Research Article

Predicting Overburden-Induced Failure Height of Large Mining Face in Shallow Coal Seam Based on Half-Plane Theory

Shuai Di

Information Research Institute of the Ministry of Emergency Management, Beijing 100029, China

Correspondence should be addressed to Shuai Di; 1150563280@qq.com

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Research on the overburden-induced failure height in large mining faces is important for solving the problems of sand rupture and water permeability when mining in shallow depth. In order to solve the safety and high-efficiency mining problems, this study used the 7 m-high face #52306 of the Daliuta mine as the object of research for theoretical analysis and field measurements. The results show that the traditional empirical formula is not suitable for predicting overburden on three zones of the large mining face in shallow depth. We use elastic half-plane theory to establish a mechanical model to calculate overburden-induced stress and analyze the law of transmission of abutment pressure in the overburden. We then use maximum shear stress theory to develop a method to predict the height of the overburden in the "three zones" of the mining face based on the law of change in the shear stress. According to the characteristic whereby shear stress at the rear of the working face increases first and then decreases, the area where the shear stress was greater than zero was defined as the zone of water conduction-induced fracture, the area where the shear stress increased was identified as the zone of collapse, and the area where the shear stress decreased was defined as the fracture zone. A comparison of the theoretically calculated values with empirically measured data showed that the relative error was controlled to within 6%, verifying the reliability of the proposed method. The results here can provide useful information for the efficient and environmentally friendly mining of shallow coal seams.

1. Introduction

The mining of shallow coal seams often leads to accidents due to sand burst and the penetration of water in the working face, where this threatens the safety of workers and significantly hinders production. Many researchers have thus investigated the overburden-induced failure of shallow coal seams: He et al. [1] used similar simulations, theoretical analyses, and engineering verification systematically to investigate the prediction of the height of the water-flowing fractured zone. Guo et al. [2] put forward a new method of predicting the height of the fractured water-conducting zone based on overburden failure transfer processes. Xu et al. [3] used similar materials to analyze the influence of the position of the key layer on the height of the water-conducting fracture zone and proposed a method to estimate this height. Fu et al. [4] used numerical simulations to analyze the influence of the mining height and the length of the working face on the height of the overburden-induced caving zone and developed a regression formula for it. Ma et al. [5, 6] conduct research around backfill mining to control rock formation migration laws. Research on the failure characteristics of the "three zones" of overburden has mainly used numerical simulations, similar materials, and field measurements [7–23], and each relevant study has shortcomings. For example, on-site measurement requires considerable human effort and financial resources, and numerical simulations lack scientificity. Predicting the height of the "three zones" of overlying strata in a high working face is an important issue.

This study takes the 7 m-high working face #52306 in the Shendong mining area as the object of research to analyze the applicability of the traditional empirical formula to the high working face of a shallow and buried mine. We use the half-plane theory of elastic mechanics to analyze the law of change in the overburden-induced stress and the
characteristics of change in the shear stress. We then use the theory of maximum shear stress to propose a method of dividing the “three zones” of the overlying rock based on changes in shear stress. The results of this study provide useful information for reasonably determining the failure characteristics of the overlying rock of a shallow but high working face.

2. Mining Conditions of the Face

Working face #52306 of the Daliuta coal mine in the Shendong mining area is located to the west of the boundary of the minefield, with working face #52307 on the north and the mined-out area of working face #52305 to the south. The working face is 1284 m long and 301 m wide, the average thickness of the coal seam is 7.35 m, and the dip is 1°–3°. The coal seam has a simple structure. The detection borehole is a postmining hole, 403 m from the cut-off hole and 26 m from the air return lane. The layout of the working face is shown in Figure 1. The direct roof is dominated by siltstone, the basic roof by fine sandstone, and the floor by siltstone. The comprehensive histogram of the working face is shown in Figure 2. According to the test results of the adjacent mine Shangwan coal mine, it is judged that the in-situ stress is about twice the gravitational stress field.

3. Applicability of Traditional Empirical Formula

Taking the height of the strata overlying working face #31401 of Bulianta in Shendong as an example, we use traditional empirical formulas and measured methods (the borehole flushing liquid lost value analysis). The traditional empirical formulas (1) and (2) are from the literature [24]. Comparing the calculated value and the actual measured value, the results are shown in Table 1.

\[ H_m = \frac{100M}{4.7M + 19} \pm 2.2, \quad (1) \]

\[ H_l = \frac{100M}{1.6M + 3.6} \pm 5.6. \quad (2) \]

where \( H_m \) is the height of the collapse zone, \( H_l \) is the height of the fracture zone, and \( M \) is the mining height.

