Hindawi
Geofluids
Volume 2020, Article ID 8871925, 16 pages
https://doi.org/10.1155/2020/8871925

Research Article

Failure Mechanism and Stability Control of Surrounding Rock of Docking Roadway under Multiple Dynamic Pressures in Extrathick Coal Seam

Xiangye Wu,[1] Shuai Wang,[1] Chen Tian,[2] Changxing Ji,[3] and Jingya Wang[1]

[1]*Institute of Mining Engineering, Inner Mongolia University of Science and Technology, Baotou 014010, China*
[2]*Bulianta Coal Mine, Shenhua Shendong Coal Group Corporation Limited, Ordos 017209, China*
[3]*Inner Mongolia Fucheng Mining Co., Ltd., Ordos 016217, China*

Correspondence should be addressed to Xiangye Wu; 2018924@imust.edu.cn

Received 23 September 2020; Revised 6 October 2020; Accepted 6 October 2020; Published 9 November 2020

Academic Editor: Yanlin Zhao

In view of multiseam mining under goaf, the surrounding rock control problem of lower coal roadway will be affected by concentrated coal pillar left in upper coal seam goaf and dynamic pressure superposition of working face in this coal seam. Under the geological environment of No. 16 extrathick coal seam in the Laoshidan coal mine and taking the working face 031604 as the research background, the reasonable docking position selection of the withdrawal roadway and the docking roadway in the middle mining period and the surrounding rock stability control problems of the withdrawal roadway and the docking roadway during the final mining period were studied by using the methods of field theoretical analysis, numerical simulation, and field measurement. The mechanical mechanism of the nonuniform failure of the retreating roadway and the docking roadway during the final mining period is shown, and the control method of the surrounding rock stability of the roadway is put forward and applied. The results show that (1) through the analysis of the superimposed stress under the concentrated coal pillar and the coal seam in advance, the specific butt joint position is arranged at 860 m away from the open-off cut, which is 10 m away from the goaf of No. 12 coal seam. (2) With the working face 031604 advancing through the process, the deviatoric stress value of the withdrawal roadway gradually increases, the maximum principal stress of the two sides of the roadway deflects clockwise from the vertical direction to the horizontal direction, its angle also gradually increases, and the shape of the plastic zone gradually expands from symmetry to asymmetry. (3) It is revealed that the peak value of deviatoric stress on both sides of the docking position of docking roadway increases gradually under the influence of mining and deflects anticlockwise to the vertical direction with the principal stress angle. The joint action of both is the mechanical mechanism that causes the plastic zone to expand in an asymmetric shape. (4) The coordinated control scheme of support (anchor bolt and anchor cable)—modified (grouting)—is adopted for the withdrawal roadway, and the coordinated control scheme of support (anchor bolt and anchor cable)—changing the cross-section shape of the roadway—is adopted for the docking roadway. The purpose of the smooth connection of working face and rapid and safe withdrawal of equipment is achieved on site.

1. Introduction

Coal is the conventional energy with the largest reserves and widest distribution in the world. Coal is China's basic energy. China's coal resource reserves are at the forefront of the world. Most coal fields are distributed with most coal seams. Field practice and research show that [1, 2], when mining multiple coal seams, deep coal roadway is often affected by the coal pillars left by the overlying coal seam and the dynamic pressure of the working face of the coal seam [3], especially in the case of extrathick coal seam, which makes the maintenance of roadway and roof management of working face more complex and difficult at the lower coal seam, and which are easy to cause roof fall, spalling, and other accidents; this situation seriously threatens the safety production of the mine.

Many experts and scholars have carried out in-depth research on the stability control of surrounding rock affected

by the dynamic pressure of roadway and have achieved a lot of research results. In China, the single coal seam with more than 8 m is defined as extrathick coal seam [4]. Kang et al. [5] analyzed the mechanism of large deformation of roadway surrounding rock under the superimposition of high crustal stress and strong mining stress of working face and put forward the coordinated control concept of support modification pressure relief for kilometer deep mine, soft rock, and strong mining roadway. Hao et al. [6] discussed the instability mechanism and main influencing factors of the mining roadway in the near distance coal seam under the goaf and considered that the instability of the mining roadway in the lower coal seam was mainly affected by the roadway layout, dynamic pressure of mining, roadway support scheme, etc. Su et al. [7] studied the stress distribution regularity of lower coal seam, deformation failure mechanism of the roadway group, and surrounding rock stability control and put forward the optimized support scheme of the arch section for a roadway in a high-stress area under coal pillar. Lv [8] analyzed the dynamic mechanical change characteristics of the remained coal pillars in the working face and deeply analyzed the mechanical action mechanism of the preexcavation and withdrawal channel. Lv et al. [9] studied the deformation and failure characteristics of surrounding rock of roadway and the mechanism of malignant expansion and failure of a plastic zone of surrounding rock of a roadway and put forward the combined support technology of bolt mesh cable and stress control technology.

To date, a great number of theories have been proposed explaining the failure mechanism of roadway surrounding rock, mainly including classical circular plastic zone theory, surrounding rock loose circle theory, natural caving arch theory, axial deformation theory, surrounding rock partition cracking theory, and "butterfly-shaped plastic zone" theory of roadway surrounding rock. In recent years, many scholars have studied the failure mechanism of surrounding rock of dynamic pressure roadway [10–13]; Wu et al. [14] analyzed the plastic zone and stress space-time evolution regularity of surrounding rock of repeated mining roadway in working face with double roadway layouts and proposed the reinforcement support method by stages. Jiang et al. [15] compared the stress distribution, failure expansion, and displacement evolution of roadway in the whole service life under different coal pillar width conditions and studied the influence of coal pillar width on ground stability. Zhang et al. [16–18] studied the stability of the roadway roof after goaf mining backfill and provided the theoretical basis for roadway roof stability control. Wang et al. [19] and others studied the mechanism of asymmetric large crushing and expansion deformation of surrounding rock of coal roadway due to excavation unloading.

In this paper, Laoshidan coal mine as the engineering background, in view of the fact that some hydraulic supports are withdrawn from the working face during the process of mining across the mountain in the lower coal seam, the docking roadway and part of the withdrawal channel are affected by multiple stress concentration, such as the left concentrated coal pillar of the overlying coal seam and the advance mining of the working face. Therefore, through the influence range and degree of concentrated coal pillar on the stress of lower coal seam when the working face is not mined, the reasonable docking position is judged, and then, the stress distribution and dynamic evolution law of plastic zone of surrounding rock of docking roadway and withdrawal channel are analyzed, the mechanical mechanism of nonuniform failure of roadway surrounding rock is revealed, the supporting methods and measures are put forward, and industrial practice is carried out. The research results have guiding significance to solve the surrounding rock stability control of mining roadway under such conditions.

2. Engineering Background

The No. 16 coal seam of the Laoshidan coal mine of Wuhai energy is the main mining coal seam. The average total thickness of the coal seam is 8.8 m, which belongs to a stable extra-thick coal seam. The dip angle of the coal seam is 1°-4°, and it is near horizontal coal seam with a buried depth of about 400 m. It is 76 m and 50 m away from the upper No. 9 coal seam and No. 12 coal seam, respectively, and all of them have been mined. There are 110 m and 130 m wide downhill protective coal pillars left in No. 9 and No. 12 coal seams, respectively. The working face 031604 is 260 m long. The working face is arranged along the strike. The strike longwall retreating the fully mechanized top coal caving mining method is adopted to mine the full height at one time, and all caving method is used to deal with the goaf. One side of the return airway of the working face is close to the goaf of working faces 031605 and 031601, and the spatial position relationship is shown in Figure 1. In order to reduce the loss of coal resources, the method of cross downhill mining is adopted to avoid connecting with the goaf, and 30 m section is reserved to protect the coal pillar. The working face should be shortened 47 m before mining to goaf 031601, and 26 extra supports should be withdrawn. Therefore, it is necessary to excavate the docking roadway and withdrawal roadway along the floor in advance. The design section of the docking roadway is rectangular, which is 4100 mm wide and 3500 mm high, and the design section of the withdrawal roadway is rectangular, which is 3500 mm wide and 3500 mm high.

Before the excavation of docking roadway 031604, the borehole peeping method was adopted on site. Three boreholes with a depth of 15 m and a diameter of 32 mm were arranged in front of the working face of haulage roadway 031601. The roof rock structure and the failure state of surrounding rock were analyzed. Combined with geological data, the comprehensive column is shown in Figure 2. Comprehensive analysis of borehole peeping results shows that solid coal is within the range of 4-5.3 m from the roadway roof, and the surrounding rock within 0-2 m from the roof is seriously broken, as shown in Figures 2(a) and 2(b); within the range of 5-6.9 m from the roadway roof, it is composed of solid coal, sand shale, and carbon shale. One or more horizontal, vertical, and inclined fractures are developed within the range of 4.2-6.9 m from the roof, and the fracture width is about 0-15 mm, as shown in Figures 2(c) and 2(d); within the range of 7-15 m from the roadway roof, it is fine sandstone with complete rock mass and relatively flat hole wall, as shown in Figure 2(e). There is a small amount of coal line at 14.0 m-14.3 m, and less fracture development, as shown in Figure 2(f).

