Estimation of Cavability by Using a Block Model and Fuzzy Sets*

by Liguan WANG¹, Fumio SUGIMOTO² and Shigeru YAMASHITA³

This paper presents a classification method for estimating the cavability class of an ore body, involving four evaluation parameters: rock quality designation (RQD), modified joint spacing, \( J_s \), modified joint set number, \( J_f \), and shear property of joints, \( \phi \). Geostatistical tools and a block modeling techniques are used to estimate the value of each parameter. Fuzzy sets are applied to estimate the cavability class of the rock mass block. The goal of this research is to develop a three dimensional (3D) block cavability model by incorporating these rock mass rating parameters. In this research work, the rock mass characterization data were derived from the rock core samples and the drill logging of a copper mine in north China.

**KEY WORDS**: Block Caving Cavability Class, 3D Block Modeling, Kriging, Semi-Variogram, Fuzzy Evaluation

1. Introduction

The development of an empirical method for the design of tunnel supports has been recognized for many decades. Terzaghi (1946) developed a method that could systematically assess ground competence based on some observational parameters and conditions. Deere (1964) introduced the RQD concept as a measure for rock core quality and proposed to use in the design and analysis of tunnel supports.

Recently, the concept of rock quality index (RQI) has been introduced to rate the most important factors that influence the behavior of rock mass. The first true classification system for rock masses, called rock structure rating (RSR), was proposed by Wickham et al. (1972, 1974). This classification system considers two general categories of factors influencing rock mass behavior in tunneling. They are geologic and construction parameters. Bieniawski (1973, 1979) developed a geomechanics classification or Rock Mass Rating (RMR) system. Barton et al. (1974) developed a method called the rock tunneling quality index (Q) for classifying rock masses. This method is based on the numerical assessment of six rock mass quality parameters.

Laubscher and Taylor (1977, 1981) developed a method for rock mass classification relevant to block caving, which is based on the RMR system. In this method, a rock mass is assigned to one of 10 rating units in two assessed RMR classes. Robert et al. (1984) developed a new rock mass classification system, called the Modified Basic RMR or MBR. It is used in estimating support requirements for mining by caving methods.

In general, the most important factors utilized in RQI approaches are intact rock strength, fracture intensity, shear strength of the fractures, the geometrical relationship among fracture patterns and the excavation boundary, and groundwater table. The rock mass rating by RQI approach is achieved by means of combining these factors.

Nguyen (1985), Sakurai and Shimizu (1989), Suzuki (1990), and Furuta et al. (1992) have proposed many methods for application to rock mass classification using fuzzy logic sets. These RQI approaches were developed mostly for the purpose of tunnel support.

Pauli and Pekka (1999) introduced a method for evaluating the behavior of rock mass in 3D. They intended to estimate the Q value of a block by using geostatistical and a block-modeling method.

The Study of Technique and Equipment in Block Caving was supported by the Eighth Five-years National Program of China. This project was executed in a North China underground copper mine (NCUC mine), Tongkuangyu copper mine, Zhongtiaoshan Non-ferrous Metal Corporation, Shanxi Province. The estimation of cavability was one of the five subjects in this project, and it was conducted by the authors. Also, this subject has got a continuous support from the National Science Foundation of China during 1998 to 2001.

In this research, four parameters (i.e., RQD, modified joint spacing, \( J_s \), modified joint set number, \( J_f \), and shear property of joints, \( \phi \)) are combined to evaluate the cavability of an ore body to be mined by block caving. Geostatistical and the block-modeling techniques were used to estimate the value of each parameter in a block, and fuzzy sets were used to determine the cavability class of the block. The goal of this research is to develop tools for predicting the distribution of cavability classes of an ore body in 3D. Data from drill core and drift logging in NCUC mine are used in this research.

2. Geologic Overviews

The NCUC mine is located in the Zhongtiaoshan Region, Shanxi Province, China. This region was formed in the Proterozoic era, and it constitutes a synclinorium structure. The trend of axes of the synclinorium structure and the orientation of the schistosity of the rocks are between northeast and north-northeast. Altered volcanic rocks in the exposed beds, which occur in this ore deposit, mainly consist of metahyalite,
metabasic volcanic rock, and metatuffaceous-semipelite.

