

HAMIDREZA TAHERKHANI*, RAMIN DOOSTMOHAMMADI*¹**INVESTIGATION OF GEOTECHNICAL PARAMETERS EFFECT ON OPEN PIT MINING OPERATION COST (CASE STUDY: ANGOURAN MINE)****BADANIE WPLYWU PARAMETRÓW GEOTECHNICZNYCH NA KOSZTY PROWADZENIA WYDOBYCIA W KOPALNIACH ODKRYWKOWYCH (STUDIUM PRZYPADKU: KOPALNIA ANGOURAN)**

Properties of intact rock and discontinuities structures are the most important variables affecting mining operations. A comprehensive review of research concerning the direct effect of geotechnical parameters on changes of total exploitation cost in open-pit mines are not provided since now. In this paper, the influence of geotechnical properties of rock mass on total cost of mining operations in Angouran (the largest lead and zinc mine in Middle East located at south western province of Zanjan, Iran) is examined. At first, the classified components of slope mass rating (SMR) and then all exploitation costs (total costs of drilling operation, blasting, bulldozers work, loading and hauling) in mining blocks are surveyed. Then, an equation for determining the correlation between geotechnical properties and total cost using multivariate linear regression is proposed. Eventually, conducting sensitivity analysis revealed that the variation of joints dip is the most effective parameter in change of mining operation cost.

Keywords: geotechnical parameters, mining operation cost, components of slope mass rating survey, sensitivity analysis, Angouran mine

Właściwości skał w pierwotnym stanie naprężenia oraz struktura nieciągłości są najważniejszymi zmiennymi wpływającymi na prowadzenie działalności górniczej. Do dnia dzisiejszego nie jest dostępny całościowy przegląd wszystkich prac badających wpływ parametrów geotechnicznych na zmiany kosztów eksploatacyjnych związanych z prowadzeniem wydobywania w kopalniach odkrywkowych. W pracy tej zbadano wpływ właściwości geotechnicznych górotworu na wysokość kosztów eksploatacji górniczej w kopalni Angouran (największa kopalnia cynku i ołowiu na Dalekim wschodzie, zlokalizowana w południowo-zachodniej części prowincji Zanjan, Iran). W pierwszej części określono podstawowe parametry klasyfikacji geotechnicznej SMR (*Slope Mass Rating*) oraz całkowite koszty eksploatacyjne (całkowite koszty prac wiertniczych, strzałowych, pracy spycharek, załadunku i transportu urobku). Następnie podano równanie określające korelację pomiędzy właściwościami geotechnicznymi skał a całkowitym kosztem

* DEPARTMENT OF MINING ENGINEERING, UNIVERSITY OF ZANJAN, IRAN.

¹ CORRESPONDING AUTHOR. E-MAIL ADDRESS: ramin.doostmohammadi@znu.ac.ir

wydobycia, z wykorzystaniem regresji liniowej wielu zmiennych. W końcowej części przeprowadzono analizę wrażliwości, która wykazała, że zmiana upadu sieci spełkań jest parametrem który w największym stopniu wpływa na zmianę kosztów wydobycia.

Słowa kluczowe: parametry geotechniczne, koszty prowadzenia wydobycia, parametry wyrobiska pochyłego, analiza wrażliwości, kopalnia Angouran

1. Introduction

The cost calculation and management is one of the most important points in mining activities. Cost management includes planning, analyzing, recording and controlling stages. The cost analysis and assessment, as the main component, provides information in regards to the level, structure and changes of production costs in various profiles and systems (Jonek-Kowalska, 2013). The received information constitutes a basis for decision making concerning changes in the cost policy of the enterprise and calculating the break-even point (BEP) (Fuksa, 2013). The main goal of those changes is to improve the operational effectiveness achieved mainly by the cost reduction or optimizing their structure. If the cost accounting systems do not provide information about the costs of particular processes or objects, therefore the assessment of their effectiveness is neglected. It causes problems such as defecting the cost management, divesting of the planning function and in result depriving of the motivational functions. These disadvantages constitute a serious threat for the operational effectiveness (Jonek-Kowalska, 2012, 2013).

All phases of mining operations, including drilling, blasting, loading and hauling are involved with rocks. Efficiency for each of these stages is influenced by intact rock properties and discontinuities structure. Rock properties and structure of discontinuities which are the geotechnical parameters of rock, have an essential role in economy of manufacturing and optimization in mining and processing operations. Blasting is one of the most important operations, which has a great technical and economical effect on the mining projects. In the mining activities the prime aim of blasting operation is rock fragmentation that is necessary for subsequent processes such as transportation, crushing, etc. hence, achieving a higher efficiency (Monjezi et al., 2013). Since the implementation quality of drilling and blasting depends on intact rock characteristics and structure of discontinuity, consideration of the rock mass characteristics is so essential to make best results in drilling and blasting operations. Loading and hauling is considered one of the major steps and its quality is so dependent on the conditions of drilling and blasting process. The amount of rock fragmentation from explosions (the amount of crushing blow), swell factor of rock mass, content uniformity and depot formation are the most important factors of blasting which significantly affects efficiency and transport operational parameters such as bucket-fill factor, loading speed and depreciation of machines.