Table 1 shows that the calculation results of the empirical formula incurred large errors compared with the field measurements. The main reasons for this are as follows:

1. The empirical formula was based on field measurements, but most of these were from working faces with mining heights shorter than 3.5 m
2. The authors mainly considered a single factor and the mining height and ignored the influence of other factors on the height of the “three zones” of the overlying strata

(3) The change in mining height had no effect on the remainder between the maximum and minimum heights of the collapse zone and the fracture zone, which were 4.4 m and 11.2 m, respectively.

Therefore, the empirical formula cannot be directly applied to the working face of large height. Given that stress is the core factor influencing the failure of the rock mass, we analyze the characteristics of changes in shear stress to determine the law of changes in overburden-induced stress from the perspective of abutment pressure. We then use the theory of maximum shear stress to develop a method for dividing the “three zones” of the overlying rock based on changes in shear stress.

4. Method to Predict Three Zones of Overlying Strata

4.1. Analyzing Law of Transfer of Abutment Pressure in Overlying Strata

We use elastic half-plane theory to analyze the problem of propagation of abutment pressure in the overlying strata [25]. If all influential factors are considered, the complexity of the problem and difficulty of mathematical derivation render it impossible to obtain a solution. We thus need to ignore less influential factors and limit the problem to a feasible range. To this end, basic assumptions are made regarding homogeneity and isotropy, continuity, a lack of initial stress, and small deformations.

Once the working face has been mined, the overlying rock layer is regarded as a homogeneous elastic half-space body, and a longitudinal section (length is in meters, height is the buried depth \( H \), and thickness is 1 m) along the advancing direction of the face is taken as the research object. The plane strain problem is used for analysis.

To simplify the analysis for convenience of calculation, a linear function is used to describe the abutment pressure as shown in Figure 3. With the original stress in the rock as a reference, the ratio of increase in stress (3) is introduced for analysis. According to the theory of the half-plane, the body is subjected to force with a normal distribution along the boundary, and abutment pressure is regarded as an external force applied to the boundary of the research object. The model to calculate overburden-induced stress is established as shown in Figure 3.

\[ \lambda = \frac{\Delta \sigma}{p} = (K - 1), \quad (3) \]

where \( \lambda \) is the ratio of increment in stress, \( p \) is the original stress of the rock, \( \Delta \sigma \) is the increase in stress, and \( K \) is the stress concentration factor.

In Figure 3, stages I and II are the range of influence of the abutment pressure in front of the working face \( (d_1 + d_2) \). The distance between the peak abutment pressure and the wall of coal is \( d_2 \). Stage III represents the range of exposed rock behind the working face \( d_3 \). Stage IV is the range of the area where gangue has been compacted in the goaf \( (d_4) \). The stress in stages I, II, and IV is expressed by a linear function, and that in Stage III has a definite value. Stress is given by
\[ \lambda = ax + b \text{ (stage I, II, IV)}, \]
\[ \lambda = -1 \text{ (stage III)}, \]

where \(x\) is an independent variable, \(a\) is the slope, and \(b\) is the intercept.

Figure 4 shows, given the force balance condition, that the increase in stress needs to satisfy the following equation:
\[ \frac{K - 1}{2} \times (d_1 + d_2) = \left( d_3 + \frac{d_4}{2} \right). \]

Table 1: Error analysis of "three zones" height in overlying strata by empirical formulas.

<table>
<thead>
<tr>
<th>Working face</th>
<th>Overlying strata</th>
<th>Empirical formula (m)</th>
<th>Measured value (m)</th>
<th>Error (m)</th>
<th>Relative error (%)</th>
</tr>
</thead>
<tbody>
<tr>
<td>31401</td>
<td>Collapse zone</td>
<td>9.01–25.62</td>
<td>17.06</td>
<td>8.05–8.61</td>
<td>50.46</td>
</tr>
<tr>
<td></td>
<td>Water-conducting fracture zone</td>
<td>36.07–52.43</td>
<td>153.90</td>
<td>101.47–117.83</td>
<td>76.56</td>
</tr>
</tbody>
</table>