More than 100 orebodies have been located in this mining district, of which No.4 and No. 5 are the two largest orebodies. The No.5 orebody is parallel to the No.4 orebody and is located on the lower side of No.4 orebody. The distance between the two orebodies averages about 100m to 130m. The No.4 orebody has been in production for many decades using sublevel caving method. The proposed caving project in this study is limited to the No.5 orebody, which is low-grade deposit and has been operated using a block caving method since 1993.

The mineralization of the No.5 orebody occurs almost totally in altered mortarite. The orebody extends from the 930 m to 575-700 m levels and deeper levels, extending 1,100 m in strike and averaging 115 m in width. The dip range of the orebody is between 45° and 60°, the strike is from northeast to southwest, and the dip direction is to the northwest.

Major prevailing joint in the orebody can be classified into two sets. The dip direction of joint set I is between 120° and 160°, and that of joint set II is between 290° to 330°. The dips of joints for both sets are between 45° and 65°.

3. Definitions of Parameters

In general, five factors greatly influence on rock mass properties. These factors are: intact rock strength, fracture intensity, shear strength of the fracture, the geometrical relationship among fracture patterns and the excavation boundary, and groundwater table. In this study, the following conditions are taken in: 1) all the parameters will have great influence on cavability and rock block size; 2) data should be easily acquired from excavation wall or drill cores; 3) data should be quantitatively given, 4) data should be sensitive on cavability rating, and 5) the acquired data should satisfy the requirements of modeling evaluation.

According to these conditions, four parameters, i.e., RQD, modified joint spacing, \( J_s \), modified numbers of joint sets, \( J_n \), and shear properties of joint, \( J_\phi \), were selected to evaluate the cavability rating of an orebody in the NCUC mine. Also, the parameter, \( J_s \), includes the influence of intact rock strength on the cavability rating.

The RQD index reflects the quality of rock masses. It is a very important factor in RQI approaches. The RQD concept follows Deere's definition and is given from observations of the drill cores and drift wall. In Deere's definition, RQD is defined as the total length of the intact cores that are greater than 100 mm in length, divided by the length of the core run.

RQD was originally introduced for NX-sized core (54.7 mm in diameter) and a threshold value of 100 mm is used. Over the years, several correction factors have been introduced to calculate RQD for other core diameters. RQD also could be estimated in areas where line mapping or area mapping has been conducted.

The modified joint spacing, \( J_s \), reflects mainly the block size characteristics of rock masses that are intersected by joint networks. It is an integrative index, involving the influences of strength of intact rock and discontinuities, in-situ stress, as well as a persistence factor of joints on the size characteristics of rock blocks.

Mukherjee and Mahtab (1987) proposed that the interaction between the geometry and strength of joints and in-situ stress conditions could be described through a concept of effective spacing. The effective spacing parameter, \( S' \), may be expressed as:

\[
S' = \left( K_c + (1 - K_c) \frac{C_c + \sigma_n \tan \phi_f}{C_c + \sigma_n \tan \phi_f} \right) S \tag{1}
\]

where \( S \) is the mean normal distance between adjacent joints of a joint set; \( K_c \) is the persistence factor of the joint set; \( C_c \) is the joint cohesion (MPa); \( \phi_f \) is the joint friction angle; \( C_c \) is the intact rock cohesion (MPa); \( \phi_f \) is the intact rock friction angle; and \( \sigma_n \) is the normal stress on joint surface, which depends on both the field stress and joint orientation.

In this study, a scanline method was used for estimating the persistence of joints. When a joint intersects an exposure region of finite size, the intersection may occur in three ways as shown in Fig.1: 1) both ends are concealed, 2) one end is concealed and one end is observable, and 3) both ends are observable. If there are \( n \) joints intersecting the scanline on a rock exposure, there should be \( r \) joints (\( r \leq n \)) with a semi-trace length less than the length of \( c \) that is parallel to the joint-trace and terminates at the both ends of the scanned width and their semi-trace lengths are \( L_1, L_2, \ldots, L_n \) (where \( L_1 \leq L_2 \leq \ldots \leq L_n \leq c \)). According to the geometrical relationship between the joint patterns and the drift wall boundary, \( c \) is assigned as 5 m in this study. The remainder of the joints on the exposure, \((n-r)\) joints, should be censored at \( c \), i.e., they have a semi-trace length beyond \( c \). The mean semi-trace length of joints can be estimated by the following formula (Priest and Hudson, 1981):

\[
L = L_n + c \left( \frac{n - r}{r} \right) \tag{2}
\]

where \( L_n \) is the mean semi-trace length of the joints, which terminates within the scanned area, and

\[
L_n = L_c \sum_{r=1}^{n} L_r \tag{3}
\]

