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# A new classification system for evaluation and prediction of unplanned dilution in cut-and-fill stoping method

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**Abstract**
Production planning in mineral exploitation should be undertaken to maximize exploited ore at a minimum unplanned dilution. Unplanned dilution reduction is among the ways to enhance the quality of products, and hence, reduce the associated costs, resulting in a higher profit. In this way, firstly, all the parameters contributing to unplanned dilution in underground stopes and specifically the cut-and-fill stoping method are identified. Secondly, the parameters are weighed using the fuzzy-Delphi analytical hierarchy process. Thirdly, the most effective parameters are selected among the pool of effective parameters. Finally, in order to present a novel classification system for an unplanned dilution assessment, a new index called stope unplanned dilution index (SUDI) is introduced. SUDI represents the extent to which a cut-and-fill stope is susceptible to unplanned dilution. That is, having the value of this index, one may classify the cut-and-fill stopes into five groups according to robustness versus unplanned dilution: very strong, strong, moderate, weak, and very weak. SUDI is applied to10 stopes in different parts of Venarch Manganese Mines (Qom, Iran). In this way, a semi-automatic cavity monitoring system is implemented in the stopes. The regression analysis method shows that there is a relationship between SUDI and the actual unplanned dilution in equivalent linear overbreak/slough with a correlation coefficient ($R^2$ = 0.8957).

**Keywords:** *Underground Mining, Cut-and-Fill Method, Unplanned Dilution, Classification System, Stope Unplanned Dilution Index.*

## 1. Introduction

In mining, the objective is to economically exploit the ore 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 geometry, size, depth, orientation, and the waste rock surrounding the ore [1]. However, defined as the contamination of the waste with actual ore, the so-called dilution drastically affects direct and indirect mining costs [2]. Dilution significantly influences the cost of a stope, and hence, mining profitability because it not only increases the associated costs 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 value. 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].

The economic effects of dilution are present in different stages of mining. These begin with the stope where unwanted waste exploitation forces mining machineries to take more time to handle the exploited material. This extra time, which is attributable to wastes, drives the cycle time of machineries, which, in turn, increases the frequency of maintenance operations on the machineries and equipment. This is further translated to the generation of excessive dust within the stope, reducing workers' eyesight that can contribute to a lower quality of products. Following the stoping stage, in the transportation phase, dilution incurs extra costs, particularly when the transportation operation is to be performed along long distances. These costs affect the transportation operation either directly or indirectly. Increased amount of the energy required to transport ore along with waste directly contributes to enhanced mining costs. Reduced transportation capacity and increased depreciation, which result in a shorter useful life of transportation systems such as conveyers, are among the direct effects of dilution on transportation cost. Moreover, addressing the points blocked by wastes and controlling excessive dust inside the stope are among the sources of indirect costs in the transportation stage. In the primary crushing stage, which is commonly undertaken inside the mine premise, dilution results in an increased cost of crushing through, for example, associated excessive costs with higher power consumption by machines, faster depreciation of machineries, and reduced operational capacity of crushers. Moreover, stacking of wastes inside the mine and the associated environmental costs represent the other cost components incurred in this stage. In the mineral processing phase, costs associated with dilution are of more significant importance, particularly in the cases where wastes exhibit similar processing properties to those of actual ore. At this stage, when exploited wastes are introduced into the processing facilities together with the actual ore, one may expect higher costs of secondary crushing, handling, screening, concentrating, dewatering, waste stacking, etc.

Dilution reduces revenues of mining projects. Waste exploitation, transportation, and processing are costly operations. In fact, each unit of dilution-resulted waste replaces a highly profitable unit of ore in the production capacity of mine. For example, in a gold mine with a processing capacity of 360,000 tons/year, taking an average grade of the mineral reserve, gold price, and total operational cost to be 0.35 ounces of gold per ton, 350 USD per ounce and 83.86 USD per ton, respectively, the difference in total revenue between two scenarios with 15% and 20% of the dilution level will exceed 4 million dollars per year; further, the recovery percentage of the corresponding mineral dressing plant reduces from 95% to 94.4% as the dilution increases from 15% to 20% [4].

A study on a metal vein mine with an annual capacity of 500,000 tons of ore indicated that 25 cm-dilution would end up with an excessive cost of 25 USD per ton. It was further revealed that as the width of the ore body became smaller, significantly higher annual costs were incurred by the same 25 cm-dilution [2]. Investigation of the effect of dilution on profitability of a gold vein mine showed that if the dilution level exceeded 40%, the mine would lose its profitability, ending up with some loss [5]. Investigating the economic losses incurred by dilution in thin vein mine of tungsten in China, where the cut-and-fill stoping method was used to exploit the mine, showed that 44.4% of the losses incurred by dilution were avoidable [6]. The associated cost with 14% dilution in a gold mine was determined to be about 38 USD per ton. In a year, this sums up at 5.4 million USD [7]. Investigations have shown that dilution has been the reason for abandoning numerous underground mines such as the Mount Todd gold mine [8].

In the present paper, expressing various definitions proposed for dilution and its types, and also introducing the factors affecting the development of dilution in underground stopes while taking a look at the dilution evaluation and prediction models proposed by different researchers, it can be seen that the dilution prediction is not a simple task to undertake; it is rather a very complex and complicated one. As such, many researchers have failed to account for some of very important factors when developed their proposed models, emphasizing on a single or just few parameters.

In the present research work, following a new approach and employing the existing literature and experts' opinions, a new classification system is presented for dilution, where an attempt is made to take into account all the parameters affecting dilution in the cut-and-fill stoping method and incorporate them into the model as much as possible. For this purpose, questionnaires were prepared based on the importance of the parameters and sent to academic and industrial experts; accordingly, the parameters were weighted using FDAHP. Following with this research work, dividing variation ranges of the parameters into specific intervals and considering their weights, a novel quantitative index, namely stope unplanned dilution index (SUDI) was defined as the sum of scores attributed to different parameters. This parameter can be used to classify the mining stopes into five classes based on their susceptibility to dilution.

## 2. Dilution definitions and models

The contamination of ore with wastes or materials of lower than cut-off grade is referred to as dilution. In the underground stoping methods, dilution happens as a result of falling of roof and walls, cutting of roof and floor, and loading of waste materials. In general categorization, dilution can be classified into two categories: planned (internal) dilution and unplanned (external) dilution. Planned dilution refers to a situation where, considering the deposit characteristics and in order to design the stope, some rock materials are removed from hanging wall and footwall. Unplanned dilution, however, happens outside stope design premise as a result of over-break of the hanging wall by undesired fractures. Final dilution can be defined as the sum of planned and unplanned dilutions [5]. The zones corresponding to these definitions are demonstrated in Figure 1.

This method has been developed based on a set of quantitative parameters of ore and surrounding rocks as well as some geometrical parameters of the layer under exploitation. Knowing the source of dilution (cutting or falling of hanging wall or footwall) represents an advantage of this method.

Popov [6] has defined dilution as the ratio of contaminated waste with total ore to exploited waste. Pakalnis [7] has proposed the definitions presented in Table 1. Among these definitions, the first two ones, Equations (1) and (2), are widely in use.

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

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

where

D represents the dilution level in%,

W is the exploited waste in tons, and

O is the exploited ore in tons.

Agoshkov et al. [8] has defined dilution as a reduction in the grade of exploited ore compared to the grade of intact ore body. Proposed by them for calculating the dilution, Equation (3) can be used for all underground stoping methods.

$$D = \frac{C_p - C_a}{C_p - C_r} \times 100 \qquad (3)$$

where

D is the percent of dilution level,

Cp is the percent of valuable metal in intact ore body,

$C_a$ is the percent of valuable metal in exploited matter, and

$C_r$ is the percent of valuable metal in the surrounding rocks.

Dunne and Pakalnis [9] have defined dilution as an expression that is independent from the stope width: average depth per unit area of stope wall fall (in meters per square meter). This way of dilution calculation is appropriate for analyzing the parameters affecting dilution, while the dilution expressed in percentage is suitable for economic analyses. Clark and Pakalnis [10] have modified the definition presented by Dunne and Pakalnis [9] by introducing the novel concept of equivalent linear over-break/slough (ELOS), which is expressed in meters and calculated as the falling or over-break volume divided by the area of stope wall.

