Journal of Geology and Mining Research Vol. 2(3) pp. 48-59, June 2010
Available online http://www.academicjournals.org/jgmr
ISSN 2006 – 9766 © 2010 Academic Journals

Full Length Research Paper

Selection of practical bench height in open pit mining using a multi-criteria decision making solution

Hossein Soltanmohammadi[1]*, Morteza Osanloo, Ali Sami[2] and Shahnam Behnam Malekzadeh[1]

Department of Mining Engineering, Aryakanyar Consulting Engineers (AKCE), Amirkabir University of Technology, Tehran, Iran.
Department of Mining Engineering, Amirkabir University of Technology, Hafez St., Enqelab Ave., Tehran, Iran.

Accepted 11 May, 2010

Determination of practical bench height is an important subject in open pit mining. This subject has always been an issue with different and sometimes conflicting criteria that have to be precisely considered during the mine design process. In this study a multi-expert multi-criteria decision making approach is used to resolve these complexities. In the proposed approach, different bench heights are firstly analyzed considering the variety of criteria such as production scheduling, dilution, costs, practicability, safety, and equipment availability. The practicability analysis is consisted of a primary sequencing method developed to compare total time needed for all bench height alternatives to reach the constant annual production. Once the criteria are weighted according to judgments by expert team, the obtained performance scores are passed to a multi-criteria model called VIKOR (multi-criteria optimization and compromise solution) to introduce the optimum alternative. This approach was utilized for a simple example with two alternatives, where the obtained results confirmed its efficiency.

Key words: Open pit mining, bench height, multi-criteria decision making.

INTRODUCTION

As a definition in open pit mining, bench height is the vertical distance between crest and toe of the bench (Fourie and Dohm, 1992). Determination of optimum bench height is a major concern in most open pit mines. It depends on various factors, such as the cutting height and the bucket capacity of the loading machines (Hustrulid and Kutcha, 1998), capacity of drilling machines, rock properties, geological characteristics of ore reserve, production parameters such as, hole diameter and road grade (Kose et al., 2005), necessity of sequencing and selective extraction, total amount of production, and pit slope stability.

In general, some advantages of designing open pit mines with higher benches can be mentioned as follows (Li, 1995):

(i) Less numbers of machinery will be utilized that are larger in size and have more capacity. Larger machinery means more productivity and efficiency and less volume of traffic;
(ii) Less time is required for set up and maintenance of equipment;
(iii) Supervision on all the operations will be more practical;
(iv) Blasting of greater blocks is possible, and as a result; more production is yielded from each level, while less number of blasts are executed.

There are also some disadvantages with utilizing higher benches (Li, 1995):

(i) Capability for selective extraction is decreased;
(ii) Dilution is increased;
(iii) Work space and as a result, flexibility of operation is decreased for the machinery;
(iv) Safety issues will be more serious.

Once the consequences of facts mentioned above are noticed well on advantages and disadvantages of greater

*Corresponding author. E-mail: hsm_mine@yahoo.com. Tel: +98-919-4351597.

Abbreviations: MRMR, Modified rock mass rating; **UCS,** uniaxial compressive strength; **RQD**, rock quality designation.

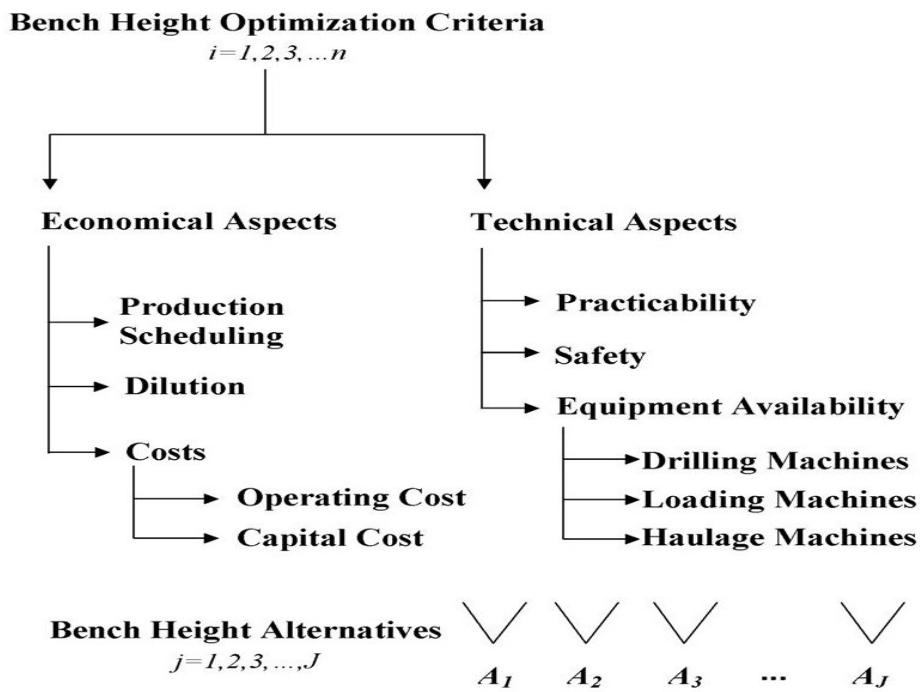

Figure 1. Hierarchical structure suggested for bench height optimization criteria.

or smaller bench heights, the significance of decision making on optimum height for benches of an open pit mine from viewpoint of multiple criteria and experts will be explicitly perceived. Each one of these criteria and experts have different performance scores and weights in the decision making process.

