

Research Article

Failure Mechanism and Control Technology of Thick and Soft Coal Fully Mechanized Caving Roadway under Double Gobs in Close Coal Seams

Shengrong Xie¹, Xiaoyu Wu¹, Dongdong Chen¹, Yaohui Sun¹, En Wang¹, Xiaowei Wu², Xiaobo Duan^{1,3}, Zaisheng Jiang¹, Pengyu Qi¹, and Songhao Shi¹

¹School of Energy and Mining Engineering, China University of Mining and Technology-Beijing, Beijing 100083, China ²Yongdingzhuang Mine, DaTong Coal Mine Group, Datong 037024, China

³Institute of Mining and Coal, Inner Mongolia University of Science and Technology, Baotou 014010, China

Correspondence should be addressed to Dongdong Chen; chendongbcg@163.com

Received 23 July 2020; Revised 1 November 2020; Accepted 24 November 2020; Published 9 December 2020

Academic Editor: Giuseppe Ruta

Copyright © 2020 Shengrong Xie et al. This is an open access article distributed under the Creative Commons Attribution License, which permits unrestricted use, distribution, and reproduction in any medium, provided the original work is properly cited.

The surrounding rock of the roadway under double gobs in the lower coal seams is partially damaged by the mining of the upper coal seam and the stress superimposition of the stepped coal pillars. What is worse, the upper layer of the roof is collapse gangue in double gobs, which makes the anchor cable unable to anchor the reliable bearing layer, so the anchoring performance is weakened. The actual drawing forces of the anchor bolt and anchor cable are only approximately 50 kN and 80 kN, respectively. The roadway develops cracks and large deformations with increasing difficulty in achieving safe ventilation. In view of the above problems, taking the close coal seam mining in the Zhengwen Coal Mine as the engineering background, a theoretical calculation is used to obtain the loading of the step coal pillars and the slip line field distribution of the floor depth. The numerical simulation monitors the stress superimposition of stepped coal pillars and the distribution of elastoplastic areas to effectively evaluate the layout of mining roadways. The numerical simulation also analyzes the effective prestress field distribution of the broken roof and grouting roof anchor cable. A laboratory test was used to monitor the strength of the grouting test block of the broken coal body. Then, we proposed that grouting anchor cable be used to strengthen the weak surface of the roof and block the roof cracks. From on-site measurement, the roadway was seen to be arranged in the lateral stress stabilization area of the stepped coal pillars, the combined support technology of the grouting anchor cable (bolt) + U type steel + a single prop was adopted, the roadway deformation was small, the gas influx was reduced, and the drawing force of the anchor bolt and the anchor cable was increased to approximately 160 kN and 350 kN, respectively. The overall design and control technology of the roadway can meet the site safety and efficient production requirements.

1. Introduction

The close coal seams are explained in the appendix of "Coal Mine Safety Regulations" (2016 Edition) in China as "coal seams where the distance between coal seam groups is relatively small and has a great influence on each other during mining." The close coal seam group is widely distributed, and its occurrence accounts for a major proportion of coal seams. Most mining areas have the problem of close coal seam group mining. The distance between coal seams decreases in close coal seams, and the mutual influence of

mining between the upper and lower coal seams will gradually increase, especially when the distance between the coal seams is extremely close. The integrity of the roof before the mining of the lower coal seam has been damaged by the strong mining of the upper coal seam [1, 2]. After the upper coal seam has been mined, the roof structure and stress environment of the lower coal seam mining area have changed [3]. The remaining sections of the coal pillars form a stress concentration, which has a strong influence on the layout space of the lower coal roadway and the stability of the surrounding rock, as shown in Figure 1.



FIGURE 1: Influence of mining in upper coal seam on damage to the lower coal seam.

Many scholars have conducted extensive research on the structure and pressure of the overlying strata caused by coal seam spacing and mining thickness in close coal seam mining. The increase in the seam thickness will cause the changes in the three strata of the overburden collapse zone, fissure zone, and curved subsidence zone to increase substantially so that the large collapse zone damages the bottom seam floor [4]. Sun et al. [5], with the help of conventional offset theory and rock movement theory, proposed that a voltage stabilization zone will form after the upper coal seam is mined. When mining the lower coal seam, the working face should be arranged within the range of this voltage stabilization zone and a reasonable staggered distance from the same coal mining under the seam. Xu et al. [6] proposed that a certain overlap of the destruction range of the collapse zone due to coal mining will lead to the expansion of the height of the lower coal collapse zone. The calculation of the mining thickness of the lower coal seam shows that the cracking ratio calculated based on the coal thickness of the lower seam is 52% higher than the cracking ratio calculated for the single coal seam with fully mechanized caving.

Many scholars have conducted extensive research on the arrangement of the coal pillars and the roadways in the upper and lower coal seam mining with close coal seams. They believe that the side below the coal pillar or coal body is a pressurized area, and the stress is higher than the original rock stress. The side is the pressure relief area, and the stress is lower than the original rock stress [7, 8]. To improve the stability of the roadway, the lower coal seam roadway is placed in a low-stress area, and the lower coal seam roadway is often arranged within a certain distance [9, 10]. Ju and Xu [11–13] proposed that when the coal pillars are out of the working face of the lower coal seam, the breaking distance of the overlying main roof is related to the distance from the opening of the lower coal seam working face to the coal pillar. The basic roof will form two different types of structures when mining under the coal seam is performed.