As of now, numerous studies have been performed on dilution in underground stopes. A lot of factors and complexities associated with the dilution mechanism have made many previous studies to focus on few parameters. The stability graph methods including Mathews' stability graph method [11], Potvin's modified stability graph method [12], and also Mathews' improved stability graph [13] are among the most known underground stope stability methods used for dilution estimation, and these methods have been accepted by both industry and academia [1]. Stability graph is composed of two parameters: hydraulic radius of stope wall (ratio of stope wall area to its perimeter) and stability number, N or N′ (modified N). The value for N (N′) is obtained from a combination of tunneling quality index, Q (Q′ modified Q) along with some other factors including stress, gravity, and orientation of discontinuities. Many researchers have tried to improve the stability graph method. Among others, one may refer to Scoble and Moss [5], who proposed the dilution lines, and Nikson [14], who proposed the effect of restrain inclination on the stope wall stability, and then Clark's dilution graph [15], who introduced the novel concept of ELOS into the field of stability graph. In addition, such researchers as Hadjigeorgiou et al. [16] and Clark and Pakalnis [10] have imposed modifications in the gravity factor. However, so far, the stability graph methods have failed to account for the drilling- and blasting-associated factors. The main barrier keeping them from applying the stability graph for dilution evaluation and prediction is the very complex mechanism through which this phenomenon develops. Indeed, a wide range of factors, rather than a single parameter or a set of few parameters potentially contributing to dilution in a stope, should be taken into account.

In general, the models already proposed for dilution determination can be classified into three categories: empirical, soft computing, and numerical modeling. Table 2 reports a summary of the studies performed on this topic so far. This table presents the type of model along with the effective factors involving it.

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 [17]. Later on, other researchers used CMS data in their studies on dilution [1, 3, 18-29]. 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. The data obtained with this system helps researchers consider the reasons behind different amounts of dilution in stopes depending on various factors such as stope geometry, geologic setting, drilling, blasting, and operational factors. As such, the back- analysis resulted from CMS data can significantly contribute to dilution prediction for designing underground stopes.

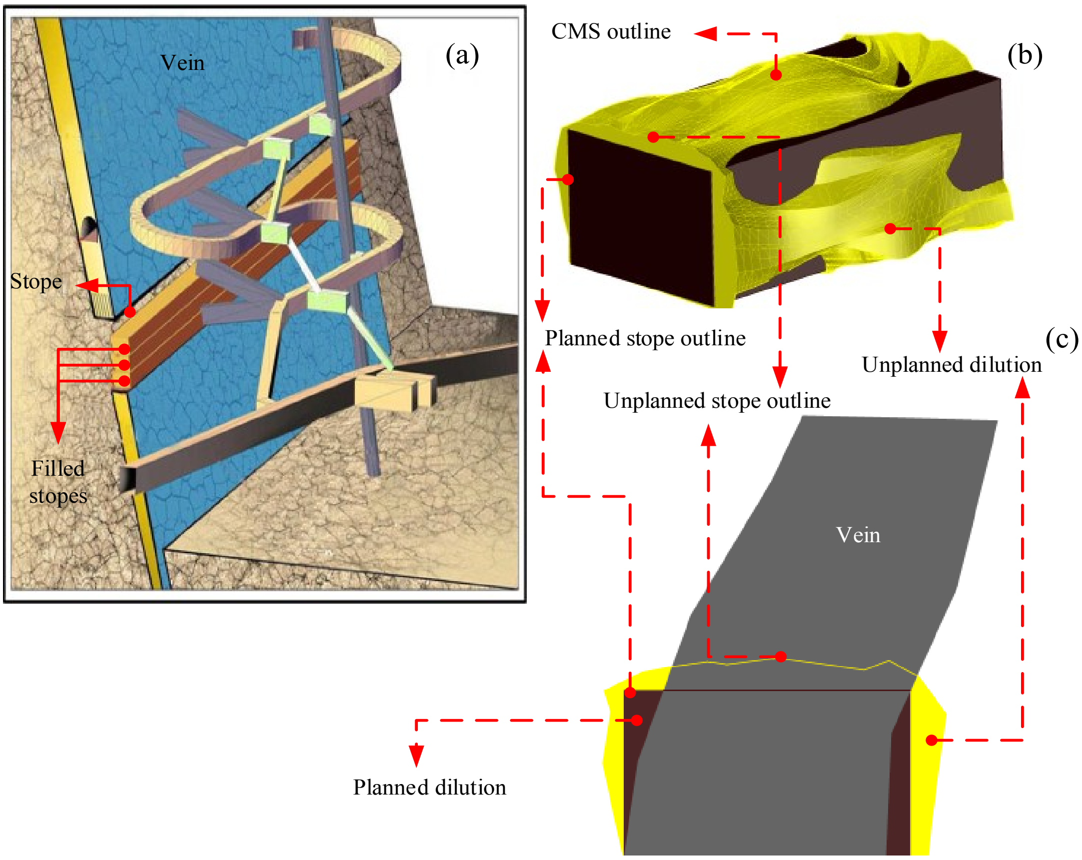



**Figure 1. a) A schematic view of cut-and-fill mining method. b) A designed and actual stope after production captured by cavity monitoring system (CMS). c) Planned and unplanned dilution in section of the stope.**

**Table 1. Variation methods of estimating dilution [7].**

| |
|---|
| % Dilution = (weight of external slough × 100)/(weigh of ore reserves) |
| Dilution = (undiluted in place grade (DDH))/(sample assay grade at draw point) |
| Dilution = (undiluted in place grade reserves)/(mill head grades obtained for same tonnage) |
| Dilution = (total wasted tonnage)/(total tons mined) |
| Dilution = (total wasted mined)/(tons of ore reserves estimated) |
| Difference between tonnage mucked and that blasted |
| Difference between tonnage of backfill placed and that theoretically required to fill "ore reserves" void |
| Dilution is visually observed and assessed |
| "x" amount of feet in the footwall plus "y" amount of feet in the hanging wall divided by the stope width |
| Historical average over past 10 years = (actual tons drawn from stopes)/calculated reserve tonnage |

### 3. Parameters affecting unplanned dilution in cut-and-fill stoping method

In this section, using the available literature and the experts' opinions, an attempt was made to identify all the parameters affecting dilution in cut-and-fill stopes. These parameters are reported in Table 3, where the parameters are grouped into four categories: design factors, drilling and blasting factors, geologic factors, and operational factors. The category of design factors includes parameters determining the stope geometry (i.e. stope length, width, and height) along with the geometry of unsupported section (hydraulic radius). Drilling and blasting factors encompass inaccuracy in drilling, powder factor, blast vibration, and hole type (horizontal or vertical). In the category of geological factors, one can find the stope wall quality, stope depth, and geological structures including foliation and fault (covering the location with respect to stope, position angle, and internal friction angle across fault plane). Finally, the operational factors include loading of wastes as well as stope filling time, filling material, and filling method (mechanical, pneumatic or hydraulic).

