Hindawi
Advances in Materials Science and Engineering
Volume 2020, Article ID 5036092, 18 pages
https://doi.org/10.1155/2020/5036092

Research Article

Deviatoric Stress Evolution Laws and Control in Surrounding Rock of Soft Coal and Soft Roof Roadway under Intense Mining Conditions

Dongdong Chen,[1] Chunwei Ji,[1] Shengrong Xie,[1,2] En Wang,[1] Fulian He,[1,2] Qiong Cheng,[1] and Qing Zhang[1]

[1]*School of Energy and Mining Engineering, China University of Mining & Technology (Beijing), Beijing 100083, China*
[2]*Beijing Key Laboratory for Precise Mining of Intergrown Energy and Resources, China University of Mining & Technology (Beijing), Beijing 100083, China*

Correspondence should be addressed to Shengrong Xie; xsrxcq@163.com

Received 2 February 2020; Accepted 23 April 2020; Published 22 June 2020

Academic Editor: María Criado

Aiming at the problem of large deformation and instability failure and its control of soft coal and soft roof roadway under intense mining, laboratory experiments, theoretical calculations, Flac^{3D} numerical simulation, borehole peeping, and pressure observation were used to study the deflection characteristics of the deviatoric stress of the gas tailgate and the distribution and failure characteristics of the plastic zone in the mining face considering the strain softening characteristics of the roof and coal of roadway, and then the truss anchor cable-control technology is proposed. The results show the following: (1) The intense mining influence on the working face will deflect the peak deviatoric stress zone (PDSZ) of the surrounding rock of the gas tailgate. The influence distance of PDSZ is about 20 m in advance and 60 m in lag; the PDSZ at the gob side of the roadway is located in the range of 3–5.5 m from the surface of the coal pillar, while the coal wall side is mainly located in the range of 3–4.5 m at the shoulder corner and bottom corner of the solid coal. (2) The intense mining in the working face caused the nonuniform expansion of the surrounding rock plastic area of the gas tailgate. The two shoulder angles of the roadway and the bottom of the coal pillar have the largest damage range, and the maximum damage location is the side angle of the coal pillar (5 m). Angle and bottom angle of coal pillar are the key points of support control. (3) The plastic failure line of the surrounding rock of the gas tailgate is always between the inner and outer contours of the PDSZ, and the rock mass in the PDSZ is in a stable and unstable transition state, so the range of anchor cable support should be cross plastic failure line. (4) The theoretical calculations and numerical simulation results agree well with the drilling peep results. Based on the deflection law of the PDSZ and the expansion characteristics of the plastic zone, a truss anchor cable supporting system with integrated locking and large-scale support function is proposed to jointly control the roof and the two sides, which effectively solves the problem of weak surrounding rock roadway under severe mining deformation control problems realizing safety and efficient production in coal mines under intense mining.

1. Introduction

The influence of mining is a major difficulty regarding the rock stability of the surrounding roadway in a mining area and is a popular research topic. Most roadways or chambers are unaffected by mining disturbances in the coal face [1]. Many roadways have achieved stability control of the surrounding rock during excavation. However, large deformations of the surrounding rock, serious floor heaving, and uncontrollable deformations of the surrounding rock still occur due to mining disturbances. The roadway is often seriously deformed and needs to be maintained and repaired frequently. Thus, the problem of roadway maintenance has affected the normal production and economic benefits of coal mines.

Experts and scholars have conducted several theoretical and experimental studies on the deformation and failure mechanism of the surrounding rock and the corresponding

supporting control technology when under the influence of mining. Based on the engineering background of a multi-roadway layout in a mining face, Kang et al. [2] analyzed the deformation and stress distribution characteristics of the surrounding rock of a retained roadway and determined that a deformation of this surrounding rock mostly occurs after the creation of a mining face. Wang et al. [3] studied the deformation and failure mechanism of the surrounding rock of a roadway under the influence of mining, and they considered that the advancement of an abutment pressure has a significant influence on the stress distribution of the surrounding rock. In addition, Li et al. [4] studied the failure characteristics of a dynamic long-span pressure mining roadway and found that the main cause of a support failure is the comprehensive effect of a complex surrounding rock environment. Moreover, Zhang et al. [5] established a mechanical model of a nonisobaric roadway and discussed the evolution mechanism of the fracturing in the surrounding rock of a roadway under a mining environment. Chen et al. [6] found that the asymmetric deformation of a roadway is caused by a change in the magnitude and direction of the stress field of the surrounding rock caused by mining stress. He and Zhang [7, 8] analyzed the failure and instability mechanism of a roadway affected by mining in an adjacent large-scale fully mechanized caving face and put forward the use of high prestressed truss anchor cable support technology. According to the deformation characteristics of the large deformations in the soft surrounding rock, Yu and Liu [9] provided a set of critical technologies and their corresponding control principles. In addition, He et al. [10] proposed the design theory of a large nonlinear deformation of the surrounding rock, designed a coupled support method consisting of bolts and a metal mesh, and developed a new type of energy-absorbing anchor cable, characterized by a constant resistance and large deformation. Li et al. [11] analyzed the serious deformation occurring in the surrounding rock in a large section chamber, as well as the frequent failures of the bolt supports; subsequently, a new high-strength bolt grouting technology was proposed. Based on the deformation characteristics of loose and fractured rock masses and their influencing factors, Fangtian et al. [12] presented an entire support technology of section bolt grouting.

It is well known that the occurrence and development of plastic deformation are determined by deviatoric stress, which can control the failure of a rock mass and has an important significance regarding the influence of the plastic failure of a rock mass [13]. Using plastic mechanics and the Mohr–Coulomb theory, Ma et al. [14] obtained a formula for calculating the deviatoric stress of the surrounding rock of a circular roadway as well as a formula for the radius of a plastic zone. Based on the rock mechanics and a numerical simulation theory, Yu and Yuan [15, 16] analyzed the distribution law of the deviatoric stress field and plastic zone during the process of deformation and failure of the surrounding rock of the roadway, aiming at the failure mechanism and stability control of the surrounding rock of the roadway. He and Xu [17, 18] used the deviatoric stress theory to study the stress distribution of the roof support and floor under a coal pillar for a large cross-section cutting hole. Xie et al. [19, 20] used a strain softening model to study the evolution law of deviatoric stress and the plastic zone of the surrounding rock of a gob-side retaining roadway. Li et al. [21] considered that adjusting the three-dimensional stress fields in the surrounding rock causes the majority of the principal deviatoric stress to concentrate on the filling body, causing the surrounding rock to fail.

In general, in the above studies, thorough analyses regarding the relationship between deformation and the failure characteristics of a roadway and the deviatoric stress were conducted. Based on the background of the gas tailgate in the 310101 working face of the Xinyuan coal mine, this paper mainly describes a study on the deflection of the PDSZ of the surrounding rock and failure rate of the plastic zone during the process of the working face. On this basis, the combined control technology of a roof truss anchor cable structure and a coal side anchor cable truss system is proposed, which effectively solves the control problem of a large deformation of the surrounding rock of a soft coal roadway under intense mining.

2. Engineering Background

2.1. Geological Conditions of Mine Production. As shown in Figure 1, the Xinyuan coal mine is situated northeast of Shouyang County, Jinzhong City, Shanxi Province, China, which is approximately 10.0 km from the center of Shouyang County. The 310101 working face, with an average depth of 500 m, is located in coal seam 3 of the first mining area in the Xinyuan coal mine, Shanxi Province, China. The average mining height is 4.0 m, and the average dip angle of the coal seam is 4° in the working face area. The coal seam is extremely soft ($f = 0.2$), and an endogenous fissure with one or two layers 0.02–0.1 m thick has developed. The immediate roof is mudstone, with an average thickness of 3.60 m. However, the thickness is unstable and reaches 10 m in local areas, whereas the seam floor is gray sandy mudstone, medium-to-thick layered, under a joint development with an average thickness of 4.50 m. Furthermore, a histogram of the regional coal and strata in the working face area is shown in Figure 2.

An inclined comprehensive long-wall backward mechanized mining method is applied in the 310101 working face; that is, the width of the working face is 240 m, the length of the working face is 1,800 m, and the cycling progress is 0.8 m. The mining roadway will continue to experience the mining influence of the working face for a long time to come. The Xinyuan coal mine is a high-gas mine, and to solve the problem of gas overruns on the working face and ensure production safety, a special gas drainage roadway is usually set up along the roof of the coal seam at a certain distance inside the tailgate, which is called the gas tailgate [22]. The gas tailgate of the 310101 working face is located 22 m from the tailgate and is retained for the next working face after mining in the 310101 working face, resulting in a lengthy period of deformation and destruction of the gas tailgate. The gas tailgate section has a rectangular shape with a width of 5 m and a height of 3 m. After excavating along the roof of

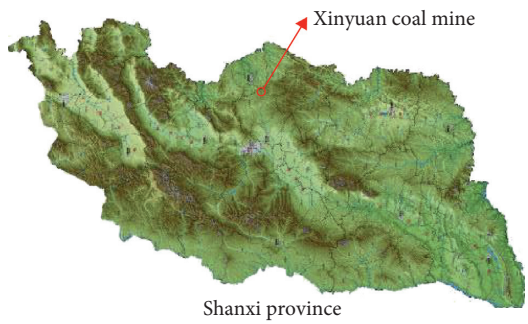


Figure 1: Location of Xinyuan coal mine.

Rock stratum	Legend of rock column	Thickness (m)	Lithology description
Fine sandstone		3.56	Gray, flat fracture, horizontal and gentle wave bedding, siliceous cementation
Medium sandstone		5.50	Dark gray, inclined bedding, joint development, brittle, calcareous cementation
Sandy mudstone		2.90	Dark gray, layered, evenly jointed, rich in plant fossil fragments, brittle and semihard, flat section
Fine sandstone		1.98	Deep gray, layered, fissure developed, containing plant rhizome fossils
Sandy mudstone		1.60	Soft, flat fracture, roof containing about 0.05 mm of coal line
Mudstone		3.60	Gray, flat fracture, horizontal and gentle wave bedding, siliceous cementation
Coal mine 3		2.50	Black, massive, vitrinite-dominated, heterogeneous fracture, endogenous and exogenous fracture
Sandy mudstone		4.50	Argillaceous structure, angular fracture, local interbedding with siltstone, undeveloped bedding

Figure 2: Coal and rock strata histogram.

coal seam 3, water leaching occurs in the local area of the roof of the gas tailgate, and the integrity of the surrounding rock is poor. The roof is easily damaged and can collapse through engineering disturbances and lateral pressure.

2.2. Severe Ground Pressure in the Soft Roadway under Intense Mining. A severe ground pressure has occurred along the roadway located at coal seam 3 during the workface operations, as shown in Figure 3. In addition, numerous roof caving accidents have occurred in the 310201 gas tailgate and 310202 belt tailgate in the west wing of the first mining face of coal seam 3 during the production of the working face. The maximum roof caving height is more than 10 m, which blocks the entire section of the roadway, preventing ventilation and the ability to walk in the area. Collapses have occurred within a considerable range of the coal side; in addition, the expansion of the area without a collapse is obvious, and the overall displacement of the coal side has reached 1-2 m. The original support system of the roadway has been seriously damaged, such as a breakage of the roof bolts, a severe bending of the pallets, and W-band folding, whereas the roadway bolts are suspended owing to the collapse of the coal body, resulting in a large number of breakage failures of the roadway bolts. For a collapsed section of a secondary reuse belt roadway, the Xinyuan coal mine has required the excavation of a supplementary roadway with a length of 380 m to replace the collapsed section. After the collapse of the gas tailgate, the use of stacking, a supplementary anchor cable, and other reinforcement support measures have been adopted to strengthen the area and maintain its normal functionality.