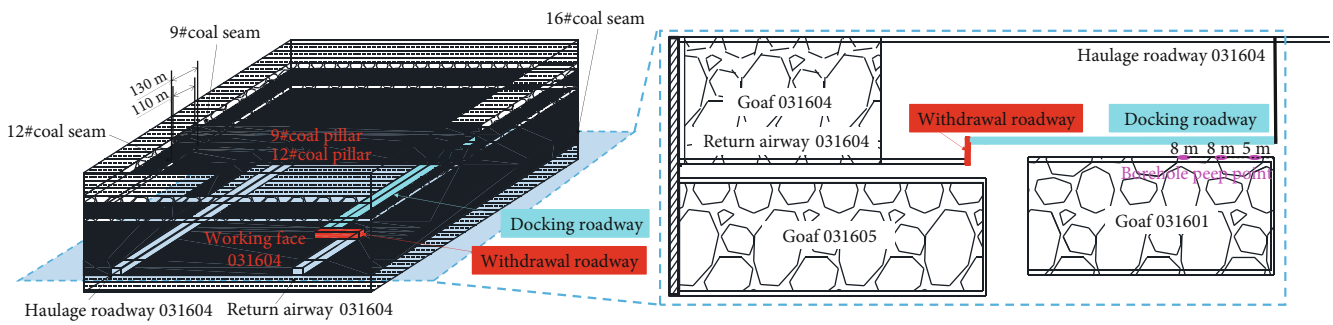
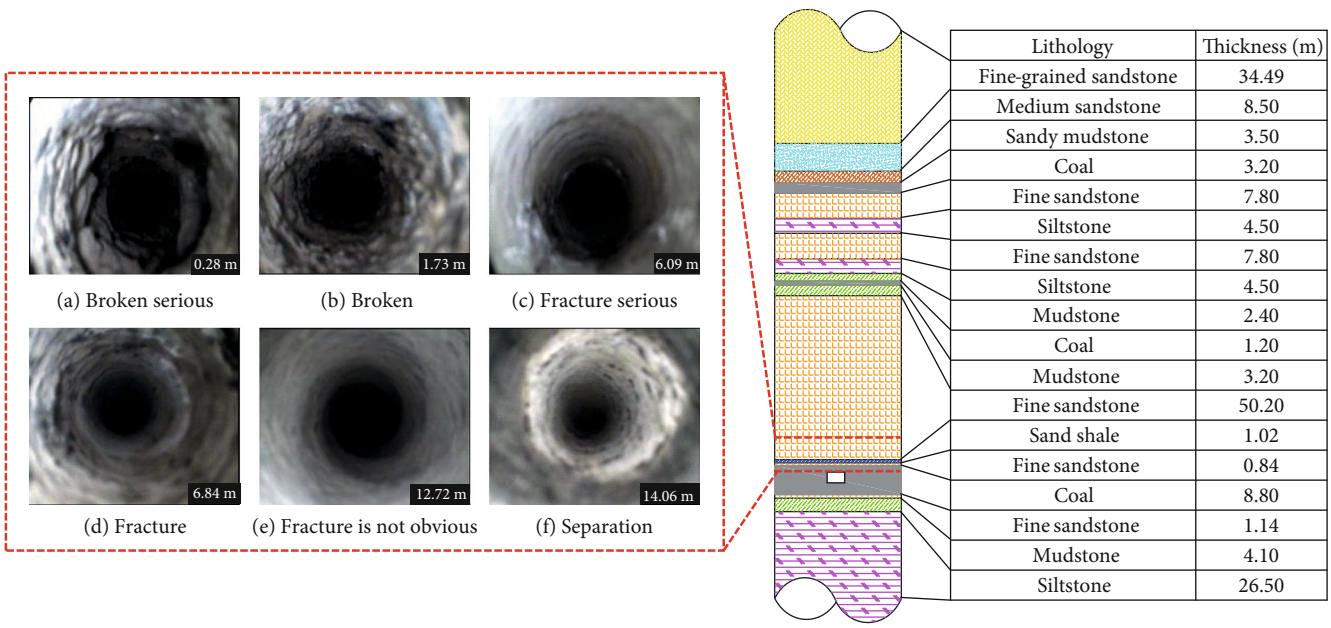


Figure 1: Spatial position relationship of coal seam and roadway.



Lithology	Thickness (m)
Fine-grained sandstone	34.49
Medium sandstone	8.50
Sandy mudstone	3.50
Coal	3.20
Fine sandstone	7.80
Siltstone	4.50
Fine sandstone	7.80
Siltstone	4.50
Mudstone	2.40
Coal	1.20
Mudstone	3.20
Fine sandstone	50.20
Sand shale	1.02
Fine sandstone	0.84
Coal	8.80
Fine sandstone	1.14
Mudstone	4.10
Siltstone	26.50

Figure 2: Observation of surrounding rock state at different depths by borehole peep.

During the mining process of the working face 031604, the docking roadway and withdrawal roadway are affected by the concentrated coal pillars of overlying coal seam and the advance mining of the working face of this coal seam, which will inevitably lead to the change of the surrounding rock stress environment of the roadway and the destruction of the surrounding rock of the roadway, which will bring hidden dangers to the safety production during the mining period. In order to ensure the smooth docking of the working face and the safe and rapid withdrawal of the fully mechanized mining equipment, it is a key problem to arrange the connecting roadway and withdrawal roadway in advance and predict the stress environment and damage degree of surrounding rock caused by mining disturbance.

3. Study on the Position of the Docking Roadway

3.1. Establishment of the Numerical Model. Based on the geological conditions of each coal seam and working face in the Laoshidan coal mine, the FLAC^{3D} three-dimensional numerical model is established, and the Mohr-Coulomb constitutive relation is adopted. The model size is 3000 m × 900 m × 230 m (length × width × height), as shown in Figure 3. The thickness of No. 9 coal seam is 3.2 m, the thickness of No. 12 coal seam is 1.2 m, the interval between seams is 26 m, the thickness of No. 16 coal seam is 8.8 m, and the distance between No. 12 coal seams is 50 m. The overburden layer of the model is 260 m, so a vertical load of 6.5 MPa is applied to the upper part of the model. The lateral pressure coefficient of the model is taken as 1.5, and the surrounding and bottom of the model are fixed constraints. The physical and mechanical parameters of rock and coal are shown in Table 1.

The double yield constitutive model can be applied to fill the gob zone. The height of the caving zone determines the stress and deformation of the goaf and affects the surrounding rock of the roadway. Therefore, it is very important to determine the height of the caving zone in the goaf. Bai et al. [20] obtained the statistical regression formula (1) for calculating the height of the caving zone through statistical

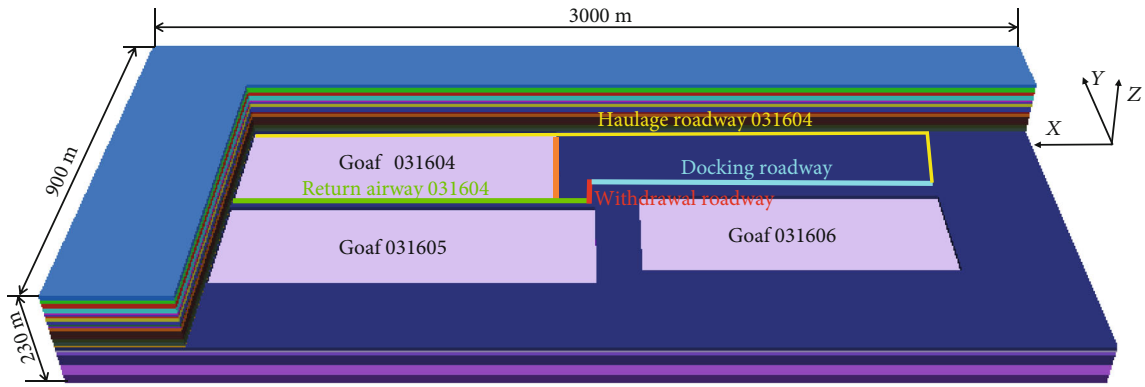


Figure 3: Numerical calculation model.

Table 1: Rock mechanics parameters of the computation model.

Lithology	Bulk modulus (GPa)	Shear modulus of elasticity (GPa)	Density (kg/m^{-3})	Cohesion (MPa)	The angle of internal friction (°)	Tensile strength (MPa)
Fine-grained sandstone	5.4	3.1	3270	9.0	34	2.67
Sand shale	2.5	2.0	3180	3.5	28	1.51
Coal shale	2.5	2.5	3180	4.6	32	1.80
Coal	2.2	1.5	2800	2.5	26	1.22
Marl	4.3	2.0	3270	4.0	30	1.40
Siltstone	4.5	3.2	3570	12.0	32	2.32

regression analysis based on different geological conditions of a large number of mines in China and the United States.

$$H = \frac{100h}{c_1 h + c_2}, \tag{1}$$

where H is the mining height, m; C_1 and C_2 are the parameters related to roof lithology [21–25], as shown in Table 2. According to formula (1) and Table 2, the average height of the caving zone in the goaf is determined by roof rock strength and coal seam mining thickness. According to the mechanical parameters of coal and rock mass in the Laoshidan coal mine, the height of the goaf caving zone in No. 9, No. 12, and No. 16 coal seams is calculated as 9 m, 4.9 m, and 14.6 m, respectively. Through the trial and error inversion method, the mechanical parameters of the double yield model of the caving zone in the goaf of each coal seam are obtained, as shown in Table 3. According to the advancing speed of the working face, each step excavation of the working face is simulated to be 10 m, and the caving zone in the goaf is filled and balanced.

3.2. Determination of Reasonable Docking Position. In order to ensure the smooth docking of the roadway and obtain the reasonable docking position, the staggered distance between the roadway and the left concentrated coal pillar in the No. 12 coal seam is determined, and the vertical stress distribution of the No. 16 coal seam under the action of the coal pillar is analyzed. The simulation results are shown in Figure 4.

As can be seen from the figure, No. 16 coal seam is affected by the superposition of concentrated coal pillars left in No. 9 and No. 12 coal seams, and the stress contour is approximately symmetrical relative to the middle position of the coal pillar. The vertical stress concentration reaches 14 MPa, which is 1.4 times of the original rock stress. The farther away from the coal pillar, the smaller the vertical stress is, and the stress environment is relatively stable. The docking joint position should be selected in the relative No. 12 coal seam goaf, while taking into account the principle that the closer to the coal pillar, the smaller the loss of coal resources. Therefore, the docking joint position is selected as the left side of the coal pillar and the side of the open hole. In this area, the minimum stress value of No. 16 coal seam is 11-12 MPa within the range of 800 m to 880 m away from the open-off cut, so as to determine the layout of docking roadway end and withdrawal roadway within 800 m to 880 m from the open-off cut.

In order to more accurately determine the position of roadway docking joint, fully consider the stress distribution of the coal seam working face affected by the superposition of advanced mining, and simulate the vertical stress distribution of surrounding rock of docking roadway during the advancing process of 031604 working face, as shown in Figure 5.

It can be seen from the figure that the vertical stress at the roadway position basically presents a single peak curve distribution under different mining distances. The working face

Table 2: Coefficients for the height of the caving zone.

Direct roof lithology	Compressive strength (MPa)	C_1	C_2
Hard	>40	2.1	16
Relatively hard	20~40	4.7	19
Soft	<20	6.2	32

Table 3: Rock mechanics parameters of the coal seam.

