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NUMERICAL MODELLING FOR ESTIMATION OF FIRST WEIGHTING DISTANCE IN LONGWALL COAL MINING - A CASE STUDY

H. Manteghi¹, K. Shahriar² and R. Torabi³

ABSTRACT: There are many parameters which affect caving processes in longwall coal mines, such as roof and floor strata conditions, thickness of immediate roof, excavation geometry and magnitude and direction of principles stresses. Estimation of first weighting distance, in this context is rather complicated. In this paper, the results of numerical modelling of the first weighting mechanism by using the finite difference code FLAC²D at the E1 longwall panel of the Parvade1 underground mine, which is located in the Tabas area at the central part of Iran, are presented. The obtained first caving distance is 11.2 m. In addition, the results of numerical modelling have been compared with some conventional methods such as Peng’s method. The results show good agreement with each other.

INTRODUCTION

Longwall mining is a method with high production capacity and mechanisation potential. Its development backs to 17 century in European collieries. At the beginning of the 19 century this method with development of self-advanced support systems also was applied in United State mining. However, in the recent years, by using the longwall mining, coal production is increasing; the main reasons can be highly mechanized procedure and safety of this method. Two essential parameters for high safety are perfect ground control and prediction of strata movements around the coal seam.

Strata control in longwall mining has been a grey area of research since its introduction in underground coal mining industry worldwide. A reliable prediction of the caving behaviour of strata and its interaction with the roof support helps in selection of sustainable mining parameters and rational capacity of supports. It is prerequisite for developing a reliable support selection tool essential for successful planning of longwall working in a given geo-mining condition (Singh and Singh, 2009).

Theoretical models for prediction of main fall (first weighting) and periodic caving distances are based on plate-beam theory and bending moment approach. A number of empirical models have been developed on the basis of either a certain concept or some field experience to assess the caving behaviour of strata. Some of these approaches suggest roof classifications for qualitative assessment of caving behaviour. Some other models propose quantitative relation to predict the span of main fall. Similar relations have been proposed by various researchers to estimate the span of periodic caving. A few models give both the options of qualitative assessment of roof caving and the quantitative assessment of caving span (Singh and Singh, 2009).

Although application of the numerical modelling technique for strata control in longwall workings is not a new topic of research, a study has not been done to assess the strata–support interaction with progressive face advance in most of the cases. Most of these studies have been done using elastic analysis where simulation of face advance bears no importance. This paper describes a numerical modelling to assess the first caving of strata and the results will be compared with theoretical models and real results. Then, the performance of the numerical model; to assess the acceptability of the model for predicting the caving behaviour of strata in a given strata condition, is compared with field observations at the E1 longwall panel of the Parvade1 underground mine which is located in the Tabas area in the central part of Iran.

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CONVENTIONAL METHOD

Peng’s theoretical method

When a longwall panel of sufficient width and length is excavated, the overburden roof strata are disturbed in order from the immediate roof toward the surface. The overburden roof strata are divided in to four zones shown in Figure 1: the caved zone, the fractured zone, the continuous deformation zone and the soil zone. The caved zone is created as the mining face advances and the immediate overburden falls and fills the void created by the extraction of the coal. The caved zone extends upwards, two to ten times the extraction thickness. The caved zone is characterized by irregular rock fragments that may have rotated relative to their initial locations, resulting in relatively high void ratios and permeability. The caved rock (goaf) is re-compacted by the weight of the overburden. The amount of re-compaction depends on the depth of overburden and the strength of the gob material.

