trend etc.), and local distribution of interest. This study covers a geostatistical multivariate data analysis to determine the grade parameters distribution of a feldspar deposit by collocated co-kriging estimation procedure since all secondary variables are available on same set of sample points.

2 THE STUDY AREA

The feldspar deposit is located in Sarpdere area, 20 km west of the Karpuzlu village in Çine (Aydın, TURKEY). The location map of the study area is given in Figure 1. The area is formed on Menderes Massif, where metamorphic rocks are exposed. The feldspar deposit occurs in augen gnesis, quartzite, garnet-mica schists and meta-granite rocks. These formations are related to Precambrian age.

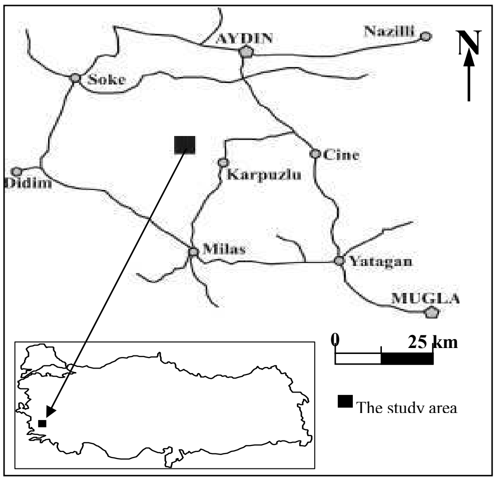

Figure 1. Location map of the study area

The main mineralization of the deposit is albite (Na-feldspar) which is produced. The albite is commonly associated with gangue minerals such as mica (muscovite and biotite) which affects the quality of the production (purity). Thus, the existence of such minerals in the area takes a significant role to determine their distribution. The open pit mining method has been employed for the production which is carried out through a valley due to the nature of the deposit.

Main production scheme is given in Figure 2. The produced material is transported to the bulk location in the plant. Then, the material is crushed and grinded for granulometric distribution. The grinded material is classified depending on granulometric distribution and its chemical content. The classified material is sold without mineral processing treatment. The feldspar is mainly used in glass and ceramic industries.

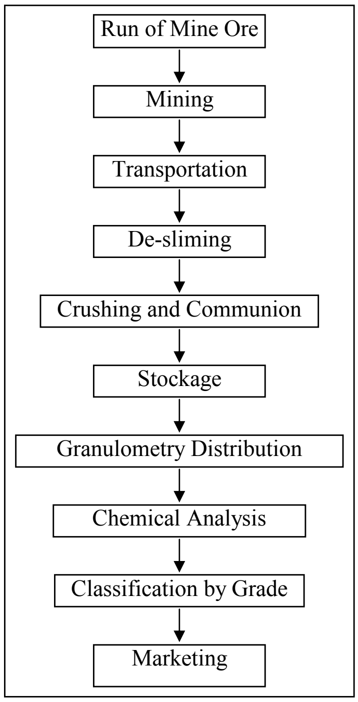

Figure 2. Production flowsheet of feldspar

The data consists of core samples obtained from 31 drill holes. The data locations and study polygon area are shown in Figure 3. The exploration boreholes intersected with the feldspar were logged. These logs include the thickness of the intersection and chemical contents of the feldspar (the grade) and coordinates of the drill holes. Chemical analysis was carried out in order to determine the concentrations of the following compounds: Na_2O, Al_2O_3, SiO_2, CaO, MgO, K_2O, Fe_2O_3 and TiO_2. The feldspar quality depends on its content in terms of these compounds. Thus, as quality variables, the deposit content in terms of the compounds Na_2O, SiO_2, K_2O, and TiO_2 vary greatly depending on the part of the deposit where the sample is obtained. These compounds determine the feldspar quality. The CaO, MgO, Fe_2O_3 and Al_2O_3 values do not vary greatly in the deposit and remain within the admissible market values for applications in the glass and ceramics industries.

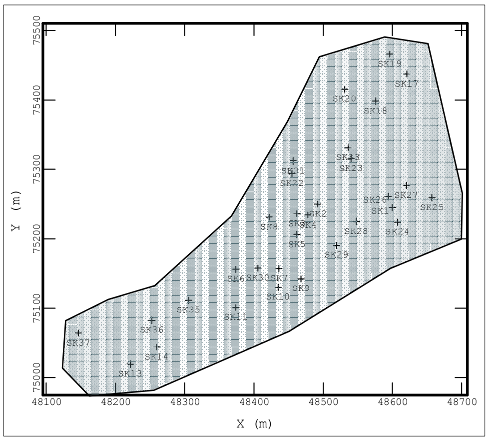

Figure 3. Location map of the drill holes and polygon area in the study field

The quality classification of the feldspar according to the chemical concentrations of the compounds is presented in Table 1.

Table 1. Quality classification of the feldspar products

Product Code	Compound Content (%)			
	Na_2O	SiO_2	K_2O	TiO_2
Ultrawhite–75	10.71	69.08	0.30	0.10(max)
Superwhite–75	10.90	68.78	0.13	0.15(max)
Extra–CG75	10.60	69.63	0.14	0.25(max)
Std–CG 75	9.65	70.00	0.17	0.35(max)
Extra – GQ	11.04	67.41	0.14	0.30
Std – GQ	10.87	67.98	0.14	0.28

3 METHOD

Co-kriging estimation technique takes into account a secondary or auxiliary variable to estimate a primary variable at a target point. Sampling point locations of the primary and secondary variable have a great importance in terms of co-kriging process to be applied. Variables in a given domain may be located either at the same points or at different points for each variable.

Data situations may be classified according to their locations (Wackernagel, 1995):

1. *Complete Heterotopy:* The variables have been measured on different sets of sample points and have no sample locations in common;
2. *Partial Heterotopy:* Some variables share same sample locations;
3. *Isotopy:* Data is available for each variable at all sampling point.

The working data set is classified as isotopic, since variables are available at all sampling points and in common. The idea of collocated co-kriging technique is to enhance the co-kriging process by adding, for each target grid node, the value of the secondary variable at this location. The system resembles the traditional co-kriging technique where one additional sample is added which coincides with the target grid node and for which both value is provided. This is therefore an isotopic case, as both variables are informed at the target grid node (Geovariances, 2006).

Collocated co-kriging is followed by derivation of variograms of variables and cross-variograms between primary and auxiliary data. In practice, the individual variograms are typically estimated by the following equation:

$$\gamma(h) = \frac{1}{2N_{(h)}} \sum_{i=1}^{N_{(h)}} \left\{ z_{(x_i)} - z_{(x_i+h)} \right\}^2 \qquad (1)$$

Where;

z_{x_i} : Sampling value at sampling point x_i

$N_{(h)}$: Number of sampling point pairs at a lag distance of h.

The cross-variogram may be estimated on the base of Eq.2 as follows.

$$\gamma_{uv(h)} = \frac{1}{2N_{(h)}} \sum_{i=1}^{N_{(h)}} \left\{ z_{u(x_i)} - z_{u(x_i+h)} \right\}\left\{ zv_{(x_i)} - z_{v(x_i+h)} \right\} \qquad (2)$$

Here u and v are the primary and auxiliary variables respectively.

As for collocated co-kriging, as mentioned before, it is the extension of kriging to more than one variable. Collocated co-kriging is a reduced form of full co-kriging (Xu et al., 1992).

Let Z (Primary variable)= 1 and Y= (Auxiliary variable)= 2. Then, collocated co-kriging with Markov-type approximation of attribute Z at location x is given by:

$$Z^*_{CK}(u) = \sum_{i=1}^{n(0)} \lambda_i^{CK}(u) Z(u_i) + \lambda^{CK}(u)\left[y(u) - m_y + m_z \right] (3)$$

Where:

$Z^*_{CK}(u)$ = Co-kriging estimator of Z(u),

$\lambda_i(u)$ = Co-kriging weight associated to neighbouring datum Z(u) for estimation at location u,

$\lambda(u)$ = Co-kriging weight associated to collocated secondary datum Y(u),

m_z = Mean of the primary variable,

m_y = Mean of the secondary variable.

4 EXPLORATORY DATA ANALYSIS

There were 31 drilling well locations on study field and no regular grid design was followed in sampling scheme. Two main feldspar variables were selected for analyses (Na_2O and SiO_2).

Regularisation is a common task for a geostatistician to get composites of the same length from an irregular line sampling. The algorithm of the regularisation process is just a weighted average of the original property. The categorical rational distribution of the core lengths of samples is given in Table 2. This table indicates that measured core sample lengths are mainly changing around 4-6 m. It is expected in a regularisation process that the mean to remain unchanged, but the variance to decrease at the end. Thus, 4m compositing length gave the best statistical results and as a result of regularisation, a total of 219 data was produced. The scatter plot of the Na_2O and SiO_2 is given in Figure 4. It can be seen in figure that two variables have a high correlation coefficient although some slight deviations between high SiO_2 and low Na_2O values, and the Figure validates the bond between the variables.

Table 2. The categorical rational distribution of the core lengths

Core length (m)	Frequency	Percentage (%)
1	8	5.16
2	12	7.74
3	19	12.26
4	33	21.29
5	23	14.84
6	33	21.29
7	10	6.45
8	5	3.23
9	3	1.94
10	2	1.29
11	1	0.65
13	1	0.65
14	1	0.65
15	1	0.65
17	1	0.65
19	1	0.65
21	1	0.65

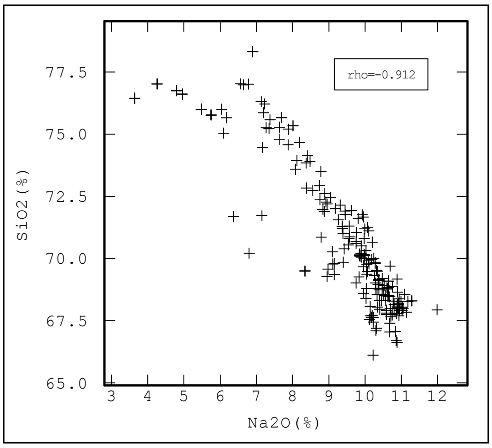

Figure 4. Scatter plot of variables

The summary statistics of regularised variables is given in Table 3 and histograms are given in Figure 5. Na_2O exhibits a slightly left tailed distribution and SiO_2 shows reasonably close to Gaussian distribution. As a result, the data provide that geostatistical analyses can be proceed over composited data without any transformation process (e.g. Gaussian and lognormal etc.).

Table 3. Summary statistics after the composition of the raw data

VARIABLE	**Na_2O (%)**	**SiO_2 (%)**
Number of samples	219	219
Minimum	3.64	66.12
Maximum	11.99	78.32
Mean	9.15	71.01
Median	9.96	70.01
Std. Deviation	1.89	3.14
Variance	3.59	9.83
Coefficient of Variation	0.21	0.04
Skewness	-1.18	0.71
Kurtosis	3.39	2.17

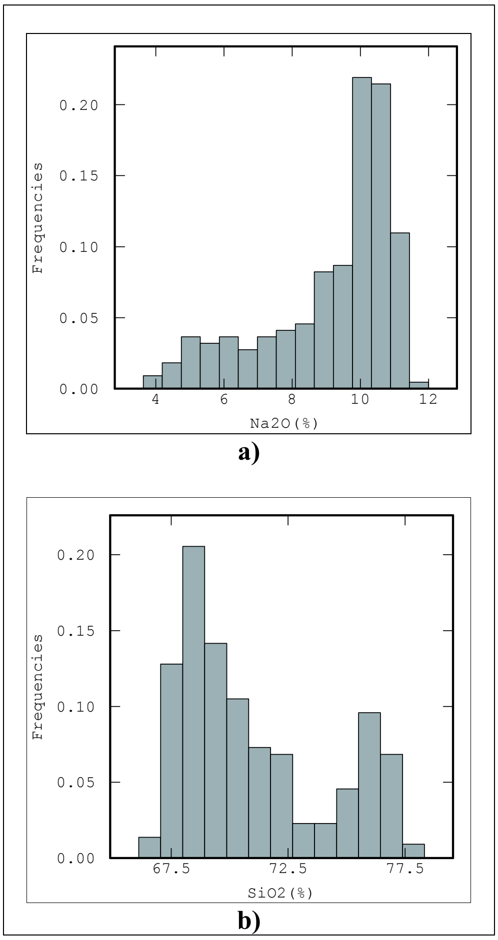

Figure 5. Histograms of the compounds: a) Na_2O, b) SiO_2

5 SPATIAL STRUCTURE ANALYSIS

Co-kriging technique requires variogram analyses of individually and cross type of variables. Directional variogram analyses (for the directions 0^o, 45^o, 90^o, 135^o) were performed separately to reveal the anisotropy axes if any. However, both variables exhibited neither severe geometric nor strong zonal anisotropy exist in the data. Thus, the directional variogram graphs were not given here.

The horizontal and vertical variogram graphs and related variogram parameters were presented in Figure 6 and Table 4 respectively. The variogram graphs mostly show a clear structure in both horizontal and vertical plane reflecting the spatial structure of variables. Numbers of pairs on the variogram graphs are satisfactory for an adequate sampling. The variogram graphs imply that theoretical Spherical and Gaussian variogram models are in agreement with experimental variograms of the variables in horizontal and vertical plane respectively. Also, the sampling design is consistent since a clear spatial structure is distinguished at such distances between the samples.

The maximum distance between two variables in the study area is about 780 m. Thus, approximately a 400 m distance was favourable to identify the spatial structure in horizontal plane. Similarly, maximum vertical sample length is about 20 m and 12-14 m is an optimal distance for variographic analyses in vertical plane.

6 CROSS VALIDATION

The cross validation test is used to check the consistency between a set of data and related their spatial structural model. This procedure works as follows:

A sample point is estimated from the neighbouring data and the theoretical variogram model computed from the whole data set by kriging at each sampling point in turn after excluding the sample value there.

There are important diagnostic statistics standing from the results. Mean and variance of standardised errors are performance indicators of a variogram model and its neighbouring parameters for estimation. Mean and variance of standardised errors are expected to be 0 and 1 respectively to satisfy the unbiassedness of kriging. Cross validation test results are given in Figure 7 and 8 and tabulated results are given in Table 5. Each model reasonably satisfies the global unbiassedness condition, where distributions of errors are centred on a zero mean.

Table 5. Cross validation results of the experimental variogram models

Variable	**Standard Error**	
	Mean	**Variance**
Na_2O (%)	-0.03162	0.97879
SiO_2 (%)	0.01226	1.08325

A sample was considered as an outlier as soon as its standardised estimation error (SEE) exceeds a given threshold (such as confidence interval) in absolute value. Outliers were indicated as bold points on base map (Figure 7a-8a), scatter and error distribution maps. It can be seen on the Figures that outliers of the compounds did not follow a particular pattern. Scatter plots of observed data versus estimated value (Figure 7b-8b) with conditional expectation line (45^o diagonal line) on the Figure show a reasonably good distribution around diagonal line. As the subsequent step; the collocated co-kriging procedure is implemented over study area with validated spatial structure model. Consequently, the cross validation graphs of estimation value versus SEE prove that the controversy in estimation is within the acceptable limits, excluding the outliers (bold dots) (Figure 7d-8d). Histograms of SEE for all the variables (Figure 7c-8c) were in good agreement with the findings of cross validation analyses mentioned above. In theory, it is expected that the mean of the estimation and standard deviation or variance equal to 0 and 1 respectively. The cross validation results showed that the estimation is reasonably acceptable.