This study employs the VIKOR (Vlse Kriterijumska Optimizacija I Kompromisno Resenje in Serbian) method, presented by Opricovic and Tzeng (2002; 2004; 2007) which is a compromise ranking method to optimize aggregation of different criteria scores of different bench heights simultaneously. The criteria scores and weights will be obtained from field quantitative observations and questionnaires sent to the most related experts. The result of this approach will be a consensual bench height alternative that has placed in the first rank of the VIKOR compromise ranking solution.

Establishing bench height optimization criteria

The criteria distinguished as most influential have been categorized in two economical and technical clusters of the hierarchy shown in Figure 1. The following sections discuss how these criteria can affect the optimum bench height.

Production scheduling

A bench height alternative is the most suitable from production scheduling point of view, when the grade variability of the produced ore from the designed push-backs is the slightest. This criterion can be expressed as standard deviation for ore grade of different push-backs of the pits designed with alternative bench heights, while the other parameters such as overall stripping ratio and total annual production remained constant. The standard deviation (*SD*) of ore grade can be calculated by Equation (1). This statistical equation is famous in mathematics as Bessel's correction (Reichmann, 1961):

Equation (1): $$SD = \left[\sum_{i=1}^{n} \frac{(G_i-\mu)^2}{(n-1)}\right]^{\frac{1}{2}}$$

Where, n is the number of push-backs $i=1,2,\ldots n$, G_i is the mean value of ore grade for push-back i, and is the expected mean value for ore grade during mine life cycle.

Dilution

The bench height also has an impact on the recovery of ore and therefore, dilution and ore loss should be precisely estimated for practical bench geometries. This estimation can be made through conditional simulation introduced by Glacken et al. (2000); where, the calculation of dilution and ore loss percentages for various ore-zones, cut-off grades, and simulation *scenarios*, can appropriately show the sensitivity of the ore-zones to bench heights and cut-off grade combinations.

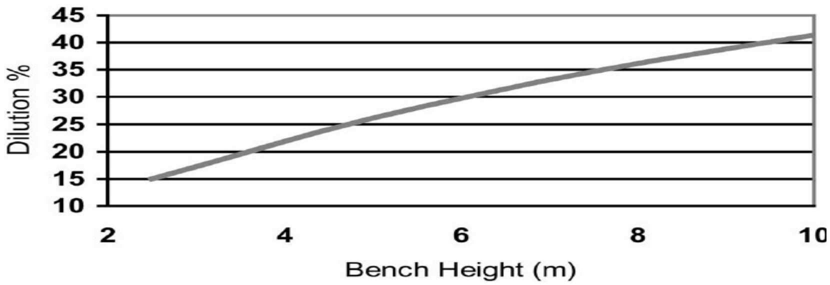

Figure 2. Dilution versus bench height (Bozorgebrahimi et al., 2005).

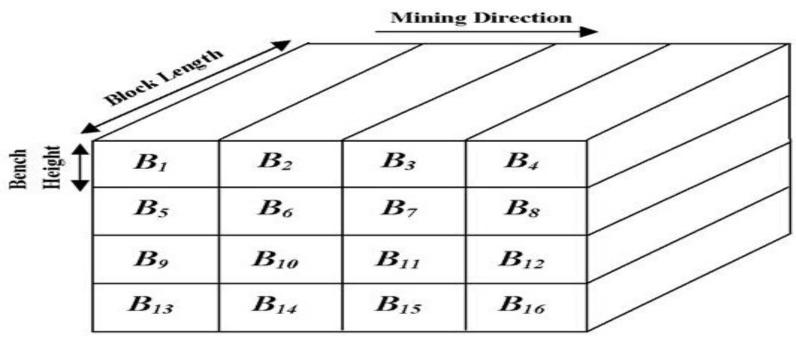

Figure 3. Large block of the simple sequencing method.

In open pit mining, factors that can affect the degree of dilution can be regarded as deposit-related or mine-related. The bench height is a mine-related parameter that can be studied using a simple geometrical model of the deposit. By changing this parameter individually, while the deposit-related parameters (such as ore distribution, ore slope and ore thickness), and mine-related parameters (such as the mining method, the mine geometry, the mining direction, the equipment size and the skill of the operators) are kept constant, the sensitivity of dilution to bench height parameter can be calculated by the following equation (Bozorgebrahimi et al., 2005):

Equation (2):

$$D = \frac{W}{(W+O)} \times 100$$

Where, D is dilution (%), W is tones of waste and O is tones of ore. It has been proved that, greater bench heights will result in poorer dilution control and reduced selectivity, especially for non-homogenous deposits. Figure 2 shows that for a particular ore body where other ore geometry parameters are constant, increasing the bench height results in an increase in dilution.