Based on the characteristics of the strata pressure in the first mining face of the west side of the Xinyuan coal mine, we can see that the deformation and failure of the surrounding rock at the mining roadway are extremely critical during the production process of the working face. The support system has been damaged in several places, such as a bolt breakage and a warping of the W-band of the anchor cable. The mining roadway is also being continuously renovated to maintain its normal functionality. However, in the tailgate located at the 310101 working face on the eastern wing, a malignant roof fall and a serious roof subsidence have occurred. The roof caving height of the roadway is 7-8 m, the roof caving length is more than 15 m, the roof subsidence of the roadway is more than 0.5 m, and the

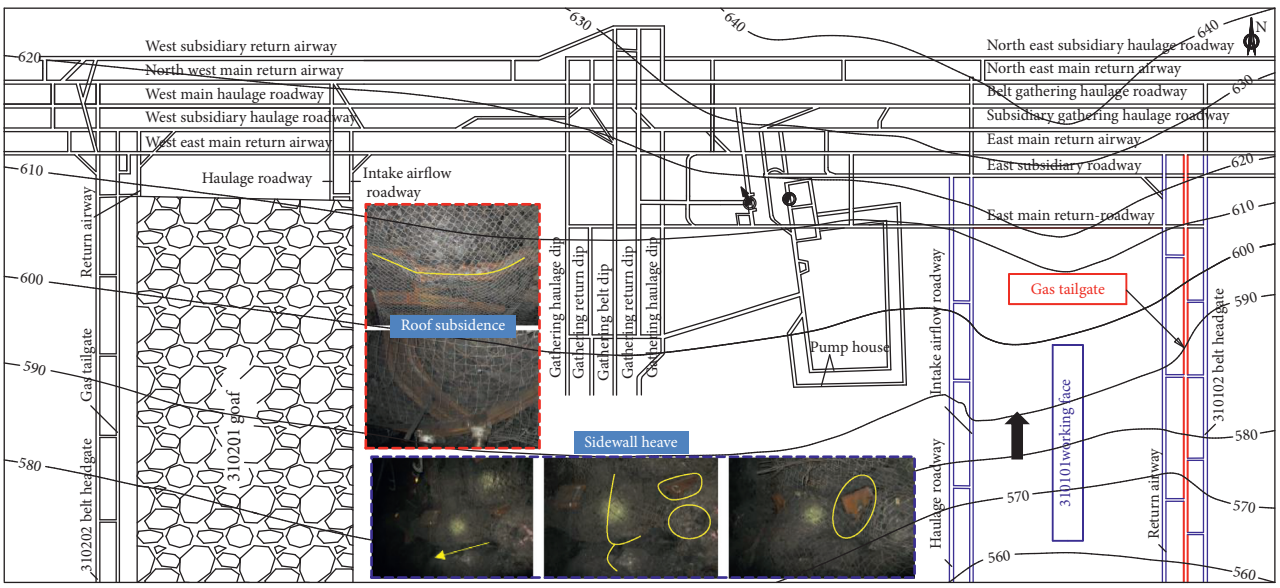


Figure 3: Rock pressure failure of soft coal roadway.

convergence rate of the cross-section is 42%. Faced with a serious roof caving and roof subsidence in the eastern coal roadway where the mining influence is small, it is difficult to ensure the stability of the surrounding rock and avoid repeated renovations during the production process of the working face. Clearly, conventional comprehensive control technology measures cannot solve this problem. In combination with the deflection law of the PDSZ and the failure rate of the plastic zone of the gas tailgate during the mining process of the 310101 working face, a comprehensive control technology that employs a roof truss anchor cable structure and a coal side anchor cable truss system is proposed to solve the problem of controlling large deformations of the surrounding rock of the soft coal roadway and realize the safety and efficient production in coal mines under severe mining conditions.

3. Roadway Surrounding Rock Failure and Plastic Area

3.1. Damage Range of Roadway Surrounding Rock. After the excavation of the gas tailgate, the plastic surroundings of the shallow surrounding rocks are deformed and destroyed under the pressure of the overlying strata, the surrounding rocks are extruded into the roadway, and finally a natural balance arch is formed in a certain depth, as shown in Figure 4. The range and stress of surrounding rock failure area are important basis for roadway support. Therefore, the elastoplastic theory is used to determine the range and stress of surrounding rock failure. According to the theory of natural balance arch, the rock mass above the natural balance arch is stable, and the area within the balance arch is the surrounding rock damage area. The damage range of the surrounding rock of the roadway is determined by equations (1)~(2)

Depth of coal roadway coal seam damage C is as follows:

$$C = \left(\frac{k\gamma HB}{10^4 f_y} - 1\right) h \tan\left(45^\circ - \frac{\varphi}{2}\right). \quad (1)$$

In the formula, K is extrusion stress concentration factor around the roadway, according to the numerical simulation of the peak stress of the two sides; take $K = 2.3$; γ is overburden bulk density, 25 kN/m^3; H is burial depth of the roadway, 500 m; and B is characterized mining. The dimensionless parameter of the influence degree is 1.4 under the influence of multiple mining; f_y is coal seam hardness coefficient, 0.9; h is roadway height, 3 m; and φ is coal seam friction angle, 28°. It can be calculated that $C = 1.83$ m; that is, the damage depth of the side of the gas tailgate is 1.83 m.

Destruction depth of roof rock layer b is as follows:

$$b = \frac{(a + C)\cos\alpha}{k_y f_n}. \quad (2)$$

In the formula, a is semicolumn distance of roadway, 2.25 m; α is inclination of coal seam, 3°; k_y is stability coefficient of rock to be anchored; mudstone roof is generally taken as 0.5; and f_n is hardness coefficient of anchored rock, 4.3. It can be calculated that $b = 2$ m; that is, the roof failure depth of the gas tailgate is 2 m.

3.2. Width of Roadway Coal Side Plastic Zone. After the excavation of the roadway, the two sides of roadway were continuously damaged during the stress rebalance distribution of the surrounding rock until the boundary of the elastic stress zone, as shown in Figure 5. According to the theory of limit equilibrium zone, the width x_0 of the coal

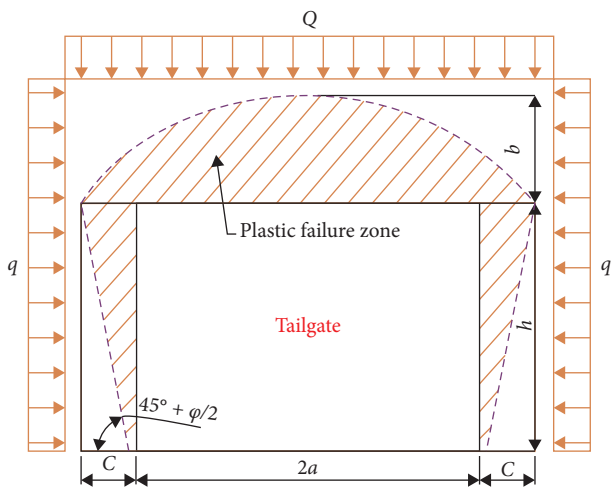


Figure 4: Calculation chart of surrounding rock damage range of roadway.

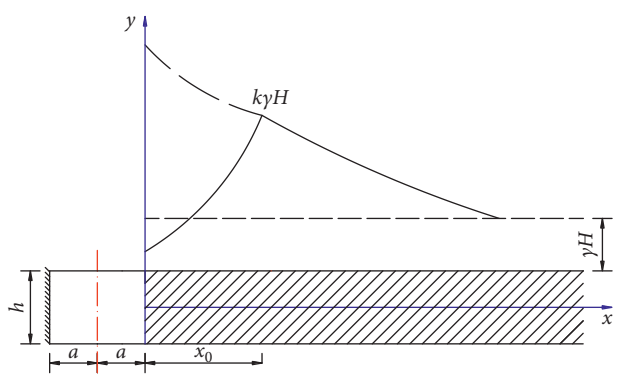


Figure 5: Calculation model of plastic zone width of coal seam.

tunnel plastic zone of the roadway can be obtained by the following formula:

$$x_0 = \frac{\lambda h}{2\tan\varphi_0}\ln\left(\frac{k\gamma h + (c_0/\tan\varphi_0)}{(c_0/\tan\varphi_0) + (p/\lambda)}\right), \tag{3}$$

where x_0 is the width of the limit equilibrium zone; h is the height of the roadway, 3 m; k is the stress concentration factor at the peak of the stress, 2.3; γ is the average bulk density of the overlying rock layer on the roadway, 25 kN/m^3; H is burial depth of the roadway 500 m; λ is the lateral pressure coefficient of the area; P is the resistance of the support body to the coal support; the two sides of the roadway are supported by bolts, taking 0.243 MPa; and internal friction angle at the coal seam interface is 27°:

$$\lambda = \frac{1-\sin\varphi}{1+\sin\varphi}. \tag{4}$$

After calculation, $x_0 = 4.78$ m.

4. Numerical Simulation

4.1. Significance of Deviatoric Stress and Establishment of Numerical Simulation. At present, the ground pressure theory of the roadway mostly focuses on the influence of the surrounding rock stability, which is caused by gravity stress and the abutment pressure. However, theoretical studies and engineering practice [18] have shown that the failure and deformation of the surrounding rock are not the result of a certain role of the vertical or horizontal stress, but the result of multiple stresses acting together. According to the geotechnical plasticity mechanics [13], the state of stress at any point in the surrounding rock of the roadway can be decomposed into the sum of the spherical and deviatoric stresses.

The state of stress at any point in the surrounding rock of the roadway can be expressed through multiple forms, and a special characteristic among all of the descriptions, namely, three principal stresses σ_i ($i = 1, 2, 3$) arranged in vertical pairs, is found:

$$\begin{bmatrix} \sigma_1 & 0 & 0 \\ 0 & \sigma_2 & 0 \\ 0 & 0 & \sigma_3 \end{bmatrix} = \begin{bmatrix} P & 0 & 0 \\ 0 & P & 0 \\ 0 & 0 & P \end{bmatrix} + \begin{bmatrix} \sigma_1 - P & 0 & 0 \\ 0 & \sigma_2 - P & 0 \\ 0 & 0 & \sigma_3 - P \end{bmatrix}, \tag{5}$$

where $\sigma_1 - P$ indicates the principal deviatoric stress, which is recorded as σ'. This is an essential source of the deformation and failure of the surrounding rock. In this study, σ' is used as an index to analyze the stability of the surrounding rock, and its value can be obtained through the following equation:

$$\sigma' = \sigma_1 - \frac{1}{3}(\sigma_1 + \sigma_2 + \sigma_3). \tag{6}$$

To solve the problem of a gas overrun in the coal face, a gas tailgate is equipped in the 310101 working face for gas drainage and also functions as an air return in the next working face, which indicates that the gas tailgate will be affected by severe mining during the processing of the 310101 working face. In addition, the surrounding rock of the roadway is soft, which will create a strong deformation and failure in the surrounding rock of the roadway. To better study the essential causes of a large deformation and roof caving of a gas tailgate under severe mining, FLAC^{3D} numerical software was used to simulate and analyze the evolution law of the deviatoric stress of the surrounding rock of the gas tailgate in the 310101 working face. The simulation object is working face 310101 of the Xinyuan coal mine, and the model size is 350 m × 200 m × 90 m, as shown in Figure 6. Mohr–Coulomb model is adopted for each stratum in the model, and strain softening model [23, 24] is adopted for soft coal and soft roof. The characteristics of strain softening of soft coal and soft roof are not the same in each mining area, so in order to carry out numerical calculation accurately, it is necessary to do specific experiments to obtain the characteristic parameters of strain softening of soft coal and soft roof and carry out numerical calculation verification. The characteristic parameters of strain softening of soft coal and soft roof are embedded in the fish statement and calculated in FLAC^{3D}. The lateral boundary of the model is fixed along the horizontal direction. The boundary of the floor of the model is fixed along the vertical direction.

