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Dilution Risk Ranking in Underground Metal Mines using Multi-Attributive Approximation Area Comparison

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Abstract

The contamination of ores with wastes or materials of lower than the cut-off grade is referred to as dilution. Dilution is an undesirable phenomenon that, on one hand, reduces the product grade and, consequently, reduces the sales prices and, on the other hand, adds an extra cost to waste production. Therefore, studying and evaluating the dilution risk is important in mining, and especially in underground mining. In this work, using a powerful decision-making method, i.e. Multi-Attributive Approximation Area Comparison (MABAC), the dilution risk and ranking it in underground mines are assessed. For this purpose, the most important parameters affecting the dilution in 10 mines of the Venarch manganese mines are first identified and then weighed using the Fuzzy Delphi Analytical Hierarchy Analysis (FDAHP) method. Then using the MABAC method, the dilution risk score for each mine is estimated, and subsequently, various mines are ranked as the dilution risk. Then with the implementation of the Cavity Monitoring System (CMS) and measurement of the actual dilution values, the mines are ranked in dilution. The correct matching of the results of these two rankings indicates that the MABAC method is highly effective in the ranking of the risk. At the end, the risk ranking of the mines is done using the TOPSIS method, and the lack of full compliance with the results of this method with the actual values indicates that the MABAC method is preferable to the TOPSIS method.

1. Introduction

In mining, the objective is to economically exploit ores, while taking into account the safety of work force and machineries. Various mining methods have been developed and implemented to accomplish this objective, depending on the geometry, size, depth, orientation, and waste rock surrounding the ore [1]. However, defined as the contamination of the waste with the actual ore, the so-called dilution drastically affects the direct and indirect mining costs [2]. Dilution significantly influences the cost of a stope, and hence, mining profitability since it not only increases the costs associated with the stope but also affects all the other cost components incurred by exploitation, transportation, crushing, milling, and handling as well as those of the operations to be performed on valueless wastes or low-grade rocks of insignificant values. Moreover, the extra time spent on cutting and filling large stopes developed as a result of wastes ends up with unplanned delays and renewal costs [3]. Investigation of the effect of dilution on the profitability of a gold vein mine has shown that if the dilution level exceeds 40%, the mine will lose its profitability, ending up with some loss [4]. The associated cost with 14% dilution in a gold mine has been determined to be about 38 USD per ton; in a year, this sums up at 5.4 million USD [5]. Investigations have shown that dilution is the reason for abandoning numerous underground mines such as the Mount Todd gold mine [6]. Therefore, studying the dilution risk in underground mines is very important, and it is can be one of the determining factors in the activity of a mine. The mine managers are thus able to prepare the right solutions and by controlling the risk, they can play an important role in the profitability of mines.

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So far, various relationships have been proposed by the researchers to measure dilution. Equations. 1 and 2 are the most widely used relationships proposed by Popov [7].

$$D = \frac{W}{O} \times 100 \quad (1)$$

$$D = \frac{W}{W+O} \times 100 \quad (2)$$

where D represents the dilution level in %, W is the exploited waste in tons, and O is the exploited ore in tons.

In the recent years, it has been made possible to measure an accurate area of mining stopes using automatic laser rangefinders. Cavity monitoring system (CMS) was first introduced by Miller [8]. Later on, other researchers used CMS data in their studies on dilution. This system is able to calculate the volume of cavity. Using this system, one can calculate dilution directly with known values of design and actual stope volumes.

In this work, in which the risk of dilution in underground mines was ranked, a new decision-making method called Multi-Attributive Approximation Area Comparison (MABAC) was used. For this purpose, the issue of dilution and the factors affecting it were studied in 10 mines of the Venarch manganese mines. All of these mines are underground and are extracted by the cut and fill mining method. In this regard, first, using the literature, the effective parameters in the dilution were identified, and then using the opinion of the academic and industrial experts and using the Fuzzy Delphi Analytical Hierarchy Analysis (FDAHP) method were weighed. Then by implementing the steps of the MABAC method, which consisted of formation of the initial decision matrix, normalization of the elements from the initial matrix, calculation of the elements from the weighted matrix, determining the border approximation area matrix, calculation of the distance of the alternative from the border approximation area for the matrix elements, and the final ranking the alternatives, the degree of dilution risk in the mines was assessed, as described below.

2. MABAC method

The MABAC method is one of the multi-criteria decision-making methods, first proposed by Pamukkar and Siriwik in 2015 [9]. The simplicity of the steps and its high precision has made this method one of the best decision-making methods. So far, this method has been used in a variety of issues, most notably in the ones included in Table 1.

2.1. Steps of MABAC method

The steps of the MABAC method in a multi-criteria decision problem, with n criteria and m alternatives, are shown in the Figure 1.

Table 1. Studies based on MABAC method.

