Surrounding rock control technology for the fault passing of mining face by removing gangue in advance from excavation roadways

Qin Zhongcheng¹, Cao Bin¹*, Li Tan¹, Liu Yongle¹, Guo Xin²

¹School of Energy and Mining Engineering Shandong University of Science and Technology, Qingdao 266590, China
² Zhai Zhen coal mine Xinwen Mining Group, Xintai 271204 China

Abstract. This study presents a new technology for passing through faults quickly by removing gangue in advance from driving roadways of working faces; this technology has the advantages of short moving time and good coal quality during fault passing. However, in practice, driving roadways are affected by mining pressure during the driving and mining period; thus, roadway support is difficult. This study aims to develop appropriate support for driving roadways, not only to ensure the stability of the roadway and satisfy the space requirements for the working face when it passes through the fault but also to reduce the strength of the rock pillars in the roadway. In this manner, the shearer can directly reduce the working strength during the period of fault passing. In this study, UDEC numerical simulation software is used to explore the size of rock pillars and the parameters of bolt support. In addition, the supporting effect of ordinary, high-strength prestressed, and high-strength prestressed yielding bolts is compared. On the basis of the simulation results, a reasonable control scheme of surrounding rock in advanced driving roadways is proposed. Moreover, the successful application of the technology in field practice shows the remarkable effect of this scheme on the surrounding rock control when passing faults. Thus, this study provides some references for surrounding rock control during the fault passing of the mining face.

Keywords: Coal mine, Fault, Mine pressure, Roadway support

1. Introduction

As a weak plane in rock mass media, the mechanical strength of faults is far lower than that of surrounding rock mass. Large drop faults are a common geological structure in underground coal mining. Affected by the structural and mechanical characteristics of faults, the bedding, joints, and fractures of the strata where the fault is located are developed, and the integrity of the roadway roof becomes poor. Traditional fault crossing methods have some defects. First, the working face transfer is time consuming; second, the production technology during fault crossing is complex; third, the downtime is long; lastly, the roof of the working face is difficult to control and manage during fault crossing, which causes serious damage to production equipment. Therefore, exploring a safe and efficient method of passing through large drop faults has always been an important issue in coal mining[1-4].

This study presents a method of quick drop fault passing, that is, digging out a roadway in
advance and removing gangue to facilitate mining. In other words, the rock fault is dug out in advance, and the excavated space (roadway) is supported when the fault is found in front of the working face. When the working face is pushed to the fault, passing through the fault becomes passing through an empty roadway. A schematic of the technology is shown in Fig. 1. Many problems caused by coal quality, low yield, slow advance speed, difficult roof control, and complicated production technology are solved when the working face passes through faults. However, in practice, pre-excavation roadways are influenced by two types of mining pressure—roadway excavation and working face mining. Supporting pre-excavation roadways is difficult; thus, the surrounding rock control of the pre-excavation roadway is key to achieving the technology.

Through a numerical simulation experiment, this study investigates a reasonable surrounding rock control technology of pre-excavation roadways and verifies it in field practice, thereby providing reference for coal mine production under the same conditions.

2. Project introduction
The 3304W working face of the third mining area of Zhaizhen Coal Mine is located west of a ginnery and north of the transportation roadway of the 3303E working face. The underground elevation is −219.33 m to 276.3 m, the ground elevation is +169.7 m to +171.75 m, and the average buried depth is approximately 420 m. The thickness of the coal seam in the 3304W working face is 2.6 m to 4.1 m, with an average thickness of 3.35 m; the dip of the coal seam is 5° to 22°, with an average of 13°. The main rock stratum of the working face roof is siltstone, and the floor is sandstone interbed. The geological structure of the 3304W working face is developed, and 27 faults are exposed in the mining process of the working face. Among these faults, the F27 fault has the largest drop, i.e., more than 10 m. The F27 fault cuts through the entire 3304W working face, thus considerably affecting the mining of the working face. The working face must push through the F27 fault to carry out subsequent coal mining. Therefore, the fault crossing problem of the 3304W working face becomes the main problem in the production of this face. The location of the F27 fault is shown in Fig. 2.