Suppose that the half-plane body is distributed on the boundary and its concentration at each point is \(\sigma\). The coordinates of point \(N\) are \((x, y)\), where \(x\) and \(y\) are the vertical and horizontal distances between \(N\) and the origin of the coordinate system \(O\). At distance \(\xi\) from the origin \(O\), we consider the infinitesimally small length \(d\xi\) and regard the force \(dF = d\xi \sigma\) on it as a small concentrated force. The vertical and horizontal distances between \(N\) and the concentrated force \(dF\) are \(x - \xi\) and \(y - \xi\), respectively. By superposing all stresses caused by small concentrated forces, stresses caused
Figure 3: Distribution characteristics of mining abutment pressure after simplified.

by all distributed forces at point \( N \) at this stage can be obtained, as shown in the following equation:

\[
\begin{align*}
\sigma_x &= -\frac{2}{\pi} \int_1^{y_2} \frac{\lambda x^3 d\xi}{\left[ x^2 + (y - \xi)^2 \right]^2}, \\
\sigma_y &= -\frac{2}{\pi} \int_1^{y_2} \frac{\lambda x(y - \xi)^2 d\xi}{\left[ x^2 + (y - \xi)^2 \right]^2}, \\
\sigma_{xy} &= -\frac{2}{\pi} \int_1^{y_2} \frac{\lambda x^2(y - \xi) d\xi}{\left[ x^2 + (y - \xi)^2 \right]^2},
\end{align*}
\]  

(6)

where \( \sigma_x, \sigma_y, \) and \( \tau_{xy} \) are the components of stress caused by a small concentrated force at point \( N \).

Therefore, assuming that the increase in stress is the distributed force on the boundary of the half-plane of the overlying strata, the coordinate axis used to find the change in stress at a point \( N \) in the overlying strata is shown in Figure 4. According to (6), the degree of change in stress caused by each value of the ratio of increase in stress at any point \( N \) in the overburden of the working face is shown in the following equation:

\[
\begin{align*}
\lambda_x &= -\frac{2}{\pi} \int_1^{y_2} \frac{\lambda x^3 d\xi}{\left[ x^2 + (y - \xi)^2 \right]^2}, \\
\lambda_y &= -\frac{2}{\pi} \int_1^{y_2} \frac{\lambda x(y - \xi)^2 d\xi}{\left[ x^2 + (y - \xi)^2 \right]^2}, \\
\lambda_{xy} &= -\frac{2}{\pi} \int_1^{y_2} \frac{\lambda x^2(y - \xi) d\xi}{\left[ x^2 + (y - \xi)^2 \right]^2},
\end{align*}
\]  

(7)

where \( \lambda_x, \lambda_y, \) and \( \lambda_{xy} \) are components of each ratio of increase in stress at any point \( N \) in the overburden of the working face.

The difference in the lithology of the overlying strata leads to stress transfer. The coefficient of decline in the weak face of the rock formation is used to describe this characteristic (see Table 2).

We introduce the coefficient of decline in the weak face of the rock formation (\( \eta \)) to describe the influence of lithology on the failure characteristics of the overburden. Equation (7) is modified as

\[
\begin{align*}
\lambda_x &= \eta \int_1^{y_2} \frac{\lambda x^3 d\xi}{\left[ x^2 + (y - \xi)^2 \right]^2}, \\
\lambda_y &= \eta \int_1^{y_2} \frac{\lambda x(y - \xi)^2 d\xi}{\left[ x^2 + (y - \xi)^2 \right]^2}, \\
\lambda_{xy} &= \eta \int_1^{y_2} \frac{\lambda x^2(y - \xi) d\xi}{\left[ x^2 + (y - \xi)^2 \right]^2},
\end{align*}
\]  

(8)

We add the ratio of increase in stress at each stage at any point \( N \) in the overburden of the working face as

\[
\begin{align*}
\sum \lambda_x &= \lambda_{xI} + \lambda_{xII} + \lambda_{xIII} + \lambda_{xIV}, \\
\sum \lambda_y &= \lambda_{yI} + \lambda_{yII} + \lambda_{yIII} + \lambda_{yIV}, \\
\sum \lambda_{xy} &= \lambda_{xyI} + \lambda_{xyII} + \lambda_{xyIII} + \lambda_{xyIV},
\end{align*}
\]  

(9)

where \( \lambda_{xI}, \lambda_{yI}, \) and \( \lambda_{xyI} \) are the components of each ratio of increase in stress in Stage I; \( \lambda_{xII}, \lambda_{yII}, \) and \( \lambda_{xyII} \) are the components in Stage II; \( \lambda_{xIII}, \lambda_{yIII}, \) and \( \lambda_{xyIII} \) are the corresponding components in Stage III; \( \lambda_{xIV}, \lambda_{yIV}, \) and \( \lambda_{xyIV} \) are the components of each ratio of increase in stress in Stage IV.
4.2. Analyzing Movement of Overlying Strata and Failure Zone in Working Face. According to Section 4.1, the rock layers at different locations have different forces, resulting in different deformations and failures in them. Therefore, it is necessary to study the characteristics of changes in the principal stress caused by this to clarify the law of overburden-induced changes in stress.