The fractured zone is located above and around the caved zone and is characterized by near vertical fractures and bedding plane shearing caused by the passage of the longwall face (Peng and Chang, 1984). Bed separation can occur in this zone. The fractured zone can extend 30 to 60 times the extraction thickness. Above the fractured zone is the continuous deformation zone. The rock is essentially un-fractured, but can experience shearing along bedding planes as they are deflected over the edges of the extracted longwall panel. Bedding plane shear will affect the horizontal conductivity of the rock. Field observations have shown that water levels and ground movements occur up to 60 m (200 ft) ahead of an advancing longwall face. These movements can be associated with shear along weak clay filled bedding planes. On the surface, there is a soil zone of varying depth depending on the location. In this zone, cracks open and close as the longwall face comes and goes (Peng and Chang, 1984).

![Figure 1 - Overburden movement resulting from longwall mining](image_url)

Movement of the zones has different degrees of effect on roof control at the longwall face. The effect decreases as the strata are located farther upward from the roof line. Those strata, the movement of which will affect roof control at the long wall face, can be classified into two types, immediate and main roofs. The immediate roof is that portion of the overburden strata lying immediately above the coal seam top, approximately two to ten times that of the mining height. Above the immediate roof, the strata in the lower portion of the fractured zone are called the main roof (Figure 2) (Peng, 2008).

Formulation of caving height or thickness of the immediate roof

The thickness of the immediate roof is the basis for designing roof control techniques. Normally, caving initiates from the lowest strata in the immediate roof and propagates upward into the fractured zone. The process of caving in each stratum is that the stratum sags downward as soon as it is undermined. When the downward sagging of the stratum exceeds the maximum allowable limit, it breaks and falls. As it falls, its volume increase; therefore, the gap between the top of the rock piles and the sagged but un-caved stratum continues to decrease as the caving propagates upward. When the gap vanishes, the caving will be stopped. Thus the height of the caving must satisfy the following condition (Peng, 2008).

\[ H - d = h_{im} (K-1) \]  
\[ \text{And } d \leq d_0 \]
Where $H$ is the mining height, $d$ is the sagging of the lowest un-caved strata, $d_0$ is the maximum allowable sagging (without breaking) of the lowest un-caved strata, $h_m$ is the thickness of the immediate roof or caving height, and $K$ is the bulking factor of the immediate roof. Therefore, according to the Equation (1), the caving height can be determined by:

$$h_m = (H - d)/(K - 1)$$  (3)

It must be noted that if $d = d_0 = H$, then $h_m = 0$, which means no caving: that is, the roof sags gradually until it touches the floor. On the other hand, if $d = d_0 = 0$:

$$h_m = H/(K - 1)$$  (4)

**Formulation of first weighting**

There are a lot of complexities in design of roof loading in longwall coal mining when the strata above the coal seam has not moved yet and the first collapse is expected. Therefore, theoretical models must be used for prediction of main fall (first weighting). The distance is assessed based on beam fixed at both ends theory and bending moment approach. A beam fixed at both ends that is affected by uniformly distributed load is shown in Figure 3.

According to the Figure 3, the reaction forces at both ends are shown in Figure 4 (Majumdar, 1986).

Therefore, the equilibrium equations are as follows:

$$\sum F_x = 0 \Rightarrow A_x - B_x = 0$$ $$A_x = B_x$$  (5)

$$\sum F_y = 0 \Rightarrow A_y + B_y = qL = 0$$  if $A_y = B_y \Rightarrow A_y = B_y = qL/2$  (6)

$$\sum M = 0 \Rightarrow -M_A - M_B + qL . L/2 + A_y (0 - B_y . L) = 0 \Rightarrow M_A = -M_B$$  (7)
Where \( A_x \) and \( B_x \) are reaction forces in the x direction, \( A_y \) and \( B_y \) are reaction forces in the y direction, \( L \) is the length of beam, \( q \) is the force at unit of beam length and \( M_A \) and \( M_B \) are bending moments at beam supports. According to Equation (7) the bending moments have been calculated by another method that is called super position method as following:

\[
M_B = \frac{qL^2}{12} \quad (8)
\]

The maximum of bending moment occurs at both ends of supports (Equation (9)) and also the maximum of tensile stress that is sustainable for a beam fixed at both ends as shown in Equation (10).