7 MAPPING

The grid design parameters which overall reserve/grade estimations take place were given in Table 6. Also, a polygon area was selected as taking into account the sampling locations and range of variogram models. The following steps were followed at final phase:

1. Estimation was done for whole grid cells,
2. Topography was subtracted from the estimation grid system.
3. Estimation was limited to polygon area.

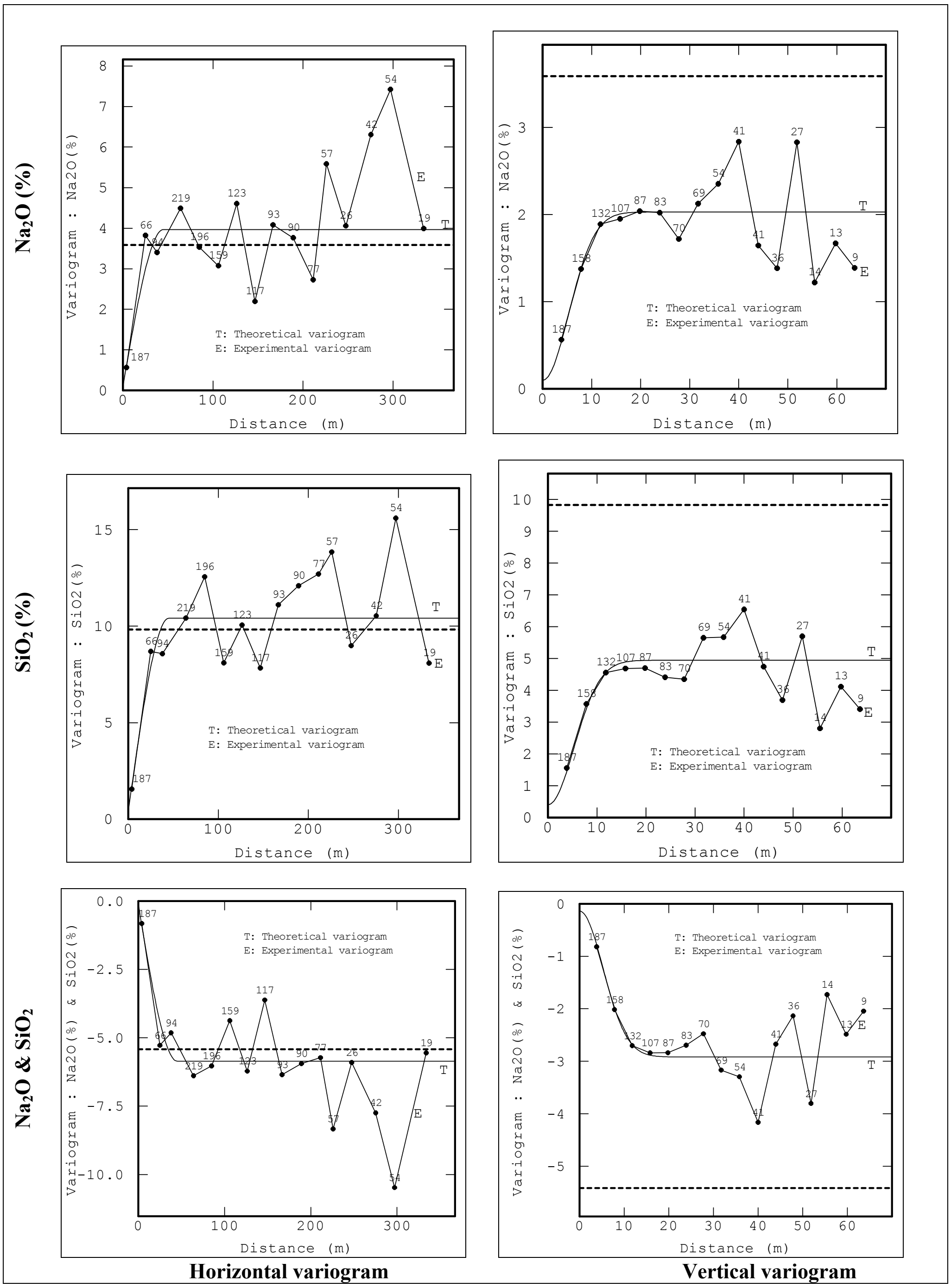

Figure 6. Individual variograms and cross-variogram of compounds in horizontal and vertical plane

Table 4. Variogram parameters of the compounds

Variogram parameters	Na_2O		SiO_2		Na_2O & SiO_2	
Plane	**Horizontal**	**Vertical**	**Horizontal**	**Vertical**	**Horizontal**	**Vertical**
Model	**Spherical**	**Gaussian**	**Spherical**	**Gaussian**	**Spherical**	**Gaussian**
Lag distance (m)	21	4	21	4	21	4
Number of lags	17	17	17	17	17	17
Range (m)	45	13	45	13	45	13
Sill (C_1)	3.87	1.93	10	4.54	-5.72	-2.78
Nugget effect (C_0)	0.097		0.4		-0.14	
Optimum points to use			12			

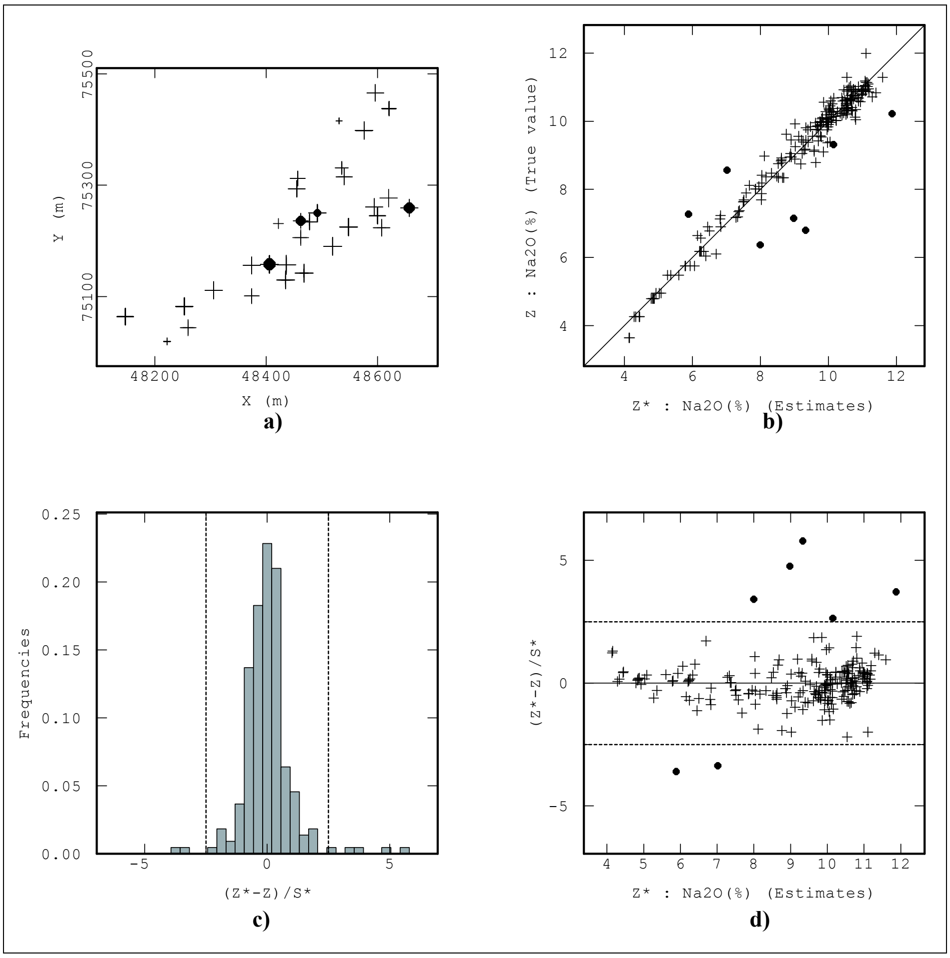

Figure 7. Cross validation results of Na_2O: a) Basemap of errors, b) Scatter diagram of observed data (Z) vs. estimated value (EV, Z*), c) Histogram of the standardised estimation errors (SSE), d) Scatter diagram of SEE vs. Z* (EV).

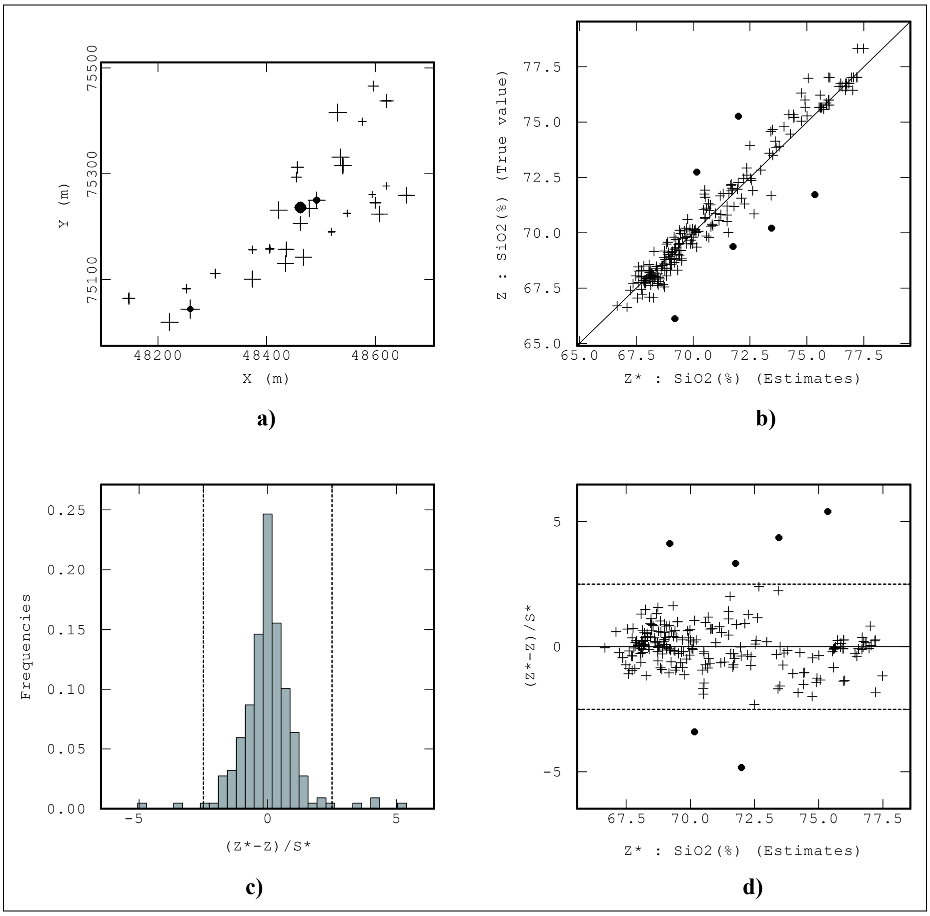

Figure 8. Cross validation results of SiO_2: a) Basemap of errors, b) Scatter diagram of observed data (Z) vs. estimated value (EV, Z*), c) Histogram of the standardised estimation errors (SSE), d) Scatter diagram of SEE vs. Z* (EV)

Table 6. Grid design parameters of the study area for estimation

	X (m)	Y (m)	Z (m)
Origin	48100	75000	340
Mesh	8	8	4
Nodes Number	77	66	46

Mapping is an important tool to reveal and examine visually the grade distribution of compounds through the study area. This part of study can be used for a short and long term of mining production planning, mixing of compounds etc. The contour maps of grade distribution of the variables are shown in Figure 9.

Contour maps on the study field were produced at 400m level. Some parts of area were disappeared due to slope of topography. High Na_2O grade distributions generally NE and SE part of study area, on the contrary, high SiO_2 grade distribution generally takes place on the west side of the area. A relatively inverse distribution characteristic is an expected result of negative correlation between compounds. The contour maps may be generated at different levels to enhance the knowledge of the deposit.

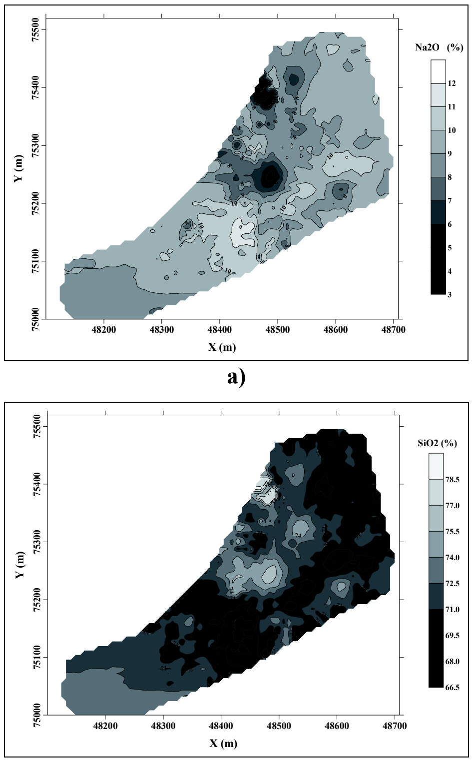

Figure 9. Contour maps of grade distribution of compounds at the 400 m level of study area: a) Na_2O, b) SiO_2

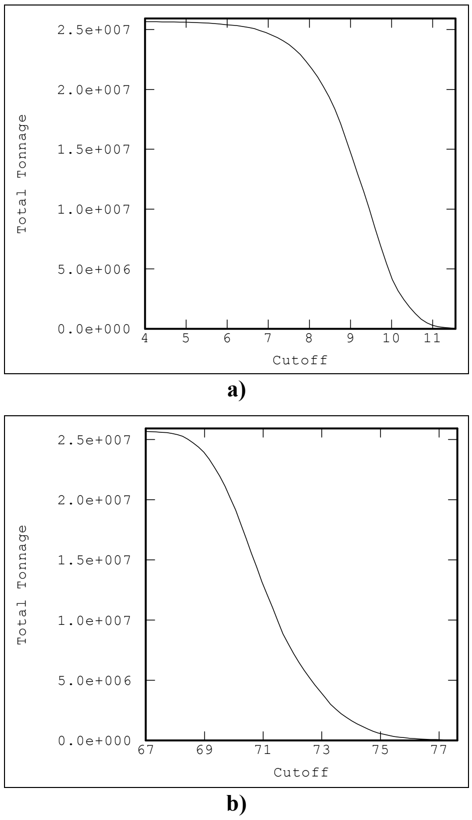

Figure 10. Grade-tonnage distribution of variables: a) Na_2O, b) SiO_2

8 GRADE/TONNAGE DISTRIBUTION

In the previous section of the paper, the distribution of the feldspar compounds visually depicted as a result of collocated co-kriging estimation. This part of the study consists of calculating the numerical grade distribution and its related total reserve on study field.

Figure 10 and Table 7-8 shows the total tonnage distribution of the variables at different sequential cut-off grade intervals as graphically and tabulated type respectively.

Total tonnage of Na_2O grade at the most grade intervals (%3-8) did not changed strongly. There is a sharp decrease in tonnage of greater Na_2O grades than %8 cut-off value. This result indicates that low Na_2O values have a short-scale distribution area and high Na_2O values have long-range spreading area. This is clearly distinguishable in Figure 10a.

