Study on support Design of 2220 roadway in Majiliang Coal Mine based on FLAC3D

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Abstract. In view of the serious deformation of roadway 2220 in Majiliang Coal Mine after excavation, the surrounding rock deformation characteristics and stress redistribution of the roadway are studied and analyzed by numerical simulation technology, and the support optimization is carried out. The research results show that the original support scheme can not effectively reduce the roadway deformation and the area of plastic zone and stress concentration area after roadway excavation. After the optimization of the support scheme, the shape variables of the roof and floor and the two sides are obviously reduced, and the plastic zone and stress concentration area are also greatly reduced, which significantly improves the overall stability of the roadway.

1. Introduction
With the continuous innovation and progress of coal mining technology, the support design of coal mine roadway has been paid more and more attention, and bolt support, as an effective form of support, has been widely used in coal mining[1-2]. Renliang Shan studies the support form of strong wall corner of roadway through model test and field measurement, and finds that this support form can effectively control roadway deformation and improve roadway stability[3]. Liang Tian analyzed the denaturation characteristics of coal mine roadway by using mechanics and numerical model, and found that the key parts of roadway support should be corners and two sides, and strengthening the two sides support can achieve the effect of strong side roof protection[4]. Wenbin Tao proposed that the support system with ground stress measurement, full monitoring of force-measuring bolts, high pretension combined with full-length anchoring technology can comprehensively and systematically understand the stress environment of deep-buried roadways, thereby optimizing support parameters[5]. Through the numerical simulation of soft rock roadway in Hongmiao Coal Mine, Wencai Wang found that the combined support effect of FRP bolt and prestressed anchor cable is better than that of screw steel bolt support[6]. Based on the analysis of the roof stability of deep mining roadway, Nianjie Ma puts forward the connectable long bolt support technology which can better adapt to the severe subsidence of roof[7]. Domestic and foreign scholars have made some achievements in the research of bolt support technology[8-10]. Through the application of numerical simulation technology and taking the 2220 roadway working face of Majiliang Coal Mine as the research background, the author optimizes the support scheme, and the research results can provide reference for roadway support with similar geological conditions.
2. General situation of the project
The topography and geomorphology of the 2220 roadway working face of Majiliang Coal Mine are hills and platforms alternating between gullies and valleys, and the gullies and branch ditches are crossed, and the gullies are developed. Most of the surface is cultivated land, young trees, barren grassland, and there is a simple cart road. The height difference of the ground is about 88m. The strike length of the working face is 1705m, the dip length is 5.40m, and the average dip angle of the coal seam is 2°. The distribution of rock strata is shown in Table 1.

| Name               | Name of rock       | Thickness (m) | Lithologic characteristics                                                  |
|--------------------|--------------------|---------------|-----------------------------------------------------------------------------|
| Main roof          | Glutenite conglomerate | 5.83 ~ 8.05   | Grayish white, composed of quartz and feldspar. Dense, hard-harder. Sub-angular-sub-circular |
|                    |                    | 6.75          |                                                                             |
| Immed-i ate roof   | sand-shale siltstone | 0.50 ~ 5.72   | Black, light gray, gray-white charcoal mudstone, sand-mudstone and siltstone. The upper part is carbonaceous-mudstone and the lower part is siltstone. Block and brittle. |
|                    | Carbon-mudstone    | 2.82          |                                                                             |
| Coal seam          |                    | 6.18 ~ 8.44   |                                                                             |
|                    |                    | 7.63          |                                                                             |
| Immed-i ate floor  | sand-shale kaolinite-rock | 3.30 ~ 7.32  | Dark gray or grayish brown, dark brown, shell-shaped fracture. Thin coal seam in the lower part. |
|                    |                    | 3.20          |                                                                             |
| Main floor         | Pebble-sandstone conglomerate | 2.50 ~ 11.37 | Off-white, mainly composed of quartz, followed by feldspar. Occasionally contain coal dust, densely cemented |