\[
M_{\text{max}} = \frac{qL^2}{12} \quad (9)
\]

\[
\sigma_{t\text{max}} = \frac{M_{\text{max}}C}{I} \quad (10)
\]

Where \( \sigma_{t\text{max}} \) is the maximum of tensile stress, \( C \) is the neutral axis distance of beam from neutral surface and \( I \) is the moment of inertia. On the other hand, \( C \) and \( I \) have been calculated by Equation (11) and Equation (12) for a rectangular beam.

\[
C = \frac{h}{2} \quad (11)
\]

\[
I = \frac{bh^3}{12} \quad (12)
\]

Where \( h \) is the beam diameter and \( b \) is the beam width. Therefore, by means of Equations (9) to (12) and to know that is \( q = \gamma b h \), where \( \gamma \) is the beam density, the maximum of allowable length beam is estimated as follows:

\[
L = \sqrt{\frac{2\sigma_{t\text{max}}h}{\gamma}} \quad (13)
\]

Where \( L \) is the maximum of the allowable length beam or distance of the first weighting, \( \sigma_t \) is the tensile strength of the immediate roof, \( \gamma \) is the density of the immediate roof and \( h \) is the thickness of the immediate roof.

**ROCK PROPERTIES**

**Assessment of material properties and rock mass strength**

Proper assessment of rock mass strength and modulus values is very important for a meaningful numerical modelling study of caving behaviour and support requirements. Therefore, physical and mechanical properties of each geological unit must be properly determined. In general, intact rock properties are determined by means of laboratory testing. However, there is an important difference between rock material and rock mass characteristics. It is compulsory to determine representative physical and mechanical properties of the rock mass instead of intact rock material. Data regarding the physical and mechanical properties of surrounding rock are given in Table 1 (Tabas Coal Project, 2003).

<table>
<thead>
<tr>
<th>Formation</th>
<th>Definition code</th>
<th>Density (MN/m3)</th>
<th>Uniaxial compressive strength (MPa)</th>
<th>Tensile strength (MPa)</th>
<th>Internal Friction angle ( \phi )</th>
<th>Cohesion C (MPa)</th>
<th>Modulus of elasticity ( E ) (MPa)</th>
<th>Poisson’s ratio ( v )</th>
</tr>
</thead>
<tbody>
<tr>
<td>Siltstone</td>
<td>1</td>
<td>0.027</td>
<td>37.38</td>
<td>2.5</td>
<td>24.12</td>
<td>1.3</td>
<td>2838</td>
<td>0.26</td>
</tr>
<tr>
<td>Sandy Siltstone</td>
<td>2</td>
<td>0.025</td>
<td>73</td>
<td>2.6</td>
<td>-</td>
<td>-</td>
<td>2987</td>
<td>0.25</td>
</tr>
<tr>
<td>Silty Mudstone</td>
<td>3</td>
<td>0.026</td>
<td>18.8</td>
<td>0.3</td>
<td>-</td>
<td>-</td>
<td>2256</td>
<td>0.28</td>
</tr>
<tr>
<td>Coal</td>
<td>4</td>
<td>0.016</td>
<td>6</td>
<td>-</td>
<td>15-25</td>
<td>0.016</td>
<td>316</td>
<td>0.25</td>
</tr>
<tr>
<td>Mudstone</td>
<td>5</td>
<td>0.026</td>
<td>24.82</td>
<td>-</td>
<td>18.62</td>
<td>0.94</td>
<td>2838</td>
<td>0.31</td>
</tr>
<tr>
<td>Sandstone</td>
<td>6</td>
<td>0.027</td>
<td>72.79</td>
<td>6.3</td>
<td>31.75</td>
<td>8.69</td>
<td>5281</td>
<td>0.25</td>
</tr>
</tbody>
</table>
Laboratory tests were carried out on core samples from exploration drilling. Rock blocks are taken directly from the mine. The data presented in Table 1 are representative only of rock material. Determination of rock mass strength characteristics is rather difficult. Therefore, it is a common practice to derive rock mass strength from rock material properties by using various failure criteria. In this study, rock material properties were converted into rock mass data by using empirical relations widely used in the literature, e.g. Hoek and Brown failure criterion, Bieniawski’s RMR classification system, and Geological Strength Index (GSI). Physical and mechanical properties of the rock mass used for numerical modelling by FLAC2D are presented in Table 2 (Tabas Coal Project, 2003).