The persistence factor \( K_c \) for the scanned area is defined as the sum of joint segments (along an imaged continuous joint) divided by the sum of joint segments and rock bridges (totaling the length of the imaged continuous joint), and calculated as follows:

![Fig.1 Diagrammatic representation of joint traces intersecting a scanline set up on a wall surface of limited extent.](image-url)
\[ K_i = \frac{r_i L}{nc} \times 100\% \]  \hspace{1cm} (4)

The modified joint spacing, \( J_f \), can be calculated by the following formula:
\[
J_f = \frac{1}{N} \sum_{i=1}^{N} S_i^0 \]  \hspace{1cm} (5)

where \( N \) is the total number of joint sets and \( S_i^0 \) is the effective spacing of the \( i \)-th joints set.

The modified joint set number, \( J_f \), reflects how the actual number of joint sets, the orientation of joint sets, and angles between two joint sets influence on the cavability and characteristics of rock blocks. It is known that larger number of joint sets, smaller dip angle of joint and larger angles between two joint sets are more favourable for orebody caving and formation of uniform size of rock blocks. The parameter, \( J_f \), is defined as the number of joint sets which is modified by two adjustment factors that consider the influences of the minimum dip angle of all the joint sets and angles between two joint sets on the cavability and characteristics of rock blocks. The parameter, \( J_f \), is quantified by the following formula:
\[
J_f = N \cdot f_\beta \cdot f_\theta \]  \hspace{1cm} (6)

where \( N \) is the total number of joint sets and \( N = 2 \) for the NCUC mine. \( f_\beta \) is an adjustment factor based on the minimum dip angle of two joint sets, and \( f_\theta \) is another adjustment factor based on the angles between two joint sets. Both factors \( f_\beta \) and \( f_\theta \) are assigned as the values ranging between 0.8 and 1.0, which is partly referenced from the MBR classification system (Robert et al., 1984).

Assuming that the dip angle of the \( i \)-th joint set is \( \beta_i \), the minimum dip angle for the two joint sets is \( \beta_{\text{min}} \), the dip direction of the \( i \)-th joint set is \( \phi_i \) and the angle between two joint sets is \( \theta \), then the angle \( \theta \) between two joint sets can be given by:
\[
\theta = \cos^{-1}(\text{n_1} \cdot \text{n_2}) \]  \hspace{1cm} (7)

where \( \text{n}_1 = (\sin \beta_1 \cdot \sin \alpha, \sin \beta_1 \cdot \cos \alpha, \cos \beta_1) \) and \( \text{n}_2 = (\sin \beta_2 \cdot \sin \alpha, \sin \beta_2 \cdot \cos \alpha, \cos \beta_2) \) are unit normal vector to joint sets 1 and 2, respectively.

According to formula (6), the parameter, \( J_f \), can be assigned as listed in Table 1 for different ranges of the parameters \( \beta_{\text{min}} \) and \( \theta \), which are related to the parameters \( f_\beta \) and \( f_\theta \), respectively.

Parameter \( J_f \) reflects the shear properties of a joint. It has been known that the shear strength of joints can be estimated by describing the roughness of joint surfaces and particularly in the case of unfilled joints, the estimation may be quite accurate (ISRM, 1978). Actual shear strength of a joint may be determined by both the friction angle and the larger asperities and it is given by the following formula:
\[
\tau = \sigma_n \tan(\phi + K_f \cdot i) \]  \hspace{1cm} (8)

where, \( \tau \) is shear strength; \( \phi \) is the friction angle without consideration of the influence of joint asperity; \( \sigma_n \) is the effective normal stress; \( i \) is the root mean square of the first derivation of the joint surface profile, and \( K_f \) is a modification factor which represents the influence of joint cohesion on joint friction angle if transform it to joint friction angle and therefore, \( K_f \geq 1 \).