3. Determination of support parameters

3.1. Bolt parameters
Based on the analysis of the surrounding rock properties and the mechanism of bolt support, the combined arch theory is applied to design the support parameters of each support method[11-12]. The specific parameters and design plan are as follows:
- **Minimum length of bolt:**
  \[ L = L_1 + \frac{b \tan \alpha + a}{\tan \alpha} \]
  In the form: \( L \)- Full length of bolt, m; \( L_1 \)- Exposed length of bolt, m; \( b \)-Thickness of composite arch, m; \( a \)- Row distance between bolts, m; \( \alpha \)- Bolt control angle, °.
  According to the measured data of mine, we take \( L_1 = 0.1 \) m, \( b = 0.9 \) m, \( \alpha = 45° \), \( a = 0.9 \) m, therefore \( L = 1.9 \) m.
- **Minimum diameter of bolt:**
  \[ d = K \sqrt{\frac{4Q}{\pi QS}} \]
  In the form: \( d \)- Bolt diameter mm; \( K \)- Surplus coefficient; \( Q \)- Anchoring force of bolt, N; \( QS \)- Tensile strength of bolt, MPa.
  According to the measured data of mine, we take \( K = 1.2 \), \( Q = 70 \) KN, \( QS = 350 \) MPa, therefore \( d = 19.1 \) mm.

3.2. Anchor cable parameters
The parameters of anchor cable support are determined according to the suspension theory, and the specific calculation method is as follows:

1. Minimum length of anchor cable:

   \[ L' = L_a + L_b + L_c + L_d \]

   In the form: \( L' \) - Anchor cable length, m; \( L_a \) - Anchorage length of anchor cable, m; \( L_b \) - Thickness of unstable strata that need to be suspended, m; \( L_c \) - Thickness of upper supporting plate and Anchorage, m; \( L_d \) - The tension length that needs to be exposed, m.

   According to the measured data of mine, we take \( L_a = 2.0 \) m, \( L_b = 5.2 \) m, \( L_c = 0.15 \) m, \( L_d = 0.25 \) m, therefore \( L' = 7.6 \) m. Therefore, the \( L' = 8 \) m anchor cable provided by the mining area meets the requirements.

2. Maximum spacing of anchor cable:

   \[ S_a = \frac{3[\sigma_a]}{4a^2\gamma k} \]

   In the form: \( S_a \) - Distance between anchor cables, m; \( a \) - Roadway width, m; \( \gamma \) - Average volume mass of overlying strata, kN/m³; \([\sigma_a]\) - Ultimate breaking force of single anchor cable, kN; \( k \) - Safety factor.

   According to the measured data of mine, we take \( a = 5.4 \) m, \( \gamma = 2599 \) kN/m³, \( k = 1.8 \), \([\sigma_a] = 490 \) kN, therefore \( S_a = 2.7 \) m.

4. Support scheme and model establishment

4.1. Determine the support scheme

According to the calculation results of the parameters, the optimization scheme is made.

1. The original support scheme:

   The diameter of the two bolts is 0.02 m, the bolt is installed horizontally, and the length of the bolt is 2.0 m; the row distance between bolts is 0.9 m; seven bolts are arranged in each row on the roof, the distance between rows is 900 × 850 mm, the bolts at the left and right corners are inserted at an angle of 80°, five anchor cables are arranged in each row, the distance between rows is 1600×1600 mm, the construction angle of shoulder angle is 70°. The specific layout is shown in Figure 1a.