<table>
<thead>
<tr>
<th>Rock Definition</th>
<th>Siltstone</th>
<th>Sandy Siltstone</th>
<th>Silty Mudstone</th>
<th>Coal</th>
<th>Mudstone</th>
<th>Sandstone</th>
</tr>
</thead>
<tbody>
<tr>
<td>Definition code</td>
<td>1</td>
<td>2</td>
<td>3</td>
<td>4</td>
<td>5</td>
<td>6</td>
</tr>
<tr>
<td>Density (MN/m³)</td>
<td>0.0272</td>
<td>0.0271</td>
<td>0.0268</td>
<td>0.016</td>
<td>0.0263</td>
<td>0.027</td>
</tr>
<tr>
<td>Internal Friction angle (φ)</td>
<td>27.42</td>
<td>31.75</td>
<td>22.17</td>
<td>15.76</td>
<td>20.13</td>
<td>43.52</td>
</tr>
<tr>
<td>Cohesion c (MPa)</td>
<td>0.357</td>
<td>0.443</td>
<td>0.257</td>
<td>0.084</td>
<td>0.231</td>
<td>0.767</td>
</tr>
<tr>
<td>Modulus of elasticity E (MPa)</td>
<td>2238</td>
<td>2818</td>
<td>1778</td>
<td>749</td>
<td>1995</td>
<td>3548</td>
</tr>
<tr>
<td>Tensile strength (MPa)</td>
<td>0.012</td>
<td>0.007</td>
<td>0.005</td>
<td>0.002</td>
<td>0.013</td>
<td>0.017</td>
</tr>
<tr>
<td>Poisson’s ratio ν</td>
<td>0.25</td>
<td>0.25</td>
<td>0.25</td>
<td>0.25</td>
<td>0.31</td>
<td>0.25</td>
</tr>
<tr>
<td>Bulk modulus &quot;K&quot; (MPa)</td>
<td>1492</td>
<td>1878</td>
<td>1347</td>
<td>499</td>
<td>1750</td>
<td>2385</td>
</tr>
<tr>
<td>Shear modulus &quot;G&quot; (MPa)</td>
<td>895</td>
<td>1127</td>
<td>695</td>
<td>299</td>
<td>761</td>
<td>1419</td>
</tr>
<tr>
<td>Uniaxial compressive strength (MPa)</td>
<td>0.273</td>
<td>0.287</td>
<td>0.114</td>
<td>0.015</td>
<td>0.165</td>
<td>1.01</td>
</tr>
</tbody>
</table>

\[ K = \frac{E}{3(1-2v)} \]
\[ G = \frac{E}{2(1+v)} \]

### NUMERICAL MODELLING

**Planning and modelling**

The E1 longwall panel of Parvade 1 coal mine with two entry system is modelled using FLAC2D software. The E1 panel the first panel extracted was 170 m in width and 980 m in length using the retreating method. According to Figure 5 the width and length of model are 25 and 50 m, respectively. In addition, there are 18 layers with six types of rock from floor to roof (according to the Table 1 and 2). The Mohr-Coulomb model is selected for this study. The thickness of the coal seam is 2 m. The immediate roof consists of mudstone with 5.5 m in height (Tabas Coal Project, 2003). Initial and boundary conditions are shown in Figure 5. The constant load of 4 GPa applied instead of strata load to ground surface.