![Figure 1. Sketch map of the working face passing through a large fault](image1)

![Figure 2. Diagram of the location of the F27 fault](image2)

3. Dimension of the rock pillar of the pre-excavation roadway
When the working face exposes the pre-excavation roadway, the rock pillars between the pre-excavation roadways bear most of the pressure of the overlying strata. In this section, based on the comprehensive analysis of the geological structure of the F27 fault, combined with the experience of the size of the rock pillar reserved in the pre-excavation roadway when the 1401E and 3303W working faces pass through the fault. Considering that the pre-excavation roadway is approximately equivalent to the ordinary roadway, the surrounding rock mass has a certain bearing capacity and thus can play a supporting role on the overlying rock mass and form the bearing structure of the pre-excavation roadway. A reasonable rock pillar width between the pre-excavation roadways of the 3304W working face is determined by using the UDEC numerical simulation method.

3.1. Model establishment and parameter setting
The UDEC numerical simulation software used in this section can well simulate the mechanical characteristics of discontinuous rock mass under dynamic load\(^{[5,7]}\). The model is established in
accordance with the drilling histogram of the 3304W working face (as shown in Fig. 3). The length and height of the model are 240 and 115 m, respectively. The physical and mechanical properties of the surrounding rock are determined by referring to the actual mechanical properties of the rock mass in the relevant coal mining working face (see Table 1).

Table 1. Mechanical index of the coal seam in UDEC simulation

| Rock stratum    | Bulk modulus-K (GPa) | Shear modulus-G (GPa) | Density-d (kg·m⁻³) | Friction angle-f (°) | Cohesion-C (MPa) | Tensile strength-t (MPa) |
|-----------------|----------------------|-----------------------|--------------------|----------------------|------------------|-------------------------|
| main roof       | 27.1                 | 20.9                  | 2447               | 28                   | 27.5             | 7.39                    |
| Immediate roof  | 15.6                 | 11.1                  | 2408               | 29                   | 35.1             | 2.11                    |
| coal            | 11.5                 | 5.5                   | 1348               | 29                   | 1.3              | 1.49                    |
| Immediate floor | 13.7                 | 9.9                   | 2247               | 15                   | 0.6              | 1.3                     |
| Hard floor      | 25.1                 | 19.5                  | 2408               | 29                   | 27.9             | 3.09                    |

3.2 Analysis of numerical simulation results

3.2.1 Stress analysis.

The internal stress of the rock pillar with different widths is shown in Fig. 4.

![Figure 3 Numerical simulation model widths](image)

![Figure 4 Stress of the rock pillar with different](image)

Under the condition of different rock pillar widths, the distribution rule of vertical stress in the rock pillar varies, and the maximum value also differs. According to the actual situation of the working face and combined with previous experience in designing rock pillar sizes, this numerical simulation determines three rock pillar widths (i.e., 2, 3, and 4 m) of the pre-excavation roadway. Through the analysis of the stress, plastic zone, and the roof and floor displacement of the rock pillar with different widths, a reasonable rock pillar width is determined.

Considering the buried depth of the working face and the average gravity density of the strata, the in-situ rock stress around the pre-excavation roadway should be 11.3 MPa. Under the condition of 2, 3, and 4 m rock pillar widths, the maximum vertical stress in the rock pillar is 5.7, 9.7, and 13.9 MPa, respectively, which are 50.4%, 86.1%, and 122.6% of the in-situ rock stress. When the width of the rock pillar is 2 and 3 m, the rock pillar is completely in the stress-relaxed area. When the width of the rock pillar increases to 4 m, a certain range of stress rising area is observed near the center line of the rock pillar.

3.2.2 Plastic zone analysis.

The plastic zones of the rock pillar of different widths are shown in Table 2.
### Table 2 Plastic zone of the rock pillar with different widths

| Rock pillar width/m | Plastic zone range/m | Proportion of plastic zone |
|---------------------|----------------------|-----------------------------|
| 2                   | 2                    | 100%                        |
| 3                   | 2.8                  | 93%                         |
| 4                   | 2.6                  | 65%                         |

Table 3 shows that under different rock pillar widths, the range of plastic zone in the rock pillar also changes accordingly. Under the condition of 2 m rock pillar width, the entire rock pillar reaches the plastic state, the rock pillar is destroyed, and the bearing capacity is lost. Under the condition of 3 m rock pillar width, although 93% of the rock pillars reaches the plastic state, the rock pillars are not completely destroyed and have a certain bearing capacity. Under the condition of 4 m rock pillar width, 65% of the rock pillars reaches the plastic state, but the rock pillars in the center part remain in the elastic stage, and the rock pillars have good bearing capacity.