Any research to investigate the geotechnical parameters effect of rock mass on the rate of change in total cost of mining operations in open pit mines has never been conducted. Most studies have focused on amount of effects in the field of geotechnical parameters on the outcome of rock fragmentation and/or classifications of Blastability.

A number of researchers have long been studied about the influence of rock mass properties on blasting operations. Bond (1952) proposed for combination which was based on feed size, product size and a rock property factor.

Bond's theory is a compromise between Rittinger's and Kick's theories and is generally recognized to be the best model to describe blasting operations (Da Gama, 1983). McKenzie

(1966) found, in the studies at Quebec Cartier Mines, that the efficiency of all the subsystems is dependent on the fragmentation. Da Gamma (1983) encouraged for blast prediction to engineers understanding the role of in-situ rock mass geometry in terms of block sizes in mine production. Estimating equations of the undersize fragment percentage were developed by Da Gamma and Jimeno (1993). Jurgensen and Chung (1987) and Singh (1991) also opined that the blast results were influenced directly by the overall formational strength of rock. Hagan (1995) concluded that the results of rock blasting were affected more by rock properties than by any other variables.

Pal Roy and Dhār (1996) proposed a fragmentation prediction scale based on the joint orientation with respect to bench face. Scott (1996) reported that the blast-controlling rock mass properties include the strength parameters, the mechanical properties like modulus of elasticity, Poison's ratio, shock wave transmission capability, the size and the shape of the natural block and the required fragment size reduction by blasting. Chakraborty et al. (2002) found the joint orientations can considerably influence the average fragment size and shape. Thornton et al. (2002) categorized the parameters influencing fragmentation in three groups like; (i) rock mass properties, (ii) blast geometry and (iii) explosive properties. Hamdi and Mouza (2005) studied a methodology for rock mass characterization and classification to improve blast results. They aimed the characterization of the two rock mass components which are discontinuity network and rock matrix. The discontinuity network was described using the 3D stochastic simulations of discontinuity networks using the SIMBLOC program methodology.

In this paper, operating costs of 103 blocks for each stage of the drilling, blasting, loading, hauling and bulldozers work in waste zone of zinc and lead Angouran mine which contains schist and limestone, is calculated separately. Then according to results, effect of geotechnical parameters and engineering properties of the rock mass on the total cost of exploitation (mining operation) is investigated. Finally the sensitivity of geotechnical parameters on changes of mining operation costs is assessed.

2. Classification of rock mass in open pit mines

Rock mass comprises several different rock types and is affected by different degrees of fracturing in varying stress condition. A number of rock mass classifications have been developed for Geo-technical purposes like Q-Index (NGIQ; Barton et al 1974), Rock Mass Rating (RMR; Bieniawski et al 1973), etc. The rock mass rating system (RMR) was presented by Bieniawski. The purpose of this system is determining the engineering properties of the rocks in the shallow tunnel that was excavated in sedimentary rocks. The rock mass classification system is one of the best ways to investigate stability and determine the support system in open pit mines. In addition, in 1974, the system of rock mass quality (Q), by Barton et al was presented for the classification of hard rock tunnels and Caverns with curved roof has been recommended and used for the design and maintenance of underground excavation (Hari & Sharma, 2011).

In 1985, Romana introduced slope mass rating (SMR), for evaluating rock slope stability (Hudson, 1993). In 1989, Bieniawski presented the modified classification of the rock mass rating (MRMR), with changes in RMR system. Geological strength index (GSI) presented by Hook and Brown (1997) that was the classification of the rock mass as an observation takes place and Usage gradients and is designed excavating in rock (Hari & Sharma, 2011).

The SMR system provides adjustment factors, field guidelines and recommendations on support methods which allow a systematic use of geomechanical classification for slopes (Hudson, 1993). The adjustment rating for joints in this system represents the joint conditions in the study area. Due to the influence of rock mass properties and discontinuity structure in SMR, using this method is considered later in this paper.

2.1. Slope Mass Rating (SMR)

The proposed 'Slope Mass Rating' (SMR) is obtained from RMR by subtracting a factorial adjustment factor depending on the joint-slope relationship and adding a factor depending on the method of excavation (Hudson, 1993):

$$SMR = RMR + (F_1 \cdot F_2 \cdot F_3) + F_4 \quad (1)$$

TABLE 1

Bieniawski [4] Ratings for RMR (Hudson, 1993)

Parameter	Ranges of values							
Strength of intact rock Material (MPa)	Point load index	> 10	4-10	2-4	1-2	For this low range uniaxial compressive test in preferred		
	Uniaxial compressive	>250	100-250	50-100	25-50	5-25	1-5	<1
	Rating	15	12	7	4	2	1	0
Rock quality designation (%)		90-100	75-90	50-75	25-50	<25		
	Rating	20	17	13	8	3		
Spacing of discontinuities (m)		>2	0.6-2	0.2-0.6	0.06-0.2	<0.06		
	Rating	20	15	10	8	5		
Condition of discontinuities		Very rough surfaces. Not discontinuous. No separation. Unweathered wall rock	Slightly rough surfaces. Separation < 1 mm. Slightly weathered walls	Slightly rough surfaces. Separation < 1 mm. Highly weathered walls	Slickensided surfaces or gouge < 5 mm thick or separation 1-5 mm. Continuous.	Soft gouge > 5 mm or separation > 5 mm. Continuous		
	Rating	30	25	20	10	0		
Groundwater in joints		Completely dry	Damp	Wet	Dripping		Flowing	
	Rating	15	10	7	4		0	

The RMR (see Table 1) is computed according to Bieniawski's 1979 proposal, adding rating values for five parameters: (i) strength of intact rock (UCS); (ii) Rock quality designation (RQD); (iii) spacing of discontinuities (SP); (iv) condition of discontinuities (CD); and (v) water inflow through discontinuities (GW). RMR has a total range of 0-100.