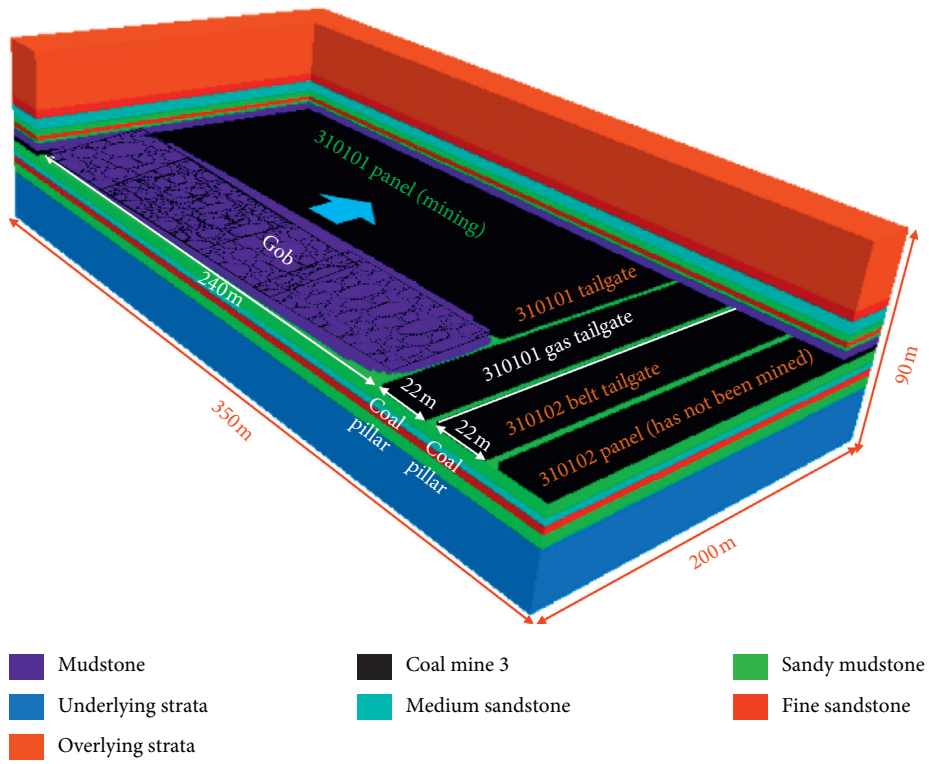


Figure 6: Numerical model diagram.

Moreover, the normal stress is applied at the top of the model. The average bulk density of the rock is 25 kN/m^3.

In order to verify the rationality of the strain softening model, a standard cylindrical numerical specimen with a height of 100 mm and a diameter of 50 mm was established in FLAC^{3D}. A constant velocity was applied on the top of the specimen, and the stress-strain curves of the standard test specimen of coal and roof mudstone in the simulation and laboratory test are shown in Figure 7. The results show that curves in laboratory test and simulation agree well, verifying the rationality of the strain softening model. The parameters of cohesion and friction angle of coal body and roof mudstone with plastic strain are determined through laboratory test (Table 1).

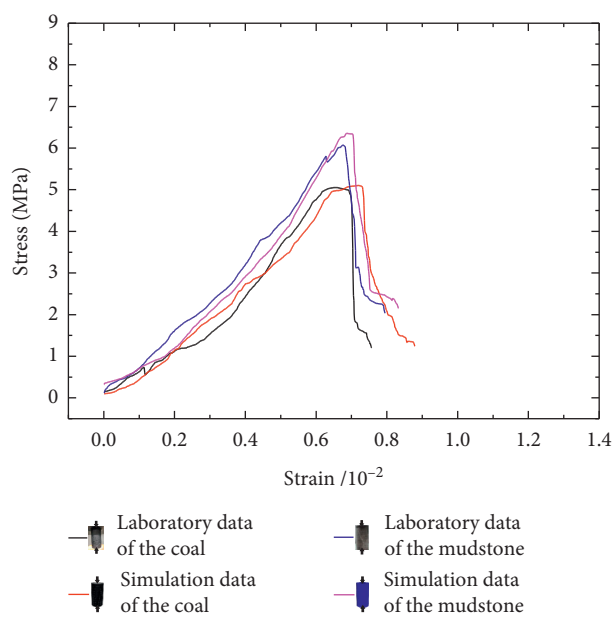


Figure 7: Stress-strain curves of the coal and mudstone.

4.2. Analysis of the Deflection of PDSZ in Gas Tailgate. The properties of the surrounding rock of the roadway deteriorate under the stress of a mining disturbance during the processing of the 310101 working face because both sides of the gas tailgate are soft and the bearing capacity of the retained coal pillars up to the roof is limited. As the working face continuously advances, a periodic fracture and caving are formed in the main roof strata above the stope. The fractured main roof touches the falling gangue in the gob and gradually begins to stabilize under the load-bearing effect of the gangue, whereas the stress of the surrounding rock of the roadway will reach a new static equilibrium under the combined action of the pillar support force and the stress of the mining disturbance [25–27]. The shallow surrounding rock of the gas tailgate undergoes dilatation and deformation under the excavation unloading. The deviatoric stress of the shallow surrounding rock is low and develops in the deep rock stratum. The deviatoric stress deflects until stabilizing, whereas the rock mass is weakened by the mining disturbance.

To better understand the deflection law of the PDSZ with the advance of the 310101 mining face, in this study, the deviatoric stress evolution law of surrounding rock of gas

Table 1: Parameters of cohesion and friction angle between coal and mudstone as a function of plastic strain.

	ε	C (MPa)	φ (°)
Coal	0	1	20.0
	0.0066	0.91	19.1
	0.0071	0.28	13.0
	0.0075	0.14	4.0
Mudstone	0	1.1	23
	0.0079	0.97	20.2
	0.0086	0.37	16
	0.0092	0.23	6

tailgate during the mining process is analyzed from two aspects. ① When the working face advances to a different distance, a comparative analysis of the deviatoric stress of the surrounding rock is conducted at the same position as the gas tailgate. ② A comparison of the deviatoric stress of the surrounding rock is also conducted under different positions of the gas tailgate when the working face advances to a certain distance. The evolution characteristics of the deviatoric stress field with the advancement of the working face are comprehensively obtained.

4.2.1. Nephogram of the Deviatoric Stress Distribution of the Same Cross-Section ($y = 80$ m) in the Gas Tailgate at Different Advancement Positions of the Working Face. From Figure 8, along the face advancing direction, the PDSZ in the surrounding rock of the gas tailgate deflects with the advancement of the working face. A deflection of the PDSZ in the surrounding rock of the gas tailgate occurs throughout the entire mining process. Before the working face advances 60 m, the distribution of the deviatoric stress in the surrounding rock of the roadway does not significantly change. The PDSZ is at the center of the roadway floor. The deviatoric stress field has a symmetrical annular distribution, and the value of the deviatoric stress is low within the area of 1.5–2 m near the surface of the roadway, whereas a high deviatoric stress distribution area is formed within a certain range of a deep roadway, which indicates that the surrounding rock of the roadway is slightly disturbed by the mining of the working face at this time, and the roadway remains consistently in a state of stability. When the working face advances 60–140 m, the distribution of high deviatoric stress in the surrounding rock of the roadway changes, and the high deviatoric stress of the roof necks at the shoulders of both sides, transforming from an annular stress area to a U-shaped high deviatoric stress area. The degree of deflection of the PDSZ of the roadway surrounding the rock increases with the distance of the coalface advancement, whereas the degree of deflection of the peak deviatoric zone of a coal pillar wall no longer changes during the 100–140 m mining process; in addition, the PDSZ gradually appears on the side of the solid coal, and the distribution range of the peak stress zone continues to deflect vertically downward with the advancement of the working face. When the mining face reaches 140 m, an asymmetric "sandwich" peak in the deviatoric stress zone is formed on both sides of the roadway and the variation of the peak stress zone tends to be stable. The peak in the deviatoric stress zone at the gob side of the roadway is located within the range of 3–5.5 m from the surface of the coal pillar with a large distribution, whereas the coal wall side is mainly located within the range of 3–4.5 m at the shoulder and bottom corners of the solid coal, and the shoulder angle distribution range is larger than that of the bottom corner. As the working face continues to advance, the distribution form of the peak in the deviatoric stress zone of the surrounding rock on both sides of the roadway no longer changes significantly. Only a strip of an increased deviatoric stress zone is formed on the upper part of the coal pillar, which obliquely runs the entire width. That is, the deviatoric stress field of the surrounding rock of the gas tailgate basically reaches a state of stability when the advancement distance of the working face is 140 m because the outer side of the coal pillar is under unidirectional or bidirectional compression, and the mechanical property of the coal pillar is poor. Under roof subsidence, a plastic failure easily occurs, shifting the high deviatoric stress of the coal pillar side to the solid coal side and allowing the deviatoric stress of the surrounding rock of the roadway to reach a new static level under the combined action of the support force of the coal pillar and the disturbance stress of the mining.

4.2.2. Nephogram of the Deviatoric Stress Distribution of the Surrounding Rock at Different Positions of the Gas Tailgate When the Working Face is at a Certain Position. Considering the deviatoric stress distribution nephogram of the surrounding rock at different positions of the gas tailgate when the working face is at a certain position ($y = 40$, 80, and 160 m), the typical characteristics of the deviatoric stress field at different evolution stages are obtained, as shown in Figure 9. When the working face progresses at 40 m, as shown in Figure 9(a), the distribution of the deviatoric stress field in the surrounding rock of the roadway does not change, and only the deviatoric stress in the coal pillar expands horizontally toward the direction of the goaf with an increase in the lagging mining distance; however, the deviatoric stress distribution form in a coal pillar of the advanced mining remains almost unchanged. This indicates that the 22 m pillar has a certain bearing capacity for the roof during the initial stage of the working face mining, while effectively weakening the advancing influence of the working face mining on the gas tailgate. When progressing at 80 m, as shown in Figure 9(b), the PDSZ of the surrounding rock in the gas tailgate shows an obvious advancement distribution, whereas the deflection trend stops at 20 m ahead of the working face, and the deviatoric stress distribution remains unchanged as the advancement distance continues to increase, which indicates that the degree of disturbance of the surrounding rock in the gas tailgate is gradually weakened by the mining in advance of the working face and tends to maintain the stability of the initial excavation. Similar to the influence of the advanced mining of the working face, the PDSZ shows an obvious deflection phenomenon behind the working face. The degree of deflection and range of distribution of the PDSZ increase with an increase in the lag