Authors	Year	Subject of study
Pamucar & Cirovic [9]	2015	Selection of transport and handling resources in logistics centers
Peng & Yang [10]	2016	Selecting a suitable project
Xue *et al.* [11]	2016	Material selection with incomplete weight information
Debnath *et al.* [12]	2017	Strategic project portfolio selection of agro by-products
Yu *et al.* [13]	2017	Selecting hotels on a tourism website
Shi *et al.* [14]	2017	Assessing healthcare waste treatment technologies from a multiple stakeholder
Goorchi *et al.* [15]	2018	Rating of risk of rock slopes
Liu *et al.* [16]	2018	Environmental engineering geological patterns in underground coal mining areas
Pamučar *et al.* [17]	2018	Selecting the best firefighter helicopter.
Liang *et al.* [18]	2019	Risk assessment of rock-burst in underground mines

2.1.1. Step 1. Formation of initial decision matrix (*X*)

The first step is to evaluate m alternatives according to n criteria. We show the alternatives in the form of vectors $A_i = (x_{i1}, x_{i2}, \ldots, x_{in})$, where x_{ij} is the value of the ith alternative according to the jth criterion ($i = 1, 2, \ldots, m$; $j = 1, 2, \ldots, n$).

$$X = \begin{array}{c} \\ A_1 \\ A_2 \\ \ldots \\ A_n \end{array} \begin{array}{c} \begin{array}{cccc} C_1 & C_2 & \ldots & C_n \end{array} \\ \begin{bmatrix} x_{11} & x_{12} & \ldots & x_{1n} \\ x_{21} & x_{22} & \ldots & x_{2n} \\ \ldots & \ldots & \ldots & \ldots \\ x_{m1} & x_{m2} & \ldots & x_{mn} \end{bmatrix} \end{array} \quad (3)$$

Where m indicates the number of the alternatives and n indicates the total number of criteria.

2.1.2. Step 2. Normalization of elements from initial matrix (*X*)

$$N = \begin{array}{c} \\ A_1 \\ A_2 \\ \ldots \\ A_n \end{array} \begin{array}{c} \begin{array}{cccc} C_1 & C_2 & \ldots & C_n \end{array} \\ \begin{bmatrix} n_{11} & n_{12} & \ldots & n_{1n} \\ n_{21} & n_{22} & \ldots & n_{2n} \\ \ldots & \ldots & \ldots & \ldots \\ n_{m1} & n_{m2} & \ldots & n_{mn} \end{bmatrix} \end{array} \quad (4)$$

The elements of the normalized matrix (N) are determined using the following equations.

(a) For the Benefit type criteria (a higher value of the criterion is preferable):

$$n_{ij} = \frac{x_{ij} - x_i^-}{x_i^+ - x_i^-} \tag{5}$$

(b) For the Cost type criteria (a lower value of the criterion is preferable):

$$n_{ij} = \frac{x_{ij} - x_i^+}{x_i^- - x_i^+} \tag{6}$$

where x_{ij}, x^+_i, and x^-_i are the elements from the initial decision matrix (X), for which x^+_i and x^-_i are defined as:

$x^+_i = \max(x_1, x_2, \ldots x_m)$ and is the maximum value of the observed criterion according to the alternatives;

$x^-_i = \min(x_1, x_2, \ldots x_m)$ and is the minimum value of the observed criterion according to the alternatives.

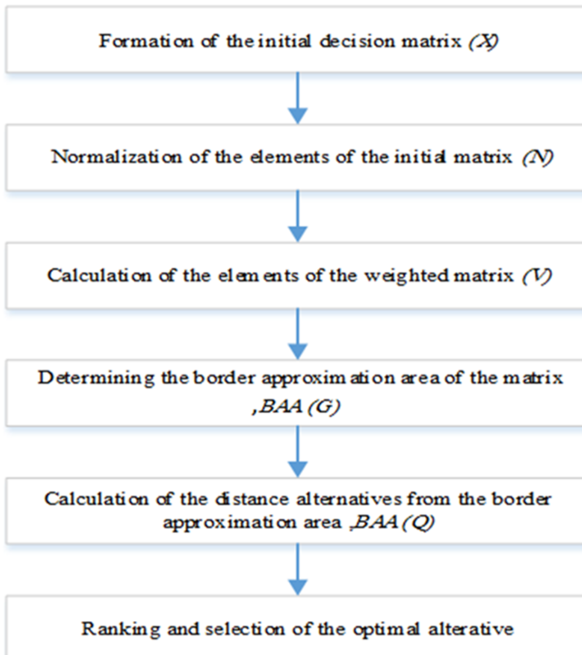

Figure 1. Steps of the MABAC method.

2.1.3. Step 3. Calculation of elements from weighted matrix (V)

The elements from the weighted matrix (V) are calculated on the basis of the following expression:

$$v_{ij} = w_i \times (n_{ij} + 1) \tag{7}$$

where n_{ij} is the elements of the normalized matrix (N) and w_i is the weight coefficients of the criteria. Using Equation (7), we obtain the weighted matrix (V).

$$V = \begin{bmatrix} v_{11} & v_{12} & \ldots & v_{1n} \\ v_{21} & v_{22} & \ldots & v_{2n} \\ \ldots & \ldots & \ldots & \ldots \\ v_{m1} & v_{m2} & \ldots & v_{mn} \end{bmatrix} = \begin{bmatrix} w_1.(n_{11}+1) & w_2.(n_{12}+1) & \ldots & w_n.(n_{1n}+1) \\ w_1.(n_{21}+1) & w_2.(n_{22}+1) & \ldots & w_n.(n_{2n}+1) \\ \ldots & \ldots & \ldots & \ldots \\ w_1.(n_{m1}+1) & w_2.(n_{m2}+1) & \ldots & w_n.(n_{mn}+1) \end{bmatrix} \tag{8}$$

where n is the total number of criteria and m is the total number of alternatives.

2.1.4. Step 4. Determining border approximation area matrix (G)

The border approximation area (BAA) for each criterion is determined according to Equation (9):

$$g_i = \left(\prod_{i=1}^{m} v_{ij} \right)^{\frac{1}{m}} \tag{9}$$

where v_{ij} is the elements of the weighted matrix (V) and m is the total number of alternatives.