In formula (8), the root mean square of the first derivation of the joint surface profile \( i \) is a single-parameter measure that characterizes a profile based on its average gradient. The profile was measured in the laboratory by using a joint surface profiler and the parameter \( i \) was calculated by using the following formula:
\[
i = \left[ \frac{1}{n(n-1)} \sum_{i=1}^{n} (dy)^2 \right]^{1/2} \]  \hspace{1cm} (9)

where \( n \) is the number of evenly-spaced sampling points; \( dx \) is the distance along the sampling line between the points and \( dy \) is the distance normal to the sampling line between the points.

A series of shear tests for joints was performed in the laboratory (Liu and Pan, 1993). The relationship between the apparent joint friction angle \( \phi \) of a joint and the root mean square of the first derivation of the joint surface profile \( i \) was fitted with the following function:
\[
\phi = \phi_0 + K_f \cdot i \]  \hspace{1cm} (10)

As the results of laboratory tests, it was found that \( \phi_0 \) equals 18.76, \( K_f \) equals 1.28, and the fit correlation factor \( r \) is 0.9941. Thus, the apparent joint friction angle could be estimated from the parameter \( i \), by using the following relationship:
\[
\phi = 18.76 + 1.28 \cdot i \]  \hspace{1cm} (11)

In order to estimate joint shear strength of joints directly by using the description on joint roughness observed in-situ or from drill core examination, the surface profiles of joints are classified into three types, those are roughly undulate, smoothly undulate and nearly plane. The profile for each set of joint surface was measured in laboratory, and the mean value of the parameter \( i \) for each roughness set is estimated as shown in Table 2.

| Table 1 | Relationship between parameter \( J_f \) and orientations of joint sets. |
|---------|---------------------------------|---|---|---|---|---|
| \( \theta \) | \( \theta < 30^\circ \) | \( 30^\circ < \theta < 45^\circ \) | \( 45^\circ < \theta < 60^\circ \) | \( 60^\circ < \theta < 90^\circ \) |
| \( J_f \) | \( f_\beta = 0.8 \) | \( f_\beta = 0.88 \) | \( f_\beta = 0.95 \) | \( f_\beta = 1.0 \) |
| \( J_f \) | \( f_\theta = 1.0 \) | \( f_\theta = 1.0 \) | \( f_\theta = 1.0 \) | \( f_\theta = 1.0 \) |
| \( i_1 \) | 1.00 | 1.76 | 1.90 | 2.00 |
| \( i_2 \) | 1.44 | 1.60 | 1.71 | 1.80 |
| \( i_3 \) | 1.30 | 1.41 | 1.52 | 1.60 |

<table>
<thead>
<tr>
<th>Table 2</th>
<th>Mean value of parameter ( i ) for each roughness profile set of joint in the NCUC mine.</th>
</tr>
</thead>
<tbody>
<tr>
<td>Asperity</td>
<td>Rough</td>
</tr>
<tr>
<td>Parameter ( i )</td>
<td>18.3</td>
</tr>
</tbody>
</table>
The parameter  was defined as the average friction angle of joints for a specified interval along the measuring line, provided that the rock mass has similar features. Following formula (11), the parameter  may be derived from the following formula:

\[ J_\phi = 18.76 + \frac{1.28}{n} \sum_{k=1}^{n} i_k \]  

where  is the number of joints in the statistical length;  is the root mean square of the first derivation of the profile for the  joint, and the value of  is selected from the parameter  shown in Table 3.

4. Data Collection

The principle of the rock mechanical data collection and processing system developed in this research is shown in Fig. 2.

The original data used in this research was collected by the rock mechanics work team, which consists mainly of the researchers in geo-engineering and rock mechanics from the NCUC mine, Central South University and China Non-ferrous Metals Institute of Engineering Design (Liu and Pan, 1993).

Data with respect to RQD,  and  were collected from the drill cores of which total length is about 7,800 meters and along the excavated drifts in the NCUC mine, of which total length is about 1,500 meters. At the drifts located on the 810 m and 870 m levels, the data were obtained through detailed line-sketching method. The surface location of the drill holes and the underground drifts are shown in Fig. 3, in which the modeling range is indicated as the ore body boundary.