![FLAC2D model geometry and boundary condition](image)

In the retreating longwall working, following the excavating of main and tail entry tunnels, the setup room tunnel will excavated between them approximately 7 to 8 m in length. When the coal extraction is beginning, the distance between end of the power support canopy and wall is almost 1.5 to 2 meters. Therefore, the first step will be modelling of this section (Figure 6a). A general layout for the longwall panel is described in Figure 6b.
Model formulation

Formulation of the two dimensional Mohr-Coulomb plasticity model consists of construction of model geometry, definition of the constitutive relation and material properties for rock mass and parting planes, \textit{in situ} stress initialization and assignment of boundary conditions of the model. In addition to this, some monitoring parameters like history of unbalanced force for a given point have also been introduced to check the convergence requirement of the virgin model after its solution.

The model geometry of a longwall panel consists of 5.5 m floor rock overlain by a coal seam. The coal seam is overlain by roof consisting of two layers namely immediate roof and main roof. These layers comprise the caving zone of the overlying strata. The element size along X direction in the mining zone is 0.25 m. In Y direction, the size of elements in each layer has been kept equal. A schematic layout of one of the models showing the grid density in its different zones is shown in Figure 7.

Figure 7 - Schematic layout of one of the models showing the grid density in its different zones

The floor boundary is fixed in both the X and Y directions. The sides are fixed in X direction only (roller boundary) till the virgin model gets converged after initialization of \textit{in situ} stress. The side boundaries are later on locked after initializing the X and Y displacements and the velocities to zero, simulating the clamped side boundary in the virgin model. These boundary conditions allow vertical and shear displacements in the model without affecting its external geometry. The X and Y displacement and velocity values in the virgin model are initialised to zero after achieving the equilibrium condition to simulate the virgin ground condition.
RESULTS AND DISCUSSION

For estimation of the vertical load variations on the coal production area with increase in the area width, before the first weighting occurs, pressure arch, beam theory and Terzaghi methods were employed. In Figure 8, the modelled vertical load on the working area based on these methods is illustrated.

According to the curves, pressure arch and beam theory methods show good agreement with each other. Also, in the Terzaghi method, vertical load on the tunnel is more than from other methods. By using the beam fixed at both ends theory and Peng’s equation (Equation (13)), the first weighting distance obtained 10.9 m and by means of site observations, this value has been reported as 12 m. By using numerical modelling, the progressive face advance is simulated in stages of 0.75 m till the main fall caving (first weighting) are observed within the two dimensional modelling limitations (Figure 9).

(a) 5.25 m Face advance at 5 steps

![Figure 8 - Vertical load variations on the working area with increase in area width before the first weighting](image-url)
In this figure (Figure 9.c) red and brown colours indicate the highest and lowest rate of displacement, respectively. As expected, the displacement progress increases with face width advancing and expands toward its centre. In addition, in Figure 9, the state of displacement distribution, displacement vectors and state of plasticity in longwall model, are presented.

CONCLUSIONS

In this study, 2D modelling of the first weighting distance with the longwall caving method applied at the E1 panel of the Parvade1 Underground Mine was carried out by FLAC2D. For realistic modelling, material properties were derived for the rock mass from laboratory data by using Mohr-Coulomb failure criterion, the RMR and GSI systems together with empirical equations. Results of this modelling study revealed that the first weighting distance was found to be 11.2 m. Also the first weighting distance obtained by using analytical method was 10.9 m. According to the site observations, on the other hand, the first weighting distance has been reported as 12 m. Comparison of the results obtained from numerical approach, analytical calculations and observed data show good agreement. From the point of view of forces affecting the production area, the vertical load on the area based on pressure arch,
beam theory and Terzaghi methods were compared with each other. All these three methods show a consistent increase in the pressure with increasing coal face advance before the first weighting occurs. Pressure arch and beam theory methods show better agreement in comparison with the Terzaghi method.

REFERENCES