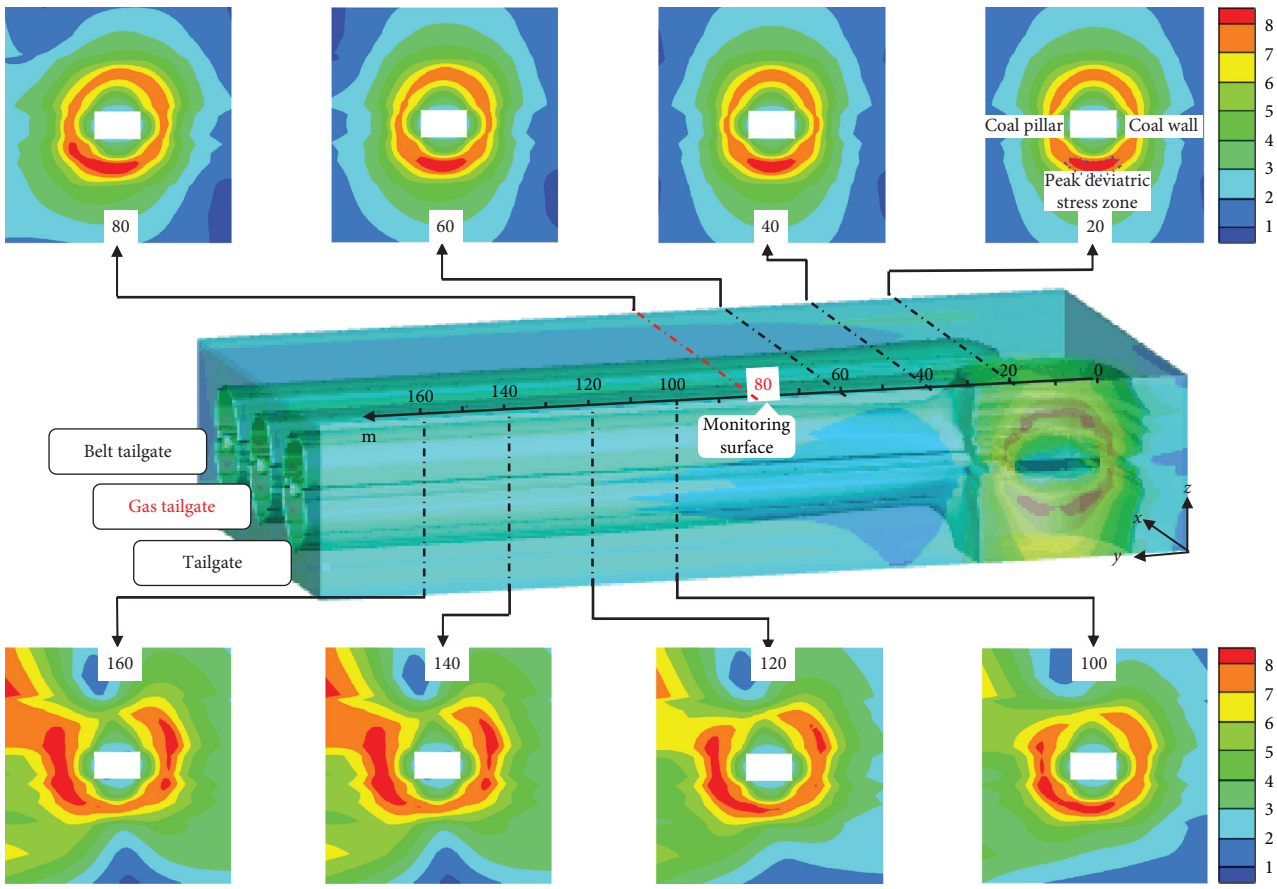
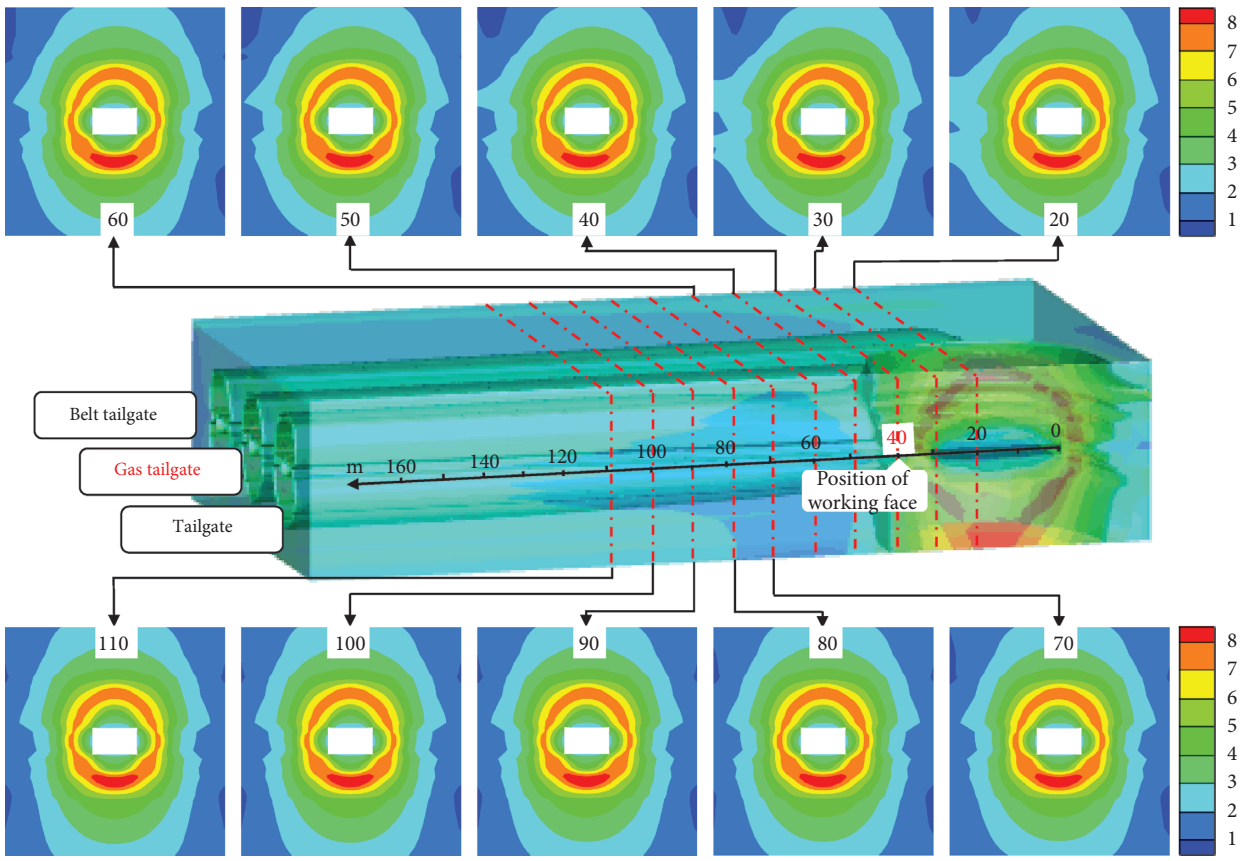


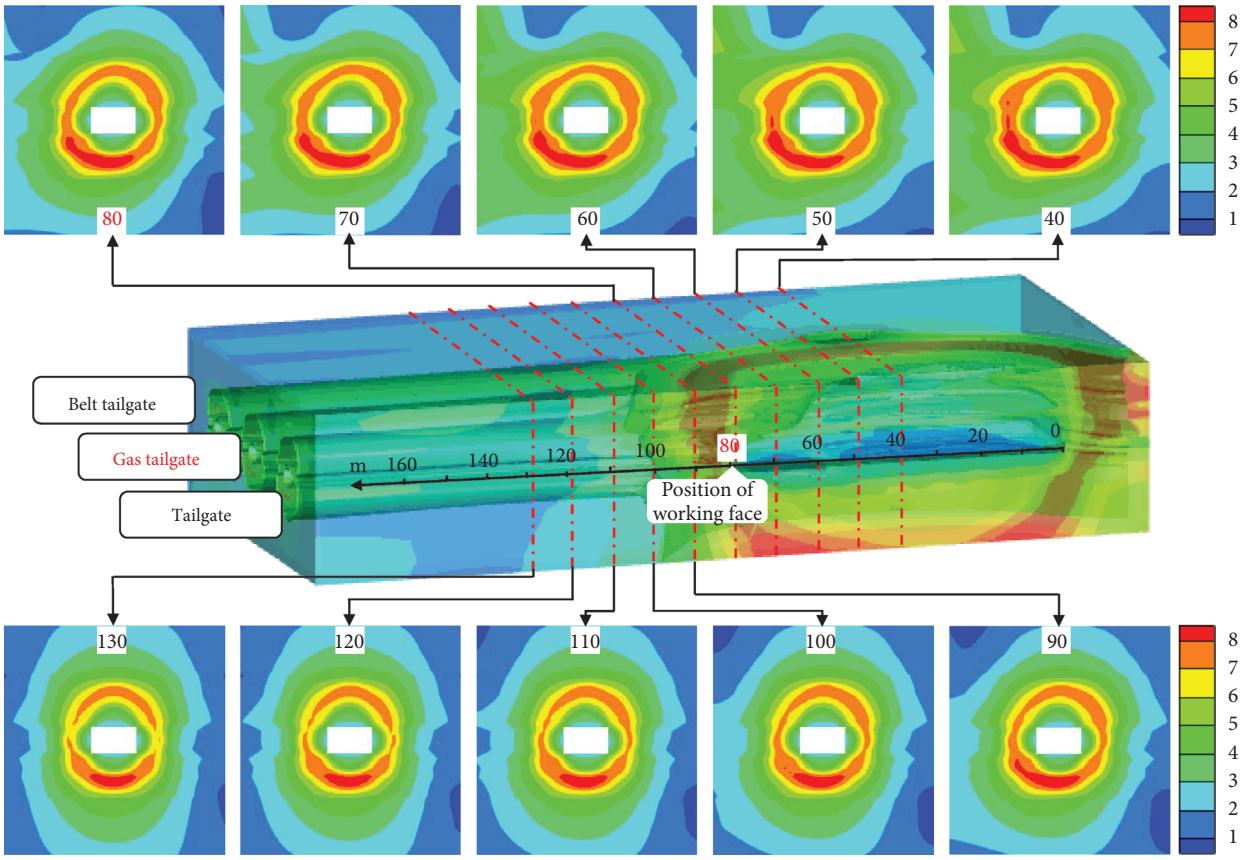
Figure 8: Deviatoric stress distribution of surrounding rock with advancement distance of the working face ($Y = 80$ m; unit: MPa).

mining distance, the concentration of deviatoric stress gradually appears in the shoulder of the coal pillar in the gas tailgate, and the degree of stress concentration increases with the distance from the working face. As shown in Figure 9(c), the distribution form of the deviatoric stress field in the surrounding rock of the roadway changes dramatically under the influence of lag mining pressure after mining. The lateral side of the high stress zone at the shoulder corner of the roadway continues until it connects with the high stress zone at the goaf side. The range of distribution of the PDSZ in the surrounding rock of the gas tailgate increases significantly and transfers with an increase in the lag mining distance, although there is no significant change in the PDSZ of the goaf side behind the working face. Only the PDSZ of the solid coal side transfers with the increase in the lag mining. When the lag mining is at 60 m, an asymmetric "sandwich" peak in the deviatoric stress zone is formed on both sides of the roadway. This indicates that the bearing capacity of the coal pillar up to the roof is limited and that the deviatoric stress of the coal pillar reaches the ultimate bearing state after mining of the working face. With an increase in the lag mining distance, the distribution form of the PDSZ no longer changes. Only the high deviatoric stress zone at the shoulder corner of the coal pillar necks with an increase in the lag mining distance, which indicates that the deviatoric stress field of the surrounding rock reaches a stable state at a 60 m lag from the mining face.