According to the stress state at any point on the plane, the principal stress can be obtained from (10) as

\[
\begin{align*}
\lambda_1 &= \frac{\lambda_x + \lambda_y}{2} + \frac{1}{2} \left( \frac{\lambda_x - \lambda_y}{2} \right)^2 + \lambda_{xy}, \\
\lambda_2 &= \frac{\lambda_x + \lambda_y}{2} - \frac{1}{2} \left( \frac{\lambda_x - \lambda_y}{2} \right)^2 + \lambda_{xy},
\end{align*}
\]

(10)

where \(\lambda_1\) is the maximum principal stress and \(\lambda_2\) is the minimum principal stress.

For the convenience of calculation, we assume that the rock layer mainly undergoes shear failure. According to the theory of maximum shear stress, we assume that the rock layer reaches the maximum shear stress in a complex stress state and fails. The maximum shear stress at any point \(N\) in the overburden of the working face is

\[
\tau_{\text{max}} = \frac{\lambda_1 - \lambda_2}{2} \quad \text{(maximum shear stress)}. \tag{11}
\]

To improve the reasonableness of the three-zone division of the overlying strata, we use the law of changes in the abutment pressure and shear stress to propose criteria for dividing the three zones of overlying strata.

(1) Selection of characteristic points: According to the law of change in the abutment pressure, after stress returns to its original state on the goaf side, the abutment pressure cannot influence the law of overburden-induced change in stress, and thus, the rear is relatively stable. Therefore, the position at which the original stress of the rock was restored on the goaf side (i.e., the increase in stress was zero) was selected as the characteristic point for the division of the overlying strata into three zones.

(2) Division method: The law of change in shear stress at different positions of the working face can be obtained using equations (10) and (11). The shear stress in the goaf first increased and then decreased. We analyzed the extent of rock failure and determined the height of the three overlying strata. The area where the shear stress was greater than zero was defined as the water-conducting fracture zone, the area where the shear stress increased was identified as the collapse zone, and the area where the shear stress decreased was the fracture zone. The area where the shear stress was zero was defined as the bending subsidence zone, as shown in Figure 5.

The law of overburden-induced failure was found to be mainly related to the lithological characteristics of the strata, the stress concentration factor, the range of changes in abutment pressure in front of the work face, the range of exposed rock strata, and the area where the gangue had been compacted in the goaf. These could be obtained by measurements and calculation. The stress concentration factor is a comprehensive manifestation of such factors as the mining height, buried depth, and advancing speed. This method can be used to easily obtain related measurements. It is also more efficient than competing methods in terms of human labor and material resources.

5. Analyzing the Effects of Application

5.1. Law of Transmission of Abutment Pressure in Overburden of Working Face. According to the column chart of coal seams, the overlying rock in working face #52306 was mainly siltstone. Lithology therefore had no prominent influence on the effect of transmission of stress. We set the coefficient of decline in the weak face of the rock formation to one.

According to measured data on-site, \(d_1 = 50\, \text{m}, d_2 = 10\, \text{m}, K = 2.0,\) and \(d_3 = 15\, \text{m}.\) Then, \(d_4 = 30\, \text{m}\) by (8). Therefore, the distance between the working surface and the origin of the coordinate system was 60 m. The law of increase in stress in the overlying strata was obtained by using MATLAB, and the result is shown in Figure 6. The abscissa \(x = -60\) represents the position of the working face, and the ordinate represents the height of the overburden.
5.1.1. Characteristics of Variation in Vertical Stress in Overlying Strata. The overlying rock strata on the goaf side were in the stress reduction zone. We used a 20 m range as an example. Along the longitudinal direction, the vertical stress of the rock formation within 0~2 m above the coal seam was close to zero. Its vertical stresses within the range of 2~5 m and 5~10 m were 0.8~0.6 and 0.6~0.4 times the stress of the original rock, respectively, and reached its minimum value at about 10 m above the coal seam. The vertical stresses of the rock formation within the range of 20~30 m and 30~50 m were 0.4~0.6 and 0.6~0.8 times the stress of the original rock, respectively. Along the transverse direction, the vertical stresses within horizontal distances of 2~4 m and 4~8 m were 0.8~0.6 and 0.6~0.4 times the stress of the original rock, respectively, and reached its minimum value at about 10 m above the coal seam.
rocks, respectively. The vertical stresses within horizontal distances of 25∼35 m and 35∼50 m from the coal seam were 0.4∼0.6 and 0.6∼0.8 times the stress of the original rock. Therefore, along the longitudinal direction, as the vertical distance between the working face and the overlying strata increased, the vertical stress first decreased and then rose; in the transverse direction, as the horizontal distance from the working face increased, the vertical stress first decreased and then rose.