SiO_2 values were generally distributed more uniformly next to Na_2O grades in the study area. There are no sharp changes in total tonnages cut-off intervals. Similarly Figure 9b validates the distribution behaviour presented in Table 8.

Table 7. Tonnages with different cut off grade intervals for the Na_20

Cut Off Grade (%)	Total Tonnage Above (ton)	Mean Grade Above (%)
3	25692160	9.07
4	25663539	9.07
5	25610957	9.08
6	25407283	9.11
7	24659149	9.19
8	22014054	9.38
9	14727066	9.77
10	4287795	10.43
11	292198	11.31
12	17306	12.58

Table 8. Tonnages with different cut off grade intervals for the SiO_2

Cut Off Grade (%)	Total Tonnage Above (ton)	Mean Grade Above (%)
67	25670195	71.22
68	25454541	71.25
69	23886387	71.43
70	19502080	71.85
71	12954573	72.53
72	7406131	73.32
73	3887104	74.09
74	1674650	74.96
75	585728	75.94
76	193690	77.02
77	75213	78.04

9 CONCLUSIONS

In this study, collocated co-kriging technique was applied using 31 well logs. After a data processing by both conventional statistics and geostatistical methods, the ultimate grade/reserve distributions were obtained both visually and numerically through the area.

The borehole logs at different sampling length were regularised by 4m benches. A total of 219 data for each variable were used to characterise the study area.

Basic findings of study may be concluded as follows:

1. Sampling distances are identical to reveal the spatial structure through the area.
2. Collocated co-kriging technique is applied since both variables share same locations in the sampling set.
3. Experimental variograms constructed and theoretical variograms were fitted onto them.
4. No anisotropy was detected in both horizontal and vertical plane during directional variogram analyses in the study field.
5. Visual and numerical variable grade distribution results of variables were presented as graphically and tabulated.
6. Feldspar tonnage distribution at various grade intervals has been calculated.
7. The study gives an idea about the distribution behaviour of the reserve grades and reveals the main grade/ tonnage relations through study area.
8. The study may be useful for practitioners who have primary and secondary variable in a study field. This method can be applied to either two-dimensional (environmental and agricultural) or three-dimensional (mining and petroleum) data.

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The Impact of Interaction between Mine Facility Location Selection Criteria on Final Ranking of Site Alternatives

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ABSTRACT Mine Facility Location Selection (MFLS) is one of the most commonly encountered problems in open pit mine planning and design in line with sustainable development. This is a critical decision which must be simultaneously considered a number of criteria such as economical parameters, environmental aspect, stability condition, technical factors, social-economic and facility characteristics to find the best location among feasible alternatives. Likewise, according to the sophisticated structure of the problem, imprecise data, lack of sufficient information, and inherent uncertainty, the usage of the fuzzy sets can be useful to solve this problem. The aim of this study is to propose a new hybrid Fuzzy Multi Attribute Decision Making (FMADM) model considering interaction between mine facility location selection effective criteria.

In this paper Fuzzy Analytical Network Process (FANP) has been used to calculate the interaction between attributes and combination of Fuzzy Analytical Hierarchy Process (FAHP), FANP and entropy applied for obtaining the overall precise weight of attributes. Proposed model analysis showed that considering the interdependency of criteria changes the final weight of attributes. The proposed model has been applied for processing plant location selection of Sangan open pit mine of Iran.

1 INTRODUCTION

The purpose of mining is to meet the demands of metals and industrial minerals of human to develop infrastructure and improve the quality of life of the population. The extracted substances are in many cases the raw materials for the manufacture of many goods and materials (Kumral et al. 2008). The increasing demand and ascending price of minerals in many cases make it possible to process lower grade ores, which means more production of material and more environmental disturbance. Nowadays it is believed that the public expect the mining industry to care the environmental issues and try to eliminate the adverse environmental impacts or at least minimize the intensity as well as the extent of them. Sustainable development requirements finally lead to using improved and environmentally friendly technologies. Using sustainable development principles must be started at the beginning of the project by selecting suitable locations for mine facility installation. To put mining operation in line with sustainable development throughout its life and also after mine closure especial arrangements must be made (Naraei et al. 2011).

Mine planning and design are very complex engineering subjects and require engineering knowledge and good understanding of many issues. One of the most important issues is decision making about MFLS. The goal of MFLS is to find the best location that should comply with sustainable development principles so as to ensure sustainable development of mine and unify economic, social and the environmental efficiency. MFLS is a very important decision for mining companies because it is costly and difficult to reverse, and it entails a long term commitment so that, a poor choice of location might result in excessive transportation costs, adverse environmental impacts, or some similar conditions that would be detrimental to mining activities (Stevenson 1993).

In the past, MFLS was a simple procedure on the basis of economic criteria and ease of operation. The process was to estimate the costs for each alternative, and the lowest cost alternative would ordinarily be the hands down winner (Caldewell et al. 1983, Robertson 1982 & Magda 1985), but effect of multiple criteria on MFLS makes it complex as the conventional procedures therein would result in incorrect results. Thus, the MFLS can be viewed as a multi attribute decision making (MADM) problem that helps decision makers select the most preferable decision and provide the basis of a decision support system. In MADM problem, a decision maker has to choose the best alternative that satisfies the evaluation criteria among a set of candidate solutions. In classical MADM methods ratings and the weights of the criteria are known precisely, whereas in the real world, in an imprecise and uncertain environment, it is an unrealistic assumption that the knowledge and representation of a decision maker or expert are so precise.

The fuzzy set theory could resemble human reasoning in use of approximate information and uncertainty to generate decisions. It was specifically designed to mathematically represent uncertainty and vagueness and provide formalized tools for dealing with the imprecision intrinsic to many problems (Zadeh 1965).

There is no well defined process to consider the weight of MFLS criteria on final ranking of site alternatives, because not only various options should be considered as potential locations, but also there are a large number of effective which are in conflict with each other.

The articles listed in Table 1 address some different type of facility location selection models in the context of MADM in the recent years and weighting methods containing their advantages and disadvantages. Based on Table 1, there are many shortcomings to these models that have been used for facility location selection, among them limitation in variety of attributes such as weakness in usage of linguistic and fuzzy attributes, ignoring the interdependency of attributes and ignoring entropy in decision matrix are the most important shortcomings that have been considered in this study.

The main objective of this paper is to present a powerful fuzzy MADM tool for making an appropriate decision in complex problems featuring uncertainty and contradictory goals. To make this study more sensible and gain a more representative description of MADM process, we would apply a hybrid model of FAHP, FANP and entropy to weight MFLS criteria.

FAHP is an application of the combination of Analytic Hierarchy Process and fuzzy set that the linguistic scale of traditional AHP method could express the fuzzy uncertainty when a decision maker making a decision. Moreover, ANP would apply to calculate the interdependency between attributes. The proposed model is able to calculate and consider entropy in decision matrix. Finally, the processing plant location of Sangan open pit mine of Iran was selected using the TOPSIS method under a fuzzy environment due to its rational structure, simplicity, good computational efficiency and capability to determine the relative performance for each option in a simple mathematical form.

2 BASIC OF FUZZY SET THEORY

Uncertainty is a major part of decision making problems in real world that is resulted from two areas (Fouladgar et al. 2011): (1) uncertainty in subjective judgments (2) uncertainty due to lack of data or incomplete information. The first is due to first is due to expert judgment. He/she may not be 100% sure when making subjective

Table 1. Facility location selection methods and weighting methods (advantages and disadvantages)

Author	Year	Ranking method	Weighting method			No limitation in number of alternatives	No limitation in number of attributes	No limitation in Variety of attributes (fuzzy, linguistic, crisp)	Considering the interdependency between criteria	Considering Entropy in decision matrix
			Subjective		Objective					
			Pairwise	ANP	Entropy					
Osanloo	2003	SAW	√	--	--	√	√	--	--	--
Akbari	2007	AHP	√	--	--	√	√	--	--	--
Hekmat	2007	FAHP	√	--	--	√	√	√	--	--
Shahriar	2007	Yager	√	--	--	√	√	√	--	--
Hekmat	2008	SAW, AHP ,TOPSIS	√	--	--	√	√	--	--	--
Yavuz	2008	Yager	√	--	--	√	√	√	--	--
Ataei	2008	ELECTRE	√	--	--	√	√	--	--	--
Golestanifar	2008	FTOPSIS, Fuzzy-WP	√	--	--	√	√	√	--	--
Athawale	2010	PROMETHEE II	√	--	--	√	√	--	--	--
Yazdani	2012	FVIKOR	√	--	--	√	√	√	--	--
Anand	2012	ANP	√	√	--	√	√	--	√	--

judgments. The second one is caused by insufficient information of some attributes

The fuzzy set theory, introduced by Zadeh (1965), deal with vague, imprecise and uncertain problems. A fuzzy set is a class of objects with continuum of grades of membership. Such a set is characterized by a membership function, which assigns to each object a grade of membership ranging between zero and one.

2.1 Linguistic Variables

A linguistic variable is a variable whose values are words or sentences in a natural or artificial language and provides a means of approximate characterization of phenomena which are too complex to be amenable to description in conventional quantitative terms. The main applications of the linguistic approach lie in the realm of humanistic systems (Zadeh 1975).

2.2 Fuzzy Numbers

A fuzzy number $\tilde{M}$ is a convex normalized fuzzy set $\tilde{M}$ of the real line R such that:

- It exists such that one $x_0 \in R$ with $\mu_{\tilde{M}}(x_0)=1$ (x_0 is called mean value of $\tilde{M}$).
- $\mu_{\tilde{M}}(x)$ is piecewise continuous.

In this paper, we use Triangular Fuzzy Numbers (TFNs) because of their computational simplicity and they are useful in promoting representation and information processing in a fuzzy environment. There are various operations on TFNs. Here, only important operations used in this study are illustrated. If we define, two positive TFNs $\tilde{M}_1=(l_1,m_1,u_1)$ and $\tilde{M}_2=(l_2,m_2,u_2)$ then:

1- Inverse: $\tilde{M}^{-1}=(1/u,1/m,1/l)$

2- Addition: $\tilde{M}_1 \oplus \tilde{M}_2=(l_1+l_2,m_1+m_2,u_1+u_2)$

3- Multiplication: $\tilde{M}_1 \otimes \tilde{M}_2=(l_1.l_2,m_1.m_2,u_1.u_2)$

4- Division: $\tilde{M}_1 \% \tilde{M}_1 = l_1/u_2, m_1/m_2, u_1/l_2)$

The distance between two triangular fuzzy numbers can be calculated by vertex method as follow:

$$d_v(\tilde{M}_1,\tilde{M}_2)=\sqrt{\frac{1}{3}\left[(l_1-l_2)^2+(m_1-m_2)^2+(u_1-u_2)^2\right]} \quad (1)$$

An important concept related to fuzzy numbers application is defuzzification. This study adopted the simple center of gravity method which converts a TFN into a crisp value as follows:

$$\bar{X}(\tilde{M}_1)=(l,m,u)/3 \quad (2)$$

Where $\bar{X}(\tilde{M})$ is a crisp value.

3 PROPOSED METHODOLOGY OF MFLS

The general procedure for MFLS in this paper divided into the following steps:

1- Determination of MFLS criteria and their interdependency
2- Determination of criteria weight by hybrid model

3- Alternatives evaluation and ranking procedure by FTOPSIS (Fig. 1)

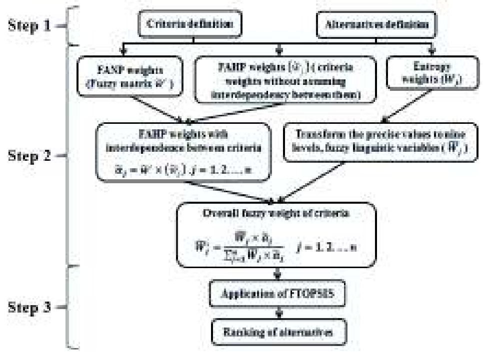

Figure 1. Mine Facility Location Selection (MFLS) framework

3.1 Determination of MFLS criteria and their interdependence

There are many criteria that influence the MFLS process. In assessing a site as a possible location for mine facilities, many factors should be considered. These factors may be presented in many ways; however, the most useful way is the one that may be easily understood by the community (Tchobanoglous et al. 1993). The number and significance of these factors can be different for each mine in each country because the process of MFLS involves a number of stakeholders and sets of requirements such as legislation, restrictions, rules, local expertise and experience. In our study fifty four numbers of extracted leading evaluative attributes from comprehensive literature review, advices by experts and sustainable development framework introduced by Azapagic (2004) are grouped into six main categories, including (1) Economic parameters, (2) Stability conditions, (3) Social-economic factors, (4) Technical parameters, (5) Environmental concerns, and (6) Facility characteristic, then, each main category was protracted to subs criteria as shown in Figure 2.

Based on the literature review in introduction, in existing methods, the influence of each criterion is verified separately and the interdependency between criteria is ignored. For example, the parameters "distance from pit" and "environmental impacts" are simultaneously effective on selecting site No. 1. As the distance from pit increases, the operational costs will also increase; thus, the probability of choosing site No. 1 decreases. On the other hand, increasing distance from pit can be compensated by eliminating the adverse environmental impacts or at least minimizing the intensity and the extent of them. In this condition site No. 1 may be preferred. This paper has discussed how to determine the interdependency between criteria in fuzzy environment.

3.2 Determination Criteria Weight by Hybrid Model

Since the criteria of evaluation have diverse significance and meanings, we cannot assume that each criterion has equal importance. Weighting methods try to define the importance of each criterion in decision making process. Changing the weight in decision making process has a great influence on ranking results.

There are many methods that can be employed to determine the weights of criteria, such as the eigenvector, weighted least square method, entropy, AHP, ANP and linear programming techniques for multidimensional analysis of preference (LINMAP).

Since in decision making problems, there are a direct access to the values of the decision matrix, the entropy and LINMAP methods can be commensurate. Entropy and LINMAP methods both work based on a decision matrix, whereas AHP, ANP, eigenvector method, and weighted least square method follow a set of judgment based pairwise comparison matrices. The selection of a method depends on the nature of the problem. These methods were neither enough nor complete, as it is not possible to design a methodology that will present a perfect weight for each criterion. In this study; a hybrid model based on entropy, FANP and FAHP methods used to obtain the overall fuzzy weight of each criterion. The

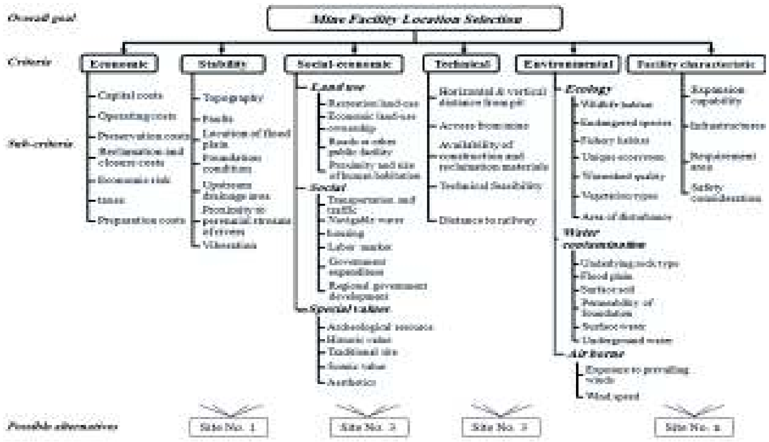

Figure 2. Structure of MFLS hierarchy

overall fuzzy weight of each criterion $\bar{\bar{W}}_j^*$ divides into the:

1- Entropy weight ($\bar{\bar{W}}_j$) based on a decision matrix.