After calculating the value g_i for each criterion, a border approximation area matrix (G) (10) is formed with the format $n \times 1$ (n is the total number of criteria according to which the selection is made from the alternatives offered).

$$\begin{array}{cccc} & C_1 & C_2 & \dots & C_n \\ G = & [g_1 & g_2 & \dots & g_n] \end{array} \tag{10}$$

2.1.5. Step 5. Calculation of distance of alternative from border approximation area for matrix elements (Q)

The distance of the alternatives from the border approximation area (q_{ij}) is determined as the difference between the elements in the weighted matrix (V) and the value of the border approximation area (G). According to Equation (11):

$$Q = V - G = \begin{bmatrix} v_{11} & v_{12} & \dots & v_{1n} \\ v_{21} & v_{22} & \dots & v_{2n} \\ \dots & \dots & \dots & \dots \\ v_{m1} & v_{m2} & \dots & v_{mn} \end{bmatrix} - \begin{bmatrix} g_1 & g_2 & \dots & g_n \\ g_1 & g_2 & \dots & g_n \\ \dots & \dots & \dots & \dots \\ g_1 & g_2 & \dots & g_n \end{bmatrix} = \begin{bmatrix} q_{11} & q_{12} & \dots & q_{1n} \\ q_{21} & q_{22} & \dots & q_{2n} \\ \dots & \dots & \dots & \dots \\ q_{m1} & q_{m2} & \dots & q_{mn} \end{bmatrix} \tag{11}$$

where gi is the border approximation area for criterion C_i, v_{ij} is the weighted matrix of the elements (V), n is the number of criteria, and m is the number of alternatives.

Alternative A_i could belong to the border approximation area (G), upper approximation area (G^+) or lower approximation area (G^-), i.e. $A_i \in \{G \vee G^+ \vee G^-\}$. The upper approximation area (G^+) is the area that contains the ideal alternative (A^+), while the lower approximation area (G^-) is the area that contains the anti-ideal alternative (A^-) (Figure 2).

The belonging of alternative A_i to the approximation area (G, G^+ or G^-) is determined on the basis of Equation 12.

$$A_i \in \begin{cases} G^+ & if \quad q_{ij} > 0 \\ G & if \quad q_{ij} = 0 \\ G^- & if \quad q_{ij} < 0 \end{cases} \tag{12}$$

For alternative A_i to be selected as the best in the set, it is necessary for it to have as many criteria as possible belonging to the upper approximate area (G^+). If, for example, alternative A_i has 5 criteria (out of a total of 6 criteria) belonging to the upper approximate area, and one criterion belonging to the lower approximate area (G^-), it means that according to 5 criteria, the alternative is near or equal to the ideal alternative, while for the one criterion, it is near or equal to the anti-ideal alternative. If $q_{ij} > 0$, that is $q_{ij} \in G^+$, then alternative A_i is near or equal to the ideal alternative. If $q_{ij} < 0$, that is $q_{ij} \in G^-$, it shows that alternative A_i is near or equal to the anti-ideal alternative.

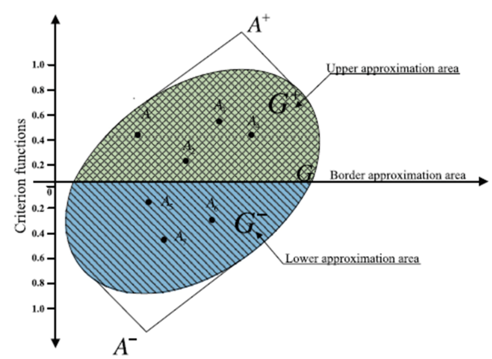

Figure 2. Presentation of the upper (G^+), lower (G^-), and border (G) approximation areas [9].

2.1.6. Step 6. Ranking alternatives

A calculation of the values of the criterion functions for the alternatives (13) is obtained as the sum of the distance of the alternatives from the border approximation areas (q_i). By calculating the sum of the elements of matrix (Q) by rows, we obtain the final values of the criterion functions of the alternatives.

$$S_i = \sum_{j=1}^{n} q_{ij}, \qquad j = 1,2,\dots,n \qquad i = 1,2,\dots,m \tag{13}$$

3. Ranking of dilution risk

In this work, the ranking of the dilution risk in the Venarch manganese mines was done using the MABAC method, which is described below.

3.1 Studied mines

Ranking of the dilution risk was done in 10 mines from the Venarch manganese mines, as listed in Table 2. The Venarch manganese mines are located 27 km to the western south of Qom within 2 km of Venrach village (longitude: 50°45′42″; latitude: 34°25′3″) [19]. With a reserve of more than 8.6 million tons, as of now, the mines are the largest manganese mines across the Middle East, and produce about 100,000-110,000 of manganese ore per year to be the largest manganese production site across Iran. The deposit is extended over an area of 40 km^2 with an ore zone length of about 12 km. The deposit was identified down to a depth of about 400 m. Thickness of exploitable ore ranges from 0.5 m to 5 m, and sometimes thicker. Manganese-bearing layers dip at 65°–90°, while the surrounding rocks are composed of tuffs along with andesite lavas and porphyries. All mines are being exploited via the cut-and-fill stoping method [20].

3.2. Studied parameters

Using the literature, it can be seen that many parameters are effective in the development of the dilution. Of these, 10 parameters can be identified as the most important ones that affect the dilution. These parameters are given in Table 3 [21]. As shown in this table, these parameters are grouped into four categories: stope design parameters, drilling and blasting parameters, geologic parameters, and operational parameters.

Table 2. Studied mines for ranking risk dilution using the MABAC method.