5.1.2. Characteristics of Variation in Horizontal Stress of Overlying Strata. Within 40 m on the goaf side, the horizontal stress of the rock formation within 0∼20 m above the coal seam was in a state of pressure relief, and the degree of relief first increased and then decreased. The horizontal stress concentration was smaller than the concentration of vertical stress above the coal body.

5.1.3. Characteristics of Variation in Shear Stress of Overlying Strata. Shear stress was observed in a certain range in the goaf and in front of the coal wall, and variations in it were symmetrically distributed along a straight line. The maximum value of shear stress was 1.06 times the stress of the original rock, and it was inclined to the front of the working face. Shear stress weakened the rock formation and led to the destruction of the overlying rock formation.

To sum up, the abutment pressure had a greater impact on the changes in vertical stress in the overlying strata and a smaller impact on changes in the horizontal stress and shear stress.
5.2. Characteristics of Distribution of “Three Zones” of Overlying Strata in Working Face. Equations (10) and (11) were solved in MATLAB to obtain the characteristics of the distribution of principal stress and shear stress. The results are shown in Figures 7 and 8.

Characteristics of changes in the shear stress of the rock at different vertical distances from the working face on the goaf side, in longitudinal sections with horizontal distances of 10 m, 15 m, and 20 m from the working face, were used for analysis as shown in Figure 9.
The following conclusions can be drawn from Figure 8:

(1) As the vertical distance between the overlying rock strata and the working face increased, the overall shear stress first increased and then decreased, and finally tended to become stable.

(2) The maximum shear stresses at 10 m, 15 m, and 20 m were 0.42, 0.36, and 0.33, respectively. Therefore, as the horizontal distance from the working surface increased, the peak shear stress on the rock layer continued to decrease.

We used MATLAB to calculate the height of the three zones of overlying strata on working face #52306, as shown in Figure 10.

Figure 9 shows that the collapse zone was 21 m, the fracture zone was 118 m, and the water-conducting fissure zone was 139 m. The fissures developed directly to the surface without a bending subsidence zone.

According to the literature [26], the distribution of the three zones of overlying strata on working face #52306 was measured by the borehole flushing fluid method. In the hole depth of 18.00–31.25 m, the leakage of the flushing fluid is 0.03–0.06 L/(m·s). In the hole depth of 31.25–35.75 m, the flushing fluid leaks out. In the hole depth of 35.75–46.25 m, the leakage of the flushing fluid is 0.64–1.20 L/(m·s). In the hole depth of 31.25–35.75 m, the flushing fluid leaks out. The results show that the collapse zone was 19.82 m, the fracture zone was 117.50 m, and the water-conducting fracture zone was 137.32 m. A comparison between the theoretical results and field measurements is shown in Table 3.

Table 3 shows that the relative error between the theoretical values and the field measurements was small. This shows that the proposed method can be used to guide field practice and provides a theoretical foundation for the three-zone division of the overlying strata. This is useful for preventing rushing water and other disasters in mines.

### 6. Conclusion

(1) We used the half-plane theory of elastic mechanics to establish a model to calculate overburden-induced stress. We used this to analyze the law of changes in overburden-induced stress and changes in shear stress. Following this, we used maximum shear stress theory to develop a method to divide the overlying rock into three zones based on the characteristics of changes in shear stress.

(2) The law of overburden-induced failure is mainly related to the lithology of the strata, the stress concentration factor, the range of changes in abutment pressure in front of the working face, the range of exposed rock strata, and the area where the gangue has been compacted in the goaf.

(3) We used the working face #52306 of the Daluata Mine as an example. The error between the theoretical values and the field measurements was small, which verifies the reliability of the theoretical analysis.

### Data Availability

The data of this study are available from the corresponding author upon request.

### Conflicts of Interest

The authors declare that they have no conflicts of interest.

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