The RQD, spacing of joints, surface properties of joints, and uniaxial compressive strengths of rock were obtained from drill cores. A number of laboratory tests were carried out to determine the strength of the intact rock and the shear strength of joint for various types of rock by using the rock mechanics test system. Furthermore, friction angles and cohesion characteristics of intact rocks were obtained from triaxial compression tests, and those of joints are obtained from direct shear tests.

In the drifts, RQD data, the geometric parameters of joints, and the surface properties of joints were obtained. RQD was calculated from the spacing of joints on the main measuring-line direction with a length of 5 ~ 10 meters. The geometric parameters of joints include the spacing, dip angle, dip direction and trace length. The surface properties include roughness, underground water conditions, fill characteristics, and weathering state.
ing conditions.

The parameters RQD, Jφ, Jf, and Jp were estimated from observation over a specified length of 5–10 meters on the measuring line, under assumption that the rock mass has similar features.

The orientations of joints were analyzed using the contours of joint pole concentrations (Wang and Zhong, 1993), and the mean and standard deviation of the orientation (dip angle and dip direction) for each joint set were also calculated.

The positions of samples were represented by the coordinates of their centers, and the coordinates were calculated by using the measured data that include the positions of the borehole mouths and the drifts with those orientations, and dips, and the surveying data.

These parameters are original ones, but the lengths for which each original parameter was derived differ. The term "composite value of parameters" is introduced for structural analysis and 3D modeling, in which all of the composite values of parameters are calculated in the same composite sample length that equals to the dimension of the block.

The formula for calculating the composite values of a parameter from the original parameter is given by:

\[
P_c = \frac{1}{n} \sum_{i=1}^{n} (L_c P_i)
\]

where \(P_c\) is a composite parameter value; \(L_i\) is the measurement interval; \(P_i\) is the original value of the parameter in the domain where the rock mass has similar features; \(L_c\) is the composite sample length; and \(n\) is the number of the original value of the parameter included in the statistical length.

The in-situ rock stress states were measured by using the overcoring method with a triaxial strain cell. In-situ stress measurements were conducted at the eight sites on three levels: 870 m, 810 m, and 800 m. The in-situ stresses at each site are shown in Table 3 (Liu and Pan, 1993).

5. Evaluations of Parameters in a 3D Model

Block modeling is a suitable way to make a 3D model forming a rock mass. Every block has its values of parameters, which will be evaluated from logged data by geostatistical methods such as Kriging and the inverse distance to a power and polygon. In this study, the values of the parameters are interpolated by applying the Kriging method to the lattice points in three dimensions. The Kriging weights are based on a semi-variogram and anisotropy factors for the search ellipsoid. The semi-variogram is calculated by averaging the squared differences between the pairs of samples, which are at a given distance apart:

\[
\gamma(h) = \frac{1}{2N} \sum_{i=1}^{N} (g_i - g_{i+h})^2 \quad i=1,2,3, \ldots, N \quad (14)
\]

where \(h\) is the distance between sampling points; \(\gamma(h)\) is the semi-variogram for distance \(h\); \(N\) is the number of pairs at distance \(h\) and \(g\) is the sampled value.

The bar charts derived from the logged data are shown in Fig.4. It can be seen from the figure that the values of RQD, Jφ, Jf, and Jp in the NCUC mine may approximately have normal distributions, although the scale of the cross axis of Fig.4(b) is represented logarithmic. A practical method for calculating the semi-variogram in 3D space was proposed by Chen, Deng and Wang (1994). Using this method, the experi-
mental semi-variograms for RQD, \( J_s \), \( J_f \), and \( J_g \) were calculated in three directions, those are the ore body strike direction (azimuth 40°), the ore body dip direction (azimuth 310°, dip 45°) and the direction perpendicular to the ore body dip direction (azimuth 130°, dip 45°).