Mine	Symbol
Piroozi-340-E	M_1
Athari-290-W	M_2
Doctor-140-W	M_3
Athari-290-E	M_4
Doctor-140-E	M_5
Jalal-390-W	M_6
Piroozi-340-W	M_7
Piroozi-240-W	M_8
Doctor-240-E	M_9
Jalal-390-E	M_{10}

Table 3. Most important parameters affecting the dilution [21].

Category	Parameters	Symbol
Stope design	Hydraulic radius	P_1
	Stope width	P_2
Drilling and blasting	Inaccurate drilling	P_3
	Powder factor	P_4
	Blast vibration	P_5
Geology	Walls quality	P_6
	Foliation	P_7
	Stope depth	P_8
Operation	Filling materials	P_9
	Filling time	P_{10}

3.2.1. Hydraulic radius

Laubscher [22] has proposed the hydraulic radius (HR) as the ratio of stope surface area to stope perimeter, Equation (14). This factor measures stope dimension and form of breast because a cross-section of a stope alone is not an appropriate measure of the stope size, as indicated by the difference in stability between two stopes of the same cross-section but different widths and/or heights. Indeed, as HR increases, a further falling and over-breaking, and hence, a higher dilution risk level is expected.

$$HR = \frac{a.b}{2(a+b)} \tag{14}$$

3.2.1 Stope width

As a geometrical parameter, the stope width can play a significant role in the wall and roof stability so that at a constant height, the wider the stope, the higher is the risk of falling and dilution [23].

3.2.3. Inaccurate drilling

Inaccurate drilling, particularly for the holes near the hanging wall and footwall, is among the important parameters contributing to dilution. An inappropriate configuration in drilling can end up with considerable results. Inappropriate setting of collaring, drilling angle, and diversion of the drilled hole significantly contribute to unplanned dilution. Inappropriate drilling pattern, operator’s skill, physical limitation drilling machine, drill bit diameter, and geological conditions can influence the hole deviation.

Investigations undertaken in an underground stope have shown that the drilling of holes of 15-20 m in depth and 64 mm in diameter are associated with about 0.5 m of deviation. Assuming this amount of deviation on both sides of a stope of 3 m in width, the resulting dilution has been estimated to be about 16% [23].

3.2.4. Powder factor

The powder factor (PF) shows the amount of explosives consumed per unit volume or unit weight of crushed rock in a blasting operation. It is measured in grams of explosive per cubic meter/ton of rock. PF is a function of the type of explosive material, specific gravity of the rock, and regional geology. In underground stopes, an increase in PF with respect to the optimum amount results in an excessive over-break, and hence, dilution risk, while a lower PF than the optimum amount will end up with a loss in the exploited ore. As such, the amount of PF can play a significant role among the other dilution-generating parameters.

3.2.5. Blast vibration

The damage incurred by blast vibration in stopes is determined by the physical damage to exposed rock mass near the blasting location. Blast vibration is measured by peak particle velocity (PPV) because when the resulting shockwave reaches a point, it makes the particles at the point to vibrate, and since the magnitude of strains in an elastic material is proportional to the particle vibration velocity, PPV serves as an appropriate measure for determining the blast vibration damages in rocks [24]. The PPV value can be determined using a seismogram device. By increasing the amount of PPV, the risk of dilution also increases.

3.2.6. Stope wall quality

In this work, the modified stability number (N') was used to determine the stope wall quality. The modified stability number was first introduced by Diederichs and Kaiser [25], who applied it to determine the rock mass quality and the bearing capacity. Indeed, this number shows the rock mass stability under the existing stress configuration. The value for N' can be calculated using Equation (15).

$$N' = Q' \times A \times B \times C \tag{15}$$

where:

Q': modified rock tunneling quality index or Q' classification system;

A: a factor related to mining stresses, rock stress factor;

B: a factor related to critical discontinuities on the wall under consideration, joint strike correction factor;

C: a factor related to the direction of the considered wall, gravity correction factor;

The coefficients A, B, and C can be determined via the corresponding relationships or graphs.

Obviously, with the increase in the modified stability number, the risk of dilution will be reduced.

3.2.7. Foliation

Foliation is another parameter contributing to the stope wall rock fall and dilution. Where foliation orientation is oblique with respect to the wall direction, one may end up with a minimum stability and a maximum dilution risk. However, where foliation is developed parallel to the stope wall, one may expect just a fair stability and a fair dilution risk, and finally, where foliation is developed normal to the stope wall, a maximum wall stability, and hence, a minimum dilution risk is expected [23].

3.2.8. Stope depth

An increased level of in-situ stresses is directly related to the stope depth. Furthermore, induced stresses within the drilling space are related to the stope depth so that the depth can be effective on the amount of over-break and dilution risk.

3.2.9. Filling method

Stability of hanging wall and dilution control are among the most important reasons for filling in the underground mining methods. Filling can be performed via various methods such as hand filling, gravity filling, mechanical filling, pneumatic filling, and hydraulic filling. However, hand and mechanical filling methods fail to effectively accomplish the filling task, leaving the exploited site with a large convergence. However, the level of compaction increases by filling materials via the mechanical, pneumatic, and hydraulic filling methods, respectively [26].

3.2.10. Filling time

Since in the underground mining methods one of the reseals for filling cavities is to support walls, the filling time is very important because the filling materials should be in place timely to prevent excessive wall fall, and hence, dilution risk [26].