From the above analysis, we can see the following. (1) Considering the deviatoric stress distribution nephogram of the same cross-section ($y = 80$ m) in the gas tailgate at different advancement positions of the working face, the PDSZ of the surrounding rock in the gas tailgate rotates clockwise with the increase in the advancing distance of working face, and the deflection of the deviatoric stress peak zone on both sides of the roadway tends to be stable when the advancing distance of the working face is 140 m. Among them, the PDSZ at the gob side of the roadway is located within the range of 3–5.5 m from the surface of the coal pillar with a large distribution, whereas the coal wall side is mainly located within the range of 3–4.5 m at the shoulder and bottom corners of the solid coal, and the distribution range of the shoulder corner is larger than that of the bottom corner. (2) Considering the deviatoric stress distribution nephogram of the surrounding rock at different positions of the gas tailgate when the mining face is at a certain position ($y = 40$ m, 80 m, and 160 m), affected by the intense mining of the working face, the PDSZ of the surrounding rock of the roadway shows an obvious advancement and lagging distribution; in addition, the deflection trend of the PDSZ stops at 20 m ahead of the working face, whereas it stops at a 60 m lag of

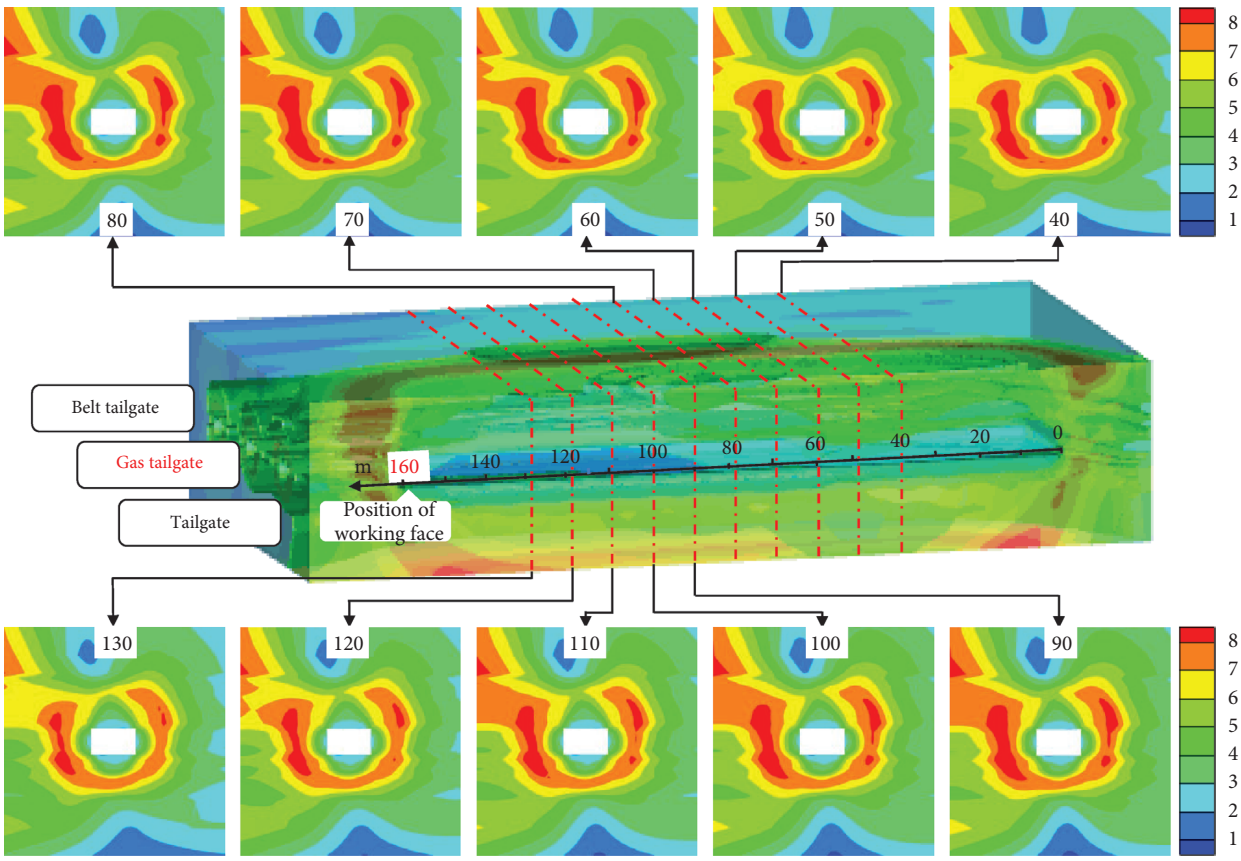


(a)



(b)

Figure 9: Continued.



(c)

Figure 9: Deviatoric stress distribution of surrounding rock of gas tailgate when the working face is at a certain position. (a) 40 m advancement of the working face. (b) 80 m advancement of the working face. (c) 160 m advancement of the working face (unit: MPa).

the mining face, and the distribution form of the PDSZ on both sides of the roadway no longer changes to a significant degree.

4.3. Analysis of the Evolution of Plastic Zone in Gas Tailgate. After the influence of the mining of the 310101 working face on the gas tailgate, the deviatoric stress field of the surrounding rock of the roadway will tend to deflect toward the goaf along with the mining in the working face. Under a high deviatoric stress, the surrounding rock of the roadway will produce a certain range of the plastic zone, and the deflection of the deviatoric stress field will directly affect the expansion and development of the plastic zone of the surrounding rock. A plastic failure of the surrounding rock forms in the middle section ($y = 80$ m) of the gas tailgate along the advancing direction of the working face at different mining distances, as shown in Figure 10(a). It can be seen that the main failures of the surrounding rock at the bottom corner of the coal pillar and the shallow shoulder corner of the coal wall are tensile and shear failures, whereas the other parts of the rock mass are mainly subjected to a single shear failure. The plastic failure zone of the surrounding rock of the roadway is small and uniform with an elliptical distribution at the initial stage of mining. With the advancement of the working face, the damage range of the plastic zone is evidently not coordinated with the development. When the advancing distance of the working face is 140 m, the range of change of the plastic zone reaches the maximum value and tends to reach stability. The distribution form of the plastic zone changes from elliptic to irregular, whereas the depths of the plastic zone of the roof and floor remain unchanged throughout the mining process, and the rock mass in the rest of the surrounding rock of the roadway shows an obvious nonuniform failure. Statistical curves of the plastic failure rate of the surrounding rock of the roadway varying with the mining distance of the working face are shown in Figure 10(b). The ratio of the change in the range of the plastic zone and the initial range is defined as the failure rate coefficient, which is recorded as η.

According to the failure coefficient curve of the surrounding rock of the roadway, the failure of the plastic zone of the surrounding rock of the gas tailgate can be roughly divided into three categories under intense mining: the failure rate of the shoulder corner of the roadway and the bottom corner of coal pillar is the fastest, the failure rates of the two sides and the bottom corner of the coal wall are the second fastest, and the failure rates of the roof and floor are the slowest. For a type-I failure rate curve, the failure rates of the left-bottom and right shoulder corners are much higher

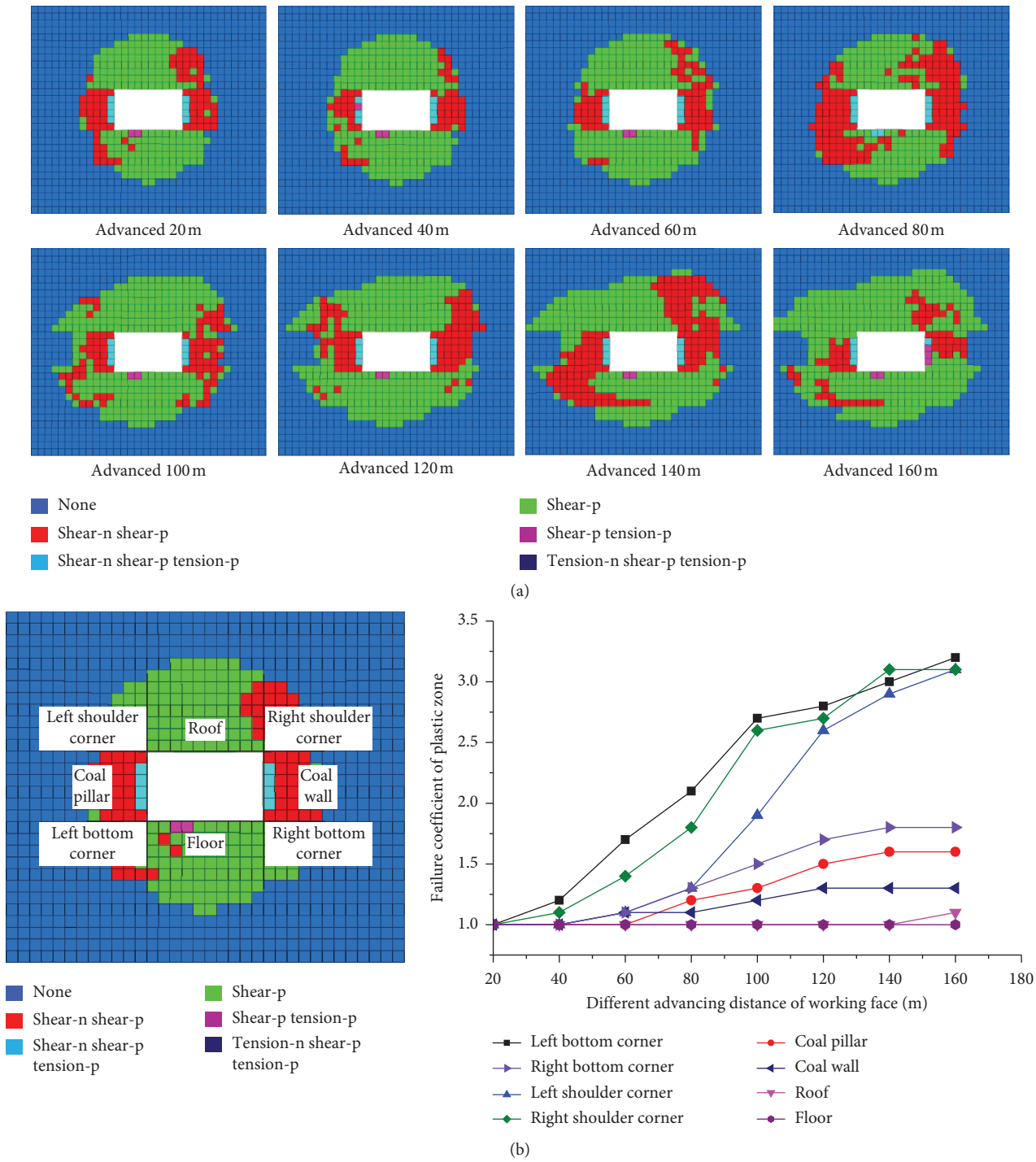


Figure 10: Evolution distribution shapes of plastic zone with advancement of the different working face. (a) Plastic failure form and (b) statistical curves of the plastic failure rate.

than that of the left shoulder corner at the initial stage of mining. However, with the advancement of the working face, the failure rate of the left shoulder corner increases significantly, whereas the failure rates of the left-bottom and right shoulder corners tend to be flat. Finally, the failure rate curves of type-I tend to coincide, where the failure depth of the left-bottom corner is 4 m, that of the left shoulder corner is 5 m, and that of the right shoulder corner is 4.5 m. For type-II failure rate curves, the failure rate of the plastic zone shows a radial distribution with the advancement of the working face, and the failure depth occurs in order of the right bottom corner (3.75 m) > pillar side, (3.5 m) > solid coal (3 m). For the type-III failure rate curves, the failure rate of the plastic zone of the roof and floor shows little change with an increase in the advancing distance during the entire mining process; however, the failure depths of the roof and

floor are as high as 4 m, which means the failure of the plastic zone of the roof and floor is mainly related to the property of the surrounding rock of the roadway.

According to the above analysis and drilling peep [28], the failure rate of the plastic zone and the deflection law of the PDSZ maintain a synchronous and coordinated relationship with the advancement of the working face. The failure range of the plastic zone is stable with the stability of the PDSZ, which indicates that the PDSZ dominates the growth rate and development of the plastic zone. The relationship between the distribution of the deviatoric stress and the range of the plastic zone is more intuitive, as shown in Figure 11. The maximum depth of the plastic failure of the surrounding rock of the gas tailgate is 5 m, which is less than the outline of the PDSZ (5.5 m), but larger than the inner outline of the PDSZ (3 m). The plastic failure line remains between the inner and outer outlines of the PDSZ. A plastic failure occurs in the rock mass within the inner edge of the PDSZ and extends outward to the plastic failure line. Therefore, the rock mass in the range of layered deviatoric stress isosurface within and outside the PDSZ is in the state of elastic-plastic deformation, and the rock mass is in the transitional state of stability and instability. To achieve the stability of the surrounding rock of the gas tailgate, it is necessary to control the stability of the unstable rock mass.