#### 3.2.3 Displacement analysis of the roof and floor of the pre-excavation roadway

The displacement of the roof and floor of the pre-excavation roadway is shown in Fig. 5.

![Figure 5 Displacement of the roof and floor under different rock pillar widths](image)

As shown in Fig. 5, the displacement of the roof and floor of the roadway changes accordingly with the change of the rock pillar width. Under the condition of 2 m pillar width, the displacement of the roof and floor of the pre-excavation roadway is 454 mm. Under the condition of 3 m pillar width, the displacement of the roof and floor of the pre-excavation roadway is 263 mm, which is 42.1% less than that of the 2 m pillar width. Under the condition of 4 m pillar width, the displacement of the roof and floor of the pre-excavation roadway is 245 mm, which is 46% less than that of the 2 m pillar width.

By analyzing the stress, plastic zone, and displacement of the roof and floor under different rock pillar widths, the following conclusions can be drawn: Under the condition of 3 m rock pillar width, the maximum vertical stress in the rock pillar is 86.1% of the in-situ rock stress, and the rock pillar is completely in the stress-relaxed area. Although the rock pillar has a certain bearing capacity, it is almost in the plastic state, making it convenient for the shearer to break the rock pillar during mining. The displacement of the roof and floor of the pre-excavation roadway is 263 mm under the condition of 3 m rock pillar width. Compared with the 4 m rock pillar width, the displacement of the roof and floor of the pre-excavation roadway increases but has minimal effect. Therefore, the reasonable rock pillar width of the pre-excavation roadway is 3 m.

#### 4. Support mode and parameters of the pre-excavation roadway

Before the working face pushes through the pre-excavation roadway, the pre-excavation roadway is influenced by two stages of mine pressure, namely, the excavation and mining influence stages. The support strength of the pre-excavation roadway must meet the requirements of the stability of the pre-excavation roadway under the influence of mine pressure to ensure that the working face passes through the pre-excavation roadway smoothly. Bolt support is a kind of active support method, which
has good surrounding rock control effect, fast construction speed, simple construction, and wide application. This section refers to the support design experience of other coal production enterprises in pre-excavation roadways when passing through the fault and combines such experience with the production geological conditions of the 3304W working face of the Zhai Zhen Coal Mine. Bolt support is then taken as the support method of the pre-excavation roadway when the 3304W working face passes through the fault, and the design parameters of bolt support are optimized.

4.1. Design of the general bolt support scheme

The orthogonal test method is a mathematical statistical method that uses an “orthogonal table” to arrange and analyze test data. The advantages of this method are less test times, simple method, convenience, and high efficiency[8,9].

According to the existing design experience in bolt supporting parameters, the following are the three main supporting parameters: bolt length, bolt diameter, and bolt spacing. Taking these three parameters as factors, each factor selects three levels, carries on the orthogonal experiment design, and simulates the roadway surrounding rock deformation situation under different parameter combinations. Bolt parameters slightly differ in terms of roof bolt support and two sidewalls and thus should be studied accordingly. The factors and horizontal changes of the two sets of experiments are shown in Table 3, and the specific test scheme is shown in Table 4.