2. New support scheme:

   The diameter of the two bolts is 0.02 m, the middle 3 are installed horizontally, the upper and lower roots go up and down at an angle of 20°, the length of bolt is 2.0 m, the row distance between bolts is 0.9 m. Seven bolts are arranged in each row on the roof, the distance between rows is 500 × 850 mm, five anchor cables are arranged in each row, the distance between rows is 1000×1600 mm. The construction angle of shoulder angle is 70°. The specific layout is shown in Figure 1b.
4.2. Establishment of numerical model

The finite difference software FLAC\textsuperscript{3D} is used to model the numerical simulation analysis, the model is Mohr-Coulomb Model, the two schemes are simulated and analyzed in the model, the simulation model constructed by FLAC\textsuperscript{3D} is shown in Figure 2. In the model, the coordinate origin is the center position at the bottom of the roadway, the x direction is the roadway width direction, the y direction is the roadway vertical direction, and the z direction is the roadway depth direction. In this model, a total of 44800 grids and 49419 grid nodes are divided. For the surrounding limited displacement boundary, the corresponding stress boundary is set at the top according to the buried depth of the roadway. The physical properties of each rock stratum in the roadway are shown in Table 2. Through the comparison of nephogram, the optimization of the new scheme is illustrated.

Table 2. Table of physical and mechanical parameters of rock strata

| Rock stratum     | $K$/GPa | $G$/GPa | $\rho$/kg $\cdot$ m$^{-3}$ | $C$/MPa | $\sigma_t$/MPa | $f$/° |
|------------------|---------|---------|----------------------------|---------|---------------|------|
| Main roof        | 15.73   | 11.02   | 2703                       | 34.68   | 2.50          | 32.01|
| Immediate roof   | 2.42    | 1.83    | 2496                       | 2.74    | 2.74          | 28.07|
| Coal seam        | 1.12    | 1.09    | 1598                       | 1.76    | 0.73          | 27.04|
| Immediate floor  | 2.51    | 1.76    | 2502                       | 2.78    | 2.78          | 27.98|
| Main floor       | 13.17   | 8.91    | 2698                       | 34.73   | 2.51          | 31.98|

5. Numerical simulation analysis

After the completion of the simulation, the two support schemes are compared by analyzing the roadway deformation, plastic zone distribution and stress redistribution of the two support methods.
As can be seen from the displacement nephogram Figure 3, when the original scheme is used for support, the roadway roof subsidence is 29.14mm, the floor uplift is 15.67mm, and the approach of the two sides is 45.11mm; while when the new scheme is used for support, the roadway roof subsidence is 26.35mm, which is 10% less than that of the original scheme, the floor uplift of 13.77mm is 12% less than that of the original scheme, and the approach of the two sides of 35.78mm is 22% less than that of the original scheme. The support effect of the new scheme is better and the effect is obvious.

As can be seen from the nephogram Figure 4 of the plastic zone, the support mode of the original scheme will produce a large plastic failure area above the roadway surrounding rock roof and below the floor, which will cause some dangerous factors in the construction environment, which is unsafe and unreasonable, while it is found that the plastic area produced by the new scheme is obviously reduced, and the overall strength of the surrounding rock is significantly improved, which can ensure the safety of the construction environment.

As can be seen from the nephogram Figure 5 of the major and minor principal stress, after the roadway is excavated, the forms of bolt support are different, and the stress redistribution around the surrounding rock of the roadway is also different. The stress redistribution will have a great impact on the safety of roadway mining. Comparing the stress redistribution of the two support schemes, it can be found that after the roadway excavation, the new scheme can better improve the stress state of the
surrounding rock of the roadway and reduce the stress concentration area caused by the shoulder corner.

![Image](image1.png)

**Fig 5. Nephogram of the major and minor principal stress**

### 6. Conclusion

After the excavation of the roadway, the application of unreasonable support schemes cannot effectively suppress the overall deformation of the roadway. After the stress is redistributed, a larger stress concentration area will be generated at the shoulder corners, and a larger area of plastic failure area will appear at the same time. After the optimization of the support, the displacement of the roof and the two sides of the roadway is significantly reduced, and the plastic failure and stress concentration areas are also significantly reduced. Therefore, compared with the original plan, the new plan has better support safety, has a better effect on improving the stability of the overall surrounding rock of the roadway, and can provide a guarantee for safe construction.

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