2- FAHP weight with interdependency between criteria for fuzzy data ($\tilde{\alpha}_j$).

The first step is to calculate the weights based on a decision matrix. The speed of the LINMAP method is, however, lower than the entropy method. Furthermore, using the entropy method, it is possible to combine the other weights (Samimi et al. 2012). In the second step FAHP and FANP was used to calculate the relative importance and interdependency between criteria given by experts in pairs.

As for the experts' opinions, this study adopted the Similarity Aggregation Method (SAM) proposed by Hsu and Chen (1996) to integrate experts' weight values for various evaluation criteria. Moreover, ANP is used because there are nonlinear relationships among hierarchical levels which make some problems in implementation of AHP method. Eventually, a set of entropy weights (W_j) that transform to nine level, fuzzy linguistic variables (W_j) and AHP weights with interdependency between criteria ($\tilde{\alpha}_i$), can be used to determine the overall fuzzy weights ($\bar{\bar{W}}_j^*$) by using:

$$\bar{\bar{W}}_j^* = \frac{\bar{\bar{W}}_j \times \tilde{\alpha}_j}{\sum_{j=1}^{n} \bar{\bar{W}}_j \times \tilde{\alpha}_j} \qquad j = 1, 2, \ldots, n \tag{3}$$

3.2.1 Entropy weighting

Shannon and Weaver (1947) proposed the entropy concept for deciding the objective weights of attributes. Entropy weighting used to determine the importance weights of decision attributes by directly relating a criterion's importance weighting relative to the information transmitted by that criterion. For example, in a given decision matrix with column vector $x_j = (x_{1j}, x_{2j}, \ldots, x_{mj})$ that shows the contrast of all alternatives with respect to jth attribute, an attribute has little significance when all alternatives have similar outcomes for that attribute. Mathematically this means that the projected outcomes of attribute j, P_{ij}, are defined as:

$$P_{ij} = x_{ij} \Big/ \sum_{i=1}^{m} x_{ij} \tag{4}$$

The entropy E_j of the set of projected outcomes of attribute j is:

$$E_j = -(1/\ln m) \sum_{i=1}^{m} P_{ij} \ln\left(P_{ij}\right) \tag{5}$$

Where m is the number of alternatives and guarantees that E_j lies between zero and one. The degree of diversification d_j of the information provided by outcomes of attribute j can be defined as $d_j = 1 - E_j$.

Hence, the entropy weighting of an attribute is calculated as follows:

$$W_j = d_j \Big/ \sum_{j=1}^{n} d_j \tag{6}$$

In situations where a decision maker has an priori α_j subjective weighting for attribute, a compromise weighting, W_j^*, that take into account both an expert's opinion and the objective entropy weighting of the attribute is calculated as follows:

$$W_j^* = W_j \times \alpha_j \Big/ \sum_{j=1}^{n} W_j \times \alpha_j \quad j = 1,2,\ldots,n \qquad (7)$$

3.2.2 Weighting with FAHP

This method has been developed by Saaty and Vargas (1994) that is a mathematical tool for solving the MADM problems. For a matrix of order n, ((n) × (n - 1)/2) comparisons are required. The fundamental scale used for this purpose is based on Saaty 1-9 scale. AHP method is combined with fuzzy set to solve the problem of the conventional AHP in handling uncertainty. For achieving the aim, a scale of $\tilde{1}-\tilde{9}$ can be defined for TFNs instead of traditional scale1–9, as presented in Table 2.

Table 2. The definition of fuzzy number

Intensity of importance	Fuzzy number	Linguistic variable
9	$\tilde{9}$ = (8,9,9)	Perfect (P)
8	$\tilde{8}$ = (7,8,9)	Absolute (A)
7	$\tilde{7}$ = (6,7,8)	Very Good (VG)
6	$\tilde{6}$ = (5,6,7)	Fairly Good (FG)
5	$\tilde{5}$ = (4,5,6)	Good (g)
4	$\tilde{4}$ = (3,4,5)	Preferable (PR)
3	$\tilde{3}$ = (2,3,4)	Not Bad (N)
2	$\tilde{2}$ = (1,2,3)	Weak advantage (W)
1	$\tilde{1}$ = (1,1,1)	Equal (E)

This section calculated the weight value of criteria $(\tilde{w}_j)$ by "Column Vector Geometric Mean Method" proposed by Buckley, because the steps of this approach are relatively easier, less time taking and less computational expense than the other fuzzy AHP.

$$\tilde{Z}_i = (\tilde{a}_{i1} \otimes \tilde{a}_{i2} \otimes \ldots \otimes \tilde{a}_{ij})^{1/n} \ , \ i,j = 1,2,\ldots,n. \qquad (8)$$

$$\tilde{w}_j = \tilde{Z}_i \,\%\, (Z_1 \oplus Z_2 \oplus \ldots \oplus Z_n) \ , \ i,j = 1,2,\ldots,n \qquad (9)$$

Where the column vector mean value of fuzzy number is $\tilde{Z}_i$ and the weight of No. i criterion is $\tilde{w}_j$.

3.2.3 Weighting with FANP

The Analytical Network Process is one of the most comprehensive frameworks for the analysis of corporate decisions. It allows both interaction and feedback within clusters of elements (interdependency). ANP introduced by Saaty (1996), is a generalization of the AHP to generate priorities for decisions without making assumption about a unidirectional hierarchy relationship among decision levels. The major difference between AHP and ANP is that ANP is capable of handling interdependency between the decision levels and attributes.

Because there is a high degree of interdependency between MFLS criteria, this study adopted the Fuzzy ANP (FANP) method to calculate the relative importance of the evaluation criteria. The weighting by FANP can be divided into three steps, which are described as follows:

Step 1: Without assuming the interdependency between criteria, the expert is asked to weight all proposed criteria with FAHP method described earlier. The result of the SAM and FAHP methods on *n* criteria can be summarized in a weight vector ($\tilde{w}_j$) with the help of Equations 8-9.

Step 2: The effects of the interdependency between the criteria are resolved. The expert will examine the impact of all criteria on each other by pairwise comparisons as in AHP method. A couple of questions such as: 'which criterion will influence criterion C_2 more; C_3 or C_5? And how much more?' are answered. For achieving the aim, a scale of $\tilde{1}-\tilde{9}$ can be defined for TFNs (Tab. 2) and 0 where C_3 and C_5 is independent of C_2. Various pairwise comparison matrices are constructed for each criterion C_k as follow:

$$\tilde{C}_{C_k} = [\tilde{C}_{C_k}]_{(n-1)\times(n-1)}, \ k = 1,2,\ldots,n \qquad (10)$$

The local weight vectors for these matrices are calculated by using "Column Vector Geometric Mean Method" proposed by Buckley and shown as column components in fuzzy interdependency weight matrix ($\tilde{w}'$).

Step 3: Now we can obtain the interdependency priorities of the MFLS criteria by synthesizing the results from previous two steps as follow:

$$\tilde{\alpha}_j = \tilde{w}' \times (\tilde{w}_j) \ , \ j = 1,2,\ldots,n \qquad (11)$$

Finally, the overall fuzzy weights of evaluation criteria can be determined by Equation 3.

4 ALTERNATIVES EVALUATION AND RANKING PROCEDURE

The full ANP and AHP solution is only partially usable if the number of criteria and alternatives is low. In this paper, we use FTOPSIS and apply it to achieve the final ranking result to avoid a large number of pairwise comparisons and also four advantages addressed earlier. Also, the basic concept of this method is that the chosen alternative should have the shortest distance from the positive ideal solution and the farthest distance from negative ideal solution. Positive ideal solution is a solution that maximizes the benefit criteria and minimizes cost criteria, whereas the negative ideal solution maximizes the cost criteria and minimizes the benefit criteria. Therefore, this method is suitable for cautious (risk avoider) decision maker(s), because the decision maker(s) might like to have a decision which not only makes as much profit as possible, but also avoids as much risk as possible. So it is suitable for those situations in which the decision maker wants to have maximum profit and also the risk of the decisions is important for him/her (Aghajani et al. 2011).

4.1 Fuzzy TOPSIS method

The TOPSIS method is a technique to calculate the preferences by similarity to ideal solution and it was proposed by Hwang and Yoon (1981). In the classical TOPSIS method, the weights of the criteria and the ratings of alternatives are known precisely and crisp values are used in the evaluation process. However, under many conditions crisp data are inadequate to model real life decision problems. In such cases, the FTOPSIS method is proposed where the weights of criteria and ratings of alternatives are evaluated by linguistic variables represented by fuzzy numbers to deal with the deficiency of the traditional TOPSIS. In general, a MADM problem can be concisely expressed in matrix format as:

$$\tilde{D} = \begin{array}{c} \\ A_1 \\ A_2 \\ A_3 \\ \vdots \\ A_m \end{array} \begin{array}{c} \begin{array}{ccccc} C_1 & C_2 & C_3 & \cdots & C_n \end{array} \\ \begin{bmatrix} \tilde{x}_{11} & \tilde{x}_{12} & \tilde{x}_{13} & \cdots & \tilde{x}_{1n} \\ \tilde{x}_{21} & \tilde{x}_{22} & \tilde{x}_{23} & \cdots & \tilde{x}_{2n} \\ \tilde{x}_{31} & \tilde{x}_{32} & \tilde{x}_{33} & \cdots & \tilde{x}_{3n} \\ \vdots & \vdots & \vdots & \cdots & \vdots \\ \tilde{x}_{m1} & \tilde{x}_{m2} & \tilde{x}_{m3} & \cdots & \tilde{x}_{mn} \end{bmatrix} \end{array} \tag{12}$$

Where possible alternatives are $A_1, A_2, \ldots, A_n$, $C_1, C_2, \ldots, C_n$ are criteria which measure the performance of alternatives and $\tilde{x}_{ij}$ is the rating of alternative A_i with respect to criterion C_j which could be in three main types of information (linguistic terms, fuzzy numbers and deterministic data). The step of FTOPSIS method can be defined as follow (Chen, 2000):

Step 1: Construct the normalized decision matrix $\tilde{R}$

The first step concerns the normalization of the judgment matrix $\tilde{D} = \left[\tilde{x}_{ij}\right]$. Generally there are two kinds of attributes, the benefit type and the cost type. The higher the benefit type value is, the better it will be. While for the cost type, it is the opposite. This study adopts normalization methods for deterministic number, triangle fuzzy number and linguistic terms that introduced by Aghajani (2011):

4.1.1 Deterministic numbers normalization

Let r_{ij} be the normalized form of the number k_{ij}, then for benefit index, we have:

$$r_{ij} = k_{ij} \Big/ \max_j k_{ij} \tag{13}$$

And for cost index:

$$r_{ij} = \min_j k_{ij} \Big/ k_{ij} \tag{14}$$

4.1.2 Triangular fuzzy number normalization

Let $\left[r_{ij}^L, r_{ij}^M, r_{ij}^U\right]$ be the normalized form of the triangle fuzzy number $\left[k_{ij}^L, k_{ij}^M, k_{ij}^U\right]$, then for benefit index, we have:

$$\left(\frac{k_{ij}^L}{\max_j k_{ij}^U}, \frac{k_{ij}^M}{\max_j k_{ij}^U}, \frac{k_{ij}^U}{\max_j k_{ij}^U}\right), \quad 1 \le i \le m, j \in I_1 \tag{15}$$

And for cost index:

$$\left(\frac{\min_j k_{ij}^L}{k_{ij}^U}, \frac{\min_j k_{ij}^L}{k_{ij}^M}, \frac{\min_j k_{ij}^L}{k_{ij}^L}\right), \quad 1 \le i \le m, j \in I_2 \tag{16}$$

Where I_1 associated with the benefit criteria and I_2 associated with the cost criteria. With benefit and cost attributes, we discriminate between criteria that the decision maker desire to maximize or minimize, respectively.

4.1.3 Linguistic terms normalization

Linguistic terms can be transferred into TFNs, then use the Equations 15-16 to normalize them. Table 3 is applied to transform linguistic terms into TFNs.

Table 3. Transformation linguistic terms to triangular fuzzy numbers

Linguistic terms	Triangle fuzzy numbers
The worst (TW)	(0.0,0.1,0.2)
Worse (W)	(0.1,0.2,0.3)
Very bad (VB)	(0.2,0.3,0.4)
Bad (B)	(0.3,0.4,0.5)
Normal (N)	(0.4,0.5,0.6)
Good (G)	(0.5,0.6,0.7)
Very good (VG)	(0.6,0.7,0.8)
Better (BE)	(0.7,0.8,0.9)
The best (TB)	(0.8,0.9,1.0)

The normalized decision matrix is as follows:

$$\tilde{R} = \left[\tilde{r}_{ij}\right] \quad , \quad i = 1,2,\ldots,m \quad , j = 1,2,\ldots,n \tag{17}$$

Step 2: Constructing the weighted normalized decision matrix $\tilde{V}$

A set of weights$\left(\tilde{W}_j^*\right)$, where $\tilde{W}_j^*$ is the weight of the *j*th attribute, is incorporated to form the weighted normalized decision matrix $\tilde{V}$ as follows:

$$\tilde{V} = \tilde{r}_{ij}.\tilde{w}_j \quad , i = 1,2,\ldots,m \quad , j = 1,2,\ldots,n \tag{18}$$

Step 3: Define the Fuzzy Positive Ideal and the Fuzzy Negative Ideal Solutions

Let us suppose that $\tilde{A}^+$ identifies the fuzzy positive ideal solution (FPIS) and $\tilde{A}^-$ the fuzzy negative ideal solution (FNIS). They are defined respectively as follows:

$$\begin{aligned} &\tilde{A}^+ = \left\{\tilde{v}_1^+, \tilde{v}_2^+, \ldots, \tilde{v}_n^+\right\}, \\ &\tilde{v}_j^+ = \operatorname*{Max}_i \left\{\tilde{v}_{ij}\right\}, i = 1,2,\ldots,m \ , j = 1,2,\ldots,n \end{aligned} \tag{19}$$

$$\begin{aligned} &\tilde{A}^- = \left\{\tilde{v}_1^-, \tilde{v}_2^-, \ldots, \tilde{v}_n^-\right\}, \\ &\tilde{v}_j^- = \operatorname*{Min}_i \left\{\tilde{v}_{ij}\right\} \ i = 1,2,\ldots,m \ , \ j = 1,2,\ldots,n \end{aligned} \tag{20}$$

Step 4: Measure the distance between alternatives and ideal solutions

To calculate the distance of each alternative from $\tilde{A}^+$ and $\tilde{A}^-$ the following equations can be easily adopted:

$$S_i^+ = \sum_{j=1}^{n} d\left(\tilde{v}_{ij}, \tilde{v}_j^+\right) \ , \ i = 1,2,\ldots,m \tag{21}$$

$$S_i^- = \sum_{j=1}^{n} d\left(\tilde{v}_{ij}, \tilde{v}_j^-\right) \ , \ i = 1,2,\ldots,m \tag{22}$$

$d(\ldots,\ldots)$ is distance between two TFNs obtained from Equation (2).