Coal seam	Bulk modulus (GPa)	Shear modulus of elasticity (GPa)	Density (kg/m^{-3})	The angle of internal friction (°)	Dilatancy angle (°)
9#	6.86	5.53	1800	10	4
12#	6.86	5.53	1800	11	6
16#	6.86	5.53	1800	9	3

position stress is 0, the vertical stress reaches the maximum within 10 m before the working face and then gradually decreases, and the stress concentration coefficient is about 3.2-3.5 times of the original rock stress. The larger the peak value is, the smaller the vertical stress distribution is near the coal pillar. Therefore, 860 m away from the open-off cut of the working face 031604 is the docking position of docking roadway, and the docking roadway and withdrawal channel are arranged in advance.

4. Evolution Regularity of the Plastic Zone of Withdrawal Passage and Docking Roadway during the Final Mining

The withdrawal roadway and the docking roadway are vertically connected and arranged at 860 m in front of the working face 031604, which is bound to be affected by the overlying mining in advance of the working face, when the withdrawal roadway enters the influence range of mining, that is, during the final mining period [26], in order to ensure the safe and smooth withdrawal of the fully mechanized mining equipment and safe mining. Therefore, it is necessary to study the evolution regularity of stress and plastic zone of surrounding rock of docking roadway and withdrawal roadway during the end mining period and then put forward the control scheme of surrounding rock stability to ensure the stability of surrounding rock during the end mining period.

4.1. Study on the Mechanical Mechanism of the Plastic Zone of Nonequal Pressure Circular Roadway in Surrounding Rock. Due to the influence of the loading and unloading effect caused by the advance coal mining face, the surrounding rock of the mining roadway is generally in a nonuniform pressure stress environment, and the surrounding rock of the roadway will produce irregular damage. Past theories have basically shown that the failure of surrounding rock is caused by the formation and development of the plastic zone of the surrounding rock. The malignant expansion of plastic zone is the direct cause of large deformation, loose failure, and support failure of surrounding rock, and the formation of the plastic zone is directly related to surrounding rock stress. In order to obtain the relationship between the stress and the plastic zone, the recessive equation of the plastic zone boundary of the surrounding rock of the circular roadway in the nonuniform pressure stress field was derived in reference [27] based on the Mohr-Coulomb failure criterion, as shown in

$$\begin{aligned}
&9(1-\eta)^2\left(\frac{a}{r}\right)^8+\left[-12(1-\eta)^2+6\left(1-\eta^2\right)\cos 2\theta\right]\left(\frac{a}{r}\right)^6\\
&\quad+\left[10(1-\eta)^2\cos^2 2\theta-4(1-\eta)^2\sin^2\varphi\cos^2 2\theta\right.\\
&\quad\left.-2(1-\eta)^2\sin^2 2\theta-4\left(1-\eta^2\right)\cos 2\theta+(1+\eta)^2\right]\left(\frac{a}{r}\right)^4\\
&\quad+\left[-4(1-\eta)^2\cos 4\theta+2\left(1-\eta^2\right)\cos 2\theta\right.\\
&\quad\left.-4\left(1-\eta^2\right)\sin^2\varphi\cos 2\theta-\frac{4C(1-\eta)\sin 2\varphi\cos 2\theta}{\sigma_3}\right]\left(\frac{a}{r}\right)^2\\
&\quad+\left[(1-\eta)^2-\sin^2\varphi\left(1+\eta+\frac{2C\cos\varphi}{\sigma_3\sin\varphi}\right)^2\right]=0.
\end{aligned}\tag{2}$$

In the equation, a is the roadway radius, θ is the polar coordinates of any point on the boundary of the plastic zone, r is the boundary of the radial plastic zone, α is the angle between the maximum principal stress and the vertical direction (positive clockwise), C is the rock cohesion, φ is the internal friction angle of the rock, σ_3 is the minimum principal stress, and η is the ratio of the maximum principal stress to the minimum principal stress of σ_1/σ_3.

Elastoplastic mechanics thinks that the stress state at a certain point can be divided into two parts: one is the spherical stress tensor, which mainly produces the change of volume; the other is the deviatoric stress tensor. In the triaxial axisymmetric space, the deviatoric stress tensor is the main reason for the shape change of the plastic zone of the surrounding rock of the roadway. The deviatoric stress is an indicator, which is used to analyze the relationship between the stress environment and the plastic zone of the surrounding rock. In order to better analyze the relationship between the radius of the plastic zone and the maximum and minimum principal stress under mining stress conditions, the parameters of the surrounding rock are fixed ($\gamma = 25$ kn/m^3, $r = 3$ m, $C = 3$ MPa, $\varphi = 25°$) and the surface is generated by changing two variables of σ_1 and σ_3 by using formula (2) and origin drawing software, as shown in Figure 6.

The curve of the relationship between the radius of the plastic zone and the principal stress shows a semi-U-shaped distribution, and the surface rises sharply from the middle to the wing end. When the minimum principal stress is fixed, the deviator stress increases with the increase of the maximum principal stress, and the radius of the plastic zone increases sharply with the increase of the deviator stress. The plastic zone showed heterogeneous malignant expansion.

In the process of mining, when the stress of surrounding rock changes, the direction of principal stress also changes. Using formula (2) to draw the shape diagram of the plastic

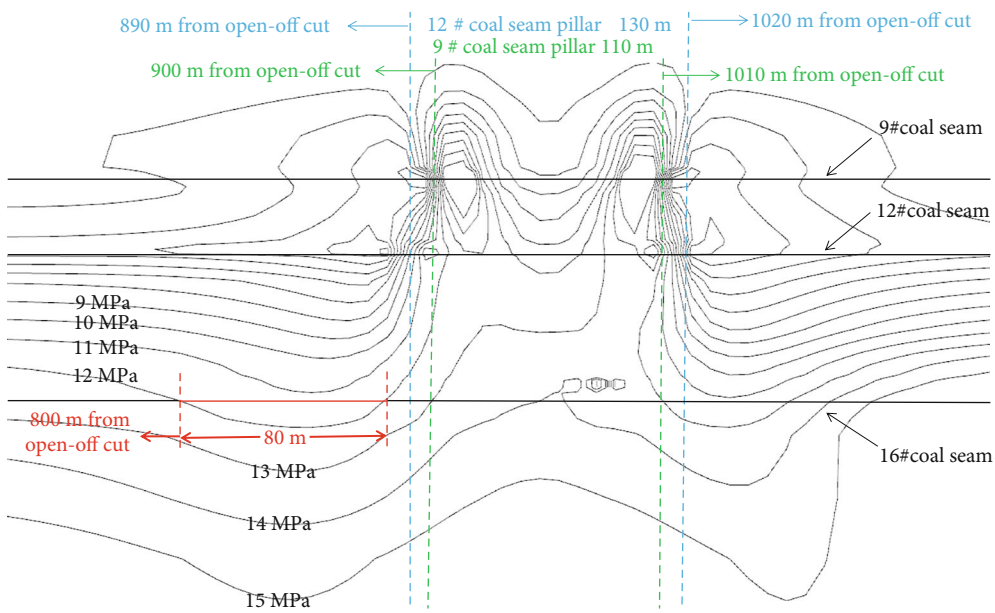


Figure 4: Vertical stress distribution curve of No. 16 coal seam under the action of the coal pillar.

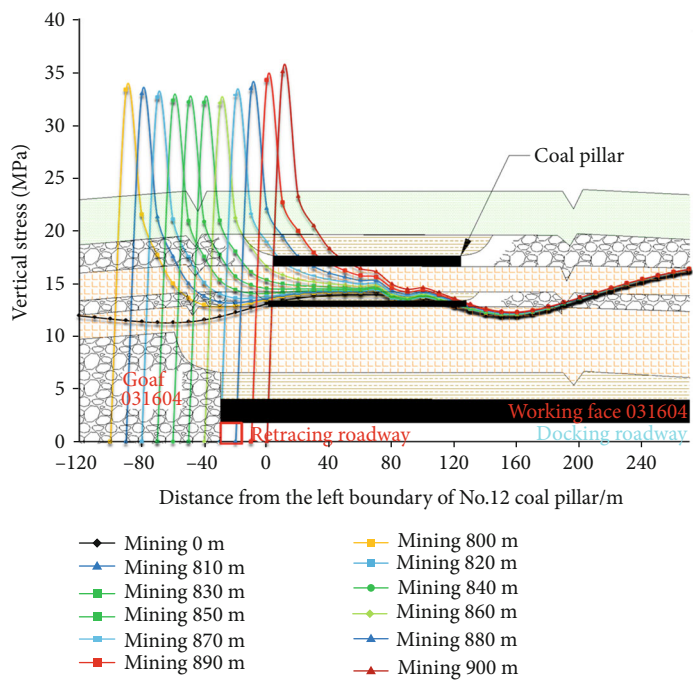
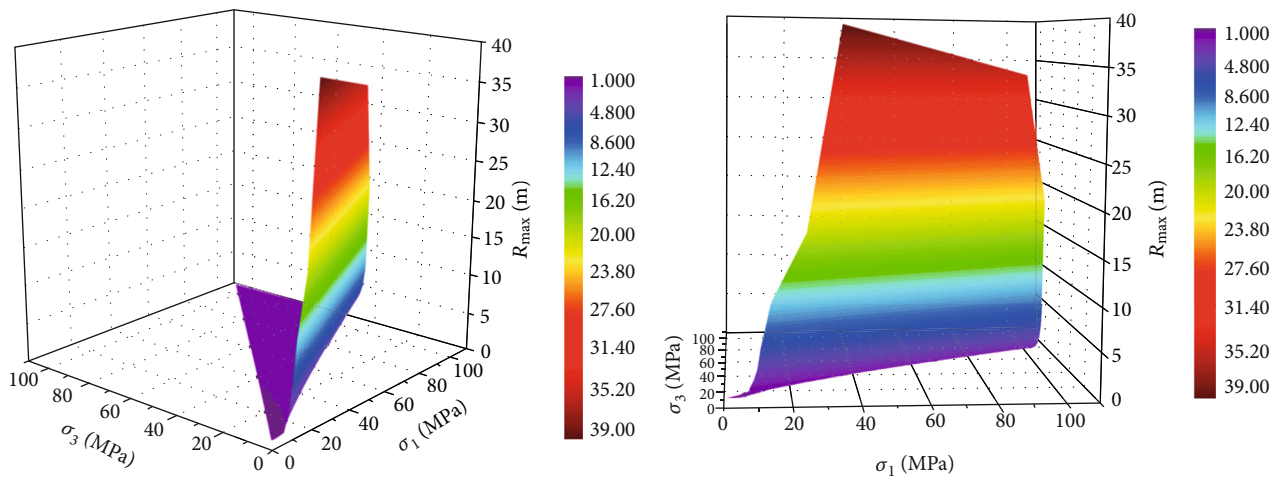


Figure 5: Distribution curve of vertical stress of surrounding rock mass in front of the working face under different mining distance.

zone of the surrounding rock, and verify it by numerical simulation. α is the angle between the maximum principal stress and the vertical direction, and the needle is positive in time, as shown in Figure 7. The results show that when the principal stress direction of the surrounding rock deflects, the distribution of the plastic zone of the surrounding rock will change accordingly, which is an important mechanical mechanism of nonuniform failure of the roadway.