The composite sample length greatly influences the calculation results of the variograms. Fig 5(a)-(c) show the experimental semi-variograms of RQD and its fit models that calculated with different composite sample lengths. They indicate that when the composite sample length increases, the range of the variogram increases. When the composite sample lengths of RQD are 2 m, 5 m and 10 m, the range of the variogram are 20.4 m, 96 m and 110 m, respectively. The longer composite sample length may cause a smaller variation on the sampling data, and thereby make the range of the variogram larger. In this study, the size of modeling block is 10 m in the three directions and the composite sample length is assigned as 10 m. The dotted line in Fig 6(a)-(c) show the results of the variograms for parameters \( J_s \), \( J_f \), and \( J_g \). In Fig 5(a)-(c) and 6 (a)-(c), the spherical model is adopted, which is defined by the following equation:

\[
\gamma(h) = \begin{cases} 
0 & \text{if } h = 0 \\
C_0 + C \left( \frac{3h}{2a} - \frac{h^3}{2a^3} \right) & \text{if } 0 < h \leq a \\
C_0 + C & \text{if } h > a 
\end{cases} \tag{15}
\]

where \( C_0 \) is the nugget; \( C_0 + C \) is the sill; and \( a \) is the range.

The slopes and shapes of the semi-variogram vary greatly depending on the evaluation parameters RQD, \( J_s \), \( J_f \), and \( J_g \). RQD has a higher range because a good sampling technique was used and the parameter changes continuously. The parameters \( J_s \) and \( J_g \) have middle ranges, as sampling techniques for these parameters also may be good and the parameters may have a locally continuous structure. The parameter \( J_f \) has a smaller range. It may be caused by variableness over short-distance of the factors such as irregular sampling error. Although the apparent nugget effect seems to be high for parameter \( J_f \), the range can still be estimated.

Cross-validation for the variograms was performed to check whether the variogram model fits the experimental variograms. A scatter diagram was used to display visually how two variables are related, in which a pair of the estimated value and the actual value are represented as a point at the coordi-
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Fig. 7 shows the relationship between the Kriged estimates and the actual values for each pair of the parameters, and Table 4 shows the statistical results in cross-validation analysis. The analytical results show that the value of each parameter has a smaller difference between the estimated value and the known one and that the applied models of a variogram fit better.

6. Estimating Cavability

There are three important aspects in creating a cavability class model. Firstly, the standard characteristic values of each parameter for each cavability class must be defined. Secondly, the weight of each parameter in consideration of its contribution to the classification results must be determined. Finally, the cavability class for each block must be estimated.

For the evaluation of the cavability rating of a block, a fuzzy method was used. It is assumed that a parameter symbol is $X_i$, where $i = 1, 2, ... , M$ and $M$ is the number of blocks. The fuzzy set for the standard characteristic values of the parameters in each cavability class has the symbol $X_{ik}$, where $k = 1, 2, ... , N$ and $N$ is the number of the cavability classes. The membership function of $\mu_{X_{ik}}$ is then defined as follows:

$$X_i \leq X_{ik}, \quad \left\{ \begin{array}{ll} \mu_{X_{ik}} = 1 & (k = 1) \\ \mu_{X_{ik}} = 0 & (k = 2, 3, ..., N) \end{array} \right. \quad (16)$$

$$X_i \geq X_{ik}, \quad \left\{ \begin{array}{ll} \mu_{X_{ik}} = 1 & (k = N) \\ \mu_{X_{ik}} = 0 & (k = 1, 2, ..., N-1) \end{array} \right. \quad (17)$$

$$X_{ik} < X_j \leq X_{(i+1)k}, \quad \left\{ \begin{array}{ll} \mu_{X_{ik}} = 1 & (k < N) \\ \mu_{X_{(i+1)k}} = 1 & (k = N) \\ \mu_{X_{ik}} = 0 & (k \neq k, k+1) \end{array} \right. \quad (18)$$

The membership grade of $\mu_{X_{ik}} (C_i)$ in the cavability class $C_i$ is given by:

$$\mu_{X_{ik}} (C_i) = \frac{1}{M} \sum_{k=1}^{M} (w_i \mu_{X_{ik}}) (i = 1, 2, ..., N) \quad (19)$$

where $w_i$ is the weight of parameter $X_i$.

The result can be obtained according to the following for-
the symbol that indicates the maximum value of \( C_k \) when \( k = 1, 2, \ldots, N \). In this way, the cavability class of a block could be regarded as \( C_k \) if the \( \mu_{x_i}(C_k) \) receives the maximum value.