The adjustment rating for joints (see Table 2) is the product of three factors as follows:

- (i) F_1 depends on parallelism between joints and slope face strikes. Its range is from 1.00 (when both are near parallel) to 0.15 (when the angle between them is more than 300 and the failure

probability is very 10o). These values were established empirically, but afterwards were found to approximately match the relationship:

$$F_1 = (1 - \sin A)^2 \quad (2)$$

where A denotes the angle between the strikes of the slope face and the joint.

- (ii) F_2 refers to joint dip angle in the planar mode of failure. In a sense it is a measure of the probability of joint shear strength. Its value varies from 1.00 (for joints dipping more than 45°) to 0.15 (for joints dipping less than 200). Also established empirically, it was found afterwards to match approximately the relationship:

$$F_2 = \tan^2 \beta \quad (3)$$

where β denotes the joint dip angle. For the toppling mode of failure F_2 remains 1.00.

- (iii) F_3 reflects the relationship between the slope face and joint dip. Bieniawski's 1976 figures have been kept. In the planar mode of failure F_3 refers to the probability that joints 'daylight' in the slope face. Conditions are fair when slope face and joints are parallel. When the slope dips 100 more than joints, very unfavorable conditions occur.

For the toppling mode of failure, unfavorable or very unfavorable conditions cannot happen in view of the nature of toppling, as there are very few sudden failures and many toppled slopes remain standing. The Goodman-Bray [10] condition has been used to evaluate toppling probability, with the hypothesis that this failure is more frequent in weathered slopes and there is a small reduction (around 50) of shear strength due to rotational friction, as proposed by Goodman [11].

The adjustment factor for the method of excavation (see Table 3) has been fixed empirically as follows:

- (i) Natural slopes are more stable, because of long time erosion and built-in protection mechanisms (vegetation, crust desiccation, etc.): $F_4 = +15$.
- (ii) Presplitting increases slope stability for half a class: $F_4 = \pm 10$.
- (iii) Smooth blasting, when well done, also increases slope stability: $F_4 = \pm 8$.
- (iv) Normal blasting, applied with sound methods, does not change slope stability: $F_4 = 0$.
- (v) Deficient blasting, often with too much explosive, no detonation timing and/or nonparallel bores, damages stability: $F_4 = -8$.
- (vi) Mechanical excavation of slopes, usually by ripping, can be done only in soft and/or very fractured rock, and is often combined with some preliminary blasting. The plane of slope is difficult to finish. The method neither increases nor decreases slope stability: $F_4 = 0$.

A tentative description of the SMR classes is given in Table 4 (Hudson, 1993).

TABLE 2

Adjustment Rating for Joints (Hudson, 1993)

Case	Very favorable	Favorable	Fair	Unfavorable	Very unfavorable	
1	2	3	4	5	6	7
P	$A = \alpha_j - \alpha_s$	$>30^\circ$	30-20°	20-10°	10-5°	5°
T	$A = \alpha_j - \alpha_s - 180 $	$>30^\circ$	30-20°	20-10°	10-5°	5°
P/T	$F_1 = (1 - \sin A)^2$	0.15	0.40	0.70	0.85	1.00

1	2	3	4	5	6	7
P	$ \beta_j $	$<20^\circ$	$20-30^\circ$	$30-35^\circ$	$35-45^\circ$	45°
P	$F_2 = \tan^2\beta$	0.15	0.40	0.70	0.85	1.00
T	$F_2 = \tan^2\beta$	1	1	1	1	1
P	$\beta_j - \beta_s$	$>10^\circ$	$10-0^\circ$	0°	0° to -10°	$<-10^\circ$
T	$\beta_j - \beta_s$	$<110^\circ$	$110-120^\circ$	$>120^\circ$	—	—
P/T	$F_3 = \beta_j - \beta_s$	0	-6	-25	-50	-60

P: plane failure; T: toppling failure; α_j : joint dip direction; α_s : slope dip direction; β_j : joint dip; β_s : slope dip.