Step 5: Measure the relative closeness to ideal solution and final ranking

The final ranking of alternatives is obtained by referring to the value of the relative closeness to the ideal solution, defined as follows:

$$CC_i = \frac{S_i^-}{S_i^+ + S_i^-} \quad , i = 1,2,\ldots,m \tag{23}$$

According to the closeness coefficient, determine the ranking order of all alternatives.

5 EVALUATING MODEL APPLICATION AND RESULTS

In order to verify the proposed model, the selection of a processing plant location of Sangan open pit mine of Iran was evaluated. Sangan iron mine project is located 16 km north of Sangan and 300 km southeastern of Mashhad in Khorasan Razavi province in Iran. Three feasible alternatives were selected for the processing plant installation using Geographical Information System. The locations of alternatives are shown in Figure 3 with letter 'A', 'B' and 'C'. These locations are entered into the model as alternatives. For this case we screened proper criteria by opinions of decision group. Finally, 10 main criteria involved in this selection include belt conveyor length (C_1), distance from railway (C_2), distance from tailing dam (C_3), preparation costs (C_4), distance to main roads (C_5), reclamation and closure costs (C_6), stability condition (C_7), social-economic factors (C_8), water contamination (C_9) and ecology disturbance C_{10}). These parameters are entered as criteria to the model (Tab. 4).

Ten main criteria are considered in this case so that four of which are crisp values

(C_1, C_2, C_3 and C_5), four of them are linguistic terms (C_7, C_8, C_9 and C_{10}) and others are fuzzy numbers (C_6 and C_4). In this process, stability condition and social economic factors are entered as benefit criteria (positive effect on decision making) and the other criteria are supposed as costs.

Figure 3. Location of alternatives for the processing plant site ('A', 'B' and 'C')

5.1 Calculate criteria weights by entropy

Using Table 4 and Equations 13-16, normalized decision matrix calculated as Table 5.

Due to complex calculations of entropy method for TFNs, traditional entropy method for deterministic number was applied in this step. Therefore, the simple center of gravity method applied, the results are shown in Figure 4.

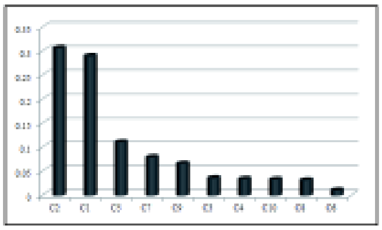

Figure 4. Ranking of main criteria by entropy method (W_i)

5.2 Calculate criteria weights by FAHP

An evaluation team of six members included two mine planning engineers, two academic professors and two environmental agencies was used. In this case, the relative importance of experts is equal. Note that, experts construct pairwise comparison matrix without assuming the interdependency between criteria. This section adopts SAM and FAHP to integrate experts' opinions to obtain the relative importance of evaluation criteria given by experts' in group decision (Fig. 5).

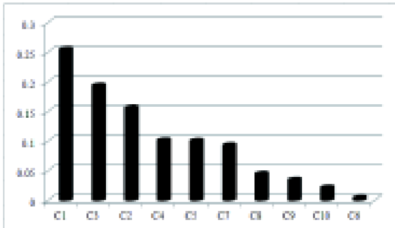

Figure 5. Ranking of main criteria by SAM and FAHP

5.3 Calculate interdependency between criteria by FANP

In this section, the interdependency between the 10 main criteria is analyzed (Fig. 6). The experts will examine the impact of all criteria on each other by pairwise comparisons. Considering Figure 10, four pairwise comparison matrices for reclamation and closure costs, preparation costs, water contamination and ecology were developed. These criteria have dependency with other criteria. The local weight vectors for these matrices obtained by using column vector geometric mean method and shown as rows in matrix $\bar{w}'$ (Tab. 6).

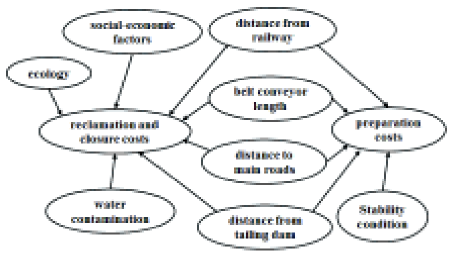

Figure 6. Main criteria interdependency network

5.4 Calculate FAHP weights with interdependency between criteria

The relative importance of the criteria considering interdependence can now be obtained by multiplying the $\bar{w}_j$ by the weight

Table 4. Evaluation value for processing plant location selection in hypothetical open pit mine

	C_1(km)	C_2(km)	C_3(km)	C_4(M$)	C_5(km)	C_6(M$)	C_7	C_8	C_9	C_{10}
A	3.6	3.2	7.4	(1.95, 2.1, 2.25)	2.6	(0.19, 0.21, 0.23)	VG	N	VB	N
B	6.4	1.4	7.8	(2.44, 2.71, 3)	1.9	(0.22, 0.24, 0.26)	N	B	B	B
C	7.0	1.8	6.1	(2.25, 2.5, 2.75)	1.6	(0.21, 0.23, 0.25)	N	B	B	B

Table 5. Normalized decision matrix

	C_1	C_2	C_3	C_4	C_5	C_6	C_7	C_8	C_9	C_{10}
A	1.0	0.44	0.82	(0.87,0.93,1.0)	0.62	(0.83,0.90,1.0)	(0.75,0.88,1.0)	(0.67,0.83,1.0)	(0.5,0.67,1.0)	(0.5,0.6,0.75)
B	0.56	1.0	0.78	(0.65,0.71,0.80)	0.84	(0.73,0.79,0.86)	(0.5,0.63,0.75)	(0.5,0.67,0.83)	(0.4,0.5,0.67)	(0.6,0.75,1.0)
C	0.51	0.78	1.0	(0.71,0.78,0.87)	1.0	(0.76,0.83,0.90)	(0.5,0.63,0.75)	(0.5,0.67,0.83)	(0.4,0.5,0.67)	(0.6,0.75,1.0)

Table 6. Interdependency between criteria (matrix $\tilde{w}_j{}'$)

	C_1	C_2	C_3	C_4	C_5	C_6	C_7	C_8	C_9	C_{10}
C_1	1	0	0	0	0	0	0	0	0	0
C_2	0	1	0	0	0	0	0	0	0	0
C_3	0	0	1	0	0	0	0	0	0	0
C_4	0.51 0.76 1.00	0.05 0.08 0.13	0.51 0.76 1.00	1	0.85 0.88 0.13	0	0.10 0.15 0.22	0	0	0
C_5	0	0		0	1	0	0	0	0	0
C_6	0.05 0.08 0.13	0.02 0.04 0.07	0.09 0.14 0.23	0	0.82 0.84 0.87	1	0	0.08 0.13 0.18	0.09 0.14 0.21	0.08 0.13 0.19
C_7	0	0	0	0	0	0	1	0	0	0
C_8	0	0	0	0	0	0	0	1	0	0
C_9	0	0	0	0	0	0.08 0.13 0.21	0	0	1	0
C_{10}	0	0	0	0	0	0.08 0.15 0.19	0	0	0	1

of matrix $\tilde{w}_j{}'$ based on Equation 11. The FAHP weights with interdependence between criteria are shown in Figure 7.

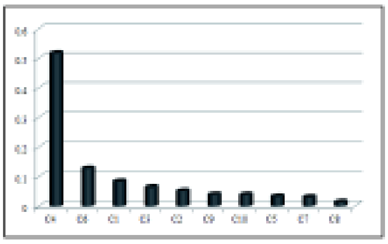

Figure 7. FAHP weights with interdependency of criteria

5.5 Calculate the overall weights of criteria

The overall fuzzy weights ($\tilde{W}_j^*$) are obtained for each criterion by using Equation 3. The weight of each criterion includes entropy of decision matrix, weight of FAHP with considering interdependency between criteria by FANP. The final precise normalized weights of processing plant location selection criteria are illustrated in Figure 8.

Based on Figure 8, belt conveyor length, preparation costs and distance from railway are the most important criteria in processing plant location selection in Sangan open pit mine, followed by distance from main roads, distance from tailing dam, stability condition and water contamination. The priority of other criteria is reclamation and closure costs, ecology and social economic factors.

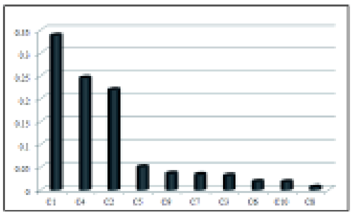

Figure 8. Overall precise weights of criteria

In order to illustrate and analyze the effect of interdependency of criteria the proposed model was solved without considering interdependency between them. To achieve this aim we assume that $\tilde{w}_j = \tilde{\alpha}_j$ (i.e. matrix $\tilde{w}_j{}'$ do not formed), the results are shown in Figure 9.

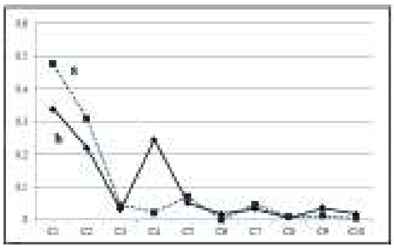

Figure 9. Criteria weights, (a) without considering interdependency of criteria (b) Considering interdependency of criteria

5.6 Final ranking of alternatives by FTOPSIS

In this section ranking process via FTOPSIS put forward. First, decision matrix (Tab. 4) normalized (Tab. 5) and then, the weighted normalized fuzzy decision matrix for the alternatives is calculated using Equation 18. After a weighted normalized fuzzy decision matrix is formed, fuzzy positive ideal solution (FPIS) and fuzzy negative ideal solution (FNIS) are determined using Equations 19-20. Then the distance of each alternative from FPIS and FNIS with respect to each criterion is calculated using vertex method (Eq. 1).

S_i^+, S_i^- and closeness coefficients of three alternatives are calculated by Equations 21-23, results are shown in Table 7.

Table 7: FTOPSIS analysis (considering interdependency of criteria)

	A	B	C
FPIS	2.16	2.32	2.36
FNIS	2.03	1.80	1.78
Closeness coefficient	0.485	0.437	0.430
Ranking	*1*	*2*	*3*

According to the closeness coefficient of three alternatives (Tab. 7), the ranking order of three alternatives is determined as A > B > C. The first alternative (A) is determined as the most appropriate facility location for the processing plant installation. In other words, the first alternative is closer to the FPIS and farther from the FNIS.

Finally, in order to illustrate the effect of interdependency between criteria on final ranking, closeness coefficient of three alternatives calculated without considering the interdependency between criteria, the ranking order of three alternatives is determined as B > C > A (Tab. 8).

Table 8. FTOPSIS analysis (without considering interdependency of criteria)

	A	B	C
FPIS	2.03	2.02	2.13
FNIS	1.65	1.8	1.76
Closeness coefficient	0.448	0.471	0.452
Ranking	*3*	*1*	*2*

6 CONCLUSION

Mine facility location selection is a complex multi person, multi criteria decision problem while sustainable development challenges facing the minerals and metals industry need a comprehensive and interdisciplinary approach based upon reliable data and transparent methodical approaches. In assessing a site as a possible location for mine facilities, many factors should be considered. In this paper fifty four numbers of evaluative attributes are grouped into six main categories as shown in Figure 2. The main objective of this paper is to present a powerful fuzzy MADM tool for making an appropriate decision in MFLS problems featuring uncertainty and contradictory goals. In this approach, a hybrid model of FAHP, FANP and entropy was used to weight the criteria. SAM and FAHP were used to integrate experts' opinions to obtain the significance evaluation of evaluation criteria given by experts in group decision. Moreover, FANP has been used to calculate the interdependency between attributes. The proposed model is able to calculate entropy in decision matrix. Finally, the processing plant location of Sangan open pit mine of Iran was selected using the TOPSIS method under a fuzzy environment. Proposed model analysis showed that considering the interdependency of criteria change the final weights of attributes (Fig. 9). As a result ignoring the interdependency of criteria can cause error in final decision making.

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Kömür Kalite Değişkenlerinin Kovaryans Eşlemeli Krigleme Yöntemi ile Kestirimi

Estimation of Coal Quality Variables by Covariance Matching Constrained Kriging

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ÖZET Kovaryans eşlemeli krigleme (KEK) yöntemi, ortalamasız krigleme'nin (OK) en küçük hata ile gerçeğe en yakın kestirici üretme özelliği ve jeoistatistiksel benzetimin uzaklığa bağlı değişkenlik üretme özelliğini birleştiren bir yöntem olarak tanımlanmaktadır. Bu yöntemde KEK kestiricisi, kriglemenin yansızlık koşuluna ek olarak "örnek noktaları ile kestirim değerlerinin varyans-kovaryans matrislerinin eşitliği" kısıtı ile elde edilmektedir. Bu çalışmada, 3 boyutlu bir kömür damarı modeline ait blokların ortalama alt ısıl değerleri (AID) KEK ve OK yöntemleri ile kestirilmiştir. Kömür damarının ortalama eğimi 7 derecedir. Damarı temsil eden blokların eğimli konumlarından ötürü kestirim sonuçlarına ait harita görüntülerinin "zebra deseni" düzeninde çıkması kaçınılmazdır ve bu görüntü gerçekçi değildir. Bu durumun önüne geçmek amacıyla kestirim işleminden önce kompozitler ile bloklar, dönüştürme bağıntıları kullanılarak yatay düzleme taşınmıştır. Kalite değişkenine ait kestirim işlemleri düzleme taşınmış bloklar üzerinde yapıldıktan sonra orijinal konumlarına geri dönüştürülmüştür.

Çalışmanın sonunda KEK ve OK yöntemlerinden elde edilen görüntü haritaları, sıklık dağılımları, variogramlar ve kalite-tonaj eğrileri karşılaştırılmıştır.

ABSTRACT Covariance matching constrained kriging (CMCK) can be described as an hybrid method that combines local accuracy propety of ordinary kriging (OK) and reproduction of spatial variability property of geostatistical simulation. CMCK estimator is built under constraints such as unbiasedness and variance-covariance matching of predictor and corresponding predictants. In this study, mean lower heating values of lignite seam blocks are estimated by using CMCK and OK. The seam has an average dip of 7^o. When estimating faulted and folded blocks, the estimates follow a "zebraic pattern" which is undoubtedly unrealistic. In order to prevent this phenomenon, blocks and composites are transformed onto the horizontal plane by using rotation equations. Estimations are for unfolded blocks and in the end blocks are back transformed into their original position.

Results of CMCK and OK are compared by means of image maps, histograms, variograms and quality-tonnage curves.

1 GİRİŞ

İşletilebilir kaynak, bir maden kaynağının belirlenmiş bir sınır değerin üzerindeki kısmı olarak tanımlanmaktadır. Bu tür kaynaklar, seçimli maden bloklarının tenörlerini sınır değerlerle karşılaştırılarak belirlenmektedir. Madencilik projesinin fizibilite, maden planlama ve üretim planlaması gibi evrelerinde yer alan işletilebilir kaynağın doğruya yakın bir şekilde belirlenmesi çok önemlidir.