| Table 3 Factors and levels of the orthogonal numerical simulation experiment |
|-----------------------------------------------|
| Level | Roof | Sidewall |
|       | Diameter/mm | Length/m | Spacing/mm | Diameter/ | Length/ | Spacing/m |
| 1     | 18 | 2.0 | 700 | 16 | 1.6 | 700 |
| 2     | 20 | 2.2 | 800 | 18 | 1.8 | 800 |
| 3     | 22 | 2.4 | 900 | 20 | 2.0 | 900 |

| Table 4 Scheme for the orthogonal experiment |
|-----------------------------------------------|
| Test number | Roof | Sidewall |
|             | Diameter/m | Length/ | Spacing/m | Diameter/ | Length/ | Spacing/m |
| 1            | 18 | 2.0 | 700 | 16 | 1.6 | 700 |
| 2            | 20 | 2.2 | 800 | 18 | 1.8 | 800 |
| 3            | 22 | 2.4 | 900 | 20 | 2.0 | 900 |
| 4            | 18 | 2.0 | 800 | 16 | 1.6 | 800 |
| 5            | 20 | 2.2 | 900 | 18 | 1.8 | 900 |
| 6            | 22 | 2.4 | 700 | 20 | 2.0 | 700 |
| 7            | 18 | 2.0 | 900 | 16 | 1.6 | 900 |
| 8            | 20 | 2.2 | 700 | 18 | 1.8 | 700 |
| 9            | 22 | 2.4 | 800 | 20 | 2.0 | 800 |

Table 3 shows that nine numerical calculation models are respectively established for the roof and sidewalls. The three support parameters of each model are determined in accordance with the corresponding test numbers in Table 4. The other conditions are identical. The test results of each parameter are shown in Fig. 6.

Fig. 6 is composed of three parts, which show the influence of different bolt parameters on the deformation of surrounding rock. At the roof and two sidewalls of the roadway, the displacement of surrounding rock with different row spacings, diameters, and lengths of bolt is shown in Figures (a), (b), and (c), respectively.

With the increase in bolt spacing, the cumulative displacement of the roof and floor increases gradually. In the roadway roof, when the spacing between bolts changes from 700 mm to 800 mm, the increase in the accumulated displacement of the roof and floor is small (i.e., 22 mm). When the spacing between bolts changes from 800 mm to 900 mm, the increase in the accumulated displacement...
of the roof and floor is large (i.e., 196 mm). In the two sidewalls of the roadway, when the bolt spacing changes from 700 mm to 800 mm, the cumulative displacement of the roof and floor increases by only 20 mm. When the bolt spacing changes from 800 mm to 900 mm, the cumulative displacement of the roof and floor increases by 120 mm.

With the increase in bolt diameter, the accumulated displacement of the roof and floor decreases gradually. In the roadway roof, when the bolt diameter changes from 20 mm to 22 mm, the accumulated displacement of the roof and floor decreases considerably by 200 mm. When the bolt diameter changes from 22 mm to 24 mm, the accumulated displacement of the roof and floor slightly changes by 30 mm. Therefore, the reasonable diameter of the roof bolt of the pre-extraction roadway should be 20 mm. In the two sidewalls of the roadway, when the bolt diameter changes from 16 mm to 18 mm, the cumulative displacement of the roof and floor decreases considerably by 80 mm. When the bolt diameter changes from 18 mm to 20 mm, the cumulative displacement of the roof and floor slightly decreases by 30 mm.

With the increase in bolt length, the cumulative displacement of the roof and floor decreases gradually. In the roadway roof, when the bolt length changes from 2.0 m to 2.2 m, the cumulative displacement of the roof and floor decreases considerably to 170 mm. When the bolt length changes from 2.2 m to 2.4 m, the cumulative displacement of the roof and floor slightly changes by only 20 mm. Therefore, the reasonable length of the roof bolt of the pre-extraction roadway should be 2.2 m. In the two sidewalls of the roadway, when the bolt length changes from 1.6 m to 1.8 m, the accumulated displacement of the roof and floor decreases remarkably by 100 mm. When the bolt length changes from 1.8 m to 2.0 m, the accumulated displacement of the roof and floor slightly decreases by 20 mm.

Based on the above numerical simulation results, the following are the reasonable supporting parameters of the pre-extraction roadway: the spacing between the bolts is 800 mm, the diameter of the roof bolt is 20 mm, the length of the roof bolt is 2.2 m, the diameter of the two sidewall bolts is 18 mm, and the length of the two sidewall bolts is 1.8 m.