In this study, the degree of cavability was classified into three classes that are noted by the symbol \( C_k \), where \( k = 1, 2, \) and 3. The standard characteristic value of each parameter for the cavability class was determined by considering mainly the probability distribution of these parameters collected in the NCUC mine. The values of each parameter at the accumulative frequency of 0.25, 0.5, and 0.75, which can be derived from Fig.4, were assigned to those for the \( C_1 \), \( C_2 \), and \( C_3 \) cavability classes. Table 5 gives the standard characteristic values of each parameter used. Fig.9 shows the membership functions for the three cavability classes.

The weight of each parameter was determined mainly according to its importance for impacting the size distribution of ore fragmentation, and is partly referenced from Q and RMR classification systems.

In fact, both the Q and RMR classification systems are based on a rating of three principal properties of a rock mass. These are the intact rock strength, the frictional properties of discontinuities and the geometry of intact blocks of rock defined by the discontinuities (Milne, 1988, 2001). The weights of the three principal properties of a rock mass are 19%, 44% and 39%, respectively, in the Q system, and 16%, 54% and 27%, respectively, in the RMR classification system.

For this study, the intact rock strength is only a factor in the context of the modified joint spacing, and the block size is the most important factor in evaluation of cavability. Therefore, the total weights of the parameters, which are concerned with block size, are adjusted up to 70%, that is the weight of parameter index \( J_s \) is assigned 40% and the RQD weight is assigned

\[
\mu_x(C_k) = \frac{1}{N} \sum_{k=1}^{N} \mu_{x_i}(C_k) \quad (20)
\]

where \( \mu_{x_i} \) is a symbol that indicates the maximum value of \( \mu_{x_i}(C_k) \) when \( k = 1, 2, \ldots, N \). In this way, the cavability class of a block could be regarded as \( C_k \) if the \( \mu_{x_i}(C_k) \) receives the maximum value.

Table 5 Standard characteristic values and weights.

<table>
<thead>
<tr>
<th>Cavability class ( C_k )</th>
<th>( J_s ) (Xa)</th>
<th>RQD (Xa)</th>
<th>( J_f ) (Xa)</th>
<th>( J_f ) (Xa)</th>
</tr>
</thead>
<tbody>
<tr>
<td>( C_1 ) (k=1)</td>
<td>0.15</td>
<td>40</td>
<td>41.3</td>
<td>1.5</td>
</tr>
<tr>
<td>( C_2 ) (k=2)</td>
<td>0.25</td>
<td>60</td>
<td>42.0</td>
<td>1.44</td>
</tr>
<tr>
<td>( C_3 ) (k=3)</td>
<td>0.40</td>
<td>75</td>
<td>42.7</td>
<td>1.34</td>
</tr>
<tr>
<td>Weight (( w_i ))</td>
<td>0.15</td>
<td>0.15</td>
<td>0.15</td>
<td></td>
</tr>
</tbody>
</table>

Fig.8 Contour plots of the evaluated parameters RQD, \( J_s \), \( J_f \) and \( J_f \) (at level 800 m in the NCUC mine, Extracted from the 3D block model).
results from the different test sites are needed to examine the applicability of the geostatistical modeling methods proposed here. The quantitative interrelationships between the measured rock mass characteristics and the caving span/sizes and locations of stable access opening are also needed to make realistic estimates in future research.

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References


3D よびる方法を提案している。この方法において,地質統計学的手法お不連続面のせん断特性 による鉱体のケーバビリティを評価す

 Parts of the analytical results for the distribution of cavability classes are shown in Fig.10. This contour map for cavability classes is at the 810 m level.

7. Conclusions

This paper explores a new method for evaluating the cavability of an ore body in 3D using a block model, the Kriging method, and a fuzzy logic technique. Preliminary research shows that geostatistical tools can be of great assistance in creating 3D rock mechanics models when a large amount of rock mechanical data has been collected in a practical project.

The quantitative interrelationship between each cavability class and the size distribution of caved ore fragmentations could be established further after the cavability model has been done. The combination of distributions of cavability classes with those of ore size fragmentations for each cavability class is beneficial in achieving more precise mine design and cost estimation.

Still, much more information and field measurement