Costs

The costs of a mine can be divided into two main portions, namely, the capital cost including equipment acquisition and the operating cost which consists mainly of labour, maintenance and consumables like electricity, explosives, fuel, etc. However, the quantity, size, and power of the ordered machinery have to be modified as the bench geometry is changed due to different bench heights. This modification has consequently a direct impact on the capital and operating costs of the mine (Roman and Daneshmend, 2000). These costs are considerably predictable and can be expressed as a currency unit per tons of extracting ore.

Practicability

Taking into consideration the inevitability of oversimplifications, a method called simple sequencing, is presented by the authors to evaluate practicability of alternative bench heights in the final pit design. In the proposed method, the ore reserve is entirely considered as a hypothetical large block with constant dimensions as illustrated in Figure 3. The large block is divided into a number of small blocks which have equal widths, lengths and heights. The large block and small blocks are equal in length; width of the small blocks should not be more than length of a regular blasting block, and height of the small blocks will match the considered bench height. In order to extract each small block shown in Figure 3, the upper and adjacent small blocks should have been extracted in previous sequences to ensure minimum required operational space.

The minimum required time cycle for "drilling and blasting" and "loading and haulage" operations of a regular blasting block can be measured from similar projects running in the area with different bench heights. These measurements then are generalized and accepted as identical for critical time cycles of the operations on small blocks of the simple sequencing method. In the proposed method, the least cumulative critical operational time $T_{Critical}$ for all the sequences $i=1,2,\ldots n$, means the most practical bench height:

Equation (3): $$T^{i}_{Critical} = \left\{max(T^{i}_{D\&B}, T^{i}_{L\&H}) \middle| i = 1,2,\ldots n\right\}$$

Equation (4): $$T_{Critical} = \sum_{i=1}^{n} T^{i}_{Critical}$$

Where, $T^{i}_{D\&B}$ and $T^{i}_{L\&H}$ are, respectively, the critical times for "drilling and blasting" and "loading and haulage" in sequence *i*.

Safety

Every bench in open pit mines composes a slope which its stability should be guaranteed to ensure safety of the entire pit. Today, the factor of safety is the most common measure of slope stability, and there is wide experience in its application to all types of geological conditions (Wyllie and Mah, 2005). For open pit mines, the factor of safety generally used is in the range of 1.2 - 1.4. The limit equilibrium analysis is usually used to calculate the factor of safety *FS* as Equations (5 and 6) (Wyllie and Mah, 2005):

Equation (5): $$FS = \frac{resisting\ forces}{driving\ forces}$$

Equation (6): $$FS = \frac{cA + W cos\psi_p tan\phi}{W sin\psi_p}$$

Where, the rock is assumed to be a Mohr–Coulomb material in which the shear strength is expressed in terms of the cohesion *c* and friction angle φ. ψ_p is the dip of the sliding surface, *A* is its area, and *W* is the weight of the block lying above the sliding surface.

Equipment availability

It is clear that, the bench geometry has a direct influence on availability of the equipment. The equipment availability can be estimated for different alternative bench heights of an open pit mine in which, different types of drilling, loading, and haulage machines are utilized being well suit for the considered bench geometry. As stated by Dhillon (2008), availability is defined as the probability that a piece of equipment is functioning satisfactorily at a specified time, when used according to specified conditions, where the total time includes operating time, logistical time, active repair time, and administrative time. Therefore, the equipment availability *AV* is simply the proportion of time the equipment is able to be used for its intended purpose and is expressed by Equation (7):

Equation (7): $$AV = \frac{(TH - DT)}{TH} \times 100$$

Where, *TH* is the total hours and *DT* is the downtime hours of equipment.

Compromise solution by the VIKOR method

The VIKOR method focuses on ranking and selecting from a set of alternatives, and determines compromise solutions for a problem with conflicting criteria. Opricovic (1998); Opricovic and Tzeng (2002; 2004; 2007) developed VIKOR, which means multi-criteria optimization and compromise solution. This method is based on the compromise programming of Multi-Criteria Decision Making (MCDM).

Decision matrix

A decision matrix (*F*) is constructed at the first step with the following structure:

Equation (8): $$F = \begin{bmatrix} f_{11} & f_{12} & \cdots & f_{1n} \\ f_{21} & f_{22} & \cdots & f_{2n} \\ \vdots & \vdots & \ddots & \vdots \\ f_{J1} & f_{J2} & \cdots & f_{Jn} \end{bmatrix}$$

Where, *J* is the number of alternatives *j*, *n* represents the number of criteria *i*, f_{ji}, indicates the performance score of alternative *j* with respect to criterion *i*. The criteria can be of cost or benefit types in the decision matrix. When a criterion is of benefit type, a larger performance is desired and conversely when a criterion is of cost type, a smaller performance is desired.