TABLE 3

Adjustment Rating for Methods of Excavation of Slopes (Hudson, 1993)

Method	Natural Slope	Presplitting	Smooth blasting	Blasting or mechanical	Deficient blasting
F4	+15	+10	+8	0	-8

TABLE 4

Tentative Description of SMR Classes (Hudson, 1993)

Class	SMR	Description	Stability	Failures	Support
I	81-100	Very good	Completely Stable	None	None
II	61-80	Good	Stable	Some blocks	Occasional
III	41-60	Normal	Partially stable	Some joints or many wedges	Systematic
IV	21-40	Bad	Unstable	Planar or big wedges	Important/corrective
V	0-20	Very bad	Completely unstable	Big planar or soil-like	Reexcavation

3. Angouran Mine, geotechnical properties, mining operation cost

Angouran lead and zinc open-pit mine with production capacity of 800 thousand tons per year and the remaining amount potential over 12 million tons with the average grade 3%-6% of lead and the average grade 25%–30% of zinc is one of the largest metal mines in Iran and also is one of the most economical lead and zinc mines in the world. Angouran mine is located in Zanjan province, 125km SW of Zanjan, and in a region with an average altitude of 3000 m. The geographical location of Angouran mine has been shown in Figure 1. The ore body is located between a limestone layer as hanging-wall and on a thick layer of schist as foot-wall. The mine was designed with an overall slope dip angle about 45° in waste and 35° in the ore zone (Behbahani, Moarefvand, Ahangari, & Goshtasbi, 2013). In Angouran mine, rocks in foot-wall are collections of metamorphic schist with an approximately thickness of 1000 m that mostly these collections have extended in the west of the mine, while rocks in hanging-wall are collections of semi-metamorphic limestone with an approximately 200 m thickness that the most extension of these collections are in the east of the mine. Limestone existing in hanging-wall has RQD near to 60% and GSI between 42 and 52 and also its UCS is approximately 75MPa. But schist which exists in foot-wall has almost RQD 51% and GSI 50; furthermore, its UCS is between 50 and 100MPa (Moarefvand & Ahmadi, 2009).



Fig. 1. (a) Geographical location of Angouran mine; (b) Zinc-lead Angouran mine

According to the mining experts and geologists' reports, geological formation containing the zinc and lead Angouran mine is Sanandaj-Sirjan belt formation. This formation was originally part of Central Iran and this formation called to the various titles such as Oromieh-Esfandaghe zone, Interior's Zagros, and finally Sanandaj-Sirjan belt.

Blast holes with an average diameter of 4.5 inches are drilled in zinc and lead Angouran mine and drilled networks typically have dimensions of 3.5×4.5 inches. The most explosive substance in this mine and its special funds medium are Ammonium-Nitrate-Fuel Oil (ANFO) and 565 g/m^3 respectively. Loading is the first step of transportation which takes place immediately after the blasting and is of particular importance. Loading in mine of Angouran mainly is done using excavator. Each device on average 160 cubic meters per hour loads the waste. Cargo is carried by 34 dump trucks with capacity from 32 to 60 tons.

3.1. Geotechnical characteristics of rock mass in Angouran mine

In order to accomplish the aim of the article, 103 extractive blocks which major mining activities including drilling, blasting, loading and hauling are done in them were selected. Studied blocks are shown in Figure 2. These blocks are located within mine waste area which are formed from schistose rocks (Schist of Footwall) and limestone (hanging wall limestone). Hanging wall limestone are a metamorphic thick limestone layer which cover ore of Angouran and are developed from north and eastern north to western south of mine. Footwall of Angouran is kind of green schist which due to containing a large amount of group minerals of mica is called mica schist. These rocks are mainly formed of quartz, feldspar and mica. This schist is the lowest part of the ore deposits. Ore deposit Zone is enclosed between the two sections of limestone hanging wall and footwall zones of schistose material.