**Table 2. Summary of studies performed on dilution.**

| **Category** | **Parameters** | **[14]** | **[9]** | **[10]** | **[11]** | **[24]** | **[25]** | **[3]** | **[2]** | **[1]** |
|---|---|---|---|---|---|---|---|---|---|---|
| | | **E** | **E** | **E** | **N** | **E** | **N** | **N** | **E** | **S** |
| **Geology** | Ore thickness | | ● | | | | | | | |
| | Stope depth | | | | | | | ● | | |
| | Angle between fault and stope surface | | | | ● | | | | | |
| | Fault friction angle | | | | ● | | | | | |
| | Fault position | | | | ● | | | | | |
| | Rock mass rating (RMR) | ● | | | | | | | | |
| | Stress environment | | | | | ● | ● | ● | | |
| | Horizontal-to-vertical stress ratio (K) | | | | | | | | | ● |
| | Adjusted Q rate (Q') | | | | | | | | | ● |
| | Modified stability number (N') | | | ● | | ● | | | ● | |
| **Stope design** | Hydraulic radius (HR) | ● | | ● | | ● | ● | | ● | |
| | Aspect ratio (length/height) | | | | | | | ● | | ● |
| | Undercutting Factor (UF) | | | | | ● | | | | |
| | Stope designed height | | | | | | | | | |
| | Stope true height | | | | | | | | | |
| | Stope strike length | | | | | | | ● | | |
| | Stope vertical length | | | | | | | ● | | |
| | Stope hanging-wall dip | | | | | | | ● | | |
| | Stope width | | | | | | | ● | | |
| | Stope breakthrough to a nearby drift and/or stope | | | | | | | | | ● |
| **Drilling and blasting** | Blast hole pattern | | | | | ● | | | ● | |
| | Length of blast hole | | | | | | | | ● | ● |
| | Diameter of blast hole | | | | | | | | | ● |
| | Inaccurate drilling | | | | | | | | | ● |
| | Powder factor | | | | | | | | | ● |
| | Space and burden ratio | | | | | | | | | ● |
| | Blast vibration | | | | | | | | | |
| **Operation** | Planned tones of stope | | | | | | | | | ● |
| | Excavation Rate (ER) | ● | | | | | | | | |
| | Stope type and extraction sequence | | | | | | ● | ● | | |
| | Expose time | | | | | ● | | | | |

E: Empirical, S: Soft Computing, N: Numerical

**Table 3. Categories and their parameters affecting unplanned dilution in cut-and-fill stoping.**

| Category | Parameters | Symbols |
|---|---|---|
| Stope design | Stope height | $F_1$ |
| | Stope length | $F_2$ |
| | Stope width | $F_3$ |
| | Hydraulic radius | $F_4$ |
| Drilling and blasting | Inaccurate drilling | $F_5$ |
| | Powder factor | $F_6$ |
| | Blast vibration | $F_7$ |
| | Horizontal hole | $F_8$ |
| | Vertical hole | $F_9$ |
| Geology | Walls quality | $F_{10}$ |
| | Stress environment | $F_{11}$ |
| | Angle between fault and stope surface | $F_{12}$ |
| | Fault friction angle | $F_{13}$ |
| | Fault position | $F_{14}$ |
| | Foliation | $F_{15}$ |
| | Stope depth | $F_{16}$ |
| Operating | Loading waste | $F_{17}$ |
| | Filling method | $F_{18}$ |
| | Filling materials | $F_{19}$ |
| | Filling time | $F_{20}$ |

## 4. Weighting of parameters affecting unplanned dilution

In this work, FDAHP was utilized to weight the parameters affecting dilution in the cut-and-fill stoping method. FDAHP is indeed a combination of AHP and FD. AHP was first introduced by Saaty [30] when parameters were weighted based on pairwise comparisons expressed in terms of pairwise matrices

Liu and Chen [31] have defined different steps to implement FDAHP, as follow. As of now, different applications have been proposed for FDAHP in mining engineering [32-42].

### 4.1. Survey of experts

In this step, a survey was undertaken to acquire, either qualitatively or preferably quantitatively (if possible), different experts' opinions on the parameters affecting a specific phenomenon or decision.

### 4.2. Calculation of fuzzy numbers

In order to calculate the fuzzy numbers ($\tilde{a}_{ij}$), the experts' opinions were directly considered. In this step, fuzzy numbers could be calculated using different membership functions such as triangular or trapezoidal methods. Considering the wide application and ease-of-calculation via the triangular method, the fuzzy numbers were calculated as shown in Figure 2. In this case, a fuzzy number was defined as in Equations (4-7):

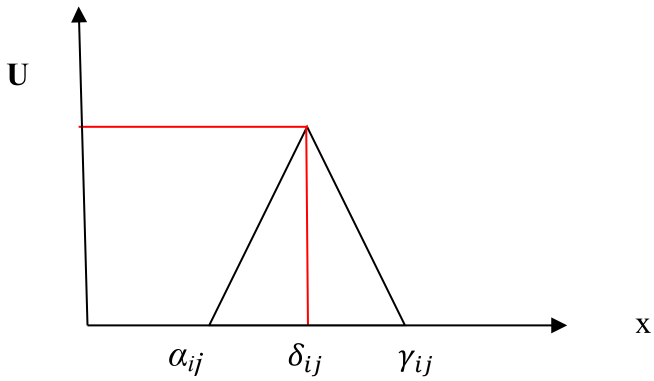



**Figure 2. Membership function of fuzzy Delphi method.**

$$a_{ij} = (\alpha_{ij}, \delta_{ij}, \gamma_{ij}) \tag{4}$$

$$\alpha_{ij} = Min(\beta_{ijk}), k = 1,...n \tag{5}$$

$$\delta_{ij} = \left( \prod_{k=1}^{n} \beta_{ijk} \right)^{1/n}, k = 1,...n \tag{6}$$

$$\gamma_{ij} = Max\left(\beta_{ijk}\right), k = 1,...n \tag{7}$$

where $\beta_{ijk}$ represents the relative importance of ith parameter on jth parameter in view of kth expert, $\gamma_{ij}$ and $\alpha_{ij}$ are the lower and upper bounds of the opinions expressed by participants, and $\delta_{ij}$ is the geometrical average of the opinions given by participants. Components of the fuzzy numbers are defined in such a way that Equation

(8) is met, with the component values always ranging within [1/9, 9].

$$\alpha_{ij} \leq \delta_{ij} \leq \gamma_{ij} \tag{8}$$

### 4.3. Formation of inverse fuzzy matrix

In this step, considering the fuzzy numbers obtained in the preceding step, a fuzzy matrix was formed between different parameters, as described by Equations (9 and10):

$$\tilde{A} \quad \left[\tilde{a}_{ij}\right] \tilde{a}_{ij} \quad \tilde{a}_{ij} \qquad i \quad j = 1,2,...n \tag{9}$$

$$\tilde{A} \quad \begin{pmatrix} (1,1,1) & (\alpha_{12},\delta_{12},\gamma_{12}) & (\alpha_{13},\delta_{13},\gamma_{13}) \\ (1/\gamma_{12},1/\delta_{12},1/\alpha_{12}) & (1,1,1) & (\alpha_{23},\delta_{23},\gamma_{23}) \\ (1/\gamma_{13},1/\delta_{13},1/\alpha_{13}) & (1/\gamma_{23},1/\delta_{23},1/\alpha_{23}) & (1,1,1) \end{pmatrix} \tag{10}$$

### 4.4. Calculation of relative fuzzy weights of parameters

In order to calculate the relative fuzzy weights of the parameters, Equations (11and12) were utilized.

$$\tilde{Z}_i \quad \left[\tilde{a}_{ij} \qquad \tilde{a}_{in}\right]^{1/n} \tag{11}$$

$$\tilde{W}_i \quad \tilde{Z}_i \quad \tilde{Z}_i \quad ... \quad \tilde{Z}_n \tag{12}$$

where

$\alpha_2, \delta_1 \times \delta_2, \gamma_1 \times \gamma_2); \otimes, \oplus$ denote the fuzzy multiplication and summation operators, respectively, and

$\tilde{W}_i$ is the row vector indicating fuzzy weight of the ith parameter.

### 4.5. Defuzzification of weights of parameters

In this step, in order to defuzzify the parameters according to Equation (13), the geometrical average of fuzzy components of the weights was obtained so that the parameters had their weights expressed as deterministic figures.

$$W_i = \left( \prod_{j=1}^{3} w_{ij} \right)^{1/3} \tag{13}$$

### 4.6. Weighting of parameters

Different FDAHP steps were taken to weight the categories of factors and parameters affecting unplanned 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 weak important, weak important, moderate important, strong important, and very strong important. Tables 4 and 5 demonstrate examples of the survey forms. Once the questionnaires were received, some scores were attributed to the descriptive terms: 9, 7, 5, 3, and 1, respectively.