4.2. Stress Analysis of Withdrawal Channel during the Final Mining. In the process of continuing to advance 50 m away from the withdrawal roadway of 031604 working face, three stress monitoring lines with a spacing of 19 m are evenly arranged in the vertical withdrawal roadway, and the monitoring line 1 is 6 m away from the docking roadway, as shown in Figure 8. The change of the deviatoric stress of coal pillars on both sides of the withdrawal roadway during the final mining period was monitored, and the generated curve is shown in Figure 9, so as to analyze the evolution law of the superimposed stress of the withdrawal roadway under advanced mining stress.

It can be seen from the figure that the distribution regularity of deviatoric stress at three monitoring lines is basically



(a) Diagonals (b) Side view

Figure 6: Right surface map of the relationship between the plastic zone radius and the principal stress.

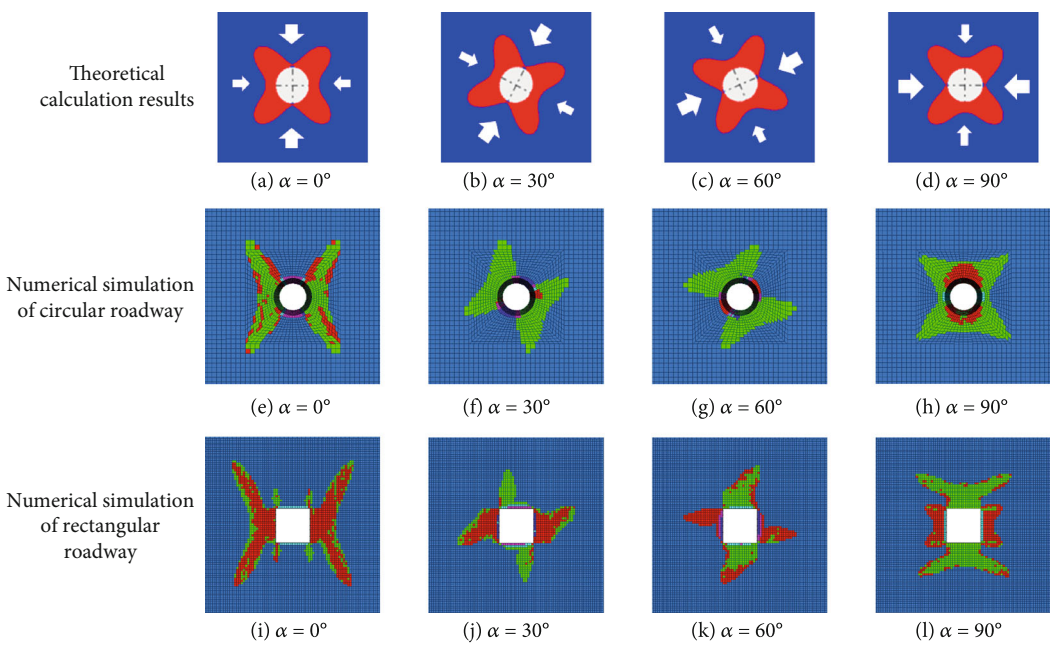


Figure 7: Principal stress deflection of surrounding rock in the circular roadway at different angles of the butterfly plastic zone distribution map.

the same, and deviatoric stress on both sides of the withdrawal roadway of line 1 is slightly larger than that of the other two measuring lines. At the side of the remained coal pillar, when the working face is 50 m-15 m away from the withdrawal roadway, the curve of the deviatoric stress presents a double peak value distribution. The two peaks appear at the position of 10 m in front of the working face and close to the withdrawal roadway, respectively. The peak value of the three measuring lines in front of the working face is about 24 MPa, and the change range is small. The peak value near the withdrawal roadway presents a gradually increasing trend. As shown in Figure 9(a), the position is 15 m away from the withdrawal roadway which deviatoric stress reaches the maximum peak that is 27.85 MPa at measuring line 1. The curve of 10 m and 5 m away from the roadway shows a single peak distribution, and the peak value gradually decreases. Figure 9(c) reaches the maximum peak value of 26.99 MPa at the distance of 10 m from the withdrawal roadway of measuring line 3.

On the right side of the withdrawal roadway, the deviatoric stress curve of the section coal pillar presents a single peak distribution, the peak position is about 5 m away from the withdrawal roadway, and the stress gradually returns to the original rock stress state with the distance from the withdrawal roadway; with the working face advancing, the peak value increases gradually, and the measuring line 1

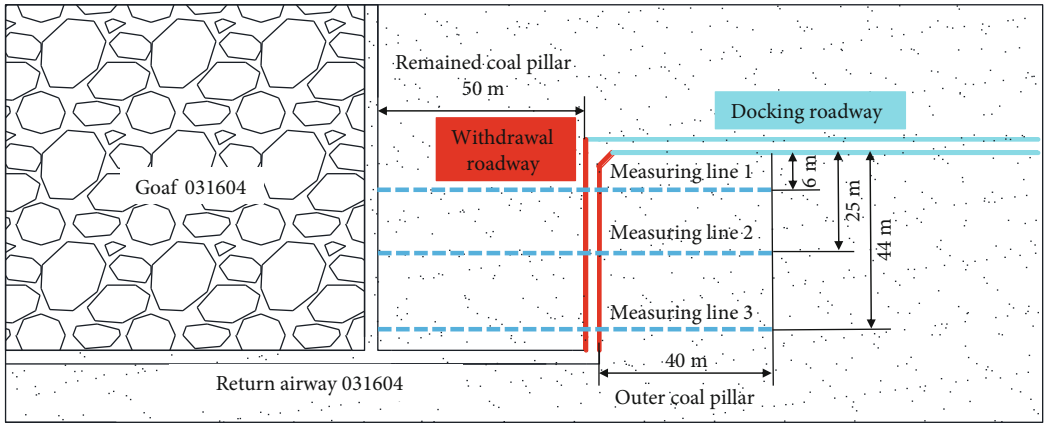


Figure 8: Monitoring position of mine pressure.

in Figure 9(a) can reach 33.22 MPa. The farther the survey line is from the docking roadway, the smaller the maximum peak is. As shown in Figure 9(c), the maximum peak value decreases to 27.88 MPa at line 3.

According to the previous analysis, with the loading and unloading action formed in front of the working face, the magnitude of the main stress in the roadway is redistributed, the direction of the principal stress also changes, and the deviatoric stress of the survey line 1 changes greatly. Therefore, the angle between the maximum principal stress and the vertical direction of the coal pillars on both sides of the withdrawal channel at the position of measuring line 1 is extracted, and the production curve is shown in Figure 10.

On the side of the remained coal pillar, from the working face to the withdrawal roadway, the angle of the principal stress increases gradually, and the maximum principal stress gradually deflects from the vertical direction clockwise to the horizontal direction. The maximum principal stress near the front of the working face is almost vertical, and the included angle is about 4°. In the interior of the remained coal pillar, the side angle decreases slightly and the maximum principal stress is vertical anticlockwise. When the distance from the retreating roadway is about 5.5 m, it reaches the trough point, then the angle increases sharply to the maximum, and the maximum principal stress continues to deflect to the horizontal direction. In the process of advancing, the maximum deflection angle of the remained coal pillar is about 60°, and when the remained coal pillar is 10 m, the angle is rotated clockwise to 88°; for the section coal pillar, the maximum principal stress is almost horizontal during the working face advancing.

As shown in Figure 11, the evolution regularity of the plastic zone of the withdrawal roadway during the advancing process of the working face 031604 is shown. In the process of working face that is 50-30 m away from the withdrawal roadway, the plastic zone of the surrounding rock of the withdrawal roadway presents a symmetrical shape, only two sides of the plastic zone expand 0.5 m each, and the principal stress angle is 70. When the working face is 30-15 m away from the withdrawal roadway, the depth of the plastic zone on the left side of the roadway extends significantly to 3.5 m, the plastic zone of the roof and the right side of the roadway are expanded and connected with each other, the overall shape changes from symmetry to asymmetry, and the angle of the principal stress in the plastic zone of the surrounding rock of the withdrawal roadway turns clockwise to 60. When the working face is 10-0 m away from the withdrawal roadway, the difference of main stress and the included angle of main stress change greatly, and the plastic zone of the goaf is connected with the plastic zone of the left side of the withdrawal roadway. At this time, the key point of support is to ensure the stability of the remaining coal pillar; the depth of the roof and right side plastic zone is obvious. After the working face is connected with the withdrawal roadway, the roof plastic zone reaches 6.5 m, the right side plastic zone reaches 6 m, and the floor plastic zone reaches 6 m, extending to 3.5 m. At this time, the failure depth and range of the roadway roof and right side are large, so the roof and right side should be taken as the support focus.

It can be seen from the above that the closer the working face is to the roadway, the deviatoric stress increases continuously, the scope of plastic zone expands obviously, and the deflection angle of maximum principal stress increases gradually, and the included angle with vertical direction is about 60°-90°. According to Figures 7(k) and 7(l), the shape of the plastic zone gradually extends from the middle of the roof to the right side, which is not conducive to the stability of the roof and the surrounding rock on the right side, and the overall stability of the surrounding rock of the roadway is more difficult to control.