3.3. Weighting of parameters

In this work, the Fuzzy Delphi Hierarchy Process (FDAHP) method was used to weight the effective parameters in the dilution. For this purpose, the survey forms were prepared and sent to the corresponding experts in academic and industrial fields. Upon these forms, the experts were asked to describe the importance of a set of categories of factors and parameters using qualitative terms indicating five intervals: very weakly important, weakly important, moderately important, strongly important, and very strongly important. Once the questionnaires were received, some scores were attributed to the descriptive terms 9, 7, 5, 3, and 1, respectively. In the following, the steps of this method including the calculation of fuzzy numbers, formation of inverse fuzzy matrix, calculation of relative fuzzy weights of parameters, and defuzzification of weights of parameters have been done. Here, for the sake of brevity, refrain from calculations of the steps of the FDAHP method, and only the final results of the weighting are given in Table 4.

Table 4. Final weight of the effective parameters [21].

Parameters	Weight
P_1	0.2199
P_2	0.0580
P_3	0.1841
P_4	0.0760
P_5	0.0545
P_6	0.1992
P_7	0.0361
P_8	0.0205
P_9	0.1092
P_{10}	0.0425

3.4. Preformation of MABAC method

According to the explanations given about the steps of the MABAC method, the ranking of dilution risk of the studied mines was done as follow.

Step 1: Formation of initial decision matrix (*X*)

After weighing the ten parameters, the quantitative and qualitative parameters were measured and evaluated in 10 mines. The quantitative parameters including the powder factor, stope depth, stope width through direct observation, blast vibration through the seismogram device, hole deviation through the laser rangefinder equipped with digital angle finder, hydraulic radius by measuring the length, and width of the stope and using Equation 14, the modified stability number, *N'*, through the modified rock tunneling quality index, *Q'*, and factors *A*, *B*, and *C*, and using Equation 15 were measured and their values were calculated. For the qualitative parameters including the foliation, filling method and filling time, which were evaluated using their descriptive expressions (for foliation parameter, expressions: perpendicular iso-strike, perpendicular unstrike, parallel, oblique iso-strike, and oblique unstrike, for the filling method parameter, expressions: hydraulic filling, pneumatic filling, mechanical filling, gravity filling, and hand filling, and for filling time parameter, expressions: continuous loading, after loading quarter stope, after loading third the stope, after loading half the stope, and after loading throughout the stope) accordingly, very low, low, moderate, high, and very high, linguistic variables have been used to illustrate their effects on the dilution risk. For these linguistic variables, according to Pamucar and Cirovic, the triangular fuzzy numbers shown in Table 5 are used. The recorded quantitative and qualitative parameters for the 10 mines, in accordance with Equation 3, are given as matrices in Table 6, and thus the first step of the MABAC method, which is the formation of the initial decision matrix (matrix X), has been done.

Step 2: Normalization of elements from initial matrix (*N*)

In order to normalize the elements of the matrix (*X*) and converted to the matrix (*N*), in the parameters P_1-P_5, P_7, P_8 and P_{10} that with increasing their amount, the dilution risk increases, Equation 5 is used, and in the parameters P_6 and P_9, that with increasing their amount, the dilution risk reduces, Equation 6 is used. For example, the element n_{11}, which corresponds to P_1, and the element n_{106}, which corresponds to P_6, are calculated as follows. The results of the normalization of all elements of matrix (*X*) are given in Table 7.

$$n_{11} = \frac{2.27 - 1.36}{3.27 - 1.36} = 0.4746$$

$$n_{106} = \frac{0.8 - 16}{0.64 - 16} = 0.9896$$

In qualitative parameters, Pamucar and Cirovic, proposed Equation 16 for defuzzification elements, where $a^{(l)}$ and $a^{(r)}$ are the left and right distribution trust intervals of triangular fuzzy number, respectively, and $a^{(m)}$ is the value in which the triangular function reaches its maximum value. In this paper, the mentioned equation is used for defuzzification of elements.

$$A = \frac{a^{(l)} + 4.a^{(m)} + a^{(r)}}{6} \tag{16}$$

Step 3. Calculation of elements from weighted matrix (*V*)

The matrix (*N*) is weighted and converted to the matrix (*V*) using Equation 7 and using the data of Table 4. For example, the element v_{11} is calculated as follows. The results of the weighting of all elements of matrix (*N*) are given in Table 8.

$v_{11} = w_{11} \cdot (n_{11} + 1) = 0.2199 \times (0.4764 + 1) = 0.3247$

Step 4. Determining border approximation area matrix (*G*)

In the following, the border approximation area of each parameter is calculated using Equation 9. For example, the border approximation area of P_1 is calculated as follows. The result of calculating the border approximation area of all parameters is a border approximation area vector, as shown in Table 9.

$$g_1 = \left(\prod_{1}^{10} v_{ij} \right)^{1/10} = (0.3247 \times 0.3224 \times 0.2660 \times 0.3189 \times 0.2199 \times 0.4398 \times 0.3511 \times 0.3143 \times 0.3097 \times 0.3891)^{1/10} = 0.3205$$

Step 5. Calculation of distance of alternative from *BAA* for matrix elements (*Q*)

The distance of the alternatives from *BAA* is calculated using Equation 11, and the matrix (*Q*) is created according to Table 10. For example, the element q_{11} is calculated as follows.

$q_{11} = v_{11} - g_1 = 0.3247 - 0.3205 = 0.0042$

Table 5. Fuzzified likert scale for evaluating the alternatives [9].

No	Linguistic terms	Triangular fuzzy numbers	Linguistic values
1	Very high influence (VH)	$\tilde{5}$	(4.50, 5.00, 5.00)
2	High influence (H)	$\tilde{4}$	(2.50, 3.50, 4.50)
3	Moderate (M)	$\tilde{3}$	(1.50, 2.50, 3.50)
4	Low influence (L)	$\tilde{2}$	(0.00, 1.50, 2.50)
5	Very low influence (VL)	$\tilde{1}$	(0.00, 0.00, 1.50)

Table 6. Initial desertion matrix (*X*).