**Table 4. Categories of factors in questionnaires sent to experts.**

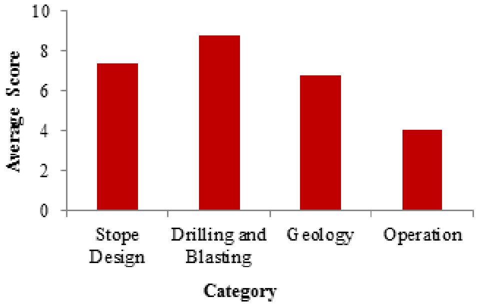
| Category | Importance of each category | | | | |
|---|---|---|---|---|---|
| | Very weak | Weak | Moderate | Strong | Very strong |
| Stope design | | | | | |
| Drilling and blasting | | | | | |
| Geology | | | | | |
| Operating | | | | | |

**Table 5. A sample of parameters in questionnaires sent to experts.**

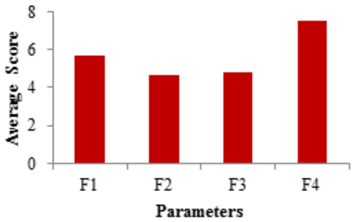
| Stope design parameters | Importance of each category | | | | |
|---|---|---|---|---|---|
| | Very weak | Weak | Moderate | Strong | Very strong |
| Stope height | | | | | |
| Stope length | | | | | |
| Stope width | | | | | |
| Hydraulic radius | | | | | |

According to the experts' opinions, the average score of each category of factors and the corresponding parameters was calculated, as is shown by the bar plot in Figure 3. As it can be seen in Figure 3(a), out of the considered categories of factors, the drilling and blasting factors and the operational factors achieved the highest and lowest average scores, respectively. In Figure 3(b), one may observe that hydraulic radius has attained the highest score among the design parameters, while Figure 3(c) shows that inaccuracy represents the parameter of highest score among the drilling- and blasting-associated parameters, with the stope wall quality obtaining the highest score among the geological parameters (Figure 3(d)). Finally, the filling time achieved the highest score among all the other operational parameters (Figure 3(e)).

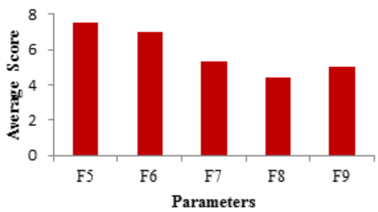
Following the completion of survey and evaluation of its results, first, the pairwise comparison matrix was developed for the categories of factors. Triangular membership function, and hence, triangular fuzzy number were used according to Figure 2 and Equations (9 and10) to form the matrix. Table 6 presents the corresponding pairwise comparison matrix to the categories of factors. In the next step, using Equation (11), the fuzzy numbers $\tilde{Z}$ and $\tilde{Z}_i$ were calculated for different categories of factors, whose fuzzy and non-fuzzy weights were then obtained from Equations (12) and (13). Table 7 reports the results of the aforementioned calculations. Further, for the parameters included in each category of factors, the Delphi-fuzzy pairwise comparison matrix was formed following a similar approach and using this matrix; the fuzzy numbers $\tilde{Z}$ and $\tilde{Z}_i$ were obtained for different parameters along with their fuzzy and non-fuzzy weights. The results of calculations are reported on the hierarchy diagram in Figure 4. Figure 5 shows the final weights of the parameters on a bar plot.



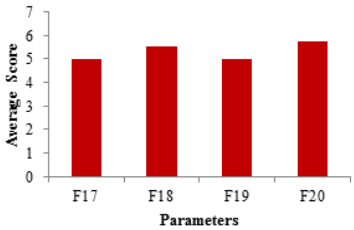

a) Average score of categories.



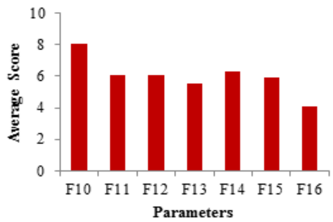

b) Average score of design parameters




c) Average score of drilling and blasting parameters




d) Average score of geology parameters




e) Average score operation parameters

**Figure 3. Average score of categories and their parameters.**

**Table 6. Pairwise comparison matrix of categories.**

| | Stope design | | | Drilling and blasting | | | Geology | | | Operating | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Stope design | 1 | 1 | 1 | 0.7778 | 0.833 | 1 | 0.78 | 1.096 | 1.8 | 1 | 1.878 | 3 |
| Drilling and blasting | 1 | 1.2 | 1.2857 | 1 | 1 | 1 | 0.78 | 1.316 | 1.8 | 1.286 | 2.2546 | 3 |
| Geology | 0.5556 | 0.91 | 1.2857 | 0.5556 | 0.76 | 1.2857 | 1 | 1 | 1 | 1 | 1.7133 | 3 |
| Operating | 0.3333 | 0.53 | 1 | 0.3333 | 0.444 | 0.7778 | 0.33 | 0.584 | 1 | 1 | 1 | 1 |

**Table 7. Results of weighting of categories.**

| | $\tilde{Z}$ | | | $\tilde{Z}_l$ | | | $\tilde{W}_l$ | | | $W_i$ |
|---|---|---|---|---|---|---|---|---|---|---|
| Stope design | 0.6049 | 1.71 | 5.4 | 0.8819 | 1.144 | 1.5244 | 0.16 | 0.274 | 0.4972 | **0.2783** |
| Drilling and blasting | 1 | 3.56 | 6.9429 | 1 | 1.374 | 1.6232 | 0.18 | 0.329 | 0.5294 | **0.3150** |
| Geology | 0.3086 | 1.19 | 4.9592 | 0.7454 | 1.044 | 1.4923 | 0.13 | 0.25 | 0.4867 | **0.2534** |
| Operating | 0.037 | 0.14 | 0.7778 | 0.4387 | 0.609 | 0.9391 | 0.08 | 0.146 | 0.3063 | **0.1521** |