Normalized decision matrix

The normalized values r_{ji} can be calculated for benefit criteria by Equation (9):

Equation (9): $r_{ji} = \frac{f_{ji}}{\sqrt{\sum_{j=1}^{J}(f_{ji})^2}}, i = 1,2,\ldots,n; j = 1,2,\ldots,J$

Similarly, the normalized values r_{ji} can be calculated for cost criteria by Equation (10):

$$r_{ji} = \frac{\frac{1}{f_{ji}}}{\sqrt{\sum_{j=1}^{J}\left(\frac{1}{f_{ji}}\right)^2}}, i = 1,2,\ldots,n; j = 1,2,\ldots,J$$

The normalized decision matrix (R) then can be expressed as follows:

Equation (11): $$R = \begin{bmatrix} r_{11} & r_{12} & \cdots & r_{1n} \\ r_{21} & r_{22} & \cdots & r_{2n} \\ \vdots & \vdots & \ddots & \vdots \\ r_{J1} & r_{J2} & \cdots & r_{Jn} \end{bmatrix}$$

Ideal and non-ideal solutions

The ideal solution A^* and the negative ideal solution A^- sets are determined as follows:

Equation (12):

$$A^* = \{\max_j r_{ji} \mid i = 1,2,\ldots,n\} = \{r_1^*, r_2^*, \ldots, r_n^*\}$$

Equation (13):

$$A^- = \{\min_j r_{ji} \mid i = 1,2,\ldots,n\} = \{r_1^-, r_2^-, \ldots, r_n^-\}$$

Criteria weighting

A compromise weighting strategy that can be adopted for VIKOR method is to obtain an aggregated set of criteria weights from questionnaires dispatched to the experts via using a group decision making system. There are different ways for aggregation of individual judgments in group decision making (Forman and Peniwati, 1998; Escobar et al., 2004). In this study, according to Equation (14), weighted geometric mean of individual judgments is suggested to be calculated in order to reach a consensual set of criteria weights.

Equation (14): $$W_i = \prod_{x=1}^{X}(W_i^x)^{W_x}$$

In this Equation, W_i refers to the group judgment on weight of criterion i, W_i^x refers to expert x's judgment on weight of criterion i, W_x is the normalized weight of expert x, and X is the number of experts.

Utility measure

The utility measure (S_j) for each alternative is given as:

Equation (15): $$S_j = \sum_{i=1}^{n} \frac{W_i(r_i^* - r_{ji})}{(r_i^* - r_i^-)}$$

Where, W_i is the weight of the ith criterion.

Regret measure

The regret measure (R_j) for each alternative is given as:

Equation (16): $$R_j = \max_i \left[\frac{W_i(r_i^* - r_{ji})}{(r_i^* - r_i^-)}\right]$$

VIKOR index

The VIKOR index (Q_j) can be expressed as follows:

Equation (17): $$Q_j = v\left[\frac{S_j - \min_j S_j}{\max_j S_j - \min_j S_j}\right] + (1 - v)\left[\frac{R_j - \min_j R_j}{\max_j R_j - \min_j R_j}\right]$$

Where, v is the weight of the maximum group utility and is usually set to 0.5 (Tong et al., 2007). The alternative with the smallest VIKOR index value is determined to be the best compromise solution.

Illustrative example

In this section, a simple example of selecting the practical bench height for a small iron mine, being extracted by open pit mining method, has been taken into consideration, taking into account the existing equipment and capabilities. The purpose is to compare two pit schemes with 10 and 12.5 m bench heights. Therefore, geological block models for the iron mine was provided for both 10 and 12.5 schemes. Total annual production estimated for this mine was 3 million tones iron ore with average anticipated Iron (Fe) grade of 60.11 and also average Phosphorus (P) grade of 0.17 as a constraint.

The project working calendar and the available equipment at the inventory of contractor was controlled following a site investigation. Then, the magnitude and capacity required for drilling, loading and haulage

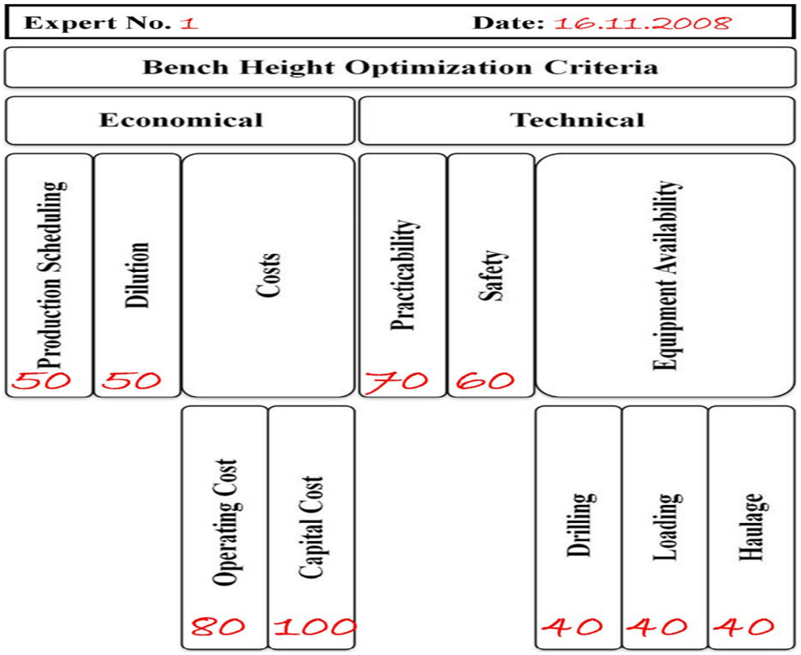

Figure 4. A sample form of questionnaires filled in by experts.