5. Control Mechanism and Technology of Surrounding Soft Coal Roadway under Intense Mining

The above analysis shows that the gas tailgate in the 310101 working face belongs to a soft coal roadway, with a developed crevice and low strength of the surrounding rock, which are vulnerable to deformation and destroyed by disturbances during an engineering project. It is inevitable that a large range of fragmentation zones and plastic zones of the surrounding rock will be formed within a short time after excavation of the roadway. In addition, the PDSZ of the surrounding rock of the roadway deflects to the side of the goaf with an increase in the advancing distance of the working face, and the maximum depth of the plastic failure of the surrounding rock of the roadway is approximately 5 m under a high deviatoric stress. However, the strength and length of the anchor bolt (cable) in the original support are too low to ensure that the anchorage foundation of each part of the anchor bolt (cable) is within the elastic zone. The broken surrounding rock is strongly extruded toward the free surface, resulting in a large convergence deformation of the surrounding rock, and even a roof caving and collapse. The support system is seriously damaged, which results in an instability of the entire section of the roadway and its continued use, as is shown in Figure 12. Therefore, a comprehensive control technology that employs a truss structure of the roof with a strong bolt system (Figure 13(a)), a multianchor cable-steel beam system (Figure 13(b)), a high prestressed anchor cable truss system, (Figure 13(c)), and a coal side anchor cable-channel steel truss system (Figure 14) is proposed.

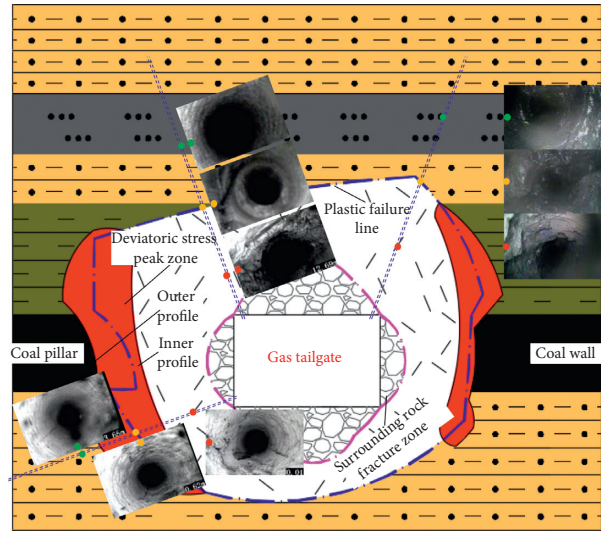


Figure 11: Contrast diagram of relationship between peak deviating stress band and plastic zone.

5.1. Roof High Prestressed Strong Supporting Technology. A surrounding rock fractured zone is formed in the shallow part of the gas tailgate of the 310101 working face when affected by the mining and roadway excavation. A strong bolt control system has been successfully applied under many complex conditions, such as multiple mining effects, soft and fractured surrounding rock, and a large span of roadway [9, 29–31]. Super strong thread steel bolts with a superior mechanical property, high prestressing, and a timely support can significantly improve the residual strength of the broken surrounding rock and form a large effective compressive stress zone in the roof, indirectly restraining the expansion of the plastic and broken zones into the deep roof. Consequently, this significantly improves the integrity and overall load-bearing capacity of the shallow surrounding rock of the roadway and promotes the formation of a prestressed load-bearing structure composed of a support and surrounding rock, as shown in Figure 13(a). Thus, based on engineering practice and a numerical simulation, super strong thread steel bolts of $\Phi = 20\,\text{mm} \times 2{,}500\,\text{mm}$ are used, with seven bolts arranged in each row. Considering that the plastic zone of the shoulder angle of the roadway has the fastest failure rate, the distance between two bolts near the two sides of the roadway is 700 mm, and the distance between the other bolts is 800 mm. The bolts for each of the sidewalls of the roadway are set at 200 mm away from the sidewalls, the angle between the bolt for each of the sidewalls of the roadway and the vertical direction is 20°, and the other bolts are arranged vertically. The line space of the strong thread steel bolts is determined to be 800 mm, and the diamond metal mesh, specified as $1{,}100\,\text{mm} \times 2{,}800\,\text{mm}$ in size, is used along with a Φ 14 mm steel ladder beam and a steel bearing plate.

A multianchor cable-steel beam system is the key to ensuring the stability of the roof. It consists of W-steel tape and several single anchor cables connected with the W-steel tape and anchored to the depth of the roof, forming a prestressed

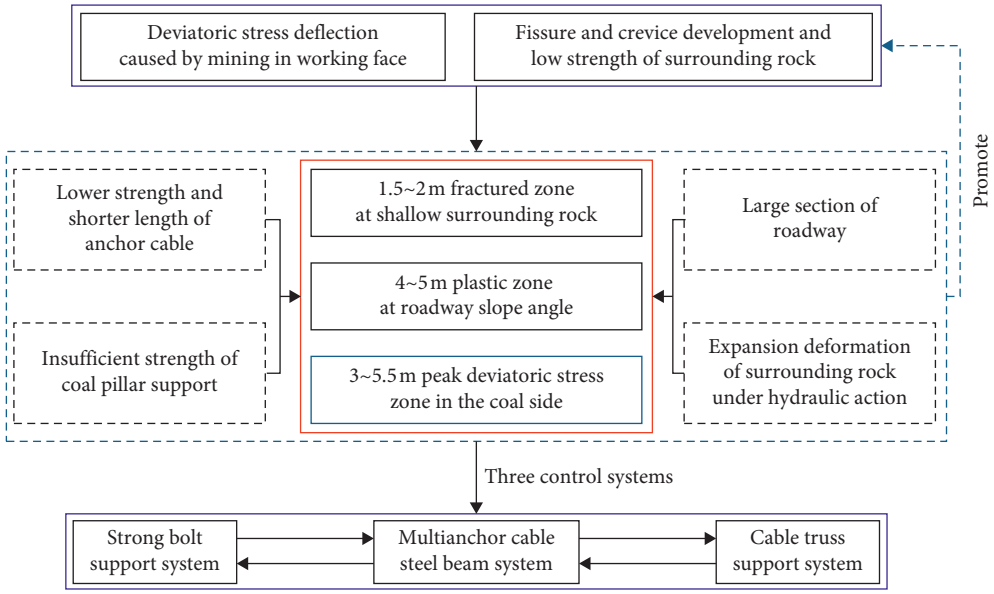


Figure 12: Roadway instability mechanism and control system.

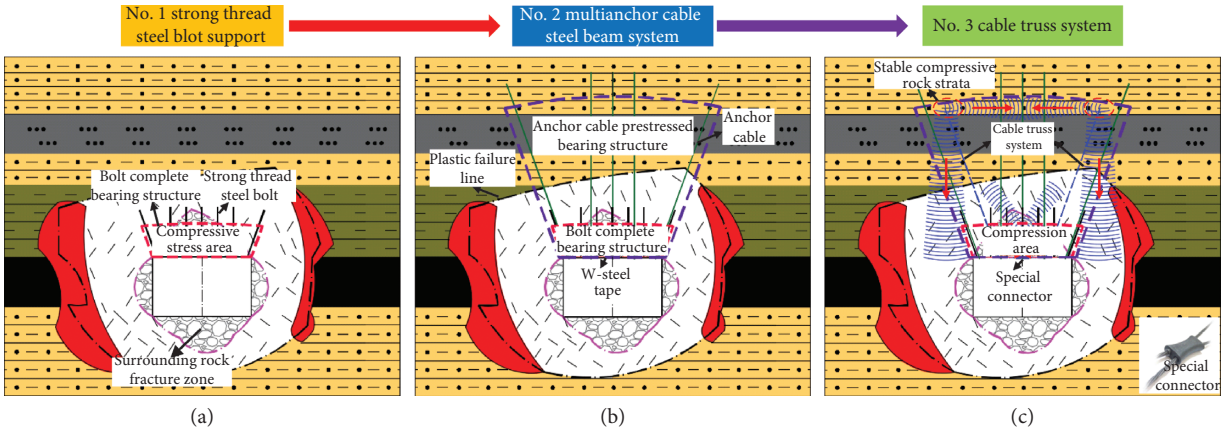


Figure 13: Principle of roof highly prestressed strong supporting roof system. (a) Strong bolt system. (b) Multianchor cable-steel beam system. (c) High prestressed anchor cable truss system.

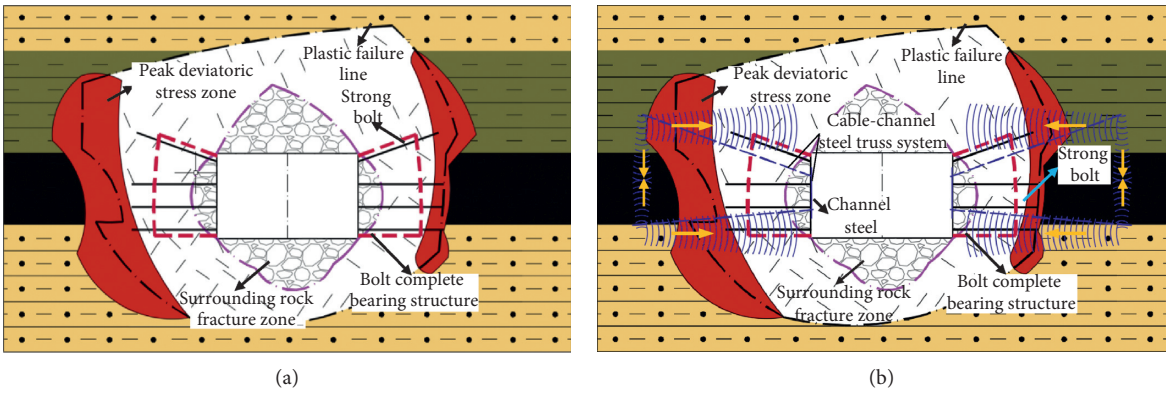


Figure 14: Principle of cable-channel steel truss system with coal side anchor.

bearing structure of multianchor cables in a steel beam support. The single anchor cable crosses the plastic failure line and connects the prestressed bearing structure formed by the shallow bolt support with a deep stability of the rock stratum of the roof. An inclined arrangement of the anchor cables near the side of the shoulder is adopted to weaken the expansion

rate of the plastic zone at the shoulder corner of the roadway. The anchor cables are joined by W-steel tape with a high tensile strength to form an integral bearing structure and jointly resist the deformation of the surrounding rock and improve the stability of the prestressed bearing structure. In the design of the anchor cable parameters, the anchor cable should cross the failure line of the plastic zone, and the anchor cable foundation should be located as far as possible above the deeply stable rock stratum of the roof, as shown in Figure 13(b). Thus, based on engineering practice, a numerical simulation, and a theoretical analysis, steel strand cables composed of 19 steel wires, specified as $\Phi = 17.8\,\text{mm} \times \text{L}9500\,\text{mm}$ in size, are used to strengthen the support, with five cables arranged in each row. The row and line spacing of the steel strand cables is determined to be $1{,}100\,\text{mm} \times 2{,}400\,\text{mm}$, and the angle between the cables, set at 300 mm away from the sidewalls, and the vertical direction is 20°. The W-steel tape, specified as $5{,}000\,\text{mm} \times 220\,\text{mm} \times 4\,\text{mm}$ in size, is used for linking the cables and operates in coordination with the steel bearing plate.