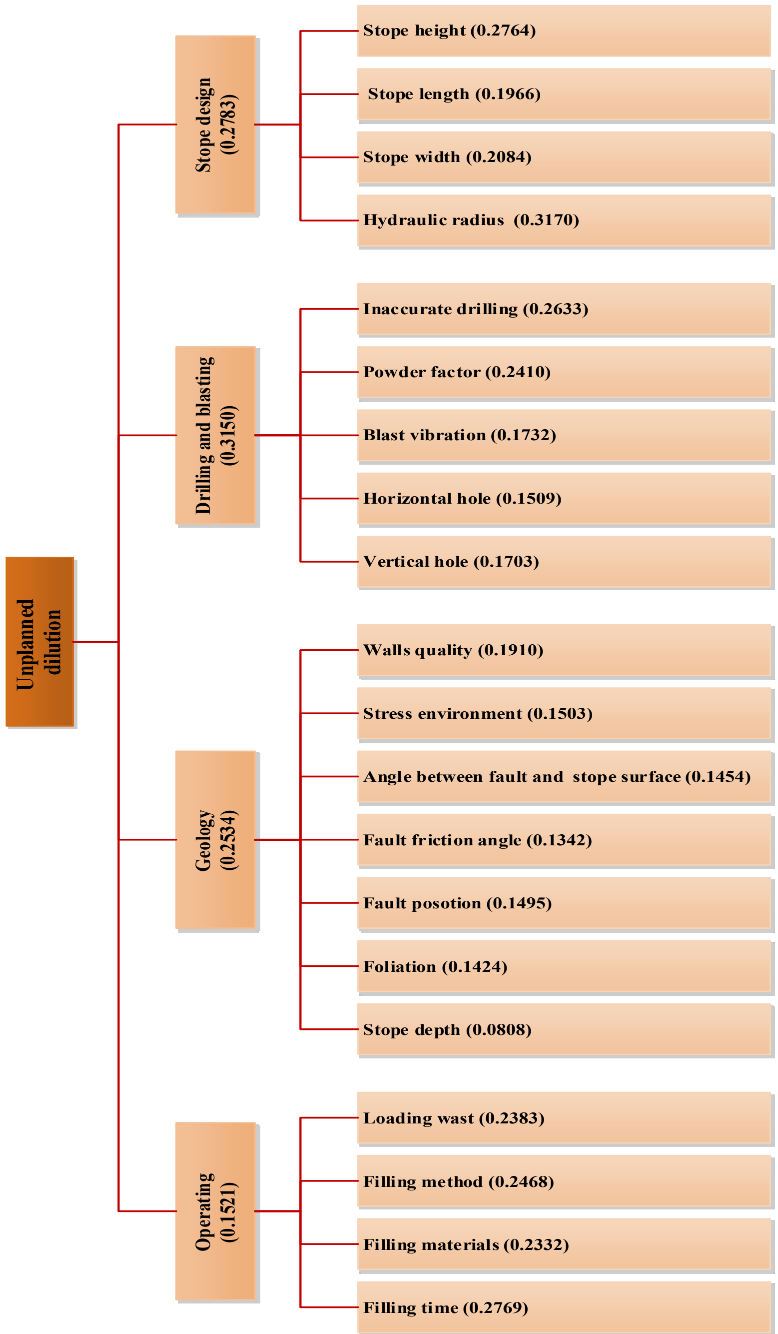



**Figure 4. Hierarchy diagram of categories and parameters of unplanned dilution.**

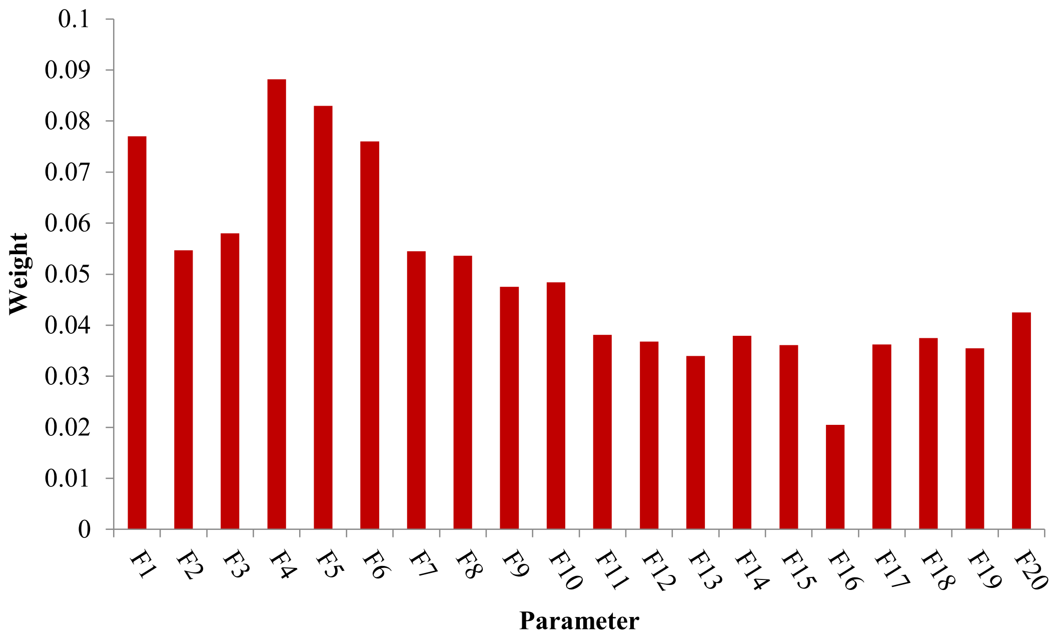



**Figure 5. Final weights of parameters.**

## 5. Parameter selection and rating for new classification system

In the discussion of the determining factors affecting unplanned dilution, an attempt was made to account for each and any parameter, even those of small contributions to unplanned dilution. However, as a matter of fact, not all of the parameters can be actually utilized in an unplanned dilution classification system because for a classification system, a significant parameter selection and composition is particularly important as it serves as one of the most important principles when designing a classification system. In other words, a classification system will be acceptable when it is not only simple but also capable of covering all factors with a minimum number of parameters, making it impractical and even unreasonable to account for each and any possible effective factor in a classification system. Moreover, there are chances that some parameters exhibit some sort of overlapping so that a single parameter alone may represent several parameters in the classification system so that the represented parameters can be omitted with their weights transferred to the representative parameter. For these reasons, in order to present the novel classification system, out of the 20 parameters identified in the previous steps, 10 parameters were selected, with the other parameters omitted. For example, in the category of stope design factors, choosing hydraulic radius parameter, tow parameters (stope height and length) were omitted as the hydraulic radius represented them well, with the weights of these two parameters added to that of hydraulic radius. Following a similar line of reasoning, in the category of drilling and blasting factors, the parameters inaccurate drilling, powder factor, and blast vibration were selected, while the parameters vertical and horizontal drillings were omitted with their weights transferred to inaccurate drilling. Further, in the category of geological parameters, the stope wall quality was expressed in terms of the modified stability number ($N'$), and since the number covers associated parameters with joints and stress.

Configurations, stress configuration, and associated parameters were omitted with fault configuration with their weights added to that of the stope wall quality. Finally, in the category of operational factors, since the filling method largely determines the filling material and loading of wastes, the filling method was selected, while omitting the other two parameters from the category with their weights added to that of the filling method. Following these selection and omission procedures, the omitted parameters were removed from calculations and the remaining parameters were used to present a new unplanned dilution classification system. The selected parameters along with their weights are brought in Table 8.

**Table 8. Parameters selected and their weights.**

| Category | Parameters | Weight |
|---|---|---|
| Stope design | Hydraulic radius | 0.2199 |
| | Stope width | 0.0580 |
| Drilling and blasting | Inaccurate drilling | 0.1841 |
| | Powder factor | 0.0760 |
| | Blast vibration | 0.0545 |
| Geology | Walls quality | 0.1992 |
| | Foliation | 0.0361 |
| | Stope depth | 0.0205 |
| Operating | Filling method | 0.1092 |
| | Filling time | 0.0425 |

### 5.1. Hydraulic radius

Laubscher [43] 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 is 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 unplanned dilution level is expected.

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

Considering the calculated weight for HR (about 0.22) and dimensions common of stopes used for applying the cut-and-fill method, rating of this parameter in the new classification system is demonstrated in Table 9.

**Table 9. Rating the HR; in new classification.**

| HR, (m) | <2 | 2-2.5 | 2.5-2.75 | 2.75-3 | >3 |
|---|---|---|---|---|---|
| Description | Very low | Low | Moderate | high | Very high |
| Rating | 2 | 5.5 | 11 | 15.5 | 22 |

### 5.2. 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 the risk of falling and unplanned dilution [44]. Considering the calculated weight for width (about 0.06) and taking into account the width of common stopes used in cut-and-fill mining, the rating of this parameter in the new classification system is demonstrated in Table 10.

**Table 10. Rating stope width; in new classification.**

| Stope width | <3 | 3-4 | 4-5 | 5-6 | >6 |
|---|---|---|---|---|---|
| Description | Very narrow | Narrow | Moderate | Wide | Very wide |
| Rating | 0.5 | 1.5 | 3 | 4 | 6 |

### 5.3. Inaccurate drilling

Inaccurate drilling, particularly for the holes near the hanging wall and footwall, is among the important parameters contributing to unplanned 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 contributes to unplanned dilution. Drilling of vertical holes is associated with smaller deals of error when compared against directional holes because the criteria taken into account by the operator for angle measurement in a directional hole are of lower accuracy than those set in the design phase.

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 unplanned dilution was estimated to be about 16% [44].

In the proposed classification system, the total inaccuracy in drilling was expressed in terms of the hole deviation percentage. Considering the calculated weight for this parameter (about 0.185), the rating of this parameter in the new classification system is demonstrated in Table 11.

