Study on Stability Analysis and Control Technology of Floor Rock Roadway in Dynamic Pressure Stope

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Abstract. Taking the floor rock roadway of 10414 working face of Yangliu Coal Mine in Huaibei as the engineering technology background, this paper uses theoretical analysis and numerical simulation technology to analyze the stability of floor rock roadway caused by stope mining. According to the dynamic pressure of stope, the support stage is divided into three stages: timely support scheme after excavation, pre-mining reinforcement support stage and post-mining repair support stage, and corresponding suggestions are put forward. Support scheme, through field practice, the overall maintenance of roadway affected by dynamic pressure is in good condition, the deformation of roof, floor and both sides are within the controllable range, and the roadway section can meet the requirements of normal use.

1. Project Overview
The first level buried depth of Yangliu Coal Mine in Huaibei is 569m, 10414 working face is the first mining face in 104 mining area of Yangliu Coal Mine. Its strike length is 1300m and inclination length is 180m. The inclination angle of working face is 2~11°, averaging 5 degrees. The roof is managed by free caving, the floor elevation of working face is -568m to -61m, and the thickness of coal seam is 2.87~3.68 m, averaging 3.31m.. The 10# coal seam in 104 mining area is determined to be the outburst dangerous area in the whole area. It needs to adopt the plan of gas drainage from floor. The maintenance time of roadway is long, and the surrounding rock deformation of roadway is strong under the influence of mining. Most of the roadway construction layers are medium and fine sandstone, with well developed cracks, low compressive strength and unstable roof.

2. Stability Analysis of Floor Rock Roadway by Mining in Dynamic Pressure Stope
The stress field of working face floor is the basic basis for the selection of floor roadway position and the design of support mode. According to the research results of soil mechanics theory, it is abstracted as the effect of concentrated load on the stress field increment of the lower semi-infinite plane body, that is, the effect of concentrated force P on the plane of the semi-infinite body. The stress distribution
formula of the floor under concentrated load is obtained by calculating the hypothesis that the lower rock stratum is isotropic intact rock mass and the lithology does not change with time.

Figure 1. Positional relationship graph of floor rock drift and upper workface

\[
\begin{align*}
\sigma_z(z, x) &= -3Pz^3 \left[ \frac{z^2}{z^2 + x^2} \right]^{3/2} \\
\sigma_x(z, x) &= -3Pz^2 \left[ \frac{z^2}{z^2 + x^2} \right]^{3/2} \\
\tau_{xz}(z, x) &= -3Pz^2 x \left[ \frac{z^2}{z^2 + x^2} \right]^{3/2}
\end{align*}
\] (1)

In practice, for rock mass with better lithology, the above formula can be used. For rock fracture development, the propagation of force will cause the opening or closing behavior of cracks. For the stress propagation formula in Formula 1 derived from micro-strain, a great change will occur. The effect of the development of floor rock fracture on stress propagation has to be considered in the calculation. For strata in tectonic influence zone, a large number of irregular cracks are developed in rock mass due to tension and compression stress caused by tectonic movement. Influenced by the advance supporting stress of working face, the cracks of coal seam floor rock layer further develop, and with the working face pushing through the process, due to the supporting stress of lateral coal pillars, the floor cracks change more complex. For fractured rock mass, the stress propagation process of coal seam floor is anisotropic under the influence of floor stress increment in the mining process of overlying coal seam due to the difference of fracture density and direction in different directions and the existence of bedding.

The deterioration coefficient $\xi$ of rock mass is defined to measure the effect of cracks on stress propagation of floor. Its value depends on the sum of crack gap distances per unit distance. It can be seen that, due to the anisotropy of rock mass, the deterioration coefficient of rock mass is also anisotropic in the process of being affected by stress. With the passage of time, cracks in rock mass open and close. Therefore, the deterioration coefficient of rock mass can be expressed as $\xi(\beta, t)$. The solution process of substitution 1 can be modified to obtain:

\[
\begin{align*}
\sigma_z(z, x, t) &= \frac{\xi(t)Pz^3}{2\pi} \left[ \frac{z^2}{z^2 + x^2} \right]^{3/2} \\
\sigma_x(z, x, t) &= \frac{\xi(t)Pz^2}{2\pi} \left[ \frac{z^2}{z^2 + x^2} \right]^{3/2} \\
\tau_{xz}(z, x, t) &= \frac{\xi(t)Pz^2 x}{2\pi} \left[ \frac{z^2}{z^2 + x^2} \right]^{3/2}
\end{align*}
\] (2)
In the formula, \( \xi(t) = \frac{1}{\pi} \int_{0}^{\pi/2} \cos^2 \beta \cdot \sin \beta \cdot \xi(\beta, t) \cdot d\beta \)