	P_1	P_2	P_3	P_4	P_5	P_6	P_7	P_8	P_9	P_{10}
M_1	2.27	4.70	12	1.50	491	1.60	H	340	H	M
M_2	2.25	4.50	10.5	1.50	482	2.80	M	290	H	H
M_3	1.76	3.40	4.5	0.86	242	10.24	VL	140	M	VH
M_4	2.22	4.20	9	1.40	423	3.60	L	290	H	H
M_5	1.36	2.80	3	0.55	210	16.00	VL	140	M	VH
M_6	3.27	5.65	16	2.00	670	0.64	VH	390	H	L
M_7	2.50	4.85	13	1.60	528	0.96	H	340	H	M
M_8	2.18	3.70	7.5	0.95	340	4.80	L	240	H	H
M_9	2.14	3.55	6	0.90	227	7.68	L	240	M	H
M_{10}	2.83	5.25	14	1.70	573	0.80	VH	390	H	L

Table 7. Normalized desertion matrix (*N*).

	P_1	P_2	P_3	P_4	P_5	P_6	P_7	P_8	P_9	P_{10}
M_1	0.4764	0.6667	0.6923	0.6552	0.6109	0.9375	0.7659	0.8000	0.0000	0.3428
M_2	0.4660	0.5965	0.5769	0.6552	0.5913	0.8594	0.5106	0.6000	0.0000	0.6857
M_3	0.2094	0.2105	0.1154	0.2138	0.0696	0.3750	0.0000	0.0000	1.0000	1.0000
M_4	0.4503	0.4912	0.4615	0.5862	0.4630	0.8073	0.2553	0.6000	0.0000	0.6857
M_5	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	1.0000	1.0000
M_6	1.0000	1.0000	1.0000	1.0000	1.0000	1.0000	1.0000	1.0000	0.0000	0.0000
M_7	0.5969	0.7193	0.7692	0.7241	0.6913	0.9792	0.7659	0.8000	0.0000	0.3428
M_8	0.4293	0.3158	0.3462	0.2759	0.2826	0.7292	0.2553	0.4000	0.0000	0.6857
M_9	0.4084	0.2632	0.2308	0.2414	0.0370	0.5417	0.2553	0.4000	1.0000	0.6857
M_{10}	0.7696	0.8596	0.8462	0.7931	0.7891	0.9896	1.0000	1.0000	0.0000	0.0000

Table 8. Weighted Normalized desertion matrix (*V*).

	P_1	P_2	P_3	P_4	P_5	P_6	P_7	P_8	P_9	P_{10}
M_1	0.3247	0.0967	0.3116	0.1258	0.0878	0.3860	0.0637	0.0369	0.1092	0.0571
M_2	0.3224	0.0926	0.2903	0.1258	0.0867	0.3704	0.0545	0.0328	0.1092	0.0716
M_3	0.2660	0.0702	0.2053	0.0922	0.0583	0.2739	0.0361	0.0205	0.2184	0.0850
M_4	0.3189	0.0865	0.2691	0.1206	0.0797	0.3600	0.0453	0.0328	0.1092	0.0716
M_5	0.2199	0.0580	0.1841	0.0760	0.0545	0.1992	0.0361	0.0205	0.2184	0.0850
M_6	0.4398	0.1160	0.3682	0.1520	0.1090	0.3984	0.0722	0.0410	0.1092	0.0425
M_7	0.3511	0.0997	0.3257	0.1310	0.0922	0.3943	0.0637	0.0369	0.1092	0.0571
M_8	0.3143	0.0763	0.2478	0.0970	0.0699	0.3445	0.0453	0.0287	0.1092	0.0716
M_9	0.3097	0.0733	0.2266	0.0943	0.0565	0.3071	0.0453	0.0287	0.2184	0.0716
M_{10}	0.3891	0.1079	0.3399	0.1363	0.0975	0.3963	0.0722	0.0410	0.1092	0.0425

Table 9. Border approximation area matrix (*G*).

	P_1	P_2	P_3	P_4	P_5	P_6	P_7	P_8	P_9	P_{10}
g_i	0.3205	0.0859	0.2706	0.1128	0.0771	0.3363	0.0518	0.0311	0.1344	0.0638

Step 6. Ranking alternatives

In order to rank the alternatives, first using Equation 10, the rank of each alternative is determined, for example, S_1, which is the rank of the alternatives M_1, is calculated as follows:

$$S_1 = \sum_{J=1}^{10} q_{1j} = 0.0042 + .0108 + 0.0410 + 0.0130 + 0.0107 + 0.0497 + 0.0119 + 0.0058 - 0.0252 - 0.0067 = 0.1150$$

Finally, by determination of the rank of all alternatives, the final ranking of alternative was carried out according to Table 11.

Table 10. Distance of the alternatives from the *BAA* matrix (*Q*).