**Table 11. Rating hole deviation; in new classification.**

| Hole deviation | <5 | 5-10 | 10-15 | 15-20 | >20 |
|---|---|---|---|---|---|
| Description | Very low deviation | Low deviation | Moderate | Much deviation | Too much deviation |
| Rating | 2 | 4.5 | 9 | 13 | 18.5 |

### 5.4. 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 gram of explosive per cubic meter/ton of rock. Powder factor is a function of 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 excessive over-break, and hence, unplanned dilution, while a lower PF than the optimum amount will end up with a loss in exploited ore. As such, the amount of PF can play a significant role among the other unplanned dilution-generating parameters.

Considering the calculated weight for this parameter (about 0.075) and taking into account the PF set in common stopes used in cut-and-fill mining, the rating of this parameter in the new classification system is demonstrated in Table 12.

**Table 12. Rating PF; in new classification.**

| PF, (kg/m3) | <0.75 | 0.75-1.25 | 1.25-1.75 | 1.75-2.25 | >2.25 |
|---|---|---|---|---|---|
| Description | Very low | Low | Moderate | High | Very high |
| Rating | 1 | 2 | 4 | 5.5 | 7.5 |

### 5.5. 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 vibrate, and since the magnitude of strains in an elastic material is proportional to particle vibration velocity, PPV serves as an appropriate measure for determining blast vibration damages in rock; the PPV value can be determined using a seismogram device. Langefors and Kihlstrom [45] have proposed a criterion for tunnels wherein a PPV of 305 mm/s results in the falling of rocks, while a PPV of 610 mm/s results in the formation of new fractures. Bauer and Calder [46] have observed that failure occurring in intact rocks of less than 254 mm/s in PPV. However, any PPV within 254–635 mm/s has been found to cause tensile surface fractures, with PPV values ranging within 635–2540 mm/s, resulting in strong tensile stresses and circular failures; ultimately, rock mass has been seen to fail at a PPV of 2540 mm/s. Most rock masses tend to be damaged at PPVs higher than 635 mm/s [44].

Based upon the researchers' findings, PPV is related to the distance and explosive charge at each delay Equation (15) [47]:

$$PPV = K.R^{-A} W^{B} \tag{15}$$

where:

*PPV:* peak particle velocity, mm/s;

*R:* distance from monitoring location to blasting location;

*W:* charge per delay;

*B, K, A:* coefficients related to local characteristics and type of explosive material calculated via regressing given the values for PPV (as recorded by seismogram) and consumed charge per delay.

As of now, a large deal of research works have been done to propose various forms of the above relationship. The relations have been obtained under different sets of conditions and via vibration intensity measurements. Of these, the most well-known is Equation (16); which was first presented by Attewel [48] and published by United States Bureau of Mines (USBM).

$$PPV = K\left(\frac{R}{\sqrt{W}}\right)^{-a} \tag{16}$$

where the constants $a$ and $K$ are related to the amplitude and damping of the shockwaves, respectively. The PPV data is scaled according to distance relationships, and then subjected to statistical analysis to determine the constants $a$ and $K$ in the PPV relationship [49].

In Equation (15), the term $\left(\frac{R}{W}\right)^{-\beta}$ is referred to as the scaled distance. As such, the scaled distance for the present relationship is $\frac{R}{\sqrt{W}}$.

Considering the two pain parameters of charge per delay and the distance from monitoring location to the blasting location and using empirical correlations while determining constant coefficients by undertaking some blasting tests, one may come with an estimation of the vibration intensity.

Considering the calculated weight for this parameter (about 0.055) and taking into account the value for this parameter in common stopes used in cut-and-fill mining, the rating of this parameter in the new classification system is demonstrated in Table 13.

**Table 13. Rating blast vibration; in new classification.**

| Blast vibration, (PPV) | <250 | 250-500 | 500-750 | 750-1000 | >1000 |
|---|---|---|---|---|---|
| Description | Very low vibration | Low vibration | Moderate vibration | Much vibration | Too much vibration |
| Rating | 0.5 | 1.5 | 3 | 4 | 5.5 |

### 5.6. 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 [50], 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 (17).

$$N' = Q' \times A \times B \times C \tag{17}$$

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.

The rock tunneling quality index or Q classification system has been introduced by Barton et al. [51] according to Equation (18).

$$Q' = \frac{RQD}{j_n} \times \frac{j_r}{j_a} \times \frac{j_w}{SRF} \tag{18}$$

where RQD represents the rock-quality designation, $j_n$ refers to the number of joint sets, $j_r$ represents the joint surface roughness, $j_w$ is the joint water reduction factor, $j_a$ denotes the weathering level of joint surfaces, and SRF is the stress reduction factor.

However, the modified rock tunneling quality index or Q′ classification system follows the same line of calculation as in the Q classification system, except that the value for SRF is assumed to be unit (1) in this case. Moreover, in cases where underground space is drilled into dry rocks, one can further set $j_w$ to 1. As such, Q′ is free of the $\frac{j_w}{SRF}$ in Q index, and can be expressed as in Equation (19).

$$Q' = \frac{RQD}{j_n} \times \frac{j_r}{j_a} \tag{19}$$

Considering the calculated weight for wall quality (about 0.2) and taking into account the possible values for modified stability number in common stopes used in cut-and-fill mining, the rating of this parameter in the new classification system is demonstrated in Table 14.

**Table 14. Rating modified stability number (N') in new classification.**

| Modified stability number (N') | >10 | 10-5 | 5-2 | 2-1 | <1 |
|---|---|---|---|---|---|
| Description | Too much stability | Much stability | Moderate stability | Low stability | Very low stability |
| Rating | 2 | 5 | 10 | 14 | 20 |

### 5.7. Foliation

Foliation is another parameter contributing to the stope wall rock fall and unplanned dilution. Where foliation orientation is oblique with respect to the wall direction, one may end up with a minimum stability and a maximum unplanned dilution. However, where foliation is developed parallel to the stope wall, one may expect just a fair stability and a fair unplanned dilution, and finally, where foliation is developed normal to the stope wall, a maximum wall stability, and hence, a minimum unplanned dilution is expected [44].

Considering the calculated weight for foliation (about 0.035) and taking into account various foliation configurations in common stopes used in cut-and-fill mining, the rating of this parameter in the new classification system is demonstrated in Table 15.

**Table 15. Rating foliation; in new classification.**

| Foliation | Perpendicular iso strike | Perpendicular un strike | Parallel | Oblique iso strike | Oblique un strike |
|---|---|---|---|---|---|
| Description | Very good | Good | Moderate | Bad | Very bad |
| Rating | 0.5 | 1 | 1.5 | 2 | 3.5 |

### 5.8. 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 unplanned dilution.

Considering the calculated weight for stope depth (about 0.02) and taking into account the possible depths of common stopes used in cut-and-fill mining, the rating of this parameter in the new classification system is demonstrated in Table 16.

**Table 16. Rating stope depth; in new classification.**

| Stope depth | <100 | 100-200 | 200-300 | 300-400 | <400 |
|---|---|---|---|---|---|
| Description | Very shallow | Shallow | Moderate | Much deep | Too much deep |
| Rating | 0 | 0.5 | 1 | 1.5 | 2 |

### 5.9. Filling method

Stability of hanging wall and unplanned dilution control are among the most important reasons for filling in the cut-and-fill stoping method. 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 large convergence. However, level of compaction increases by filling materials via the mechanical, pneumatic and hydraulic filling methods, respectively [52]. Considering the calculated weight for the filling method (about 0.11) and taking into account the common filling methods used in cut-and-fill stoping, the rating of this parameter in the new classification system is demonstrated in Table 17.

**Table 17. Rating filling method; in new classification.**

| Filling method | Hydraulic filling | Pneumatic filling | Mechanical filling | Gravity filling | Hand filling |
|---|---|---|---|---|---|
| Description | Very good | Good | Moderate | Bad | Very bad |
| Rating | 1 | 2.5 | 5.5 | 7.5 | 11 |

### 5.10. Filling time

Since in the cut-and-fill stoping method one of the reseals for filling cavities is to support walls, the filling time is very important in this stoping method because the filling materials should be in place timely to prevent excessive wall fall, and hence, unplanned dilution.

Considering the calculated weight for the filling time (about 0.04) and taking into account the common conditions in the cut-and-fill stoping method, the rating of this parameter in the new classification system is demonstrated in Table 18.