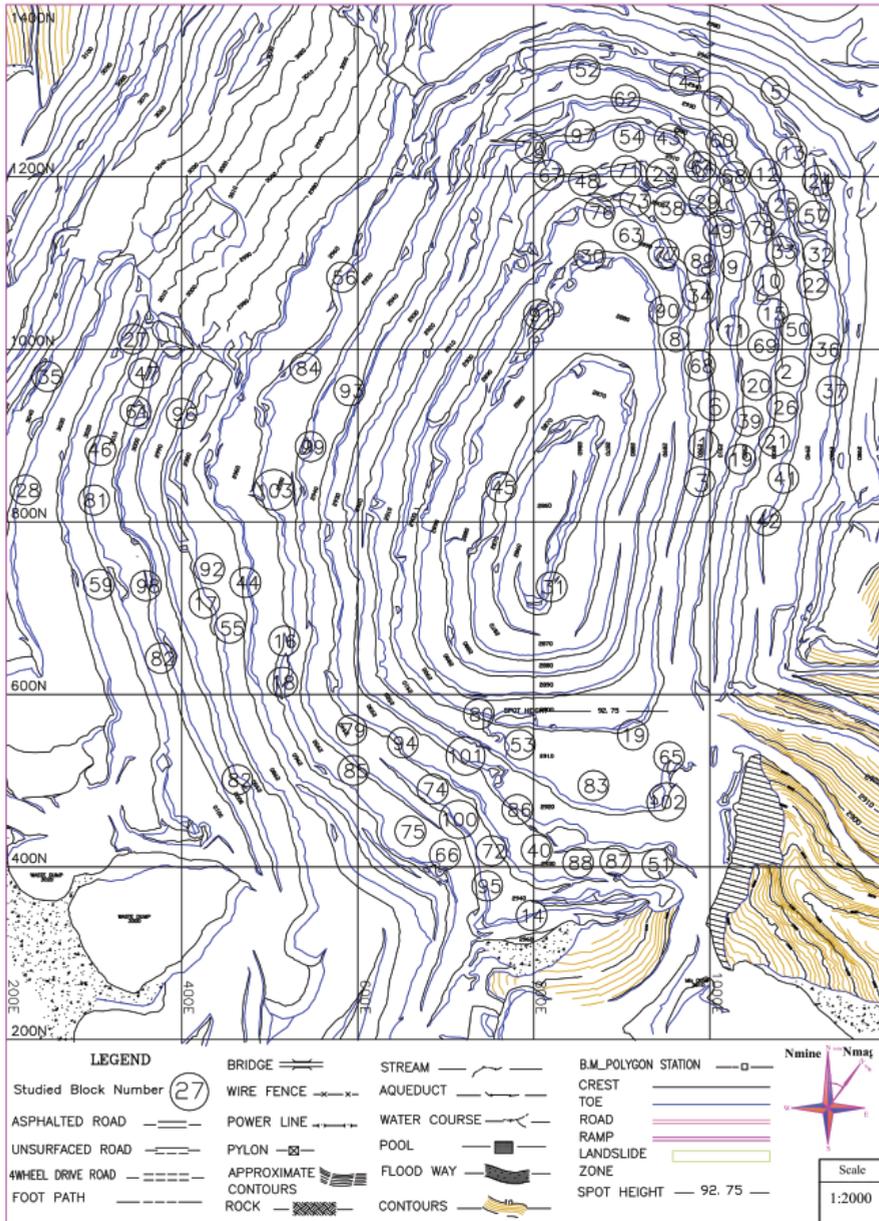


Fig. 2. The position of 103 studied blocks in Zinc-lead Angouran mine

The results of performed experiments to determine rock strength variables according to the Mining Company are given in Table 5.

According to deductions which were taken in Angouran mine a layered plate and two principal joint sets can be identified. Table 6 shows the characteristics of discontinuities in Angouran mine.

TABLE 5

Geomechanical properties of in situ formations of limestone and schistose (Moarefvand & Ahmadi, 2009)

Type of formation	geomechanical properties of in situ Schistose formations			Geomechanical properties of in situ limestone		
	Intact Rock	$\phi = 37^\circ$ $\varphi = 14^\circ$	C = 3.5 MPa m = 0.183	$\sigma_C = 50$ MPa S = 0.0009	$\phi = 52^\circ$ $\varphi = 14^\circ$	C = 6 MPa m = 0.29
Rock Mass	$\phi = 37^\circ$ $\varphi = 14^\circ$	C = 3.5 MPa m = 0.183	$\sigma_C = 50$ MPa S = 0.0009	$\phi = 24^\circ$ $\varphi = 14^\circ$	C = 1 MPa m = 0.29	$\sigma_C = 100$ MPa S = 0.0003

TABLE 6

Characteristics of discontinuities in Angouran mine

Discontinuity	Dip (°)	Azimuth (°)	Slope Orientation (°)	Joint Spacing (Cm)	Filling of Joint	Separation (Cm)
Bedding	10-15	N14W	To the east	40	Calcite cement	locally
Joint set 1	75	N55E	To the southeast	35	The local clay	1
Joint set 2	75	N18W	To the west	25	—	2

Components of 8-fold classification system SMR (UCS , RQD , SP , CD , GW , F_1 , F_2 , F_3) were deducted from the study case of mine blocks (Fig. 2) using field survey. Table 7 shows resulted variables as examples for five blocks.

TABLE 7

Rating of SMR index for 103 studied blocks of Angouran mine

No.	UCS	RQD	SP	CD	GW	F_1	F_2	F_3	SMR	Class of SMR
1	12	17	11	8	15	0.57	0.72	0	55	III
2	7	13	9	8	15	0.15	0.49	0	44	III
3	12	17	12	8	15	0.51	0.72	6	53.8	III
4	7	17	13	0	15	0.3	0.72	25	38.6	IV
5	12	17	12	17	15	0.16	0.72	25	62.1	II

Since this case study is limited for two types of limestone and schist, the first component (UCS), will have only two amounts which would be in role of excavated stope face separator for type of rock.

3.2. The cost of mining operations at the Angouran mine

To calculate the total cost of mining operation for each stope block, equation (4) is used:

$$C_{Total_i} = C_{Di} + C_{Bi} + C_{Li} + C_{bi} + C_{Hi} \quad (4)$$

where C_{Total_i} is the total cost of mining operation at stope face of number i in USD per in situ cubic meter. For exploitation of each cubic meter in situ rock in the same stope face, it obtains from

total cost of operations of drilling (C_{Di}), blasting (C_{Bi}), loading (C_{Li}), bulldozers work (C_{bi}) and hauling (C_{Hi}). As an example, the results of calculations for the five blocks are shown in Table 8.