	P_1	P_2	P_3	P_4	P_5	P_6	P_7	P_8	P_9	P_{10}
M_1	0.0042	0.0108	0.0410	0.0130	0.0107	0.0497	0.0119	0.0058	-0.0252	-0.0067
M_2	0.0019	0.0067	0.0197	0.0130	0.0096	0.0341	0.0027	0.0017	-0.0252	0.0078
M_3	-0.0545	-0.0157	-0.0653	-0.0206	-0.0188	-0.0624	-0.0157	-0.0106	0.0840	0.0212
M_4	-0.0016	0.0006	-0.0015	0.0078	0.0026	0.0237	-0.0065	0.0017	-0.0252	0.0078
M_5	-0.1006	-0.0279	-0.0865	-0.0368	-0.0226	-0.1371	-0.0157	-0.0106	0.0840	0.0212
M_6	0.1193	0.0301	0.0976	0.0392	0.0319	0.0621	0.0204	0.0099	-0.0252	-0.0213
M_7	0.0306	0.0138	0.0551	0.0182	0.0151	0.0579	0.0119	0.0058	-0.0252	-0.0067
M_8	-0.0062	-0.0096	-0.0228	-0.0158	-0.0072	0.0081	-0.0065	-0.0024	-0.0252	0.0078
M_9	-0.0108	-0.0126	-0.0440	-0.0185	-0.0206	-0.0292	-0.0065	-0.0024	0.0840	0.0078
M_{10}	0.0686	0.0220	0.0693	0.0235	0.0204	0.0600	0.0204	0.0099	-0.0252	-0.0213

Table 11. Rank of the alternatives using MABAC method.

Alternatives	*S*	Rank
M_1	0.1150	4
M_2	0.0721	5
M_3	-0.1584	9
M_4	0.0094	6
M_5	-0.3326	10
M_6	0.3640	1
M_7	0.1767	3
M_8	-0.0797	8
M_9	-0.0527	7
M_{10}	0.2476	2

4. Validation of ranking of dilution risk

After determining the rating of the dilution risk of the mines, the actual dilution of the mines was measured for the validation of this ranking. In this work, the CMS method was used to measure the amount of dilution. This system is able to calculate the volume of stopes. In order to calculate the stope volume, a cross-sectional profile of the stope was acquired at equal spacing, and then integrated into a continuous volume. A laser rangefinder with an effective range of 200 m at 1 mm tolerance equipped with a digital goniometer of an operating angle range of 360 degrees at 0.1 degree tolerance was used to acquire the profiles. In order to acquire each section, first, the rangefinder was mounted on a tripod at the center of the lower side of the section on the stope floor. Then the distance from that to points on the stope walls and roof at different angles were read until a section was recorded. Next, the tripod was shifted to the center of the lower side of the next section and the procedure was repeated to record the second section. The procedure was repeated until the required number of sections was captured. Following the investigations, the acquired data was fed into the AutoCAD.14 software, where the actual stope was modeled three-dimensionally and the stope volume was determined. With the amounts of designed volume, actual volume, andspecific gravity of the ore and waste, the amounts of dilution of stopes were calculated using Equation 2. The images of the output of the CMS method of two different mines are shown in Figure 3. The results of the dilution measurement in the 10 mines are given in Table 12. As it can be seen in this table, the ranking of the amounts of dilution measured in mines is equal to the ranking of the dilution risk in the mines, which is obtained by the MABAC method. At the end, the ranking of risk dilution using the TOPSIS method was performed as one of the most important multi-criteria decision-making methods. Since this method is known, we refused to provide explanations for the steps of the method, and only the tables of numbers for each step of the method were presented. First, by multiplying the vector of the weight of the parameters (Table 4), in the normalized matrix (Table 7), the weighted normal matrix was created in accordance with Table 13. Then using the TOPSIS relationships, the values of distance from the ideal solution (S^{+}), the distance from the anti-ideal solution (S^{-}), and the similarity index (C^{*}) were calculated, and eventually, the ranking of alternatives was done. The mentioned items are summarized in Table 14. The results of ranking of dilution risk using the MABAC and TOPSIS methods and ranking of actual values of dilution are summarized in Table 15. The results obtained indicate the precision of the MABAC method. The results indicate the precision and preference of the MABAC method.

Table 12. Rank of actual amounts of dilutions in 10 mines.

Mine	Actual amount of dilution (%)	Rank
M1	17	4
M2	15	5
M3	6	9
M4	14	6
M5	5	10
M6	27	1
M7	21	3
M8	9	8
M9	11	7
M10	23	2

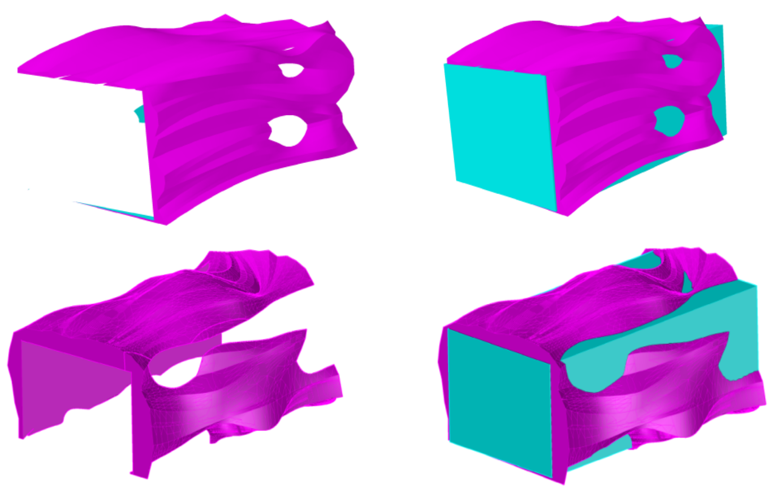

Figure 3. Images of the output of the CMS method of two mines.

Table 13. Weighted Normalized desertion matrix.