In the newly proposed classification system, in order to rate different values for each parameter, maximum score was given to the interval within the variation range of the parameter at which the parameter imposes the largest contribution to unplanned dilution (worst conditions), describing the interval as very poor. Accordingly, 70%, 50%, 25%, and 10% of the maximum score were attributed to poor, fair, strong, and very strong conditions, respectively. Following this score budgeting, an attempt was made to incorporate non-linearity into the classification system because a non-linear classification system is more likely to provide acceptable results in terms of classifying poor rock masses [53]. Following with this paper, a brief description is given on the effect of each parameter on unplanned dilution and the way its variation range is rated.

**Table 18. Rating filling delay; in new classification.**

| Filling time | Continuous loading | After loading quarter stope | After loading third the stope | After loading half the stope | After loading throughout the stope |
|---|---|---|---|---|---|
| Description | Very soon | Soon | Moderate | Late | Very late |
| Rating | 0.5 | 1 | 2 | 3 | 4 |

## 6. New unplanned dilution classification system and presentation of unplanned dilution index

The new classification system together with the value for each parameter is presented in Table 19. In the new classification system, the level of potential susceptibility of a cut-and-fill stope to unplanned dilution is investigated based on a novel quantitative index called the stope unplanned dilution index (SUDI). Using this new index, an overall score between 10 and 100 is attributed to the exploitation stopes, indicating the susceptibility of the stopes to unplanned dilution qualitatively, as expressed in terms of either of the five classes very low, low, moderate, high or very high. In other words, according to Table 20, one can qualitatively classify robustness against unplanned dilution of stopes into five classes of very strong, strong, moderate, weak, and very weak.

## 7. Case study

The proposed classification system has been used in Venarch Manganese Mine (Qom, Iran). Venarch Manganese Mine is located 27 km to the western south of Qom within 2 km of Venrach Village (longitude: 50°45′42″; latitude: 34°25′3″). With a reserve of more than 8.6 million tons, as of now, the mine is the largest Manganese mine across Middle East, and produces 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. As a result of geologic phenomena, the deposit is well-fragmented, and each fragment of the mine is exploited separately. All parts of the mine are being exploited via the cut-and-fill stoping method. The classification method has been applied in 10 stopes in different parts of Venarch Manganese Mines. For this purpose, the geometrical, geological, drilling and blasting, and operational factors of unplanned dilution of stopes are rated under consideration and then stopes have been classified. Next a semi-automatic cavity monitoring system (CMS) has been implemented in the stopes. Table 21 reports the results of these items. In order to determine the relationship between the stope unplanned dilution index (SUDI) and the actual amount of dilution (in ELOS) the regression analysis method has been used. The relationship with a correlation coefficient ($R^2$ = 0.8957) is shown in Figure 6.

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 with 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. Specifying the design and actual volumes of the stope, ELOS was calculated. Figure 7 demonstrates the workflow for calculating actual the stope volume and determining ELOS on a sample stope.

**Table19. New unplanned dilution classification system.**

| | | | | | |
|---|---|---|---|---|---|
| Hydraulic radius (HR), (m) | <2 | 2-2.25 | 2.25-2.75 | 2.75-3 | >3 |
| Description | Very low | Low | Moderate | high | Very high |
| Rating | 2 | 5.5 | 11 | 15.5 | 22 |
| Stope width (m) | <3 | 3-4 | 4-5 | 5-6 | >6 |
| Description | Very wide | Wide | Moderate | Narrow | Very narrow |
| Rating | 0.5 | 1.5 | 3 | 4 | 6 |
| Hole deviation (%) | <5 | 5-10 | 10-15 | 15-20 | >20 |
| Description | Very low deviation | Low deviation | Moderate | Much deviation | Too much deviation |
| Rating | 2 | 4.5 | 9 | 13 | 18.5 |
| Powder factor (PF), (kg/m3) | <0.75 | 0.75-1.25 | 1.25-1.75 | 1.75-2.25 | >2.25 |
| Description | Very low | Low | Moderate | High | Very high |
| Rating | 1 | 2 | 4 | 5.5 | 7.5 |
| Blast vibration, (PPV) | <250 | 250-500 | 500-750 | 750-1000 | >1000 |
| Description | Very low vibration | Low vibration | Moderate vibration | Much vibration | Too much vibration |
| Rating | 0.5 | 1.5 | 3 | 4 | 5.5 |
| Modified stability number (N') | >10 | 10-5 | 5-2 | 2-1 | <1 |
| Description | Too much stability | Much stability | Moderate stability | Low stability | Very low stability |
| Rating | 2 | 5 | 10 | 14 | 20 |
| Foliation | Perpendicular iso strike | Perpendicular un strike | Parallel | Oblique iso strike | Oblique un strike |
| Description | Very good | Good | Moderate | Bad | Very bad |
| Rating | 0.5 | 1 | 1.5 | 2 | 3.5 |
| Stope depth | <100 | 100-200 | 200-300 | 300-400 | <400 |
| Description | Very shallow | Shallow | Moderate | Much deep | Too much deep |
| Rating | 0 | 0.5 | 1 | 1.5 | 2 |
| Filling method | Hydraulic Backfilling | Pneumatic Backfilling | Mechanical Backfilling | Gravity Backfilling | Hand Backfilling |
| Description | Very good | Good | Moderate | Bad | Very bad |
| Rating | 1 | 2.5 | 5.5 | 7.5 | 11 |
| Filling time | Continuous loading | After loading quarter stope | After loading third the stope | After loading half the stope | After loading throughout the stope |
| Description | Very soon | Soon | Moderate | Late | Very late |
| Rating | 0.5 | 1 | 2 | 3 | 4 |

**Table 20. SUDI classification.**

| SUDI | 10-20 | 20-40 | 40-60 | 60-80 | 80-100 |
|---|---|---|---|---|---|
| Unplanned dilution susceptibility | Very low | Low | Moderate | High | Very high |
| Robustness against unplanned dilution | Very strong | Strong | Moderate | Weak | Very weak |

**Table 21. Application of new classification system in 10 different stopes of Venarch Manganese mines.**

| Stope name | Hydraulic radius (HR), (m) | Stope width (m) | Drilling hole deviation (%) | Powder factor (PF), (kg/m3) | Blast vibration, (PPV) | Modified stability number (N') | Foliation | Stope depth | Filling method | Filling time | SUDI | Robustness against unplanned dilution | ELOS (m) |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Piroozi-340-E | 2.27 | 4.70 | 12 | 1.5 | 491 | 1.6 | Oblique iso strike | 340 | Gravity | Third | 55.5 | Moderate | 0.20 |
| Athari-290-W | 2.25 | 4.50 | 10.5 | 1.5 | 482 | 2.8 | Parallel | 290 | Gravity | Half | 46 | Moderate | 0.25 |
| Doctor-140-W | 1.76 | 3.40 | 4.5 | 0.86 | 242 | 10.24 | Perpendicular iso strike | 140 | Mechanical | Throughout | 20.5 | Strong | 0.13 |
| Athari-290-E | 2.22 | 4.20 | 9 | 1.4 | 423 | 3.6 | Perpendicular un strike | 290 | Gravity | Half | 41 | Moderate | 0.23 |
| Doctor-140-E | 1.36 | 2.80 | 3 | 0.55 | 210 | 16 | Perpendicular iso strike | 140 | Mechanical | Throughout | 18.5 | Very strong | 0.14 |
| Jalal-390-W | 3.27 | 5.65 | 16 | 2 | 670 | 0.64 | Oblique un strike | 390 | Gravity | Quarter | 81 | Very poor | 0.38 |
| Piroozi-340-W | 2.50 | 4.85 | 13 | 1.6 | 528 | 0.96 | Oblique iso strike | 340 | Gravity | Third | 63 | Moderate | 0.33 |
| Piroozi-240-W | 2.18 | 3.70 | 7.5 | 0.95 | 340 | 4.8 | Perpendicular un strike | 240 | Mechanical | Half | 35.5 | Strong | 0.19 |
| Doctor-240-E | 2.14 | 3.55 | 6 | 0.9 | 227 | 7.68 | Perpendicular un strike | 240 | Mechanical | Half | 29.5 | Strong | 0.17 |
| Jalal-390-E | 2.83 | 5.25 | 14 | 1.7 | 573 | 0.8 | Oblique un strike | 390 | Gravity | Quarter | 69 | Poor | 0.37 |