Table 1. Aggregation of ten individual judgments into a normalized weighting set.

	1	2		10	Geometric mean	Normalized weights
Production scheduling	50	60	---	50	57	0.115
Dilution	50	30	---	50	48	0.096
Operating cost	80	70	---	60	70	0.140
Capital cost	100	70	---	60	81	0.162
Practicability	70	80	---	50	67	0.134
Safety	60	50	---	60	62	0.124
Drilling availability	40	30	---	30	37	0.074
Loading availability	40	60	---	40	47	0.094
Haulage availability	40	20	---	20	30	0.060

machines was evaluated for both of the pit schemes.

DATA GATHERING

Group criteria weighting

A group of experts consisted of 10 decision makers was selected and asked to judge significance of criteria shown in Figure 4 through scoring them from a 0 - 100 range. Subsequently, as shown in Table 1, using Equation (14), weighted geometric mean of these individual judgments was calculated and the normalized values considered as the final weights of the criteria. In order to compose a decision matrix associated to this problem, in the next step, a number of calculations for both of the bench height alternatives had to be made including; standard deviation of ore grade, dilution, cost estimation, etc.

Comparison of production scheduling

Production scheduling for 10 m bench height

Three push-backs were designed for this scheme with 70 m intervals. Table 2, shows the different Iron (Fe) and Phosphorus (P) grade of the designed push-backs. Table 2 shows the standard deviation SD of Fe and P which have been calculated according to Equation (1). The geometric mean of these values has been accepted as an indicator for production scheduling criterion of 10 m bench height scheme.

Table 2. Calculation of mean SD value for 10 meters bench height scheme.

Push-backs	Fe grade	P grade	SD of Fe	SD of P	Average SD
1	63.31	0.16	2.60	0.03	0.26
2	59.65	0.19			
3	58.37	0.20			

Table 3. Calculation of mean SD value for 12.5 meters bench height scheme.

Push-backs	Fe grade	P grade	SD of Fe	SD of P	Average SD
1	62.78	0.18	2.43	0.02	0.23
2	61.12	0.19			
3	58.19	0.19			

Table 4. Calculation of dilution percentage for 10 meters bench height scheme.

Push-backs	Total ore (Tons)	Total waste (Tons)	Dilution (%)	
1	15,301,836	2,008,129	11.60	
2	9,728,509	17,139,642	63.79	46.43
3	12,099,161	39,394,906	76.50	

Table 5. Calculation of dilution percentage for 12.5 meters bench height scheme.

Push-backs	Total ore (Tons)	Total waste (Tons)	Dilution (%)	
1	13,379,089	4,196,302	23.88	
2	11,807,858	12,589,336	51.60	50.51
3	12,945,560	43,426,749	77.04	

Table 6. Cost estimation for pit schemes with two different bench heights.

Mining costs	Mining activities	H = 10 m		H = 12.5 m	
Operating costs (US$/Ton)	Drilling and blasting	0.63	2.51	0.55	2.26
	Loading and haulage	1.67		1.52	
	Miscellaneous	0.21		0.19	
Capital costs (US$/Ton)	Drilling and blasting	0.14	1.24	0.16	1.31
	Loading	0.37		0.45	
	Haulage	0.74		0.70	

Production scheduling for 12.5 m bench height

For this scheme, three push-backs were designed similarly with 70 m intervals. The SD calculation for this scheme has been shown in Table 3.

Comparison of dilution

Utilizing the provided block models and the designed push-backs for both of the pit schemes, the dilution was calculated for each push-back of two schemes using Equation (2). Tables 4 and 5, show the obtained results for this step. As it can be seen, the dilution control seems to be more difficult for the 12.5 m bench height scheme.

Comparison of costs

The operating and capital costs of mining activities for both pit schemes were estimated separately. Table 6