TABLE 8

Calculation results of mining operation cost for 103 studied blocks

The cost of the various stages of mining operations for each Studied block(USD per in situ cubic meter)									
No.	X	Y	Z	Drilling	Blasting	Loading	Hauling	Bulldozers work	Total Cost
1	1105	975	2930	0.48	0.8	0.81	1.62	0	3.72
2	995	1125	2900	0.3	0.63	0.73	1.35	0.31	3.32
3	1069	1076	2920	0.28	0.63	0.52	1.11	0.04	2.58
4	1075	1300	2960	0.23	0.8	0.65	1.62	0.13	3.43
5	1034	876	2910	0.39	0.69	1.05	2.08	0	4.21

4. Influence of geotechnical parameters on the cost of exploitation

To describe the relationship between geotechnical parameters and mining operations costs, multivariate linear regression method is used (equation (5)):

$$\begin{bmatrix} y_0 \\ y_1 \\ \vdots \\ y_n \end{bmatrix} = \begin{bmatrix} 1 & x_{11} & \cdots & x_{1k} \\ 1 & x_{21} & \cdots & x_{2k} \\ \vdots & \vdots & \cdots & \vdots \\ 1 & x_{n1} & \cdots & x_{nk} \end{bmatrix} \times \begin{bmatrix} \beta_0 \\ \beta_1 \\ \vdots \\ \beta_k \end{bmatrix} \quad (5)$$

where X is matrix of point values for each geotechnical parameter, β is matrix of influence coefficients for geotechnical variables, and Y is extractive blocks cost matrix of mining operations.

Geotechnical variables and points (scores) used in this equation are the same classified components of SMR and advantages offered by this method. A sample of this information which is used in equation (5) as input is shown in Table 7. Due to lack of water bearing blocks (only 5 blocks contain water) and the same blasting conditions in all extractive stope face, groundwater variables and blasting conditions (F_4) are not applied to the input matrix. Therefore, the input matrix, contain the following seven components respectively: uniaxial compressive strength (UCS), rock quality designation (RQD), spacing of discontinuities (SP), condition of discontinuities (CD), azimuth difference of joints and Stope Face (F_1), joints dip (F_2), gradient between the joints dip and Stope Face dip (F_3).

Output part of the equation includes the total cost of drilling, blasting, bulldozers work, loading and hauling per exploitation of one cubic meter of the rock mass (Table 8). It has to be noted that the geotechnical characteristics and mining operation cost of 103 blocks are applied at input and output of equation (5). Table 7 and Table 8 only show details of 5 blocks as an example.

4.1. Discussion and investigation of the relationship between the cost of exploitation and geotechnical characteristics

With solving equation (5), the cost of exploitation of blocks is calculated as follows:

$$\begin{aligned}
 COST = & 6.245 - 0.093 \times R_{UCS} + 0.015 \times R_{RQD} + 0.044 \times R_{SP} + \\
 & + 0.005 \times R_{CD} - 0.083 \times R_{F_1} - 1.873 \times R_{F_2} + 0.003 \times R_{F_3}
 \end{aligned}
 \quad (6)$$

where $COST$, R_{UCS} , R_{RQD} , R_{CD} , R_{SP} , R_{F_1} , R_{F_2} and R_{F_3} are total exploitation cost (USD per cubic meter), scores of UCS , RQD , SP , CD , F_1 , F_2 and F_3 , respectively.

Figure 3 shows the actual costs of the mining operations versus the estimated costs by multivariate linear regression. According to the correlation coefficient between the estimated values and real values which is 0.715 and on the other hand, large amounts estimated are in the range of $\pm 20\%$ of real values, it can be concluded that an acceptable relationship exists between geotechnical parameters and total mining operations cost.

As mentioned in Section 3-1, the studied rock types in Angouran mine are schist and limestone. Thus two uniaxial compressive strength of intact rock are inserted in calculations. Hence the uniaxial compressive strength is the separator of extractive rock type in equation (6).

Increased extractive costs in schist rock in compare of limestone which is also confirmed in equation (6) are because of following reasons:

- 1) Inappropriate blasting due to the presence of water in the schistose rocks which results increasing in mining operations,
- 2) Higher specific gravity of schist than limestone which leads to increase in the number of cargo loading and hauling,
- 3) Loss of schist,
- 4) Sticking of drilling, cargo loading and hauling equipments in schist.

According to equation (5), the effects of RQD , SP , and CD on the costs of extraction operations are positive. Increasing of these variables means more integration and resistance of rock against fragmentation operation and therefore increasing in costs of mining operations.

Sign of the coefficient associated with variable effect F_1 on costs of mining operations is negative. Reducing of difference angle between dip direction and stope face direction (alignment of joints and stope face) means an increase in the coefficient of F_1 . Alignment of along joints and stope face minimizes escape of blasting gas and causes optimization of blasting performance and costs reduction of exploitation.

Due to higher length of blocks than their heights (length of block $\gg 5 \times$ width of blocks), trace length of joints with low dip in compare of joints with high dip in crop of stope face increases which is a factor in the loss of blasting gas, weak (poor) fragmentation and thereby increasing the cost of exploitation in the low dip joints. Eventually, increasing in the difference angle between joints dip and stope face dip causes orientation of joint direction to inward of the stope face and thus reduces the ability of fragmentation as Lilly (1976) and Mumivand (2006) have reported before. Decreasing of fragmentation ability and its negative consequent in mining operations leads to increasing of exploitation costs which is validated in equation (6).