According to Saint Venant’s principle, if only the magnitude and distribution of force are changed without changing its effect, the change of stress distribution far from the action point can be neglected except for the change of stress distribution near the action point. Therefore, the support stress distribution before and after the working face is simplified as piecewise linear function distribution.

![Simplified diagram of pressure distribution in front and behind the workface](image)

**Figure 2.** Simplified diagram of pressure distribution in front and behind the workface

The abutment pressure distribution functions of AB, BO and CD are obtained from Figure 2.

\[
\begin{align*}
q_1(\eta) &= \frac{(k-1)(a-1)\gamma H}{b} \eta + k\gamma H \\
q_2(\eta) &= \frac{k\gamma H}{a} \eta \\
q_3(\eta) &= -\gamma H \frac{d}{d} \eta - \frac{c\gamma H}{d}
\end{align*}
\]

(3)

In the formula, \( k \)-Stress Concentration Factor of Advanced Supporting in Working Face

By calculus method, the stress of each part on a certain point of the floor is obtained and the sum is obtained. The stress at any point of the floor is as follows

\[
\begin{align*}
\sigma_z(z, x, t) &= -\frac{z^3}{2\pi} \sum_{i=1}^{3} \int_{h_i}^{\eta_i} \frac{\xi_i(t)q_i(\eta)}{\left[z^2 + (x-\eta)^2\right]^{5/2}} d\eta \\
\sigma_x(z, x, t) &= -\frac{z^3}{2\pi} \sum_{i=1}^{3} \int_{h_i}^{\eta_i} \frac{\xi_i(t)q_i(\eta)(x-\eta)^2}{\left[z^2 + (x-\eta)^2\right]^{5/2}} d\eta \\
\tau_{xz}(z, x, t) &= -\frac{z^2}{2\pi} \sum_{i=1}^{3} \int_{h_i}^{\eta_i} \frac{\xi_i(t)q_i(\eta)(x-\eta)}{\left[z^2 + (x-\eta)^2\right]^{5/2}} d\eta
\end{align*}
\]

(4)

In the formula, \( z \)- The vertical distance between the point and the coal seam floor

\( x \)-The distance between the point and the horizontal direction of the coal wall in the working face

From the above formula, it can be seen that the stress of a point in the floor is closely related to the upper load on the floor and the position of the point on the top abutment pressure. At the same time, it is affected by the development of fissures in the floor rock. The upper load is determined by the burial depth, the thickness of the coal seam, the mining height and other factors. When the coal seam
occurrence conditions are certain, the optimum mining technology is basically determined, while the location of the floor rock roadway is determined. It is not affected by coal seam occurrence conditions. Therefore, in order to control surrounding rock of roadway with dynamic pressure floor, it is necessary to start with rational layout of roadway location and optimization of stress field of roadway.

The K value used in calculating the stress value in Form 4 is the stress concentration factor of the advance support of the working face. When the thickness and mining height of the coal seam are determined, the K value in front of the working face is related to the direction distance of the point from the coal pillar left behind in the working face. The stress concentration factor is larger when the point is close to the coal pillar, and gradually decreases to one when it is close to the middle of the working face. Stable value. Therefore, the change of distance between coal pillar and coal pillar has certain influence on the advance stress of working face.

Using FLAC3D numerical simulation analysis software and Mohr-Coulomb constitutive model, the dimension of the three-dimensional model in the x, y and Z directions is 130 m x 100 m x 83.3 m. There are about 350,000 meshes and 360,000 nodes. The numerical calculation model is shown in Figure 3. The top boundary of the model is designed to be buried at 600 m depth. The stress boundary is used to apply uniform load according to the weight of overlying strata, and the average density of strata is 2500 kg/m³. The lower boundary is fixed, and there is no displacement in the x, y and Z directions. The front, back and both sides of the model are set as displacement boundary, and the horizontal boundary displacement is fixed.