4.2. Selection of different types of bolts

The pre-extraction roadway is located in the fault; thus, the roof and two sidewalls of the pre-extraction roadway are relatively broken. When using only the surrounding rock supported by ordinary bolts, the deformation of the surrounding rock of the roadway becomes difficult to control, and the separation layer and the slice of the roadway roof become more serious. Therefore, a more suitable type of bolt for the pre-extraction roadway must be investigated. [10-14]

The support parameters of ordinary bolts (rod strength: Q235, yield strength: 235 MPa, tensile strength: 380 MPa) are applied to high-strength prestressed bolts (rod strength: Q500, yield strength: 500 MPa, tensile strength: 660 MPa, prestress: 50 MPa) and high-strength prestressed yielding bolts (rod strength: Q500, yield strength: 500 MPa, tensile strength: 660 MPa, prestress: 50 MPa; when the stress of the bolt reaches 80% of its own yield strength, the compression structure starts). The deformation of surrounding rock is analyzed through a numerical simulation experiment under the same support parameters. The results are shown in Fig. 7.

![Figure 6 Displacement of the roof and floor with different support parameters](image-url)
As shown in Fig. 7, under the same support parameters, the displacement of the roof and floor of the pre-excavation roadway is 230 mm when the ordinary bolt support is used. When the high-strength prestressed bolt support is used, the displacement of the roof and floor of the pre-excavation roadway is 155 mm, which is 26.2% lower than that of the common bolt support; however, the displacement of the roof and floor of the pre-excavation roadway is only 78 mm, which is 66.1% lower than that of the common bolt support. Therefore, under the same support parameters, the high-strength prestressed yielding bolt has the best support effect on the surrounding rock of the pre-excavation roadway.

5. Actual measurement of the surrounding rock control effect of the pre-excavation roadway
To verify the supporting effect of the high-strength prestressed yielding bolt in pre-excavation roadways, surface displacement observation equipment is installed on the surface of the pre-excavation roadway. The time period of monitoring data selection starts when the working face is 50 m away from the pre-excavation roadway and ends when the working face is pushed to the pre-excavation roadway. The monitoring results are shown in Fig. 8.

![Figure 7: Comparison of the support effect of different bolt types](image1)
![Figure 8: Monitoring results of the displacement of surrounding rock](image2)

**Figure 7** Comparison of the support effect of different bolt types  
**Figure 8** Monitoring results of the displacement of surrounding rock

The monitoring results in Fig. 8 reveal that when the working face is pushed to the position of the pre-excavation roadway, the displacement of the roof and floor of the pre-excavation roadway is 218 mm, and the displacement of the two sidewalls of the roadway is 96 mm. The surrounding rock control effect of the pre-excavation roadway is good, and the space of the pre-excavation roadway meets the requirements of the working face in terms of mining height.

6. Conclusion
This study investigates the width setting of rock pillars and the selection of support parameters in the pre-excavation roadway by using numerical simulation software. Based on existing research results\(^1\)\(^7\) and the field engineering practice of the Zhaizhen Coal Mine, this study concludes that when the rock pillar of the pre-excavation roadway is 3 m, most of the rock pillars are in the plastic deformation area under the action of the advanced abutment pressure of the working face; moreover, the shearer can cut the rock pillar directly. However, the residual bearing capacity of the rock pillar can support the overlying rock mass. Combined with high-strength prestressed yielding bolt support, the displacement of the roadway roof and floor is 218 mm, and the displacement of the two sides is 96 mm. The displacement of surrounding rock meets the requirements of engineering specifications. The research results are reasonable and consistent with actual situations.

In this study, UDEC numerical simulation software is used to determine the reasonable width of rock pillars from the stress situation of rock pillars, the situation of plastic zones, and the displacement of the roof and floor of the pre-excavation roadway. The orthogonal test principle is used to design the test scheme, and the influence of different bolt row spacings, bolt diameters, and bolt lengths on the surrounding rock deformation of the roadway is analyzed. In addition, a comprehensive analysis of ordinary anchor, high-strength prestressed anchor, high-strength prestressed yielding anchor bolt three bolt support methods, from the rod strength, active support, overload load relief three aspects, choose
a reasonable anchor form. Lastly, the numerical simulation results are applied in field practice to verify the rationality of the research results. The research method in this study can provide some reference for the control of surrounding rock in roadways under complex conditions.

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