4.3. Analysis of the Stress and Plastic Zone at the Docking Position of the Docking Roadway during the Final Mining. According to the above analysis, the surrounding rock stress and plastic zone change obviously within the range of 20 m from the working face to the docking position during the final mining period. The right side of the withdrawal roadway and the right side of the docking roadway are both within the section coal pillar, and the two sides interfere with each other at the docking position during the final mining. Therefore, in the process of advancing 20 m from the working face to the docking roadway, the roadway stress nephogram at the position 5 m ahead of the docking position of the docking roadway is intercepted, as shown in Figure 12.

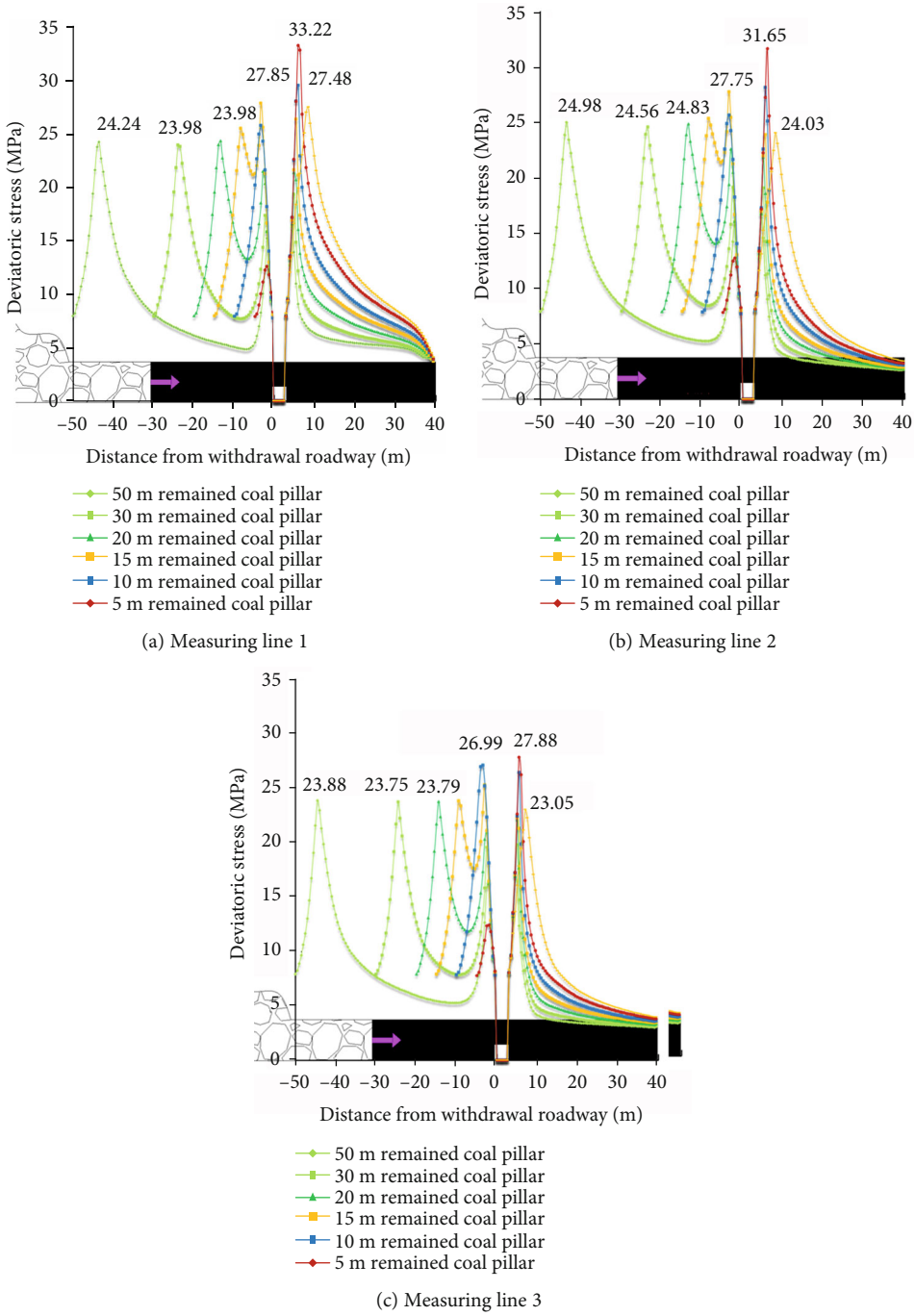


Figure 9: Variation trend of deviatoric stress of coal pillars on both sides of the withdrawal channel during final mining.

When the distance between the working face and the docking roadway is 20 m, there is no obvious deviatoric stress concentration in the surrounding rock of the roadway. With the working face advancing to 5 m, the two sides produce obvious deviatoric stress concentration, the peak value and distribution range of the two sides gradually increase, and the concentration degree of the right side slope is significantly greater than that of the left side; until the working face is connected, the peak range of the left side deviatoric stress is larger than that of the right side. During the whole process of pushing, the eccentric stress concentration of the left side extended to the top angle obviously, while the right side extended to the bottom angle obviously, and the deviatoric stress concentration position of the two sides gradually shifted to the deep.

In Figure 13, when the distance between the working face and the docking roadway is 20 MPa, the peak value of the two sides' deviatoric stress is about 20 MPa, and the peak value of the two sides appears about 2.5 m near the roadway. With the working face advancing to the 5 m position, the peak value of the two sides' deviatoric stress increases obviously, and the increased amplitude of the right side slope is obviously higher

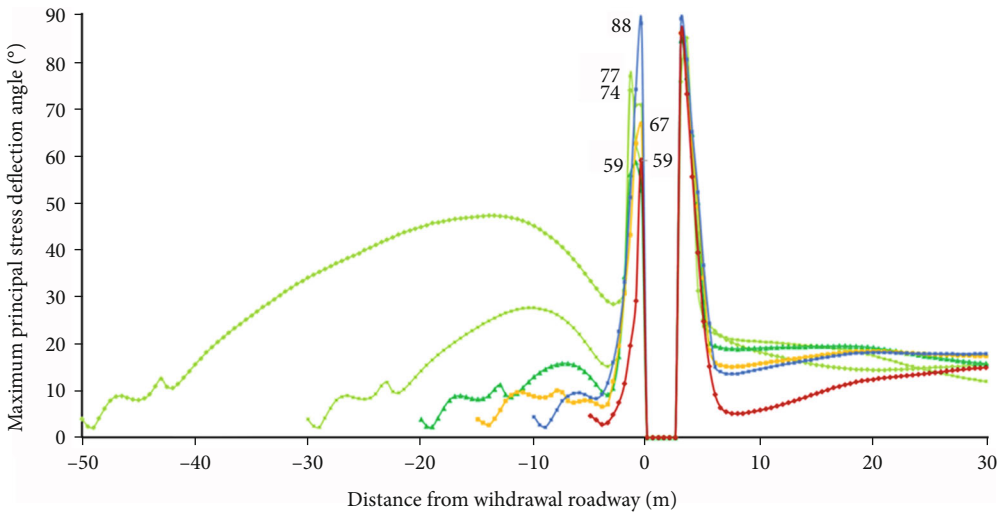


Figure 10: Variation trend of the principal stress angle of coal pillars on both sides of the withdrawal channel during final mining.

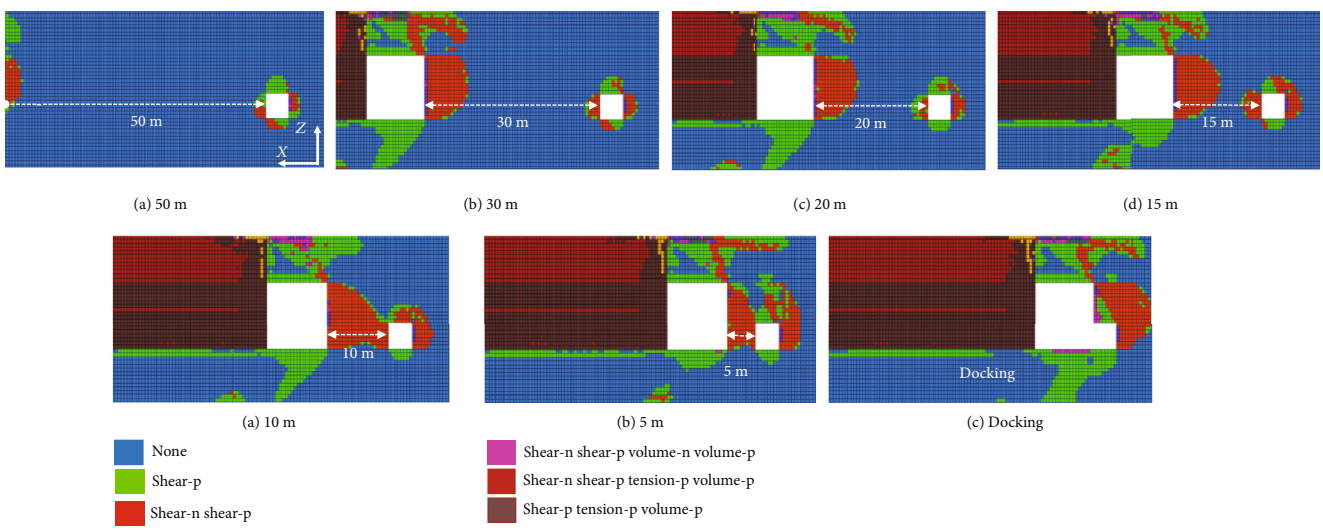


Figure 11: Evolution diagram of plastic zone distribution characteristics of remained coal pillar.

than that of the left side. Finally, the left side reaches 30.5 MPa, and the right side reaches 33.8 MPa. The peak position of the two side's deviatoric stress gradually shifts to the deep, about 1 m. When the working face is connected, the peak value of the deflection stress of the left side is almost unchanged, while that of the right side is slightly reduced to 27.4 MPa.

The curve generated by the angle between the maximum principal stress and the vertical direction is shown in Figure 14. It can be seen that the curve from the deep to the two sides shows a decreasing trend, and the deflection angle of the maximum principal stress gradually decreases and gradually deflects to the vertical direction counterclockwise. With the advancement of the working face, the angle of 0.5 m position near the two sides gradually decreases, the left side decreases from 17 degrees at 20 meters to 2.5 degrees at the docking position, the right side also decreases from 16 degrees at 20 meters to 4.3 degrees at the docking position, and the two sides are finally close to the vertical direction.