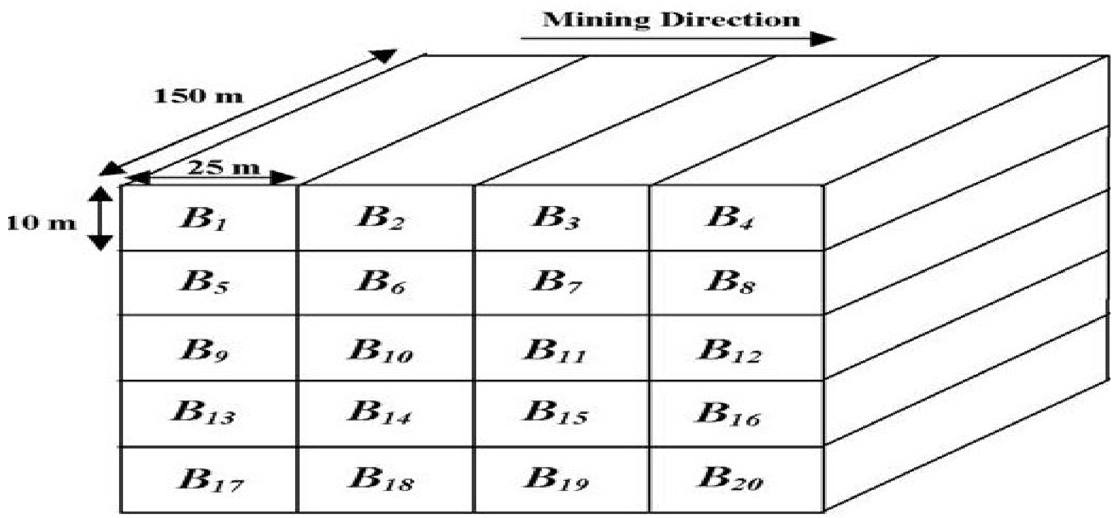

Figure 5. Large block of 10 meters bench height scheme.

shows the obtained results. As can be seen, because the greatest parts of waste zone were soil and extremely fractured rock, the total cost of drilling and blasting for the considered example was far less than cost of loading and haulage.

Comparison of practicability

The practicability of two alternative bench height schemes was evaluated through simple sequencing method. The metric dimension of large block considered for this reserve was 150 × 100 × 50, with average specific gravity of 4 tons per cubic meters to stand for 3 million tons of annual production.

Practicability of 10 m bench height

Figure 5 shows that, 20 small blocks with a 150 × 25 × 10 dimension should be extracted with the aid of predicted equipment for 10 m bench height scheme. Considering a 4 × 5 m burden and spacing with staggered pattern, a total number of 180 blast holes with 11.5 m length should be drilled in each small block. 207 h will be needed for a drilling machine with 10 m per hour drilling speed, to complete drilling of one small block. When 22 h time needed for charging of 180 blast holes is added to the time required for drilling, the total time required for drilling and blasting will be 229 h. With regard to the estimated loading-haulage time cycle for the predicted equipment, the total time needed for loading and haulage of a small block was estimated to be equal to 250 h. Table 7 shows that according to Equations (3 and 4), 17 sequences with total time of 4,229 h will be needed for extraction of all small blocks in this scheme.

Practicability of 12.5 m bench height

Figure 6 shows that the large block of this scheme, has been divided to 16 small blocks with dimension of 150 × 25 × 12.5 which will be extracted through 14 sequences and 2,583 h as shown in Table 7. A staggered pattern with burden and spacing of 5.5 × 6.6 m, and total number of 92 blast holes with average length of 14.5 m has been considered for the small blocks. Accounting on 10 m per hour drilling speed for the predicted drilling machine, a total time of 134 h will be needed for drilling of each small block. If the charging time (11 h) is added to this value, the total drilling and blasting time will be 145 h. Similar to the previous section, the total time needed for loading and haulage of a small block of 12.5 m bench height scheme was estimated to be equal to 187.5 h.

Comparison of safety

With regards to the geotechnical observations and Equations (5 and 6), the factor of safety (*FS)* for sliding blocks in the slopes composed both the bench schemes (10 and 12.5 m) was calculated. The factor of safety for the most critical sliding block of each scheme was selected as a representative factor of safety for the pit design scheme. The most critical factors of safety were 1.35 for 10 m bench height and 1.18 for 12.5 m bench height.

Comparison of availability

The average availability *AV* was calculated for the predicted equipment by use of Equation (7). Data used for availability calculation (average downtimes) were

Table 7. Time required for reaching annual production using simple sequencing method.

Sequences	Operations	Blocks	H = 10 m	H = 12.5 m
1	D and B	B_1	229	145
2	L and H	B_1	250	187.5
	D and B	B_2	229	145
3	L and H	B_2	250	187.5
	D and B	B_3	229	145
4	L and H	B_3	250	187.5
	D and B	B_4	229	145
	D and B	B_5	229	145
5	L and H	B_4	250	187.5
	L and H	B_5	250	187.5
	D and B	B_6	229	145
6	L and H	B_6	250	187.5
	D and B	B_7	229	145
7	L and H	B_7	250	187.5
	D and B	B_8	229	145
	D and B	B_9	229	145
8	L and H	B_8	250	187.5
	L and H	B_9	250	187.5
	D and B	B_{10}	229	145
9	L and H	B_{10}	250	187.5
	D and B	B_{11}	229	145
10	L and H	B_{11}	250	187.5
	D and B	B_{12}	229	145
	D and B	B_{13}	229	145
11	L and H	B_{12}	250	187.5
	L and H	B_{13}	250	187.5
	D and B	B_{14}	229	145
12	L and H	B_{14}	250	187.5
	D and B	B_{15}	229	145
13	L and H	B_{15}	250	187.5
	D and B	B_{16}	229	145
	D and B	B_{17}	229	----
14	L and H	B_{16}	250	187.5
	L and H	B_{17}	250	----
	D and B	B_{18}	229	----
15	L and H	B_{18}	250	----
	D and B	B_{19}	229	----

Table 7. Contd.