![Figure 3. Numerical calculation model and boundary condition diagram](image)

In the mining process of the working face, four monitoring lines are set at different depths of the center of the working face to monitor the vertical stress distribution in the direction of the working face, as shown in Fig. 4. It can be seen from the graph that the vertical stress of the floor in the mining process of the overlying working face tends to concentrate ahead, reduce pressure relief and recover compaction as a whole; with the increase of the depth of the floor, the stress concentration factor of the floor caused by the advance support stress gradually decreases; when the vertical distance from the floor of the coal seam is 30 m, the influence of mining on the floor begins to ease gradually. Considering that the floor roadway needs to be used to extract and shield the upper roadway, the distance between the roadway and the coal seam is restricted by the cost of drilling construction, therefore, the floor roadway is selected to be 25m below the floor of the coal seam.
Figure 4. Vertical stress distribution curves along the strike

Four monitoring lines are set in a range of 50m around the working face and 25m below the coal floor to record the vertical stress. As shown in figure 5, the vertical stress of the floor varies with the distance of the coal pillar in the direction of inclination of the working face. It can be seen from the figure that the lower part of the coal pillar always maintains a high stress value, which increases with the working face pushing. Within the range of 10-20m away from the coal pillar, the stress of the floor decreases sharply with the working face, and the stress environment remains small after the working face. The stress in the range of 25~80m away from the coal pillar decreases first and then increases, and basically returns to the initial stress when it is near the middle of the working face. Therefore, when the horizontal position of the floor roadway is 20m dislocation within the boundary of the coal pillar, it can be in a low-stress environment after the mining of the working face to reduce the roadway destruction and support difficulty.

Figure 5. Vertical stress distribution curves along the dip

3. Surrounding Rock Control Technology of Floor Rock Roadway in Dynamic Pressure Stope
In order to ensure that the roadway can meet the needs of ventilation, pedestrians and other safety production after final deformation, the roadway section is reserved to meet the requirements of deformation. The roadway section is initially defined as a straight wall semi-circular arch, with net width×medium height = 4.0m×3.4m. Based on a lot of field investigation of the test roadway, it is found that the surrounding rock of the 10414 bottom extraction roadway is soft and fractured, and will be
affected by the strong mining stress. Combining with the current situation of the strong deformation and instability of the surrounding rock of the relevant roadway after mining, the supporting scheme and supporting parameters in the process of roadway repair are determined.

3.1. Timely support scheme after roadway excavation

In the initial stage of roadway excavation, stress compensation is mainly carried out through support for three-dimensional stress transformation caused by excavation. The main body of roadway adopts bolt-mesh shotcrete support scheme. Aiming at the surrounding rock of some fault fractured zones, the roof support scheme is adopted.

(1) The roof of the roadway is supported by seven IV-grade super-strong pre-tension bolts with 4.6 m long M4 steel strip and 6# cold-drawn steel mesh. The row spacing between bolts is 800 mm x 800 mm. The bolt specifications are M20-22-L2400mm and the reinforcing steel mesh specifications are 1000mm*1000mm. The two sides of the roadway are supported by three sets of strong pre-tension bolts, such as grade II L-screw steel, plus 2.8m long M3 steel strip and steel mesh. The specifications of the bolts are M22-20-L2200mm. The bolt-mesh support arrangement is shown in Fig. a.

In special areas with sudden increase of roof watering, abnormal change of direct roof thickness, and impact tendency of sticking and suction drilling, the row spacing should be reduced in time, the density of anchor rope should be increased, and the anchor rope should be struck in time.