The plastic zone of the roadway surrounding rock during the advancing process of the intercepted working face is shown in Figure 15. When the working face is 20 m away from the docking roadway, the depth of the roadway roof and left side plastic zone are 2 m and 2.5 m, respectively. They are connected at the left side shoulder and expand each other obviously. The depth of the right side and low plate plastic zone is 3 m and 1.5 m, respectively. When the working face is 10 m away from the docking roadway, the depth of the plastic zone of the roof and the low plate of the roadway does not change, the left side continues to expand to 2.5 m, the extension range of the left side shoulder angle and the roof is gradually increased, and the depth of the plastic zone on the right side increases to 3.5 m, which extends to the roof obviously. According to the comparative analysis, the

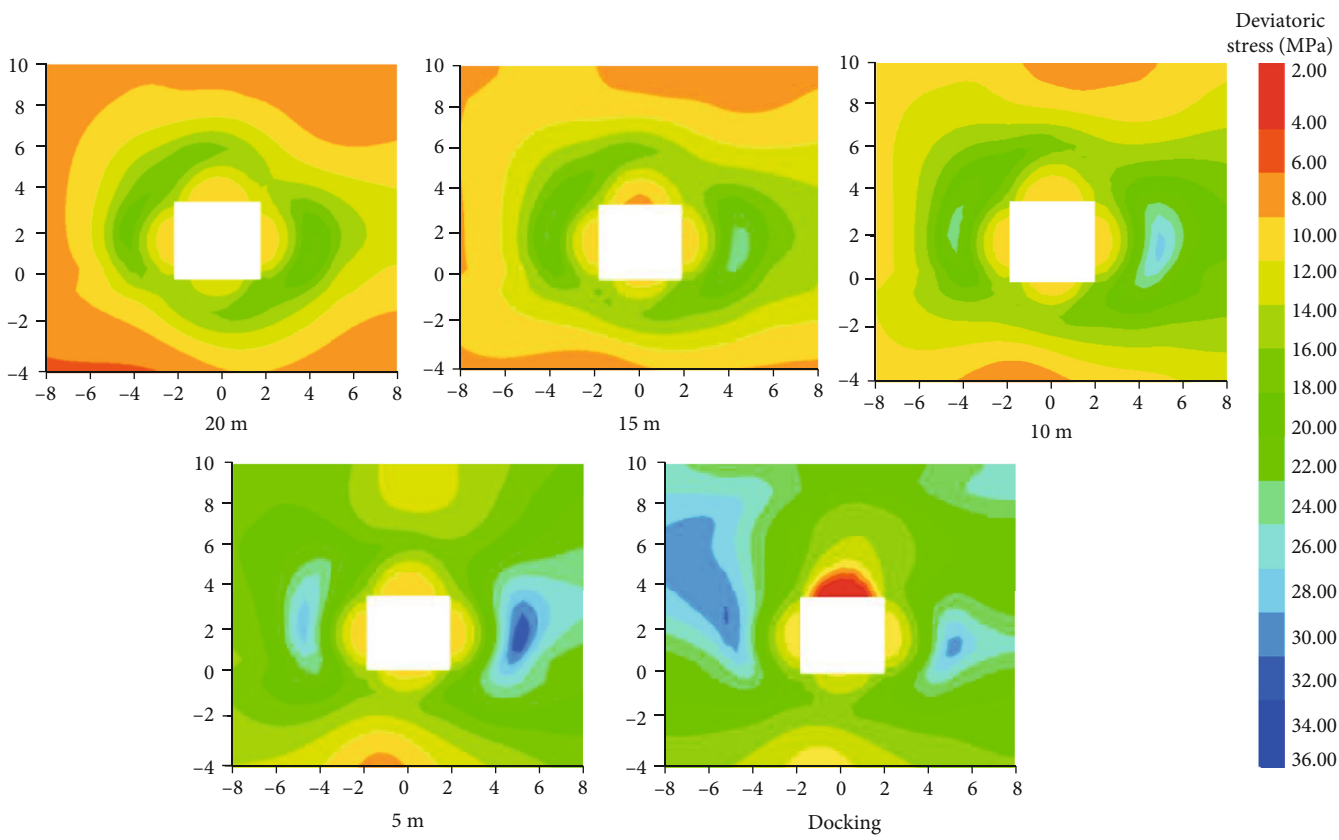


Figure 12: Nephogram of deviatoric stress.

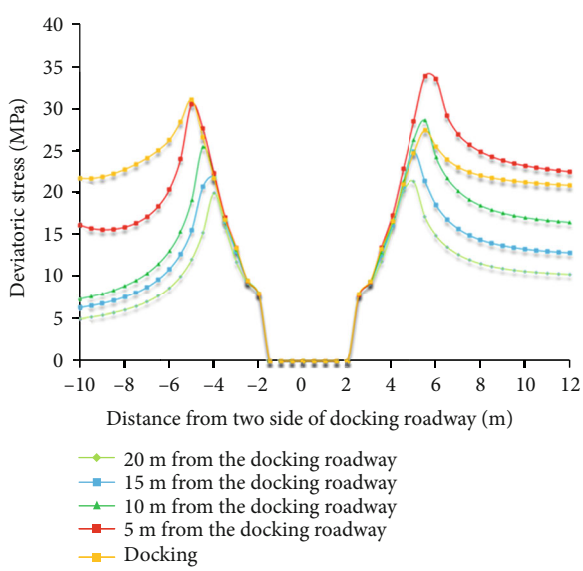


Figure 13: Deviatoric stress curve of two sides.

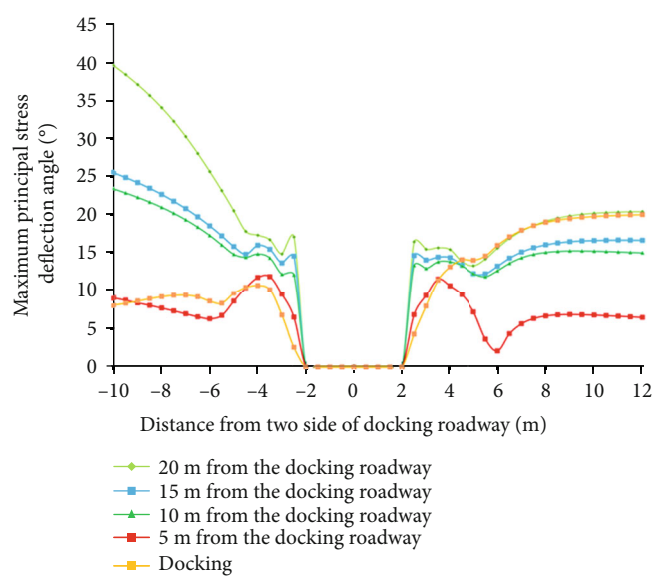


Figure 14: Variation diagram of the deflection angle of maximum principal stress of two sides.

maximum principal stress angle of the two sides deflects 3 to 4 degrees in the vertical direction. According to Figures 7(a) and 7(b), the decrease of the deflection angle causes the roof to expand to the left side, while the right side extends to the roof obviously. When the working face is 5 m away from the docking roadway, the depth of the plastic zone on the roof increases to 3 m, the scope of the plastic zone of the two sides is obviously extended, and the plastic zone of the three sides has been connected as a whole. According to Figure 15(d), at this time, the location of the internal deviatoric stress concentration in the two sides expands obviously to

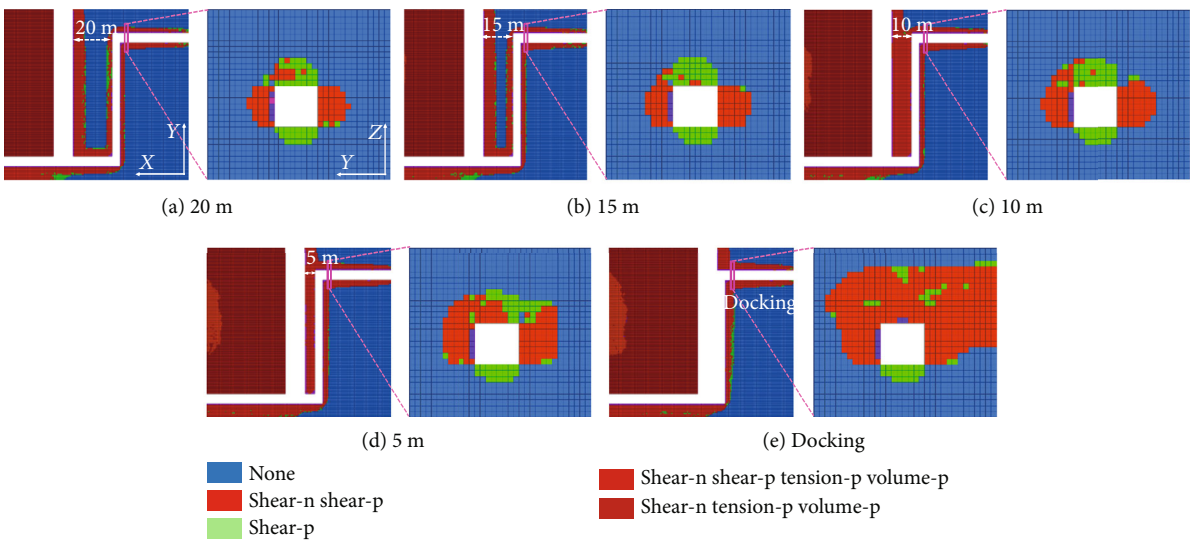


Figure 15: Evolution diagram of plastic zone distribution characteristics of docking roadway.

the bottom angle. When the working face is connected with the adjacent roadway, the plastic zone of the roof is greatly extended to a depth of 5 m. The right side has been connected with the plastic zone of the coal pillar side in the withdrawal roadway section. Therefore, it is difficult to control the stability of the roadway roof and the surrounding rock of both sides.