16	L and H	B_{19}	250	----
	D and B	B_{20}	229	----
17	L and H	B_{20}	250	----
Total time required (h)			4,229	2,583

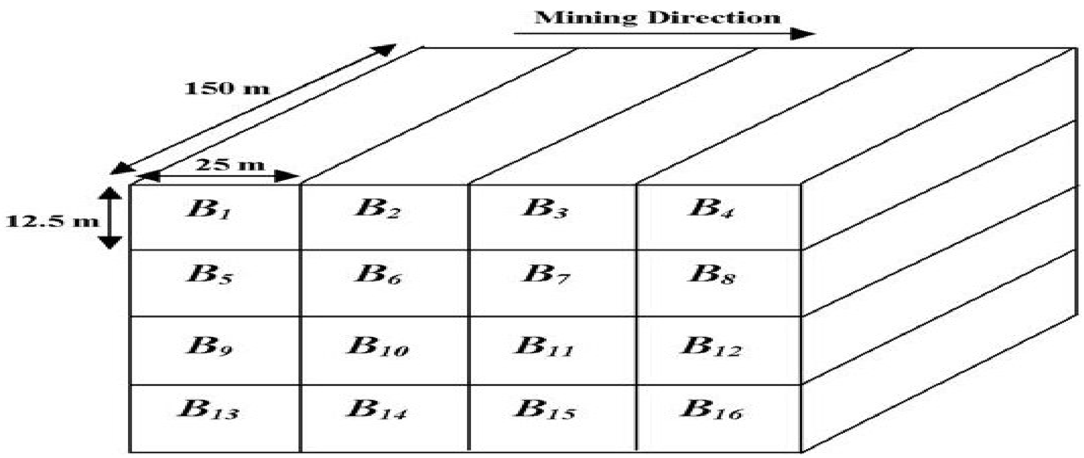

Figure 6. Large block of 12.5 meters bench height scheme.

Table 8. Availability of equipments for bench height schemes.

	Drilling machines (%)	Loading machines (%)	Haulage machines (%)
H = 10 m	52	78	76
H = 12.5 m	63	65	74

obtained from field performance of similar drilling, loading, and haulage machines in projects with 10 and 12.5 m bench height. Table 8 shows the average availability for both of the bench height schemes.

Application of the VIKOR method

According to Equation (8), the gathered data of previous sections (weights and performance scores) were placed in decision matrix shown in Table 9. These scores were normalized with the aid of Equations (9 and 11) and then were placed in the normalized decision matrix shown in Table 10. Table 11 shows that, the ideal and negative ideal solutions have been determined according to Equation (12 and 13). Finally, according to Equation (15 and 17), the utility measure, regret measure, and VIKOR index were calculated. Table 12 shows that, the 10 m bench height alternative with VIKOR index of 0, in spite of worse performance in some criteria such as production scheduling, operation cost, practicability, and drilling availability has been distinguished as the best compromise solution for this problem.

CONCLUSION

This study introduces an approach in which the advantages and disadvantages of utilizing different bench heights in an open pit mine case is aggregated in a compromising way and a practical bench height is finally selected for the case. For this purpose, the effective criteria in bench height optimization, are indicated and categorized in two economical and technical groups. However, this categorization is quite optional and flexible and depends on the accuracy of available data and also importance of each criterion. For example, a third group of criteria named as "geomechanical aspects"

Table 9. Decision matrix.

Criteria	H = 10 m	H = 12.5 m	Weights
Production scheduling	0.26	0.23	0.115
Dilution (%)	46.43	50.51	0.096
Operating cost (US$)	2.51	2.26	0.140
Capital cost (US$)	1.24	1.31	0.162
Practicability (h)	4,229	2,583	0.134
Safety	1.35	1.18	0.124
Drilling availability (%)	52	63	0.074
Loading availability (%)	78	65	0.094
Haulage availability (%)	76	74	0.060

Table 10. Normalized decision matrix.

Criteria	H = 10 m	H = 12.5 m	Weights
Production scheduling	0.655	0.756	0.115
Dilution (%)	0.736	0.677	0.096
Operating cost (US$)	0.669	0.744	0.140
Capital cost (US$)	0.726	0.687	0.162
Practicability (h)	0.521	0.853	0.134
Safety	0.753	0.658	0.124
Drilling availability (%)	0.637	0.771	0.074
Loading availability (%)	0.768	0.640	0.094
Haulage availability (%)	0.716	0.698	0.060

Table 11. Ideal and negative ideal solutions.

Criteria	Ideal solutions	Non-ideal solutions	Weights
Production scheduling	0.756	0.655	0.115
Dilution (%)	0.736	0.677	0.096
Operating cost (US$)	0.744	0.669	0.140
Capital cost (US$)	0.726	0.687	0.162
Practicability (h)	0.853	0.521	0.134
Safety	0.753	0.658	0.124
Drilling availability (%)	0.771	0.637	0.074
Loading availability (%)	0.768	0.640	0.094
Haulage availability (%)	0.716	0.698	0.060

Table 12. Calculation of the VIKOR index.