Seçimli madencilik bloklarının ortalama değerlerini belirlemede jeoistatistiksel bir kestirim yöntemi olan krigleme kullanılmaktadır. Ancak krigleme, doğası gereği, büyük değerleri olduğundan küçük, küçük değerleri ise olduğundan büyük olarak kestirmektedir. Koşullu benzetim (Deutsch and Journel, 1992), kriglemeye alternatif olarak düşünülebilir. Bu tekniğin kriglemeye göre üstün yanı, benzetim değerlerinin, kullanılan veri ile uzaklığa bağlı değişkenliğinin aynı olmasıdır. Ancak benzetimlerde girdi parametrelerinin yanlış seçimi yanlı sonuçlar elde edilmesine neden olmaktadır. Benzetim sonucunda birçok gerçekleme içinden hangisinin fiili duruma en yakın olduğunun belirlenememesi benzetimin sorunlu yanlarından bir diğeridir.

Ayrıcı krigleme *(Disjunctive kriging)*(Matheron, 1976), İki değerli krigleme *(Indicator kriging)*(Journel, 1983) ve iki değerli eşkrigleme *(Indicator Cokriging)* (Lajaunie, 1990) yöntemleri, bölgesel değişkenin doğrusal olmayan fonksiyonlarının kestiriminde kullanılan diğer kestiricilerdir.

Aldworth ve Cressie (2003), geliştirilen krigleme ve benzetim tekniklerinin avantajlı yanlarını birleştiren melez bir kestirim yöntemi geliştirmişlerdir. Bu teknik, "Kovaryans Eşlemeli Krigleme" (KEK) olarak adlandırılmaktadır. Tekniğin amacı, t sınır değerine göre $I(Z(A)\geq t)=0$ ve $I(Z(A)\geq t)=1$ ikili değerlerin doğrusal olmayan fonksiyonlarının kestirimidir. Burada Z(A): blok ortalamasını, I(.) fonksiyonu ise bu ortalamanın altında veya üstünde kalan değerleri ifade eden ikili değer fonksiyonudur.

KEK yönteminde sistem, kestirilen noktalar ile kestiriciler arasındaki varyans-kovaryans matrisleri eşitlenerek kurulmaktadır (Cressie and Johannesson 2001). Tercan (2004) iki boyutlu KEK yöntemini işletilebilir kaynak kestiriminde bir linyit sahasında uygulamıştır. Bu çalışmada OK yöntemi ile karşılaştırma yapılmış ve kalite-tonaj eğrileri karşılaştırılmıştır.

Hofer ve Papritz (2010) bir metal döküm tesisinin yol açtığı kirliliği dört tane altıgen bloğu evrensel krigleme (UK*(universal kriging)*), koşullu benzetim, kısıtlamalı krigleme (CK*(constrained kriging)*) ve KEK yöntemi ile kestirmişlerdir. Bu çalışmada blokların ortalaması (lineer) ve kritik eşiği geçen değerlerin (doğrusal olmayan) kestirimleri yapılmıştır.

Hofer ve Papritz (2011) iki boyutta CK, UK ve KEK yöntemleri ile hem nokta hem de blok kestirimler yapan, R yazılımı için "constrained Kriging" isimli bir paket sunmuşlardır.

Bu çalışmada, bir kömür damarına ait alt ısıl değer (AID) değişkeninin KEK ve OK yöntemleri ile kestirimi yapılmıştır. Bu yöntemle elde edilen sıklık dağılımları, variogramları ve kalite-tonaj eğrileri karşılaştırılmıştır.

2 YÖNTEM

Üzerinde çalışılan duruma göre kestirim kıstasları (1) gerçek değerlere yakınlık, (2) uzaklığa bağlı değişkenliğin üretilmesi veya (3) her iki kıstasın da sağlanması olabilir. Krigleme yöntemi, en az istatistiksel hata ile değerler üretmektedir. Benzetim ise, variogram ve histogram gibi temel istatistiklerin yeniden üretilmesini amaçlarken, KEK yöntemi kestirimde lokasyona bağlı değişkenliğin üretilmesinin yanı sıra en az hata ile değer üretmeyi de amaçlamaktadır. Şekil 1'de, KEK yönteminin krigleme ve benzetim yöntemleri arasındaki yerinin şematik olarak gösterimi yer almaktadır.

Şekil 1. KEK yönteminin yeri

KEK yöntemi, kestirimde kullanılan verilerin varyans-kovaryans matrisleri ile kestirimlerin varyans-kovaryans matrislerini, yansızlık koşulu altında eşitleyerek bilinmeyen değerleri kestirmektedir.

$$(\mathrm{Var}[Z_v^*]+\mathrm{Var}[\mu_{gls}]) \times \mathbf{K} = (\mathrm{Var}[Z_v]+\mathrm{Var}[\mu_{gls}]) \quad (1)$$

Burada, Z_v: gerçek blok değerini, Z_v^*: kestirilmiş blok değerlerini, $\mathrm{Var}[Z_v^*]$ ve $\mathrm{Var}[Z_v]$ sırasıyla kestirimi yapılan ve kestirimde kullanılan değerlerin varyans-kovaryans matrislerini, μ_{gls}: genelleştirilmiş en küçük kareler yöntemi ile elde edilen ortalamayı ve **K**: matris eşleme işlemini kontrol eden matrisi ifade etmektedir.

K matrisinin tanımı Eşitlik 2'de yer almaktadır.

$$\begin{aligned} \mathbf{P} &= (\mathrm{Var}[Z_v^*]+\mathrm{Var}[\mu_{gls}]) = \mathbf{L}_P^T \mathbf{U}_P \\ \mathbf{Q} &= (\mathrm{Var}[Z]+\mathrm{Var}[\mu_{gls}]) = \mathbf{L}_{QT} \mathbf{U}_Q \\ \mathbf{L}_P \mathbf{K} &= \mathbf{U}_Q \\ \mathbf{K} &= \mathbf{L}_{P-1} \mathbf{U}_Q \end{aligned} \quad (2)$$

Bu durumda KEK kestiricisi, aşağıdaki gibi ifade edilmektedir:

$$Z^*_{V(KEK)} = \mu_{gls} + \mathbf{K}C(v-x_i)'\,C(x_i-x_j)^{-1}(Z(x_i)-\mu_{gls}) \quad (3)$$

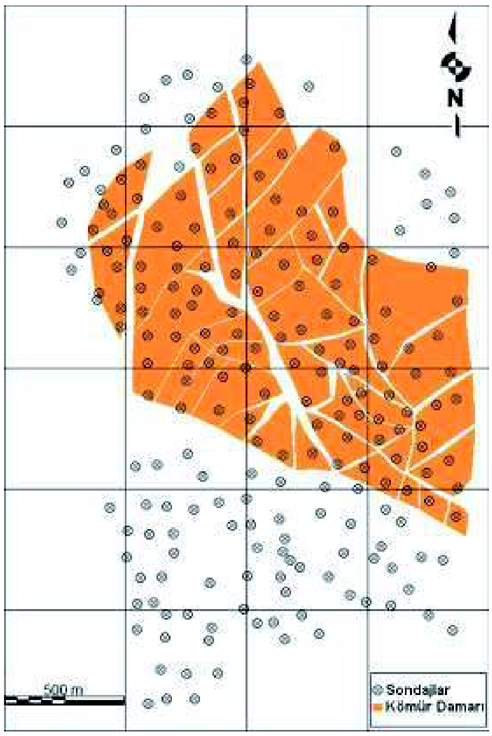
Burada $C(v-x_i)'$: elemanları v bloğu ile x_i lokasyonu arasındaki ortalama kovaryansları olan vektör, $C(x_i-x_j)$: x_i ve x_j lokasyonları arasındaki kovaryansları içeren matrisi, $Z(x_i)$: x_i mekanında ölçülen değerleri içeren vektörü, T ve -1 ise sırasıyla devrik ve matrisin tersini ifade etmektedir.

K matrisini oluşturan **Q** ve **P** matrislerinin pozitif tanımlı olması gerekmektedir. Aksi takdirde, KEK kestiricisi tanımlanamamaktadır. Bu durumda Aldworth ve Cressie (2003) Z^* vektörünü **Q** ve **P** matrislerini pozitif tanımlı yapan örnek kümelerine ayırıp çözüm elde etmeyi önermektedir. **P** ve **Q** matrislerinin pozitif tanımlı olması halinde bile bilgisayar kapasitesinin sınırlarını aşan durumlar yüzünden çözümsüzlük söz konusu olabilmektedir. Bu çalışmada **P** matrisi alt matrislere ayrılarak çözümler yapılmıştır. Kukush ve Fazekas (2005) KEK kestiricisinin elde edilmesinde kullanılan **P** ve **Q** matrislerini her zaman pozitif tanımlı yapan bir algoritma sunmuşlardır. Ancak, karmaşık bir yapıya sahip olan bu algoritma Aldworth ve Cressie'nin (2003) sonuçları ile daha az uyumlu bir sonuç üretmektedir.

3 DURUM ÇALIŞMASI

Çalışmada, kömür damarının katı modeli oluşturulduktan sonra, jeoistatistiksel çalışmalar için, damarın blok modeli elde edilmiştir.

Çalışmada kullanılan 1,5 km^2'lik yüzey alanına ve 17M m^3 hacme sahip kömür damarının doğrultusu K69^{o}B ve eğimi 7^{o}KD olarak ölçülmüştür. Kömür damarının şekli ve sondaj yerleri Şekil 2'de gösterilmişitir. Sahanın güneyinde yer alan ve daha önce üretimi tamamlanmış olan kısım modele dahil edilmemiştir.

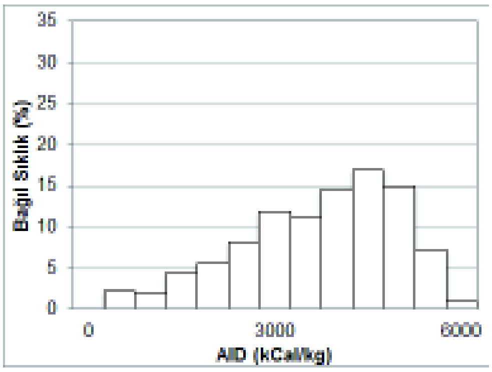


Şekil 2. Kömür damarı ve sondaj konumları

3.1 Kompozitleştirme

Çalışmada, sahada yapılmış 245 adet dik sondaja ait analiz değerleri kullanılmıştır. Sondaj karotlarının ortalama uzunluğu 0,52 m'dir. Çalışmada kompozit uzunluğu 1 m olarak alınmıştır Ara kesmelerin kompozitleme işleminde hesaba katılması gerekmektedir. Bu nedenle ara kesmelerin AID değerleri 1 kCal/kg olarak atanmıştır.

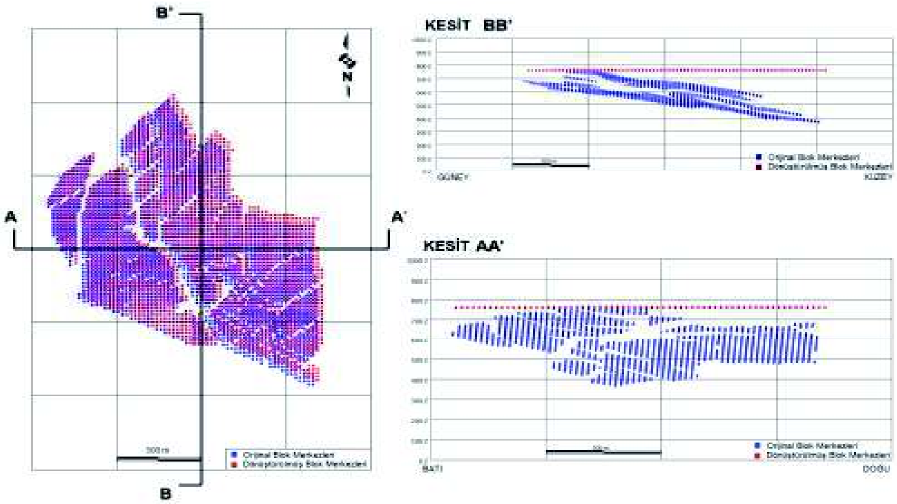
AID değişkenine ilişkin kompozitlerin sıklık dağılımı Şekil 3'te yer almaktadır. AID değişkeni sola çarpık bir dağılım ortaya koymaktadır.



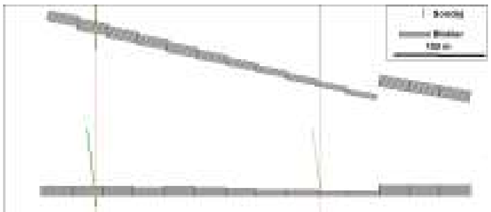
Şekil 3. Kompozitlerin sıklık dağılımı

3.2 Bloklar Ve Kompozitlerin Düzleme Taşınması

Kestirim işleminden önce kömür damarını temsil eden bloklar ile kestirimde kullanılan kompozitler, aralarındaki uzaklığa bağlı ilişki bozulmadan, aynı düzleme taşınmıştır (Şekil 4). Bu işlemin amacı, tektonik hareketler nedeniyle kıvrımlı ve faylı blokların kesitiriminde koordinat sistemindeki konumlarından ötürü kestirimlerin "zebra deseni" gibi çıkmasının önüne geçmektir. Bunun için, bloklara ve kompozitlere sadece koordinat dönüşümü değil, sündürme ve germe işlemlerinin yapılması gerekmektedir. Kestirimler bu gerilmiş ve sündürülmüş uzayda yapıldıktan sonra geri dönüşüm ile orijinal hallerine getirilmektedir. Deutsch (2005), katı modelden elde edilen kesitlerdeki kontrol noktaları aracılığıyla kıvrım düzleme yöntemi önermiştir. Bu çalışmadan farklı olarak mevcut çalışmadaki kıvrım düzeltmede doğrudan blokların merkez koordinatları ve blok içerisine düşen kompozit noktaları kullanılmıştır. Şekil 5'te blokların dönüşümden önceki ve sonraki durumları görülmektedir



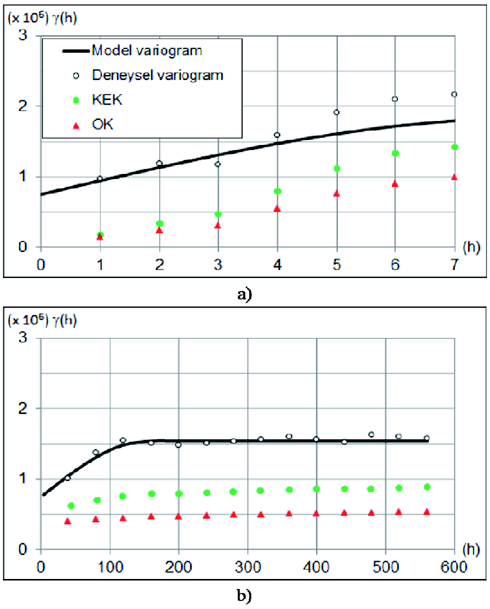
Şekil 4. Blok merkezlerinin orijinal ve dönüştürülmüş olarak plan ve kesit görünüşleri



Şekil 5. Blok dönüştürme işlemi

3.3 Variogram Analizi

Kömür damarında AID değişkeni, derinlik arttıkça değişen kömür kalitesinden ötürü yönelim *(trend)* göstermektedir. Kestirim işleminden önce yönelimler, verilerden uzaklaştırılmış ve deneysel variogramlar bu artık *(residual)* değerler kullanılarak hesaplanmıştır. Artık değer AID değişkenlerine ilişkin variogramlar Şekil 6'da yer almaktadır.