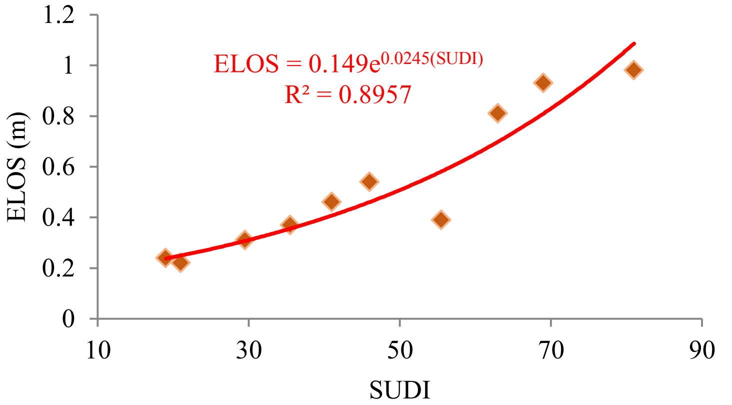



**Figure 6. Relationship between SUDI and ELOS.**

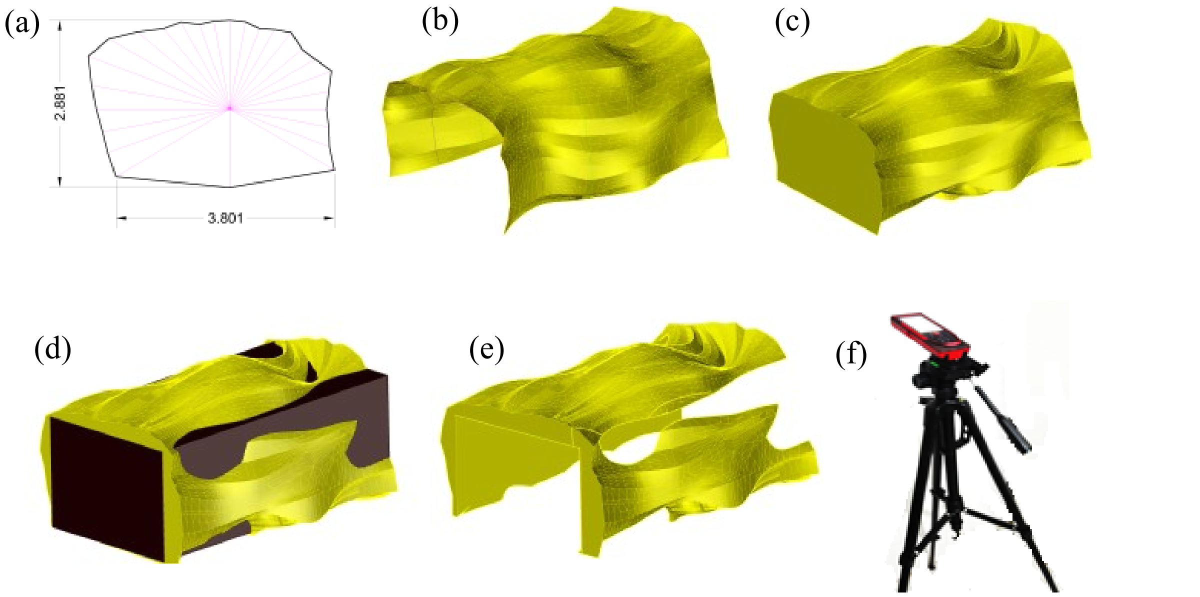



**Figure 7. Workflow for calculating unplanned stope volume and determining ELOS on a sample stope. (a) A cross-sectional profile of a stope. (b) Surface of unplanned stope. (c) Volume of unplanned stope. (d) Planned and unplanned stope. (e) Unplanned dilution. (f) A laser rangefinder mounted on a tripod.**

## 8. Conclusions

In this research work, first, categories of the factors and parameters affecting unplanned dilution in the cut-and-fill stoping method were identified. In order to weight the identified categories of factors and parameters, the Fuzzy-Delphi Analytical Hierarchy Process (FDAHP) was followed. Of all categories of factors, the factors associated with drilling and blasting were given the largest weights, while the operational factors were of the smallest weights. Further, of all the parameters found effective, the hydraulic radius and the fault friction angle were of the largest and smallest weights, respectively. Then out of the 20 identified effective parameters, 10 parameters were selected as the most effective ones, and a new classification system was presented to evaluate and predict the unplanned dilution level. In this new classification system, where susceptibility of a stope to unplanned dilution was evaluated, a new quantitative index called stope unplanned dilution index (SUDI) was used. Using this new index, an overall score between 10 and 100 was attributed to exploitation of stopes, classifying their susceptibility to unplanned dilution into the five classes very low, low, moderate, high, and very high.

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# ارائه یک سیستم طبقه‌بندی جدید ارزیابی و پیش‌بینی ترقیق برنامه‌ریزی نشده در روش استخراج کندن و پر کردن

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**چکیده:**

پدیده اضافه شکست و ریزش در دیواره‌ها و سقف کارگاه‌های استخراج زیرزمینی که موجب مخلوط شدن باطله با ماده معدنی استخراجی و ایجاد ترقیق می‌شود به دلایل گوناگونی صورت گرفته و پارامترهای بسیاری در ایجاد آن دخالت دارند. در این پژوهش به منظور ارزیابی و پیش‌بینی ترقیق برنامه‌ریزی نشده در کارگاه‌های استخراج زیرزمینی از نوع کندن و پر کردن، ابتدا تمامی پارامترهای مؤثر مورد شناسایی قرار گرفته‌اند. این پارامترها در ۴ دسته عوامل طراحی کارگاه، چال‌زنی و آتشباری، زمین‌شناسی و عملیاتی دسته‌بندی شده‌اند. سپس وزن‌دهی دسته عوامل و پارامترها با استفاده از روش تحلیل سلسله مراتبی دلفی فازی انجام شده است. در ادامه، پارامترهای برتر از میان تمامی پارامترهای مؤثر انتخاب شده‌اند. سپس به منظور ارائه سیستم طبقه‌بندی جدید ارزیابی ترقیق برنامه‌ریزی نشده، یک اندیس جدید به نام شاخص ترقیق برنامه‌ریزی نشده کارگاه، SUDI، ارائه شده است. این شاخص که دامنه آن در بازه ۱۰ تا ۱۰۰ متغیر بوده و از جمع امتیازات اختصاص داده شده به بازه‌های مربوط به دامنه تغییرات هر کدام از پارامترهای وزن داده شده حاصل می‌شود، بیان کننده میزان آمادگی کارگاه‌های استخراج به ترقیق برنامه‌ریزی نشده است. در ادامه، سیستم طبقه‌بندی جدید، در مجموعه معادن منگنز ونارچ، به کار گرفته شده و با استفاده از آن، مقادیر SUDI مربوط به ۱۰ کارگاه استخراج تعیین شده‌اند. در پایان با استفاده از سیستم مانیتورینگ فضای حفاری، مقادیر ترقیق برنامه‌ریزی نشده کارگاه‌های مذکور اندازه‌گیری شده‌اند. تحلیل رگرسیون نشان می‌دهد رابطه حاکم بین مقادیر ترقیق اندازه‌گیری شده و مقادیر SUDI کارگاه‌ها از نوع نمایی، و با ضریب تعیین ۰/۸۹۵۷ است.

**کلمات کلیدی:** معدنکاری زیرزمینی، روش کندن و پر کردن، ترقیق برنامه‌ریزی نشده ، سیستم طبقه‌بندی، شاخص ترقیق برنامه‌ریزی نشده کارگاه.