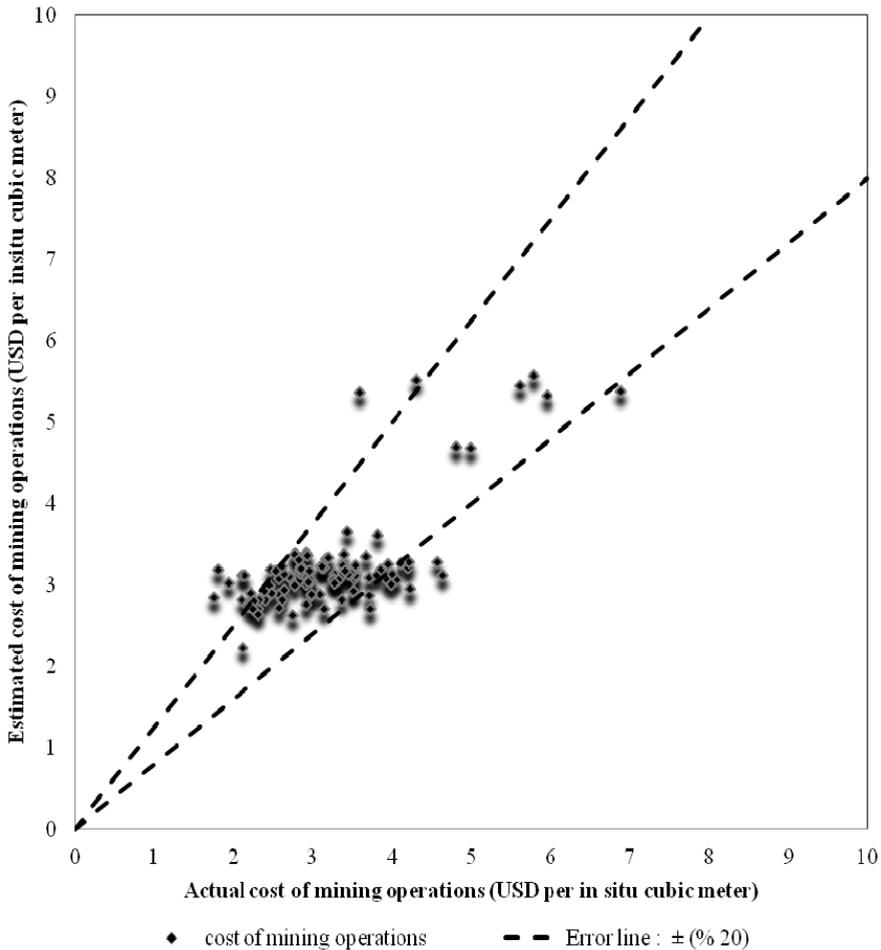


Fig. 3. Actual and estimated costs of mining operation in Angouran mine

4.2. Sensitivity analysis

In this subsection, the amount of change in the exploitation costs versus change of every single geotechnical parameters is studied and therefore the parameters effect is evaluated. Equation (7) for sensitivity analysis on effects of geotechnical parameters in mining operations at zinc and lead Angouran mine is used (utilized) (Kulatilake, Wang, & Song, June, 2012).

$$\Delta C_E = \frac{(C_{T_{\max}} - C_{\text{med}}) / C_{\text{med}}}{(R_{\max} - R_{\text{med}}) / R_{\text{med}}} \quad (7)$$

where ΔC_E , $C_{T_{\max}}$, C_{med} , R_{\max} , R_{med} are percentage change in average cost of exploitation for the parameter change, total cost of exploitation for the maximum value of the parameter of the cost

(other geotechnical parameters remain in the average value), the cost for the average value of the parameters, the maximum value of the parameter of the cost, average value of the parameter, respectively.

According to the calculations, the maximum effect of increasing mining operations costs is the joint dip parameter (F_2). Other effective variables after joint dip are: joint spacing, uniaxial compressive strength of intact rock, rock quality index, the gradient between the joints dip and stope face, condition of discontinuities and angle differences between along of joints and along of the stope face, respectively.

5. Conclusion

- The analysis and assessment of costs, as one of the most important stages of cost calculation and management, helps to improve the operational effectiveness. In this paper, the operating costs of mining in waste zone of Angouran mine has been analyzed against geotechnical parameters.
- The formula (Equation 6) to determine the relationship between geotechnical parameters and operating cost using multivariate linear regression was proposed. The proposed equation is limited to limestone and schist zone of Angouran mine. Correlation coefficient of the proposed equation and what is done in practice, is 0.715 which represents the acceptable relationship between geotechnical parameters spending mining operations.
- In order to determine the effect of geotechnical parameters on the operating cost, sensitivity analysis based on percentage change of cost for changing of each parameter was conducted. The results show a significant effect of joints dip on the final performance of mining.
- The proposed model can be used to predict the excavation cost in different parts of the mine, in advance. This helps to improve the mining plan and excavation direction to reduce the operating costs.
- It is recommended to use the mentioned method in other zinc and lead mines and mining industries to develop a general model for determining the operating costs versus geotechnical parameters.

Acknowledgments

Collaboration of the IMPASCO Company is gratefully acknowledged. Many thanks also to Mr. Karegar for his help to access the required database.

References

- Barton N., Lien R., Lunde J., 1974. *Engineering Classification of Rock Masses for the Design of Tunnel Support*. Rock Mech., 6(4): 189-236.
- Behbahani S.S., Moarefvand P., Ahangari K., Goshtasbi K., 2013. *Unloading scheme of Angooran mine slope by discrete element modeling*. International Journal of Rock Mechanics & Mining Sciences, 220-227.
- Bieniawski Z. 1973. *Engineering Classification of Jointed Rock Mass*. Trans. S. Afr. ICE, 15: 335-344.
- Bond F.C., W. B., 1959. *The Work Index in Blasting*. Quarterly of the Colorado School of Mines, 54 (3): 77-82.