Şekil 6. Değişkene ilişkin (a) düşey yönde ve (b) yatay yönde kompozit variogramlar

Variogramlar incelendiğinde, düşey yöndeki yapısal uzaklığın, yatay yöndeki yapısal uzaklığa göre çok kısa olduğu görülmektedir. Bu durum, linyit damarının yatay yöndeki yayılımı ile kalınlığı arasındaki farktan kaynaklanmaktadır. Variogramdaki külçe etkisi, eşik değerlerinin hemen hemen yarısına eşittir. Bu durum ise kömür damarının içerisinde bulunan ara kesme değerlerinin etkisinden kaynaklanmaktadır. Deneysel variograma uyarlanan küresel model variograma ilişkin parametreler Çizelge 1'de özetlenmiştir.

Çizelge 1. Variograma ilişkin parametreler

C_0	C_1	C_2	a_1		a_2
600.000	200.000	422.000	50 (yatay)	3 (düşey)	250

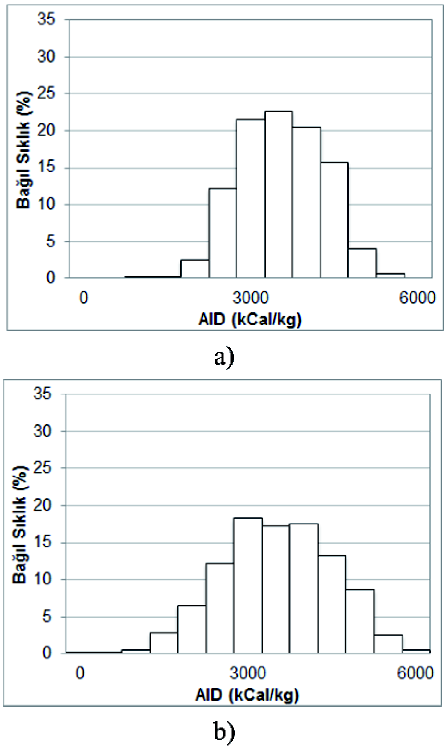
3.4 Blok Ortalamalarının Kestirimi

Kömür damarını temsil eden blok model, boyutları 25 m×25 m×1 m olan toplam 19974 bloktan oluşmaktadır. Blok ortalamalarının kestirim işleminin son aşamasında yönelim değerleri artık değerlere ilave edilmiştir.

OK ve KEK kestirim sonuçları, tanımlayıcı istatistikler, sıklık dağılımı, variogram, yönelim analizleri açısından karşılaştırılmıştır. Ayrıca iki farklı kestirim yönteminin kalite-tonaj eğrisine etkisi incelenmiştir. Çizelge 2'de blok kestirim değerleri ile kompozitlerin tanımlayıcı istatistiklerine ilişkin sonuçlar yer almaktadır. Çizelgede kestirim sonuçlarının ortalaması incelendiğinde yansız bir sonuç elde edildiği görülmektedir. Ancak blok kestirimlerinin varyansları farklılık göstermektedir. KEK yöntemi, OK yöntemine göre daha yüksek bir varyans üretmiştir. Şekil 7'de, kestirim sonuçlarına ilişkin sıklık dağılımları karşılaştırılmaktadır. OK kestirimlerinin ortalama etrafında sıklaştığı ve KEK kestirimlerinin, OK'ya göre daha geniş bir aralığa yayıldığı görülmektedir.

Çizelge 2. Kompozit ve kestirim sonuçlarının tanımlayıcı istatistikleri

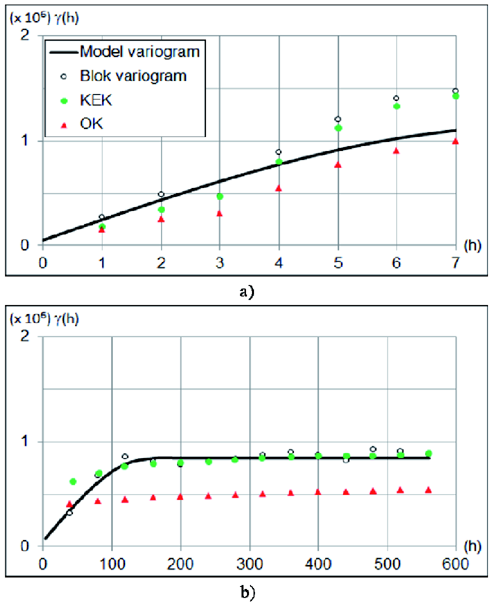
	Veri sayısı	En düşük	Ortalama	En yüksek	Varyans
Kompozit	1677	1	3454	5818	1581227
KEK	19774	1	3261	5854	836633
OK	19774	1003	3311	5361	516910



Şekil 7. (a) OK ve (b) KEK yöntemlerinden elde edilen sıklık dağılımları

Blok variogramları ve kestirim sonuçlarının variogramları Şekil 8'de verilmiştir. Blok variogramı, blok içindeki ortalama variogram değerinden kompozit variogram değeri çıkarılarak elde edilmektedir (Journel ve Huijbregts, 1978). OK kestirimlerinin variogramlarının, KEK ile karşılaştırıldığında, yapısal uzaklığın daha fazla ve düşük eşik değerli olduğu görülmektedir. KEK kestirim değerlerinin variogramları ise blok variogramlara yakın sonuçlar üretmiştir.

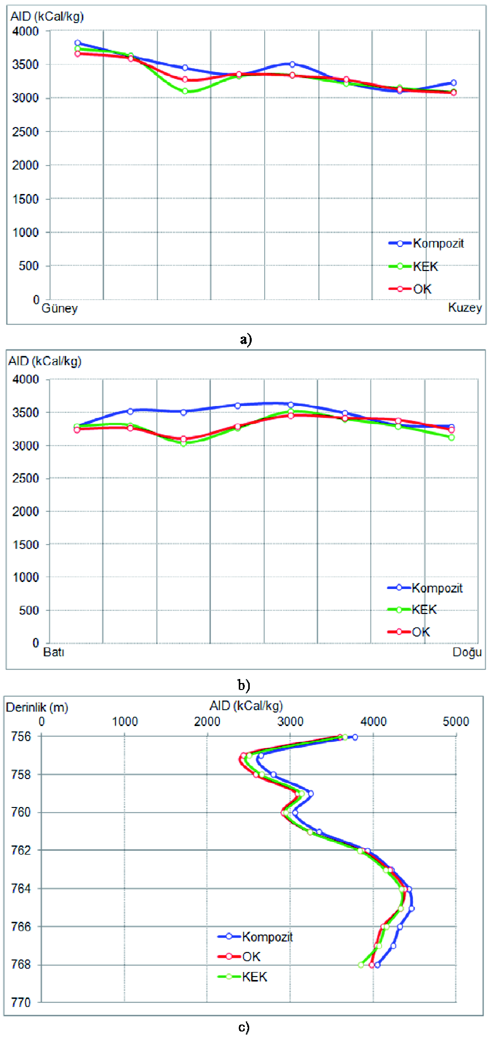
Kestirim sonuçlarının sahanın genelinde nasıl değişkenlik gösterdiğini incelemek için saha Kuzey-Güney (KG), Doğu-Batı (DB) ve dik yönlerde dilimlere ayrılmış ve bu dilimlerin içinde kalan blokların ortalama değerleri grafiğe aktarılmıştır (Şekil 9). Dik yöndeki hesaplamalar 1 m dilim kalınlığı ile KG ve DB ana yönlerindeki hesaplamalar ise 500 m'lik dilimlere göre yapılmıştır.



Şekil 8. Değişkene ilişkin (a) düşey yönde ve (b) yatay yönde blok variogramlar

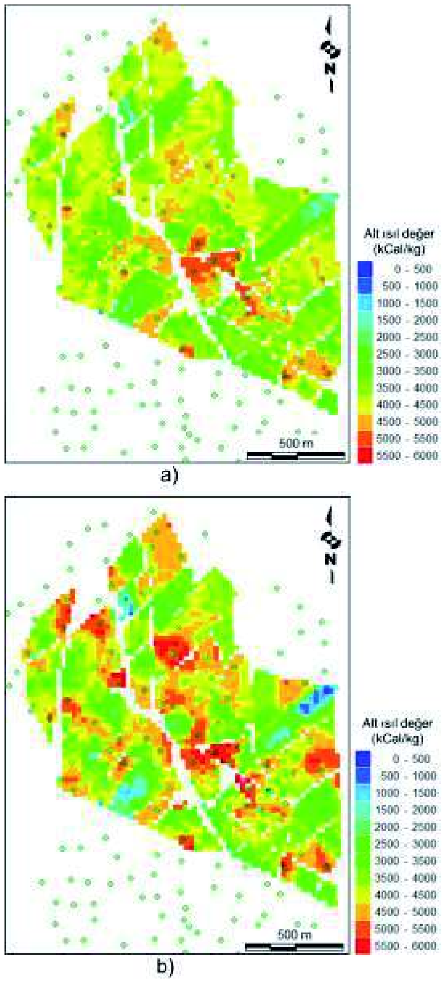
Şekil 9 incelendiğinde, sahanın uç noktalarındaki dilimler içine düşen blok sayısının diğer dilimlere göre fazla olmamasından kaynaklanan sapmalar dışında sonuçların kompozit değeri ile uyumlu olduğu görülmektedir. Şekil 9'a göre kömür kalitesi, derine gidildikçe artış göstermektedir. Bu durum OK ve KEK kestirim sonuçlarınca da üretilmiştir. KG yönünde AID değerleri 3200 kCal/kg'dan 3600 kCal/kg'a yükselmektedir.

Şekil 9'a göre OK ve KEK kestirim sonuçları benzer gözükse de, Şekil 10'da verilen AID plan görünüşlerinde lokasyona bağlı değişkenliklerin farklı olduğu görülmektedir. KEK kestirimleri, OK kestirimlerine göre daha çok değişkenlik göstermektedir. Örneğin KEK yöntemi, Kuzeybatı-Güneydoğu şeridinde yüksek değerler üretirken, OK kesitirim sonuçlarında bu durum gözlenmemektedir.

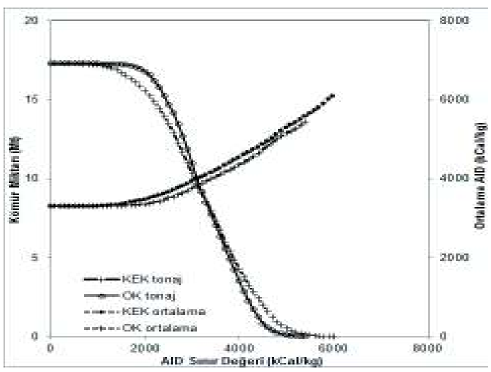


Şekil 9. AID ortalamalarının (a) GK, (b) BD ve (c) düşey yönlerdeki değişimi

OK ve KEK kestirim sonuçlarına göre çizilen kalite-tonaj eğrisi Şekil 11'de yer almaktadır. KEK yönteminde, OK sonuçlarına göre, AID sınır değer üzerinde daha yüksek ortalamalar elde edilmiştir. Bu durum, KEK yönteminin, OK yöntemine göre yüksek değerler üretebilmesinden kaynaklanmaktadır. OK kestirimlerinden, düşük sınır değerlerde yüksek tonaj; yüksek sınır değerlerde ise düşük tonaj elde edilmiştir.



Şekil 10. AID ortalamalarının (a) OK ve (b) KEK yöntemlerine ait görüntü haritaları



Şekil 11. Kalite-tonaj eğrisi

4 SONUÇ

Çalışma sahasındaki kömür, sahanın tektonik yapısı gereği, değişken kalınlıklı ara kesmeli yapısı yüzünden ve derinleştikçe değişen kalite bakımından çözülmesi zor bir kestirim problemi ortaya koymaktadır. Sahadaki faylı yapının etkisini ortadan kaldırmak için kıvrımları düzleştirme veya düzleme taşıma işlemi yapılmıştır. Ayrıca derinleştikçe artan kömür kalitesinin etkisini yok etmek amacıyla yönelim değerlerini gerçek değerlerden çıkararak artık değerler kullanılmıştır.

AID artık değerlerinin sıklık dağılımı sola çarpık bir dağılım göstermektedir. Kömür damarı içerisinde bulunan ara kesmeler, variogramda yüksek külçe etkisi olarak gözükmektedir. Kömür damarının yatay yöndeki yayılımı, düşey yöndeki yayılımına göre çok daha fazladır ve bu durum, yatay yönde hesaplanan variogramlarda daha çok devamlılığı olan yapısal uzaklık olarak açığa çıkmaktadır.

Kestirim sonuçlarına göre KEK yöntemi, OK yöntemine göre lokasyona bağlı değişkenliği daha iyi yansıtmaktadır. KEK yöntemi, aynı zamanda OK yöntemi gibi gerçeğe yakın değerler üretmeye imkân veren yansız bir kestiricidir. Hem lokasyona bağlı değişkenliğin hem de gerçeğe yakın kestirimlerin amaçlandığı durumlarda kullanılabilecek bir kestirim yöntemidir.

Kalite-tonaj eğrilerinde belirli AID sınır değer üzerinde kalan ortalamalarda KEK yöntemiyle, OK yönteminden daha büyük ortalamalar elde edilmiştir.

İlerleyen aşamalarda, sıklık dağılımında sağa veya sola çok çarpıklık gösteren veriler üzerinde çalışmala yapılarak KEK yönteminin nasıl sonuçlar üreteceği incelenebilir. Ayrıca, KEK matris sistemine doğrudan etkisi olan K matrisinin yapısı üzerinde çalışmalar yapılarak farklı ayrıştırma yöntemlerinin incelenmesi bu çalışmaya katkı sağlayacaktır.

KATKI BELİRTME

Bu çalışma 108G036 nolu TÜBİTAK projesi kapsamında desteklenmiştir.

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Geostatistical Assessment of Collapses in Gole Gohar Open Pit Mine, Kerman, South West of Iran

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ABSTRACT Almost all collapses of rock slops especially in mines are related to discontinuities which include beddings, faults and major joints. Spatial distribution of collapses in open pit mines is related to distribution of discontinuities. Geostatistical assessments can be used for understanding the distribution of regionalized variables in any spatial study. In this paper, regionalized variable theory is used for analyzing and interpreting the spatial distribution of collapses taken place at Gole Gohar iron mine, which is located south west of Kerman city, Kerman province, and south west of Iran. In order to define regionalized variable distribution, at first step, variogram functions are determined for identifying the regional behavior. Furthermore, it is possible to estimate the tonnage of collapse for every local block on pit wall and prepare maps for the interpretation of behavior of the regionalized variable. Analysis of variograms showed that the tonnage of collapses have a spatial structure that make it possible to set up a geostatistical model to make a prediction of collapses for each block on the pit wall.

Keywords: geostatistical assessment, regionalized variable, collapse, Gole Gohar open pit mine

1 INTRODUCTION

Rock slope instabilities are a major hazard for human activities often causing economic losses, property damages and maintenance costs, as well as injuries or fatalities.