(2) U29 steel roof is used to support the area where the anchorage effect of roadway crossing fault surrounding rock is worse than that of broken section. The lateral leg angle is 87°, the roof spacing is 800mm, and six 50mm×50mm angle steel rods are used to connect the roof with the roof. The steel bars are welded with 10 mm steel bars on the waist, back and top. The specifications are 700 mm×800 mm and the meshes are 60 mm×60 mm. The shack legs must be paved with stones for shoes. The size of stones is 300 mm×300 mm×150 mm in length and 200 mm×200mm×12 mm in width and thickness, and the size of iron shoes is 200 mm×200 mm×12 mm in length and width. The cross-section diagram of the roadway supported by roof is shown in Fig. B.

(3) Initial shotcrete closes the surrounding rock to prevent weathering and tidal decomposition of surrounding rock, concrete ratio, cement: yellow sand: stone = 1:2:2. The thickness of spraying layer is about 50mm, and the strength is C20. The spraying concrete work closely follows the head-on construction. After spraying, the concrete should be sprayed and maintained in time.

3.2. Pre-mining Reinforcement Support Scheme

When the trial operation of 10414 first mining face in Yangliu Coal Mine started, in order to reduce the influence of the first mining face on the ground pressure of the roadway under the face, especially on
the influence area of local fault structure and the geological abnormal complex zone, it was decided to modify and supplement the support scheme, and to design the reinforcement support. Using 300mm disc tray with short anchor cable, the specifications of the anchor cable are 18×4300mm, and the distance between rows is 900×1000mm. The thickness of the grouting layer is 70-100 mm. Shallow hole grouting with multicolored eyes has a row spacing of 2.4 m and a spacing of 1.2 m. Sulfoaluminate cement is used as grouting material. The depth of grouting hole is 1.5 m and the length of grouting bolt is 1.2 m.

3.3. Rehabilitation and Support Scheme after Mining

The net cross-section size of the repaired roadway is 4000 mm×3400 mm, and the full-section bolt belt net shotcrete scheme is adopted.

The roof of the roadway is supported by seven IV-grade ultra-strong pre-tension bolts with 4.6 m long M4 steel strip and 6# cold-drawn steel mesh. The specifications of the bolts are M20-22-L2800 mm, the row spacing between bolts is 800 mm×800 mm, and the specifications of the metal mesh are 1700 mm×1200 mm, and the mesh size is 100 mm×100 mm. Three sets of grade II leftwise threaded
steel bolts with equal pre-tension and 2.8m long M3 steel strip and steel mesh are used to support both sides of the roadway. The specifications of the bolts are M22-20-L2200mm, the row spacing between the bolts is 800 mm×800mm, and the specifications of the steel mesh are 1000mm×1000mm. Three sets of anchor cables are arranged on the roof of roadway. The type of anchor cables is 17.8×6300mm. The middle anchor cables are arranged along the middle roof, and one set of anchor cables is arranged on both sides. The distance between the anchor cables is 2000mm×1600mm.

After the repair and support installation of anchor belt mesh is completed, a re-spraying is carried out. The thickness of spraying layer is 100mm to prevent the corrosion of anchor metal mesh steel strip and other exposed air. The grouting parameters are the same as the pre-mining reinforcement support.

3.4. Supporting Effect
Yangliu 10414 Machine Drainage Roadway is located in the floor of 10414 Machine Roadway about 25m. The geological conditions are complex, the faults are developed, the ground pressure is large, the overall maintenance of roadway affected by dynamic pressure is good, the deformation of roof, floor and both sides are within the controllable range, and the roadway section can meet the requirements of normal use. In the area affected by concentrated stress in front of the working face, the vertical distance between some sections and the floor of the working face is small. With the advancing of the working face being affected by intense dynamic pressure, the support is strengthened and the whole roadway is repaired mainly by the floor heave control work.

Figure 9. The supporting effect of gas drained roadway in front of 10414 workface

4. Conclusion
Based on the engineering background of maintenance control during mining of floor rock roadway in 10414 face of Yangliu Coal Mine, this paper synthetically uses theoretical analysis, numerical simulation and field industrial test methods, divides support stages into three categories according to the idea of staged and staged support. According to the characteristics of rock pressure in each stage, it puts forward the technical scheme of strengthening control of floor roadway surrounding rock, and changes of roadway surrounding rock. The shape has been effectively controlled and the industrial test has been successful, which provides a reliable guarantee for the safe and efficient production of Mines under similar conditions.

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