	H = 10	H = 12.5
Utility measure	0.463	0.537
Regret measure	0.140	0.162
VIKOR index	0.000	1.000

could be added to the clusters of the hierarchy. In this category, parameters such as safety factor, modified rock mass rating (MRMR), uniaxial compressive strength (UCS), rock quality designation (RQD), spacing and orientation of joint planes, specific weight of rock, water conditions, cohesive and frictional strength and so many other geomecanical parameters of the host rock can be considered.

In the proposed approach, quantification of some criteria such as costs, dilution, safety, and equipment availability were comparatively easy and therefore were suggested to be estimated from the similar projects or previous experiences of applying different bench height alternatives. On the other hand, two other criteria namely; production scheduling and practicability were somewhat tricky to be quantified. Therefore, the standard deviation of targeted ore grade as a quantitative parameter was considered to be representative for production scheduling. In order to evaluate practicability of the bench height schemes, the total hours of time needed to reach annual production was considered to be representative for this criterion, and a technique called the simple sequencing method was presented for this purpose. In this study;

(i) Because of relatively small number of existing criteria, a simple technique namely, group scale method was used to assess significance of the criteria. However, given that the number of considered criteria is absolutely customizable, when there are more criteria to be weighted, the advanced weighting techniques such as Analytical Hierarchy Process (AHP) should be applied.
(ii) A mathematical analysis method called VIKOR, was used to determine the optimum bench height. There are some additional widely applied analytical methods such as SAW, TOPSIS, PROMETHEE and ELECTRE that can replace VIKOR with almost the same advantages for this kind of problem. The outranking methods (PROMETHEE and ELECTRE) do not quite fit this problem because they only present a set of best and worst solutions. Among the three ranking methods, TOPSIS and VIKOR have been argued to have better distinguishing ability than SAW. However, VIKOR might be used when many people are involved in assessment, but TOPSIS is used when few are involved (Chu et al., 2007).
(iii) The proposed approach was successfully applied on a simple Iron ore mine with two bench height alternatives. The results showed that, this approach can be generalized for more intricate situations and an optimum response can be achieved for all similar problems within a short time.

REFERENCES

Bozorgebrahimi A, Hall RA, Morin MA (2005). Equipment size effects on open pit mining performance, Inter. J. Surface Min. Reclamation Environ. 19(1): 41-56.

Chu M, Shyu J, Tzeng G, Khosla R (2007). Comparison among three analytical methods for knowledge communities group-decision analysis, Expert Syst. Appl. 33: 1011-1024.

Dhillon BS (2008). Mining Equipment Reliability, Maintainability, and Safety, Springer-Verlag, London, ISBN: 978-1-84800-287-6, p. 209.

Escobar MT, Aguaron J, Moreno-Jimenez JM (2004). A note on AHP group consistency for the row geometric mean priorization procedure, Europ. J. Oper. Res. 153: 318-322.

Forman E, Peniwati K (1998). Aggregating individual judgments and priorities with the Analytic Hierarchy Process, Eur. J. Oper. Res. 108: 165-169.

Fourie GA, Dohm GC (1992). Mining engineering handbook, Open pit planning and design Chapter 13.1, 2nd Edition pp. 1274-1290.

Glacken IM, Noppé M, Titley M (2000). Mining Bench Height Evaluation for the Wallaby Resource-A Conditional Simulation Case Study, 4th International Mining Geology Conference, Coolum, Qld.

Hustrulid W, Kuchta M (1998). Open Pit Mine Planning and Design (vol. 1). Fundamentals. Balkema. Rotterdam p. 257.

Kose H, Aksoy CO, Gönen A, Kun M, Malli T (2005). Economic evaluation of optimum bench height in quarries. The Journal of the South African Institute of Mining and Metallurgy (JSAIMM) pp. 127-135.

Li Z (1995). Truck-haulage economics of mining open pits by high bench, CIM bull. 88(993): 50-52.

Opricovic S (1998). Multi-criteria optimization of civil engineering systems, Faculty of Civil Engineering, Belgrade.

Opricovic S, Tzeng GH (2002). Multicriteria planning of postearthquake sustainable reconstruction, Comput.-Aided Civil Infrastruct. Eng.17: 211-220.

Opricovic S, Tzeng GH (2004). The Compromise solution by MCDM methods: A comparative analysis of VIKOR and TOPSIS, Eur. J. Operat. Res. 156(2): 445-455.

Opricovic S, Tzeng GH (2007). Extended VIKOR method in comparison with outranking methods, Eur. J. Operat. Res.178: 514-529.

Reichmann WJ (1961). Use and abuse of statistics, Methuen, Reprinted 1964–1970 by Pelican. Appendix 8.

Roman PA, Daneshmend L (2000). Economies of Scale In Mining-Assessing Upper Bounds With Simulation, The Engineering Economist: J. Devot. Problems Capital Invest. 45 (4): 326-338.

Tong LI, Chen CC, Wang CH (2007). Optimization of multi-response processes using the VIKOR method, Int. J. Adv. Manuf. Technol, 31: 1049-1057.

Wyllie DC, Mah CW (2005). Rock Slope Engineering, Taylor and Francis, New York p. 456.