Stability analyses are routinely performed in order to assess the safe and functional design of an excavated slope (e.g. open pit mining, road cuts, etc.), and/or the equilibrium conditions of a natural slope. The analysis technique chosen depends on both site conditions and the potential mode of failure, with careful consideration being given to the varying strengths, weaknesses and limitations inherent in each methodology.

It is less than 25 years since most rock slope stability calculations were performed either graphically or using a hand-held calculator. The engineer today is presented with a vast range of methods for the stability analysis of rock and mixed rock-soil slopes; these range from simple infinite slope and planar failure limit equilibrium techniques to sophisticated coupled finite/distinct element codes.

Geotechnical engineering is constantly evolving and its practitioners are always looking out for tools, which can improve design and help better handle the large uncertainties and variations inherent in soil and rock properties. In recent years, several authors have attempted to apply geostatistics to the problems of geotechnical engineering.

Geostatistics, as a methodology for estimating recoverable reserves in mining deposits, was mathematically formalized by French professor Georges Matheron 1963, inspired by the pioneering work of South African mining engineer D.G. Krig in the 1950's. Today it is extensively used in the mining and petroleum industries, and in recent years has been successfully integrated into remote sensing (Atkinson&Lewis, 2000, Qingmin Meng et al.,2009, Pardo-Iguzquiza et al., 2011) and Geographic Information Systems (GIS) (Choi&Park,2006), soil scientists (Choi&Park,2006, Emery, 2006, Tavares, et al., 2008), hydrologists (Hossain, et al., 2007, Chowdhury,2010) as well as statisticians, so there are successful applications to a variety of fields. Geostatistical assessments can be used for understanding the distribution of regionalized variables in any spatial study. In this paper, regionalized variable theory is used for analyzing and interpreting the spatial distribution of collapses taken place at Gole Gohar iron mine which make it possible to predict collapses for each block on the pit wall.

2 GEOLOGICAL SETTING

The Gole Gohar iron mine is located in the NW of Sanandaj-Sirjan adjacent to Zagros zone in Iran. The mining area is in 53 kilometers of South West of the Sirjan in latitudes 55° 15′ to 55° 24′ and longitudes of 29° 3′ to 29° 7′. Given the tectonic setting, the remote sensing survey of the area around the mine, the geological survey of the mine and the area around the Gole Gohar mine, following results about structural geology model of mine were obtained.

In the study area variety of faults consist of reverse, strike slip, normal faults and tensile major joints with considerable aperture are visible. Figure 1 shows distribution of faults around the pit No.1 of Gole Gohar mine.

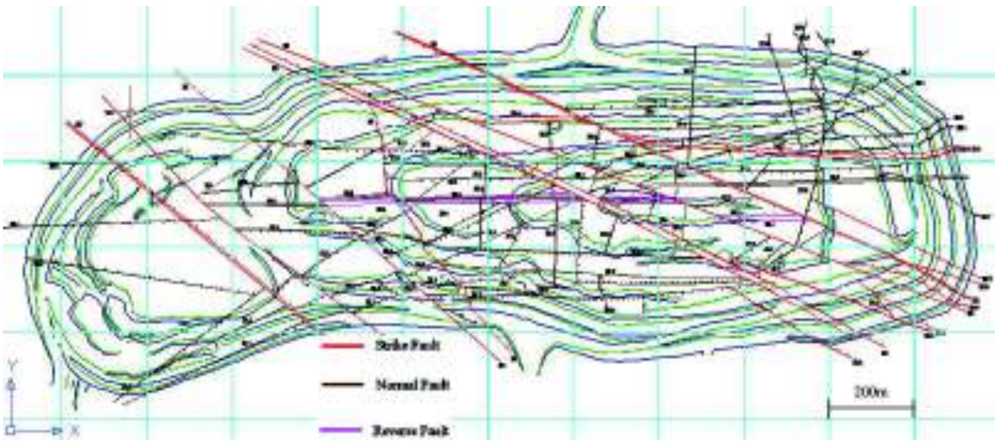

Figure 1. Distribution of faults around pit No.1 of Gole Gohar iron mine

What is causing the above faults is existence of a subsurface right lateral strike slip fault with NW-SE trend, which has to bend to the left. This situation has led a compressional lens shaped that its northeast and southwest boundary are thrust faults with dips towards the southwest and northeast, respectively.

Structural geology section perpendicular to the strike of the zone is like a flower.

Flower structure as a certain structure of deformation strike slip areas can be seen in pit No.1 (Fig. 2).

In bedrock including ore body, faults often has east-west trend with dip 45 to 80 degree toward the south. These faults almost are boundary between ore body and host rocks. Since these faults have small angle with the northern benches of mine and their slope is consistent with trenches slope, instability is inevitable(Hasanpoor,2010).

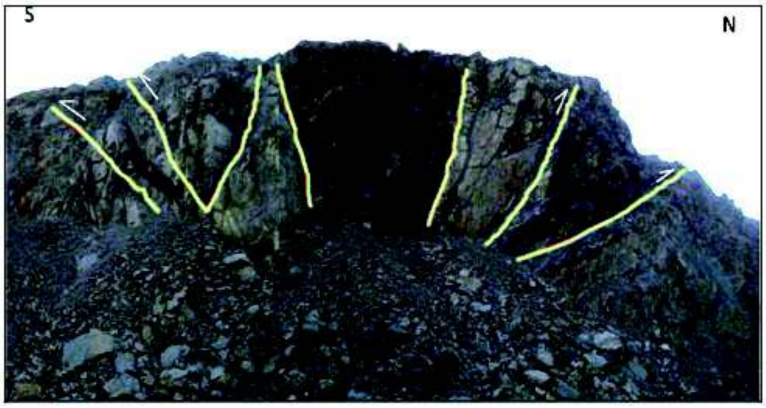

Figure 2. Flower faults structure in southwest wall of the Gole Gohar mine (Hasanpoor,2010).

3 DETERMINATION OF POTENTIAL FAILURE GEOMETRY

To determine which failure modes are possible at a particular operation the geologic parameters in various sectors of the mine need to be quantified. Collecting information such as orientation, spacing, trace length, and shear strength with respect to major structures and other geologic features is an important key to determining failure potential. The basic failure modes which may occur are planar, wedge, circular and toppling failure (Osanloo, 2005, Girard, 2001).

The types of failures occurred in pit No.1 of Gole Gohar mine which obtained in field survey is shown Figure 3. As shown in Figure 3, rock mass in Gole Gohar mine have potential of different kinds of failures.

4 METHODOLOGY

The basic geostatistical tool for characterizing spatial variability is the variogram $\gamma(h)$. $\gamma(h)$ is defined as half the average quadratic difference for N pairs of measurements of the variable z separated by a distance h (Armstrong, 1998, Isaaks&Srivastava, 1989, Journel, 1989, Journel&Huijbregts, 1978):

$$\gamma(h) = \frac{1}{2N(h)} \sum_{i=1}^{N(h)} [z(x+h) - z(x)]^2 \qquad (1)$$

After calculation the experimental variogram, it is necessary to adjust the mathematical model to represent the variable as realistically as possible. It is important that the mathematical model represents the trend of the variogram with relation to distance h. Estimates obtained from kriging will then be more precise and reliable.

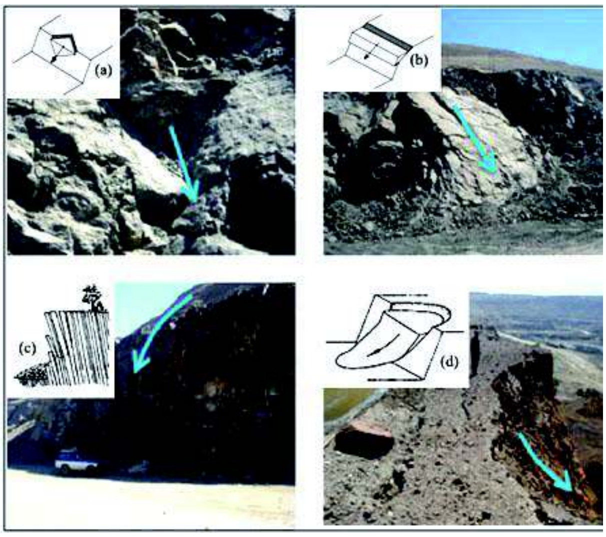

Figure 3. Types of failures occurred on the pit wall of Gole Gohar iron mine. (a) wedge failure, (b) planar failure, (c) toppling failure and (d) circular failure(Hasanpoor,2010).

Among the spatial interpolation (geostatistical estimation) techniques, a process called kriging is the best linear unbiased estimator (BLUE) of unknown characteristics (Isaaks&Srivastava, 1989, Journel, 1989), which make it possible to understand the regional behavior of the natural phenomena for every point in the study area (Krige, 1962).

If magnitudes of data are available at specific locations, it is possible to estimate the values of it at other locations through Kriging. The goal of Kriging is to predict the average value of $Z(x_0)$ at specific point of study area. If $Z(x_1), Z(x_2), Z(x_3), \ldots, Z(x_n)$ are known values of parameter, then the estimated value of parameter at point x_0 is given by:

$$Z(x_0) = \sum_{i=1}^{n} w_i Z(x_i) \quad (2)$$

Where w_i are weights applied to the respective values $Z(x_i)$, such that:

$$\sum_{i=1}^{n} w_i = 1 \quad (3)$$

The weights w_i are determined through kriging matrix (Isaaks&Srivastava, 1989, Subyani, 1997).

5 GEOSTATISTICAL MODELING AND DISTRIBUTION OF TONNAGE OF COLLAPSE

In this research the variable is the tonnage of collapses. Spatial distribution of collapses on the pit wall of Gole Gohar iron mine is shown in Figure 4.

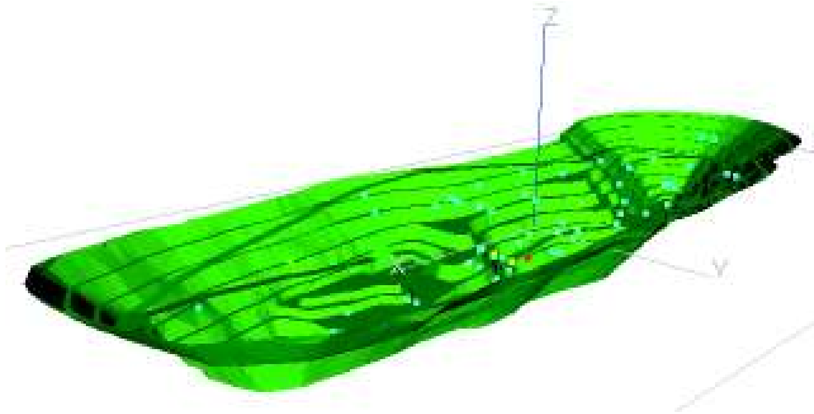

Figure 4. Spatial distribution of collapses on the pit wall of Gole Gohar iron mine.

Here, the methodology is applied in order to represent the distribution of tonnage of collapse as regional variable. Investigation of the distribution of interest regionalized variable in the given pit wall is carried out by determining the variogram function. These functions are determined for identifying the regional behavior.

Through experimental variograms calculated with all collapses data in several directions, the omnidirectional variogram which shows the best structure was chosen. The experimental variogram for collapse data was fitted using the spherical model which is presented in Figure 5. The optimum sill and range were chosen for variogram by cross validation method. The parameters of the variogram function are given in Table 1.

After determining a theoretical variogram and running kriging technique using above mentioned methodology, it is possible to estimate the tonnage of possible collapses for every local block on pit wall and prepare maps for the interpretation of behavior of the regionalized variable.

The kriging map of estimated blocks with the size of 25*25*10m in level 1594m is shown in Figure 6.

Table 1. The parameters of the variogram function

Variogram model	Nugget ($\%^2$)	Sill ($\%^2$)	Range (m)
Spherical	28.6	430	115

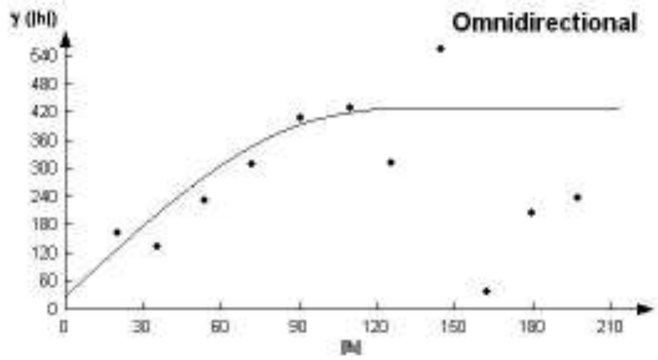

Figure 5. Spherical model fitted to experimental variogram calculated for tonnage of possible collapses occurred in Gole Gohar iron mine.

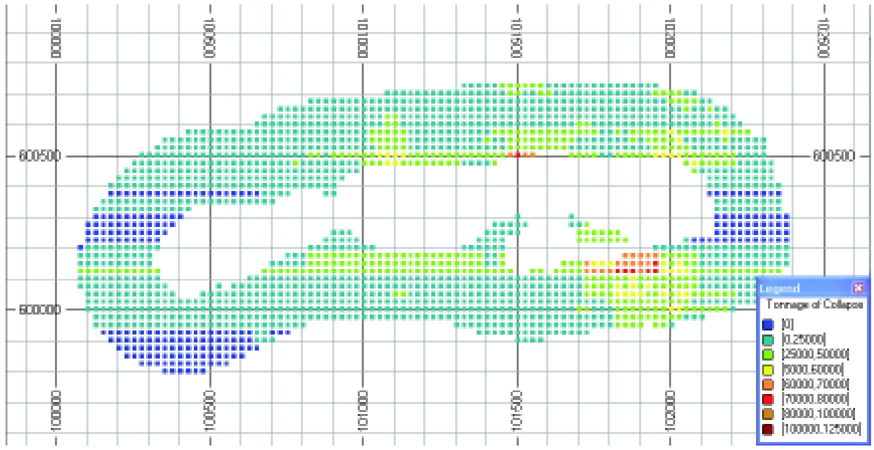

Figure 6. Tonnage Kriging map of possible collapse in level 1594m

Comparing figures 1 and 6 shows, there is a relation between tonnage of possible collapses and existence of discontinuities specially faults around the pit area. In an another word, distribution of possible collapses in pit No.1 of Gole Gohar iron mine is related to distribution of discontinuities so covariance between tonnage of possible collapses and discontinuities in area could be an approach to evaluate safety factor. As shown in Figure 6, eastern and western walls are stable but there is possibility of collapse in northern and southern walls. Therefore, for development of exploitation in north and south walls of pit mine, some particular improvement methods such as unloading and maintenance system could be considered.

6 CONCLUSION

In this paper, geostatistical analysis provides a three dimensional visualization of spatial variability of tonnage of collapse in open pit Gole Gohar iron mine. The first step of the application is to determine variogram functions for identifying the regional behavior, and the second step is to estimate the tonnage of collapse for every local block on pit wall and prepare maps for the interpretation of behavior of the regionalized variable. Results show that distribution of possible collapses in pit No.1 of Gole Gohar iron mine is related to distribution of discontinuities. Also there is possibility of collapse in northern and southern walls.

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