	P1	**P2**	**P3**	**P4**	**P5**	**P6**	**P7**	**P8**	**P9**	**P10**
M1	0.1048	0.0387	0.1275	0.0498	0.0333	0.1868	0.0276	0.0164	0.0000	0.0146
M2	0.1025	0.0346	0.1062	0.0498	0.0322	0.1712	0.0184	0.0123	0.0000	0.0291
M3	0.0461	0.0122	0.0212	0.0162	0.0038	0.0747	0.0000	0.0000	0.1092	0.0425
M4	0.0990	0.0285	0.0850	0.0446	0.0252	0.1608	0.0092	0.0123	0.0000	0.0291
M5	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.1092	0.0425
M6	0.2199	0.0580	0.1841	0.0760	0.0545	0.1992	0.0361	0.0205	0.0000	0.0000
M7	0.1312	0.0417	0.1416	0.0550	0.0377	0.1951	0.0276	0.0164	0.0000	0.0146
M8	0.0944	0.0183	0.0637	0.0210	0.0154	0.1453	0.0092	0.0082	0.0000	0.0291
M9	0.0898	0.0153	0.0425	0.0183	0.0020	0.1079	0.0092	0.0082	0.1092	0.0291
M10	0.1692	0.0499	0.1558	0.0603	0.0430	0.1971	0.0361	0.0205	0.0000	0.0000

Table 14. Values of S^+, S^-, C^*, and rank of alternatives using the TOPSIS method.

	S^+	S^-	C^*	Rank
M_1	0.2318	0.2136	0.4796	4
M_2	0.2268	0.2014	0.4703	5
M_3	0.2902	0.1425	0.3293	10
M_4	0.2323	0.1869	0.4458	6
M_5	0.3286	0.2037	0.3826	8
M_6	0.2037	0.3286	0.6174	1
M_7	0.2226	0.2380	0.5167	3
M_8	0.2419	0.1727	0.4165	7
M_9	0.2636	0.1406	0.3479	9
M_{10}	0.2109	0.2730	0.5642	2

Table 15. Comparison of the ranking of dilution risk used MABAC and TOPSIS with the ranking of actual amounts of dilution.

Rank	MABAC	The actual amount of dilution (%)	TOPSIS
1	M_6	M_6	M_6
2	M_{10}	M_{10}	M_{10}
3	M_7	M_7	M_7
4	M_1	M_1	M_1
5	M_2	M_2	M_2
6	M_4	M_4	M_4
7	M_9	M_9	M_8
8	M_8	M_8	M_5
9	M_3	M_3	M_9
10	M_5	M_5	M_3

5. Conclusions

Dilution risk ranking in underground metal mines, which results from estimating the dilution risk of various mines and is based on the existing status of the effective parameters, can play an important role in the prevention or control of dilution. Various methods can be used to achieve this ranking. In this work, a new decision-making method called MABAC was used. The results of this method showed that it had a very high accuracy because the ranking of mines in the measured dilution from the cavity monitoring system was consistent with the ranking made with this method. Also the non-conformance of the ranking resulting from the measured dilution by the TOPSIS method reflects the fact that the MABAC method can be used as a suitable and reliable alternative for ranking and selecting projects in the older decision-making methods.

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رتبه‌بندی ریسک ترقیق در معادن زیرزمینی فلزی به روش مقایسه محدوده تقریب مرزی چند شاخصه

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چکیده:

مخلوط شدن مواد معدنی استخراج شده با مواد باطله یا مواد با عیار کمتر از عیار حد را ترقیق می‌گویند. ترقیق پدیده‌ای است نامطلوب که از یک طرف سبب کاهش عیار محصول و به دنبال آن سبب کاهش قیمت فروش و از طرف دیگر سبب صرف هزینه اضافه جهت استخراج باطله می‌شود. بنابراین مطالعه و ارزیابی میزان ریسک ترقیق دارای اهمیت ویژه‌ای در معدن‌کاری و بخصوص در معدن‌کاری زیرزمینی است. در این مقاله به منظور ارزیابی میزان ریسک ترقیق و رتبه‌بندی آن در معادن زیرزمینی، از یک روش تصمیم‌گیری جدید قدرتمند با عنوان مقایسه محدوده تقریبی مرزی چندشاخصه (MABAC)، استفاده شده است. به این منظور ابتدا مهم‌ترین پارامترهای مؤثر بر ترقیق در ۱۰ معدن از مجموعه معادن منگنز ونارچ شناسایی شد، سپس با استفاده از روش تحلیل سلسله مراتبی دلفی فازی (FDAHP)، وزن‌دهی شدند. در ادامه با به‌کارگیری از روش MABAC، امتیاز ریسک ترقیق مربوط به هر معدن تخمین زده شد و از آنجا معادن مختلف به لحاظ امتیاز ریسک ترقیق، رتبه‌بندی شدند. سپس با پیاده‌سازی سیستم مانیتورینگ فضای حفاری (CMS) و اندازه‌گیری مقادیر واقعی ترقیق در معادن مذکور به لحاظ میزان ترقیق واقعی رتبه‌بندی شدند. انطباق درست نتایج حاصل از این دو رتبه‌بندی نشان دهنده کارایی بسیار بالای روش MABAC در رتبه‌بندی است. در پایان رتبه‌بندی ریسک ترقیق معادن مذکور با استفاده از روش TOPSIS نیز انجام شده است و عدم انطباق کامل نتایج حاصل از این روش با مقادیر واقعی، نشان دهنده ارجحیت استفاده از روش MABAC نسبت روش مذکور است.

کلمات کلیدی: ریسک ترقیق، رتبه‌بندی، روش MABAC، معدن زیرزمینی فلزی.