To sum up, with the 031604 working face advancing through the process, the deviatoric stress value of the withdrawal roadway and the docking roadway gradually increased, and the plastic zone scope expanded obviously. At the position 10 m away from the withdrawal roadway, the remained coal pillar was completely destroyed; the maximum principal stress of the two sides of the withdrawal roadway deflected clockwise from the vertical direction to the horizontal direction, its angle also gradually increased, and the shape of the plastic zone gradually changed from the vertical direction to the horizontal direction. The roof and section coal pillar side become the key control parts of surrounding rock stability. Under the influence of mining, the peak value of deviatoric stress on two sides of butted roadway increases gradually, and the failure range of its plastic zone gradually expands. With the principal stress angle that gradually deflects anticlockwise to the vertical direction, it gradually presents an asymmetric distribution. When the roof passes through, the plastic zone of both sides forms a whole, and the right side and the coal pillar side of the withdrawal roadway section are damaged and connected. At this time, the roof and the two sides become the key to the stability control of the surrounding rock position.

5. Study on Stability Control of Surrounding Rock in Docking Roadway

According to the production practice of the Laoshidan coal mine, in order to realize the safe and rapid withdrawal of the redundant fully mechanized mining equipment and the smooth docking of working face, it is necessary to ensure the surrounding rock stability of the withdrawal roadway and docking roadway during the final mining period. The surrounding rock of the withdrawal roadway and docking roadway is low strength coal. Based on the above theoretical analysis, based on the influence of superfront mining during the advancing process of the working face and the surrounding rock stability control problems that may be faced in the field, the reasonable support scheme is proposed.

5.1. Control Principle of Surrounding Rock of Withdrawal Channel and Docking Roadway. Bolt and cable support can effectively control the strength deterioration and plastic deformation of surrounding rock mass of the roadway, maintain the stability of surrounding rock, and avoid roof caving and collapse. Through timely and actively support of anchor cable, the principal stress difference and stress gradient change in the shallow part of surrounding rock are reduced, and the prestressed bearing structure is formed in the surrounding rock mass of the roadway to deal with the hidden danger of roof fall caused by continuous advancing of the working face.

The shape of the roadway section has a significant influence on the stability of surrounding rock, and the variation of deviatoric stress gradient and the distribution of plastic zone of the roadway with different cross-section shapes are obviously different; the smoother the boundary line of the roadway is, the smaller the scope of "invalid reinforcement area" of the surrounding rock, which can effectively reduce the uneven distribution of deviatoric stress and plastic asymmetric failure range, and the higher the stability of roadway. The optimal cross-section shape of roadway in high crustal stress is circular and elliptical, and the semicircular arch roadway with the straight wall can effectively improve the distribution of deviatoric stress in the roof and two sides and the expansion degree of the plastic zone.

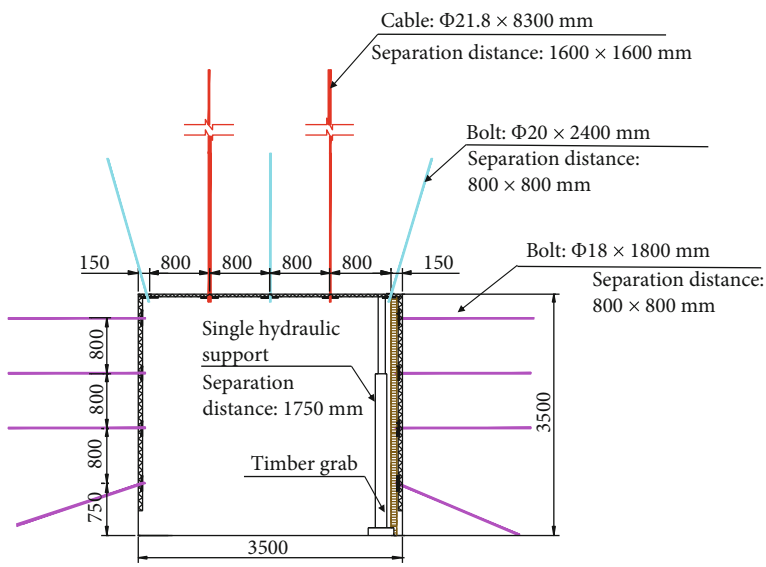


Figure 16: Support scheme for withdrawal roadway.

Grouting, as the main means to improve the properties of the damaged surrounding rock mass, plays a role in filling the space of discontinuity structural plane of the surrounding rock mass of the roadway. It can "bond" the rock mass on both sides of the structural plane, delay its strength loss, and improve the integrity of rock mass. Through grouting the coal bodies on both sides of the withdrawal channel, the overall strength and integrity of the roadway side coal body can be improved. While controlling the damage and deformation of the surrounding rock of the roadway, the support force of the roadway side to the roof can also be improved. With the support of the anchor cable, the anchoring force of the anchor cable in the coal seam can be improved, so as to ensure the smooth docking of the working face and rapid and safe withdrawal.

According to the above theoretical analysis, during the final mining period, with the continuous advancement of the working face, the plastic zone of the surrounding rock of the withdrawal roadway and the docking roadway developed maliciously, and the remaining coal pillar was completely destroyed at the position of 10 m away from the withdrawal roadway. Therefore, the support (bolt and anchor cable)—modified (grouting) collaborative control scheme—is adopted for the withdrawal channel to ensure the stability of the surrounding rock mass of the withdrawal roadway and the remaining coal pillar. The roof and surrounding rocks at the docking position of the docking roadway are seriously damaged, and the plastic zone has been connected as a whole. Therefore, the support (anchor bolt and anchor cable)—the collaborative control scheme of changing the cross-section shape of the roadway—is adopted to ensure the smooth use of the docking roadway.

5.2. Surrounding Rock Stability Control Scheme of Withdrawal Roadway and Docking Roadway. The withdrawal roadway support parameters of anchor cable support are as follows: five Φ 20 mm × 2400 mm left-hand screw thread steel bolts are arranged on the roof with a row spacing of 800 mm × 800 mm, and 2 pieces of Φ 21.8 × 8 300 mm anchor cables with the spacing of 1600 mm × 1600 mm; 4 pieces of Φ 18 mm × 1800 mm glass fiber reinforced plastic bolts are used for the remained coal pillar side; 4 pieces of left-hand screw thread steel bolts are selected for the section coal pillar side, and the row spacing between the two sides of bolts is 800 mm × 800 mm. Diamond mesh shall be laid in the range of wall height above 350 mm. In order to prevent roof fall and spalling accidents of roof and section coal pillar side during connection, a single hydraulic prop with a row spacing of 1750 mm is erected against the side of the withdrawal channel, and the gap between the pillar side and the section is filled with formwork. The cross-section support is shown in Figure 16.

As the working face is advanced to within 20 m of the withdrawal roadway, the roadway is affected by obvious mining. During the continuous advancing process, the remained coal pillars are completely destroyed. When the width of the remained coal pillars is 50 m, grouting is started for the remained coal pillars and section coal pillars. The remained coal pillars within 47 m from the working face to the return airway are divided into five grouting sections, each section is about 10 m, and three boreholes are evenly arranged. Each hole is arranged at the position of 1 m away from the roof. The elevation angle is 60°, and the hole depth is 3.5 m. Each hole is injected with 1 t reinforcement material. Every time the working face is pushed forward for one cycle, 3 tons of reinforcement materials are injected in each section, and 15 tons of grouting are injected in each cycle. When the working face is pushed to the 20 m position of the withdrawal roadway, grouting shall be conducted to the side of the section coal pillar in the withdrawal roadway, and the grouting holes shall be arranged at the position of 1 m from the roof. The drilling parameters, grouting method, and grouting amount shall be the same as the grouting measures for the remained coal pillars, and the section coal pillars shall be grouted at one time.

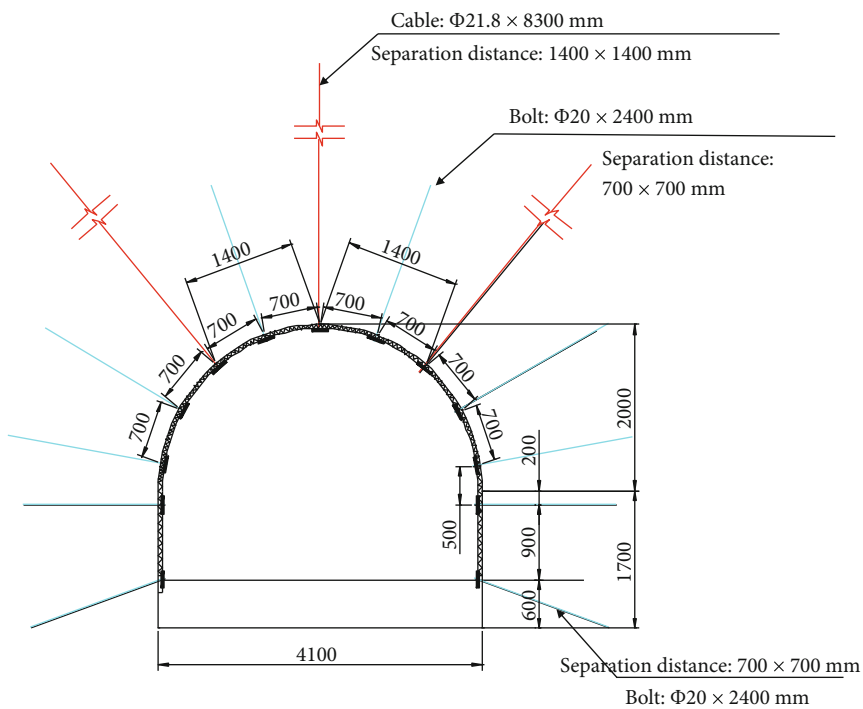


Figure 17: Support scheme within 5 m behind the docking position of ducking roadway.

In view of the problem that the depth and range of the plastic zone of the roof and the two sides of the docking roadway are large and difficult to control, the method of changing the shape of the tunnel section is adopted to improve the surrounding rock failure. The straight-wall semicircular arch roadway layout is selected within 150 m behind the docking position of the docking roadway. The section width is 4100 mm, the wall height is 1700 mm, and the arch height is 2050 mm. Nine Φ 20 mm × 2400 mm left-hand screw thread steel bolts are arranged on the roof of the roadway, the spacing between rows is 700 mm × 700 mm, and 3 pieces of Φ 21.8 × 8300 mm anchor cables are arranged with the spacing of 1400 mm × 1400 mm. Two Φ 18 mm × 1800 mm left-handed screw thread steel bolts are selected for both sides, and the spacing between rows is 700 mm × 700 mm. Diamond mesh is laid in the range of wall height above 600 mm. The support section is shown in Figure 17.