Owing to the large span of the roadway and the weak mudstone in the roof, a roof caving is prone to occur under intense mining. Therefore, it is necessary to take effective measures to prevent a roof caving based on the above support to ensure the safety of the roadway roof. The cable truss system is a new support structure developed to solve the problem of a large deformation and easy roof caving of the roadway under complex geological conditions [32]. The cable truss system consists of a long anchor cable and a special locking device. The anchor foundation is placed in deeply stable compressive rock strata in the two shoulders of the roadway. The two inclined anchor cables are connected by a special connector and form an outward inclined groove in the active support structure after prestressing, as shown in Figure 13(c). The prestressed cable truss system exerts a high pretension from the horizontal and vertical directions, realizing a three-dimensional compression of the surrounding rock. During the process of roof subsidence of the roadway, the cable truss system is gradually blocked, the pressure of the shallow surrounding rock is strengthened, and the support and surrounding rock are jointly loaded, preventing a large deformation of the surrounding rock of the roadway and providing a new means for the support of a large deformation roadway. Thus, based on engineering practice, a numerical simulation, and a theoretical analysis, a high-strength and highly elongated prestressed steel strand composed of 19 high-strength steel wires of $\Phi\,17.8 \times 9{,}500\,\text{mm}$ in size is adopted for the cable truss system, the pretightening force of which can reach no lower than 160 kN. The angle between the cables, which is set at 1,300 mm from the sidewalls, and the vertical direction is 20°. The cable truss system is connected by a lock matched with 17.8 mm steel wires and a special cable truss connector.

5.2. Control Technology of Cable-Channel Steel Truss of Coal Side Anchor. Because the outer side of the coal pillar is under unidirectional compression and the coal body is poor, a plastic failure easily occurs under roof subsidence. Therefore, a strong bolt support system should still be used on both sides of the gas tailgate. Considering the weak and broken deformation of the coal body at the side and to further control the expansion deformation of the surrounding rock of the crevices, a 3,000 mm length bolt is selected to form a large bolt support structure. Consequently, this significantly improves the stress of the surrounding rock within the peak deviator stress zone, delaying the transfer of a high deviator stress to the side of the solid coal as far as possible and weakening the shear failure from a high deviator stress of the coal mass, as shown in Figure 14. Thus, based on engineering practice, a numerical simulation, and a theoretical analysis, super strong thread steel bolts of $\Phi 20\,\text{mm} \times 3{,}000\,\text{mm}$ in size are used, with a row and line spacing of $800\,\text{mm} \times 800\,\text{mm}$. The angle between the bolt, which is set at 300 mm away from the roof, and the vertical direction is 20°. The diamond metal mesh, specified as $2{,}600 \times 1{,}100\,\text{mm}$ in size, is used along with steel and wooden bearing plates.

According to the numerical simulation, it can be seen that the PDSZ gradually deflects toward the coal pillar side and the solid coal side with an advancement of the working face, and the rock mass in the PDSZ is in a transition between stability and instability. To realize the stability of the surrounding rock of the roadway, it is necessary to control the stability of the unstable rock mass in the PDSZ. Therefore, a truss anchor cable is used to strengthen the support of the roadway side. The shear deformation of the surrounding rock at the corner of the roadway can be effectively controlled by an inclined anchor cable crossing the PDSZ. The cable truss system is composed of anchor cables connected by channel steel, which can form a continuous compressive stress zone on the shallow surrounding rock of the roadway side. This changes the support mode and structure and enhances the self-supporting ability of the coal mass on the roadway side. Thus, based on engineering practice, a numerical simulation, and a theoretical analysis, a high-strength and highly elongated prestressed steel strand composed of 19 high-strength steel wires of $\Phi 17.8 \times 5{,}300\,\text{mm}$ in size and a 1.8 m long number 14 steel channel is adopted for the cable-steel channel truss system. The angle between the cables, which is set at 800 mm away from the roof, and the vertical direction is 20°, and the angle between the cables, which is set at 1,000 mm away from the floor, and the vertical direction is 5°. The line spacing of the steel strand cables is determined to be 2,400 mm.

6. Engineering Practice

6.1. Support Parameters. According to the above theoretical analysis, numerical simulation, and engineering practice, the parameters of the gas tailgate in the 310101 working face are determined, as shown in Figure 15.

The actual roof support design was as follows: super strong thread steel bolts of $\Phi\,20\,\text{mm} \times 2{,}500\,\text{mm}$ in are used, with seven bolts arranged in each row. The line spacing of the strong thread steel bolts is determined to

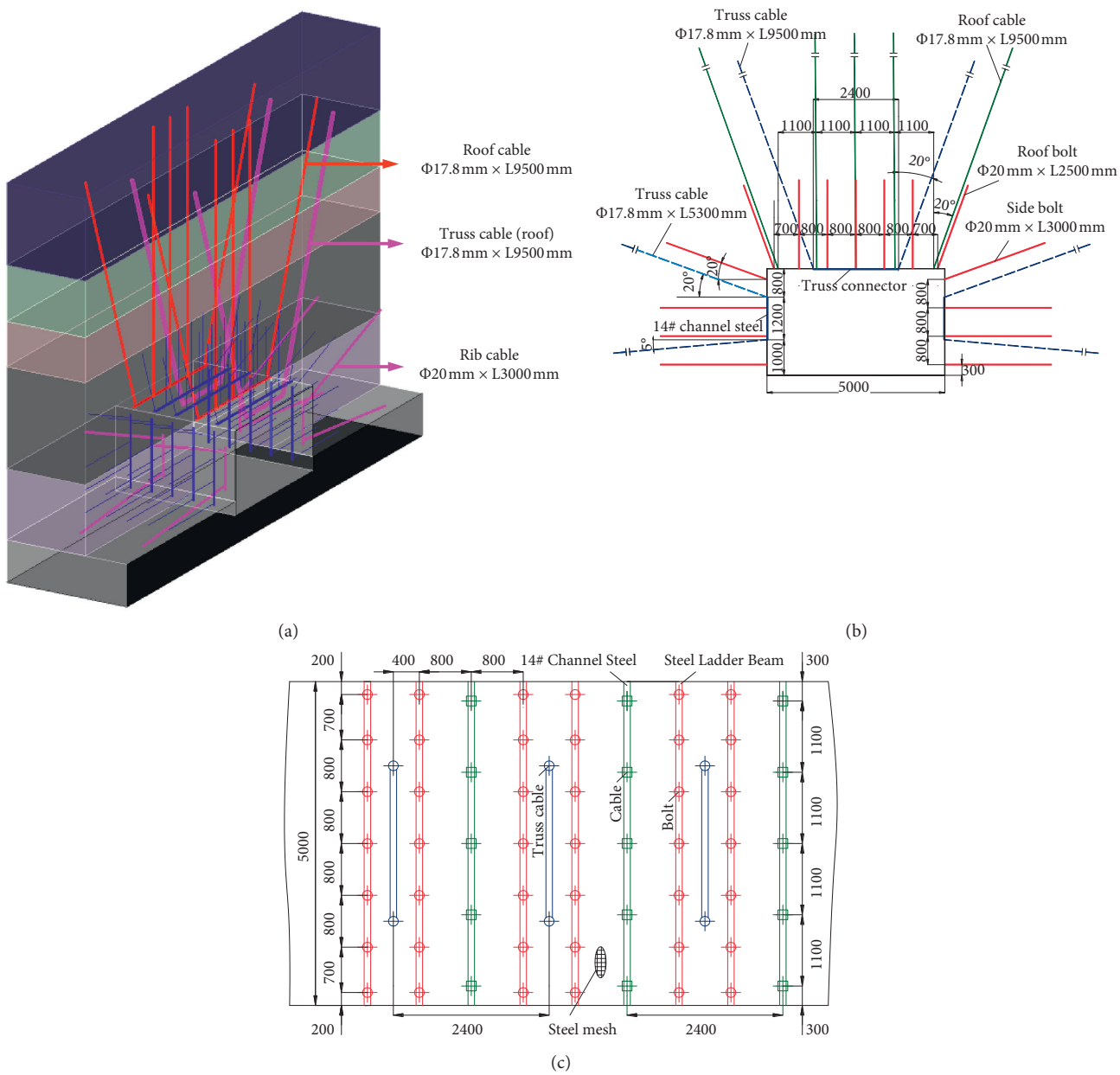


Figure 15: Support diagram of gas tailgate (unit: mm).

be 800 mm, and a diamond metal mesh specified as 1,100 mm × 2,800 mm in size is used along with a Φ 14 mm steel ladder beam and a steel bearing plate. Steel strand cables composed of 19 steel wires specified as Φ 17.8 mm × L9500 mm in size are then used to strengthen the support, with five cables arranged in each row. The row and line spacing of the steel strand cables is determined to be 1,100 mm × 2,400 mm. The W-steel tape specified as 5,000 mm × 220 mm × 4 mm in size is used for linking the cables and operates in coordination with the steel bearing plate. In addition, a high-strength and high-elongation prestressed steel strand composed of 19 high-strength steel wires of Φ17.8 × 9,500 mm in size is adopted for the cable truss system, the pretightening force of which can be no lower than 160 kN. The cable truss system is connected by a lock matched with 17.8 mm steel wires and a special cable truss connector.

For the coal side support, super strong thread steel bolts of Φ 20 mm × 3,000 mm in size are used, with four bolts arranged in each row. The row and line spacing of the strong thread steel bolts is determined to be 800 mm × 800 mm. In addition, an anchor cable-channel steel truss support system is adopted with a high-strength elongation prestressed anchor cable specified as Φ 17.8 mm × 5,300 mm in size using 1.8 m long number 14 channel steel, and the row space of the anchor cables is determined to be 2,400 mm.

6.2. Support Effect. To further test the control effect of the surrounding rock of the gas tailgate under intense mining,

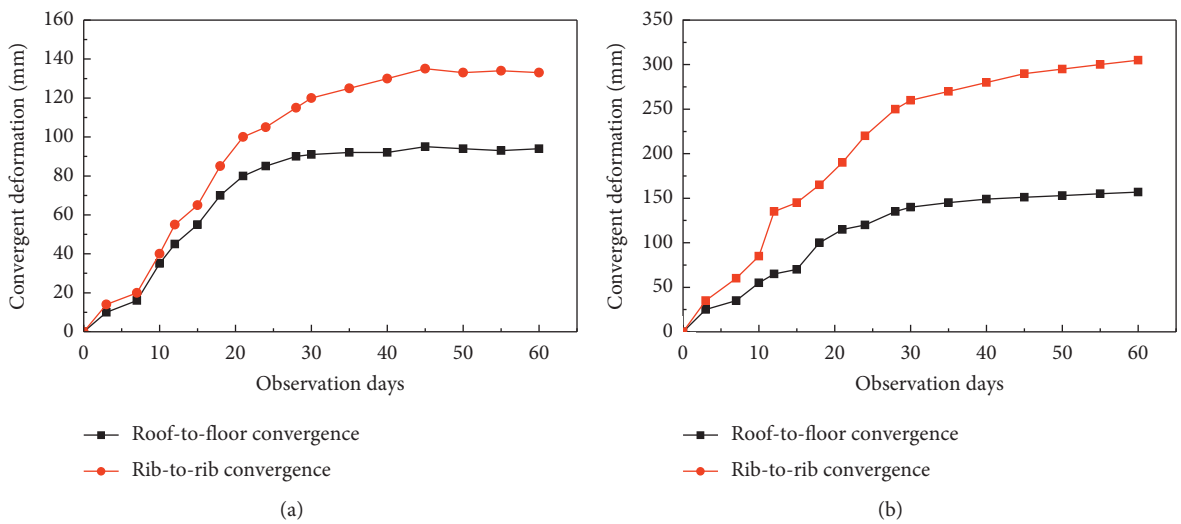


Figure 16: Convergence of surrounding rock of gas tailgate, (a) before and (b) after mining.

the surface displacement of the gas tailgate is monitored using cross point methods; the monitoring data before and after mining are shown in Figure 16. Before mining, the convergence of the surrounding rock of the gas tailgate is extremely small, the maximum displacement of the roof to floor of the gas tailgate is no more than 100 mm, and the maximum displacement of the two sides of gas tailgate is no more than 140 mm. After mining, the maximum displacement of the roof to floor of the gas tailgate is no more than 310 mm, the maximum displacement of the two sides of the gas tailgate is no more than 160 mm, the roof separation of the gas tailgate is 15 mm, and the roof is in a stable state.