- Bond F. 1952. *The Third Theory of Comminution*. Transactions of the American Institute of Mining Engineers, 193: 494-496.
- Chakraborty A., Raina A., Ramulu M., Choudhury P., Haldar A., Sahu P., et al., 2002. *Development of Innovative Models for Optimisation of Blast Fragmentation and Muck Profile Applying Image Analysis Technique and Subsystems Utilisation Concept In Indian Surface Coal Mining Regime*. Project No. MT/103, for Ministry of Coal, Govt. of India.
- Da Gama D. 1983. *Use of Comminution Theory to Predict Fragmentation of Jointed Rock Mass Subjected to Blasting*. Proc. First Int. Symp. on Rock Frag. By Blasting, Lulea, Sweden, 563-579.
- Da Gamma D., Jimeno C., 1993. *Rock Fragmentation Control for Blasting Cost Minimisation and Environmental Impact Abatement*. Proceedings of 4th. International Symposium On Rock Fragmentation By Blasting, July 5-8. Balkema, Vienna, Austria, 273-279.
- Fuksa D., 2013. *Concept of determination and analysis of the Break-Even Point for a mining enterprise*. Arch. Min. Sci., Vol. 58, No 2, p. 395-410.
- Hagan T., 1995. *The Effect of Rock Properties on the Design and Results of Tunnel Blasts*. J. Rock Mech. Tunnelling Technol., 25-39.
- Hamdi E., Du Mouza J., 2005. *Methodology for Rock Mass Characterisation and Classification to Improve Blast Results*. Int. J. Rock Mech. Min. Sci., Vol. 42, 175-194.
- Hari D., Sharma S. 2011. *Monograph on Rock Mass Classification Systems and Applications*. Central Soil and Materials Research Station, Olof Palme Marg, Outer Ring Road, Hauz Khas, New Delhi.
- Hudson J.A., 1993. *Rock testing and site characterization, A Geomechanical Classification for Slopes: Slope Mass Rating*. [In:] J.A. Hudson, Comprehensive rock engineering. London, UK: Imperial College of Science, Technology & Medicine, Pergamon press, Oxford.
- Jonek Kowalska I., 2013. *Introduction to a Methodology of Cost Management in a colliery from a perspective of longwall life cycle*. European Scientific Journal, edition vol. 9, No. 25, 380-402.
- Jonek-Kowalska I., 2012. *Cost analysis and assessment in the polish hard coal mining industry in years 2006-2011*. Journal of Interdisciplinary Research, Silesian University of Technology, ul. Roosevelta 26, 41-800 Zabrze, Poland, 35-40.
- Jurgensen G., Chung S., 1987. *Blast Simulation Surface and Underground with SABREX Model*. CIM Bul., 80(904): 37-41.
- Kulatilake P., Wang X., Song W.D., 2012. *Stability investigations around a mine tunnel through three-dimensional discontinuum and continuum stress analyses*. China, Elsevier, Tunnelling and Underground Space Technology, Vol. 32, 98-112.
- McKenzie A., 1966. *Cost of Explosives - Do You Evaluate it Properly?* Mining Congress J., 32-41.
- Moarefvand P., Ahmadi M., 2009. *Report of unstable mass condition in angouran mine after collapse occurrence*. Zanjan: Madan Zamin construction company [inPersian].
- Moarefvand P., Ahmadi M., Afifipour M., 2012. *Unloading scheme to control sliding mass at Angouran open pit mine, Iran*. [In:] Qian, Zhou, editors. Harmonising, rock engineering and the environment. London: Taylor & Francis; 1963-6.
- Monjezi M., Farzaneh F., Asadi A., 2013. *Evaluation of Blasting patterns using operational research models*. Arch. Min. Sci., Vol. 58, No 3, p. 881-892.
- Pal Roy, P., & Dhar, B. 1996. *Fragmentation Analyzing Scale - a New Tool for Breakage Assessment Fragmentation By Blasting - Fragblast 5*, Montreal, 25-29 Aug. Balkema, Rotterdam. Proc. 5th. Int. Symp. On Rock, p 448.
- Scott A., 1996. *Blastability and Blast Design*. Proc. 5th International Symp. on Rock Fragmentation By Blasting, Montreal, Canada, 25-29, Sept. Colorado School of Mines, Colorado, Balkema, Rotterdam, Netherlands, 27-36.
- Singh D., 1991. *Effect of Physico-Mechanical Properties of Rocks on Drilling and Blasting Operations in Underground Drivage*. Workshop on Tunnels, Mine Roadways and Caverns, Ooty, September, Colorado School of Mines, Colorado, Balkema, Rotterdam, Netherlands.
- Thornton D., Kanchiboltsa S., Brunton I., 2002. *Modelling the Impact and Blast Design Variation on Blast Fragmentation*. The International Journal for Blasting and Fragmentation. 6(2) Swets and Zeitlinger, The Netherlands, 171-172.