At the intersection of the withdrawal roadway and the docking roadway, and within the scope of 5 m behind the docking position of the docking roadway, six rows of Φ 21.8 × 8300 mm anchor cables with a spacing of 1400 mm × 1000 mm are used for reinforcement, and the π-shaped steel beam of 5 m length is used with the support of resin fiber flexible mesh. After adopting the above support scheme, a number of roof displacement monitoring stations are arranged in the middle of the roof at the docking position of the docking roadway. The results show that the roof subsidence is about 200 mm during the end mining period; there is no roof fall, collapse, roof crushing support, anchor cable support failure, and other phenomena; the working face is smoothly connected; and the equipment is quickly and safely withdrawn. The surrounding rock control has certain guiding significance.

6. Conclusion

(1) Through the analysis of stress environment of coal seam under the concentrated coal pillar, it is concluded that the docking position should be set in the relative No. 12 coal seam goaf. Based on the analysis of the superimposed stress in advance mining of this coal seam, the specific docking position is arranged at 860 m away from the open cut, and 10 m away from the goaf of No. 12 coal seam, which is conducive to ensuring the stability of surrounding rock of roadway docking

(2) With the working face 031604 advancing through the process, the deviatoric stress value of the withdrawal roadway gradually increases, the plastic zone scope expands obviously, the maximum principal stress of the two sides of the roadway deflects clockwise from the vertical direction to the horizontal direction, its angle gradually increases, and the shape of the plastic zone gradually expands from symmetry to asymmetry. The roof and section coal pillar side become the key control parts of the surrounding rock stability

(3) The results show that the peak value of deviatoric stress of two sides increases gradually and the angle of principal stress gradually deflects anticlockwise to a vertical direction under the influence of mining at the docking position of docking roadway. The joint action is the mechanical mechanism of asymmetric distribution and expansion of the plastic zone of docking roadway. The roof and the two sides of the plastic zone form a whole, the right side and the

withdrawal roadway section coal pillar side damage connection, and roadway roof and two sides become the key parts of surrounding rock stability control

(4) In view of the asymmetric failure characteristics of the withdrawal roadway and the docking roadway, the support (bolt and anchor cable)—modified (grouting) collaborative control scheme—is adopted for the withdrawal roadway, and the coordinated control scheme of support (bolt and anchor cable)—changing the cross-section shape of the roadway—is adopted for the docking roadway. The proposed control scheme for surrounding rock stability is applied on site, which achieves smooth docking of working face and rapid and safe equipment for the purpose of full withdrawal

Data Availability

The data used to support the findings of this study are available from the corresponding author upon request.

Conflicts of Interest

The authors declare that there are no conflicts of interest regarding the publication of this paper.

Acknowledgments

This research was supported by the National Natural Science Foundation of China (Grant No. 51964036, No. 51764044), the Natural Science Foundation of Inner Mongolia (Grant No. 2019LH05004), the Science Research Foundation of Institution of Higher Education of Inner Mongolia Autonomous Region (Grant No. NJZY20093), and the Innovation Fund Project of Inner Mongolia University of Science and Technology (Grant No. 2019QDL-B31).

References

[1] K. Xiangyong, Y. Ke, and L. Wei, "Floor failure mechanism and engineering application of downward mining of the large dip angle coal seam group," *Chinese Journal of Underground Space and Engineering*, vol. 11, pp. 394–400, 2015.

[2] L. Haodang, Y. Hanhong, and Z. Bin, "Control study of strong strata behaviors during the fully mechanized working face out of concentrated coal pillar in a shallow depth seam in proximity beneath a room mining goaf," *Journal of China Coal Society*, vol. 40, pp. 6–11, 2015.

[3] L. I. U. Jinhai, J. I. A. N. G. Fuxing, and Z. H. U. Sitao, "Study of dynamic and static abutment pressure around longwall face and its application," *Chinese Journal of Rock Mechanics and Engineering*, vol. 34, no. 9, pp. 1815–1827, 2015.

[4] X. Yongqi, *The coal mining*, China University of Mining and Technology press, 1999.

[5] K. Hongpu, J. Pengfei, H. Bingxiang et al., "Roadway strata control technology by means of bolting-modification-destress-ing in synergy in 1 000 m deep coal mines," *Journal of ChinaCoal Society*, vol. 45, no. 3, pp. 845–864, 2020.

[6] H. Dengyun, W. Yongzheng, C. Haijun, C. Xiaowei, and L. Yang, "Instabilitymechanism and prevention technology of roadway in close distanceand extra thick coal seam under goaf," *Journal of China Coal Society*, vol. 44, no. 9, pp. 2682–2690, 2019.

[7] S. Xuegui, S. Xuanmin, Y. Honghu, and D. Xianjie, "Stability controlof the roadway group under the influence of overlying-goaf," *Journal of Mining & Safety Engineering*, vol. 33, no. 3, pp. 415–422, 2016.

[8] L. Huawen, "The mechanism of stability of pre- driven rooms andthe practical techniques," *Journal of China Coal Society*, vol. 39, no. S1, pp. 41–43, 2014.

[9] K. Lv, Z. Zhichao, and Z. Zhiqiang, "Study on support technology of equipment removal channel in fully-mechanizedtopcoal caving face in ultra- thick seam," *Coal Science and Technology*, vol. 46, no. 3, pp. 39–43, 2018.

[10] W. Zhu, D. Ren, and M. Chen, "Rational buried depth for regulating roadway application during coal face withdrawal in Shendong mining area," *Journal of Mining & Safety Engineering*, vol. 32, no. 2, pp. 279–284, 2015.

[11] B.-y. Jiang, S.-t. Gu, L.-g. Wang, G.-c. Zhang, and W.-s. Li, "Strain burst process of marble in tunnel-excavation-induced stress path considering intermediate principal stress," *Journal of Central South University*, vol. 26, no. 4, pp. 984–999, 2019.

[12] S. H. Tu, F. T. Wang, F. J. Dou, Y. Yuan, and Y. Lu, "Fully mechanized top-coal caving underground stress at gateways under barrier pillars of an upper coal seam," *Journal of China University of Mining & Technology*, vol. 35, no. 2, pp. 191–193, 2010.

[13] Y. L. Zhao, Y. X. Wang, W. Wang, L. Tang, Q. Liu, and G. Cheng, "Modeling of rheological fracture behavior of rock cracks subjected to hydraulic pressure and far field stresses," *Theoretical and Applied Fracture Mechanics.*, vol. 101, pp. 59–66, 2019.

[14] X. Wu, "Space-time evolutionary regularity of plastic zone and stability control in repetitive mining roadway," *Journal of China Coal Society*, vol. 41, no. 1, pp. 1–6, 2016.

[15] L. Jiang, P. Kong, P. Zhang et al., "Dynamic analysis of the rock burst potential of a longwall panel intersecting with a fault," *Rock Mechanics & Rock Engineering*, vol. 53, no. 4, pp. 1737–1754, 2020.

[16] Z. Zhang, M. Deng, J. Bai, X. Yu, Q. Wu, and L. Jiang, "Strain energy evolution and conversion under triaxial unloading confining pressure tests due to gob-side entry retained," *International Journal of Rock Mechanics and Mining Sciences*, vol. 126, p. 104184, 2020.

[17] Z. Zhang, X. Yu, H. Wu, and M. Deng, "Stability control for gob-side entry retaining with supercritical retained entry width in thick coal seam longwall mining," *Energies*, vol. 12, article 1375, 2019.

[18] Z. Zhang, M. Deng, X. Wang, W. Yu, F. Zhang, and V. D. Dao, "Field and numerical investigations on the lower coal seam entry failure analysis under the remnant pillar," *Engineering Failure Analysis*, vol. 115, p. 104638, 2020.

[19] E. Wang, G. Chen, X. Yang, G. Zhang, and W. Guo, "Study on the failure mechanism for coal roadway stability in jointed rock mass due to the excavation unloading effect," *Energies*, vol. 13, no. 10, article 2515, 2020.

[20] M. Bai, F. S. Kendorski, and D. J. Van Roosendaal, "Chinese and North American high-extraction underground coal mining strata behavior and water protection experience and guidelines," in *Proceedings of the 14th International Conference on Ground Control in Mining*, Morgantown, WV, USA, 1995.

[21] S. S. Peng and H. S. Chiang, *Long wall minim*, Wiley, 1984.

[22] Q. S. Bai, S. H. Tu, Y. Yuan, and F. T. Wang, "Back analysis of mining induced responses on the basis of goaf compaction theory," *Journal of China University of Mining & Technology*, vol. 42, no. 3, pp. 355–361, 2013.

[23] Y. Zhao, L. Zhang, W. Wang, Q. Liu, L. Tang, and G. Cheng, "Experimental study on shear behavior and a revised shear strength model for infilled rock joints," *International Journal of Geomechanics*, vol. 20, article 04020141, no. 9, 2020.

[24] Y. L. Zhao, L. Y. Zhang, J. Liao, W. J. Wang, Q. Liu, and L. Tang, "Experimental study of fracture toughness and subcritical crack growth of three rocks under different environments," *International Journal of Geomechanics*, vol. 20, no. 8, article 04020128, 2020.

[25] Y. Zhao, L. Zhang, W. Wang, C. Pu, W. Wan, and J. Tang, "Cracking and stress–strain behavior of rock-like material containing two flaws under uniaxial compression," *Rock Mechanics and Rock Engineering*, vol. 49, no. 7, pp. 2665–2687, 2016.

[26] H. Gang, L. Xudong, Q. Xiaocheng, Z. Lei, and L. Lixin, "Study on correlation between spatial fracturing of overlying strata and distribution of mining stress field in stope," *Coal Science and Technology*, vol. 47, no. 2, pp. 53–58, 2019.

[27] Z. Zhiqiang, M. Nianjie, L. Hongtao, and G. Xiaofei, "A butterfly failure theory of rock mass around roadway and its application prospect," *Journal of China University of Mining & Technology*, vol. 47, no. 5, pp. 969–978, 2018.