After adopting the above-mentioned combined control technology, the problem of a large-area roof caving of the mining face has been effectively solved. The effective control of the gas tailgate not only provides the basic conditions for ventilation and gas control, but can also help excavate a smaller section of roadway, which resolves the situation of the excavation seriously replacing the out-of-balance state, reducing the loss of exploitable resources of the coal mine and significantly increasing the economic and technical benefits of the enterprise. The output of working face is increased from 1.2 to 1.8 Mt/a, which effectively solves the difficult problem of coal resource mining under severe conditions and realizes the safe and efficient production of the coal mine.

7. Conclusion

In this paper, it is concluded that the peak stress of the partial stress of the weak surrounding rock of the gas tailgate under the influence of strong mining will be significantly deflected, and the plastic zone will expand nonuniformly. The key control area of the anchor cable and the anchorage range of the anchor cable were verified by theoretical calculations, drilling peeping, and field practice.

(1) The strain softening model is used to numerically calculate that the strong mining influence on the working face will cause the peak stress zone of the surrounding rock of the gas tailgate to deflect. The PDSZ at the gob side of the roadway is located in the range of 3–5.5 m from the surface of the coal pillar, while the coal wall side is mainly located in the range of 3–4.5 m at the shoulder corner and bottom corner of the solid coal. The peak area of the partial stress is the key control area.

(2) Strong mining in the working face causes the plastic area of the surrounding rock of the gas tailgate to expand nonuniformly. The solid coal shoulder angle, coal pillar side shoulder angle, and coal pillar side bottom angle have the largest deformation and damage and the damage ranges are 4 m, 4.5 m, and 5 m, respectively. These three areas are the focus of support control.

(3) The plastic failure line of the surrounding rock of the gas tailgate is always between the inner and outer contours of the peak stress zone, and the rock mass in the peak stress zone is in a stable and unstable transition state. The support range of the anchor cable should cross the plastic failure line of the surrounding rock of the gas tailgate.

(4) A truss anchor cable support system with integrated locking and large-scale support function is proposed to jointly control the roof and the two sides of the surrounding rock of the weak roadway in strong mining. The maximum deformations of the roadway and the two sides of the roadway are 310 mm and 160 mm, respectively, and the control effect is good.

Data Availability

All data supporting the conclusions draw by this study can be obtained from the corresponding author upon request.

Conflicts of Interest

The authors declare that there are no conflicts of interest.

Acknowledgments

This work was financially supported by the China Postdoctoral Science Foundation (Grant no. 2019M650895), the National Natural Science Foundation of China (Grant no. 51974317), the Fundamental Research Funds for the Central Universities (Grant no. 2010QZ06), and the Yue Qi Young Scholar Project, China University of Mining & Technology, Beijing (Grant no. 800015Z1104). The authors would like to thank the continuous support from the Xinyuan coal mine during the on-site sampling and monitoring.

References

[1] M. G. Qian and P. W. Shi, *Rock Pressure and Strata Control*, China University of Mining And Technology Press, Xuzhou, China, 2010.

[2] H. P. Kang, L. X. Yan, X. P. Guo, Z. T. Zhang, and F. Q. Gao, "Characteristics of surrounding rock deformation and reinforcement technology of retained entry in working face with multi-entry layout," *Chinese Journal of Rock Mechanics and Engineering*, vol. 31, no. 10, pp. 2022–2036, 2012.

[3] W. J. Wang, Y. Yuan, W. J. Yu, and X. Chen, "Failure mechanism of the subinclined shaft in floor under mining influence and its control," *Journal of China Coal Society*, vol. 39, no. 8, pp. 1463–1472, 2014.

[4] S. G. Li, X. Y. Cheng, C. Liu, T. J. Zhang, L. J. Xing, and G. J. Wang, "Research on technology of anchor cable supporting and grouting reinforcement for dynamic pressurized tunnel with crushed surrounding rock," *Coal Science and Technology*, vol. 44, no. 1, pp. 67–72, 2016.

[5] H. L. Zhang, L. G. Wang, M. Tu, and Y. G. Zhang, "Study on failure evolution laws and control technology of roadway surrounding rock under mining circumstances," *Journal of Mining and Safety Engineering*, vol. 30, no. 5, pp. 653–664, 2013.

[6] S. Y. Chen, C. S. Song, Z. B. Guo, J. Wang, and Y. Wang, "Asymmetric deformation mechanical mechanism and control countermeasure for deep roadway affected by mining," *Journal of China Coal Society*, vol. 41, no. 1, pp. 246–254, 2016.

[7] G. Zhang and F. He, "Asymmetric failure and control measures of large cross-section entry roof with strong mining disturbance and fully-mechanized caving mining," *Chinese Journal of Rock Mechanics and Engineering*, vol. 35, no. 6, pp. 806–818, 2016.

[8] F. L. He and G. C. Zhang, "Deformation and failure mechanism and control technology of large section coal roadway subjected to severe mining dynamic load," *Journal of Mining and Safety Engineering*, vol. 33, no. 3, pp. 423–440, 2016.

[9] W. J. Yu and F. F. Liu, "Stability of close chambers surrounding rock in deep and comprehensive control technology," *Advances in Civil Engineering*, vol. 2018, Article ID 6275941, 18 pages, 2018.

[10] M. He, W. Gong, J. Wang et al., "Development of a novel energy-absorbing bolt with extraordinarily large elongation and constant resistance," *International Journal of Rock Mechanics and Mining Sciences*, vol. 67, pp. 29–42, 2014.

[11] S.-C. Li, H.-T. Wang, Q. Wang et al., "Failure mechanism of bolting support and high-strength bolt-grouting technology for deep and soft surrounding rock with high stress," *Journal of Central South University*, vol. 23, no. 2, pp. 440–448, 2016.

[12] W. Fangtian, C. Zhang, S. F. Wei, X. Zhang, and S. Guo, "Whole section anchor–grouting reinforcement technology and its application in underground roadways with loose and fractured surrounding rock," *Tunnelling and Underground Space Technology*, vol. 51, pp. 133–143, 2016.

[13] Z. G. Xu, *Elasticity*, Higher Education Press, Beijing, China, 2013.

[14] N. Ma, J. Li, and Z. Zhao, "Distribution of the deviatoric stress field and plastic zone in circular roadway surrounding rock," *Journal of China University of Mining and Technology*, vol. 44, no. 2, pp. 206–213, 2015.

[15] W. J. Yu, G. S. Wu, C. Yuan, P. Wang, and S. H. Shao, "Failure characteristics and engineering stability control of roadway," *Journal of China Coal Society*, vol. 42, no. 6, pp. 1408–1419, 2017.

[16] C. Yuan, W. Wang, T. Feng, W. Yu, H. Wu, and W. Peng, "Research on control principles of surrounding rocks of roadway based on expansion of plastic zone," *Journal of Mining and Safety Engineering*, vol. 34, no. 6, pp. 1051–1059, 2017.

[17] F.-L. He, L. Xu, H.-K. Wu, and T.-D. Li, "Deviatoric stress transfer and stability of surrounding rock in large-section open-off cut roof," *Chinese Journal of Geotechnical Engineering*, vol. 36, no. 6, pp. 1121–1127, 2014.

[18] L. Xu, H.-X. Wei, Z.-Y. Xiao, and B. Li, "Engineering cases and characteristics of deviatoric stress under coal pillar in regional floor," *Rock and Soil Mechanics*, vol. 36, no. 2, pp. 561–568, 2015.

[19] S.-R. Xie, G.-C. Zhang, S.-S. He et al., "Surrounding rock control mechanism and its application of gob-side retaining entry in deep backfilling with large mining height," *Journal of China Coal Society*, vol. 39, no. 12, pp. 2362–2368, 2014.

[20] S. Xie, S. Yue, D. Chen et al., "Deviatoric stress evolution laws and control of surrounding rock at gob-side entry retaining in deep backfilling mining," *Journal of China Coal Society*, vol. 43, no. 7, pp. 1837–1846, 2018.

[21] J. Z. Li, M. Zhang, Y. Li, and H. Hu, "Surrounding rock control mechanism in the gob-side retaining entry in thin coal seams, and its application," *Journal of the Southern African Institute of Mining and Metallurgy*, vol. 118, no. 5, pp. 471–480, 2018.

[22] K. Wang, B. Y. Hao, P. P. Zhang, and Y. F. Qiao, "Floor heave control mechanism of inner interlocked gas tail road based on reinforced concrete pillar," *Journal of China Coal Society*, vol. 41, no. 9, pp. 2180–2187, 2016.

[23] H. Wu, X. K. Wang, W. J. Wang, G. Peng, and Z. Zhang, "Deformation characteristics and mechanism of deep subsize coal pillar of the tilted stratum," *Energy Science& Engineering*, vol. 8, no. 2, pp. 544–561, 2020.

[24] S.-L. Wang, S.-P. Hao, Y. Chen, J.-B. Bai, X.-Y. Wang, and Y. Xu, "Numerical investigation of coal pillar failure under simultaneous static and dynamic loading," *International Journal of Rock Mechanics & Mining Sciences*, vol. 84, pp. 59–68, 2016.

[25] L. Q. Shi, D. J. Xu, Y. Wang, M. Qiu, and J. Hao, "A novel conceptual model of fracture evolution patterns in the overlying strata during horizontal coal seam mining," *Arab Journal of Geosciences*, vol. 12, no. 10, 2019.

[26] J. Wang, J. Ning, L. Jiang, J.-Q. Jiang, and T. Bu, "Structural characteristics of strata overlying of a fully mechanized longwall face: a case study," *Journal of the Southern African Institute of Mining and Metallurgy*, vol. 118, no. 11, pp. 1195–1204, 2018.

[27] J. Wang, J. G. Ning, P. Q. Qiu, S. Yang, and H. F. Shang, "Microseismic monitoring and its precursory parameter of hard roof collapse in longwall faces: a case study," *Geomechanics and Engineering*, vol. 17, no. 4, pp. 375–383, 2019.

[28] W. Gao, L. Shi, J. Han, and P. Zhai, "Dynamic monitoring of water in a working face floor using 2D electrical resistivity tomography (ERT)," *Mine Water and the Environment*, vol. 37, no. 3, pp. 423–430, 2018.

[29] H. P. Kang, J. Lin, and M. J. Fan, "Investigation on support pattern of a coal mine roadway within soft rocks-a case study," *International Journal of Coal Geology*, vol. 140, pp. 31–40, 2015.

[30] H. Kang, "Sixty years development and prospects of rock bolting technology for underground coal mine roadways in China," *Journal of China University of Mining and Technology*, vol. 45, no. 6, pp. 1071–1081, 2016.

[31] L. S. Jiang, P. Kong, J. M. Shu, and K. G. Fan, "Numerical analysis of support designs based on a case study of a longwall entry," *Rock Mechanics & Rock Engineering*, vol. 52, no. 9, pp. 3373–3384, 2019.

[32] H. Yan, F.-L. He, T.-F. Xu, H.-J. Jiang, and S. Gao, "Experimental study of bolt and cable truss system for large cross-section coal roadways under high stress," *Rock and Soil Mechanics*, vol. 33, no. 11, pp. 257–262, 2012.