The instability mechanisms of these two structures are not the same, but they will cause the working resistance of the working face bracket to be too large. Research on coal roadways under close coal seams includes roof span, prestressed load-bearing structure of the anchor cable, and roof tensile and shear failure [14–17]. The study of the stability of the surrounding rocks for coal seams under close coal seams includes the compression bearing capacity of the coal and rock mass, the relative proportions of the coal and rock mass, and the coal and rock roadway tensile shear failure structure [18–21].

Due to the close distance between the upper and lower layers of the close coal seams, strong mining in the upper coal seam has severely damaged the roof of the lower coal seam. Many scholars have adopted the hydrogeological borehole detection method, the borehole injection method in sections, the rock layer mobile borehole detection method, the borehole ultrasonic imaging probing method, the radio wave perspective method between holes, the seismic wave CT detecting technology, and the resistivity method to explore the damage to the bottom plate. In previous research, there has been a lack of research on the destruction mechanism and control technology of fully mechanized top coal caving roadways with thick soft coal under double gobs in close coal seams. This article combines theoretical analysis, numerical simulation, laboratory testing, and on-site monitoring. Overall analysis of the response relationship among the stepped coal pillar load, the stress superposition, the stress transfer and roof failure depth, and the failure range of the fully mechanized top coal caving roadway under double gobs in close coal seams was performed. In this way, the effective prestress field of the grouting anchor cable of the broken roof was clarified, and the surrounding rock failure mechanism of the fully mechanized top coal caving roadway with thick soft coal under the seam was clarified. This paper proposes the combined support control technology of the grouting anchor cable and the U type steel and single prop, which has important reference value for the mining geology of the thick soft coal fully mechanized caving coal roadway under double gobs in close coal seams.

2. Geological Conditions and Destruction Characteristics of the Roadway Surrounding Rock

2.1. Geological Conditions and Mining Conditions. The Zhengwen Coal Mine that was studied is located in Xiaoyi City, Shanxi Province, China. The No. 1 mining area of the Zhengwen Coal Mine produces mainly #10+11 coal, and the stratum is classified as the Carboniferous Taiyuan Formation. The coal seam elevation of the 11103 working face is between +745~+773, the corresponding ground elevation of this working face is +926~+1014, and the average coal seam inclination angle is 6.5°, which is a near-horizontal coal seam, and the coal seam is more complicated and contains multiple layers of mudstone gangue. The average thickness of the coal seam is 7.86 m. The #10 + 11 coal seam is separated from the upper #9 coal seam by 1 m of thick mudstone, and the average thickness of the upper #9 coal seam is 1.6 m. The overburden layer of the #9 coal seam is K2 limestone (7 m), mudstone (2.96 m), fine sandstone (4.1 m), and sand mudstone (2.34 m). At present, the 11103 working face is arranged under the #10+11 coal seam with an average thickness of 5.86 m. The overburden under the #10 + 11 coal seam is sand and mudstone (6.13 m), #12 coal (0.2 m), mudstone (7.44 m), and fine sandstone (3.73 m). The distribution and excavation under double gobs in close coal seams is shown in Figure 2, and the mining sequence of double gobs in close coal seams is shown in Figure 3.

2.2. Roadway Mining Pressure Monitoring. The average interval between the #9 coal seam and the #10 + 11 coal seam is 1 m, so the coal seam is close. The 11103 haulage roadway is arranged under the double gobs, and this roadway is affected by the mining of the upper coal seam and the stress of the stepped coal pillars resulting in the roof structure of the lower coal seam mining area with multiple cracks and easy instability. After excavation and recovery, the surrounding rock cracks and roof cracks of the roadway promote each other and penetrate each other. The 11103 haulage roadway also has the characteristics of close coal seams, double gobs, strong mining, thick coal seams, etc. The following rock pressure phenomenon occurs during the roadway process, as shown in Figure 4. As shown in Figure 4(a), the squeezing deformation in the plumb direction of the roadway is significant, and the plumb movement in the roof stratum is violent. The supporting structure cannot adapt to the roof movement, which causes the W-type steel belt to bend severely and exhibit a "V"-shaped squeeze failure. The roof shows that some anchor cable locks are damaged and invalid. As shown in Figure 4(b), the roof of the roadway is sinking, and the 29U-shaped steel is distorted in both vertical and horizontal directions. As shown in Figure 4(c), the side roadside bulges against the coal pillars, and the

anchor bolts are unanchored. As shown in Figure 4(d), the surface of the coal shack on the side of the working face is partially broken.

- (1) The 11103 haulage roadway is driven at the side of the coal pillar, which is affected by the concentrated high stress of the coal pillar. The upper #9 coal seam (1.6 m) and the #10+11 upper coal seam stratification (2 m) have been mined. The upper #9 coal seam reserves a protective coal pillar (16 m), and the upper #10+11 coal seam stratification reserves a protective coal pillar (28 m), forming a stepped coal pillar. The stress concentration degree of the coal pillar is high. The 11103 haulage roadway is located at the side of the coal pillar. After the roadway was excavated, the stability of the surrounding rock was destroyed. The stress of the surrounding rock changed from the original three-way stress state to the two-way stress state. The rock stratum showed plastic compression deformation damage. The deformation and expansion of the surrounding rock in the vertical and horizontal directions caused the bending and sinking of the roof coal seam, and the two sides bulged out.
- (2) The roof of the 11103 haulage roadway is a broken coal body, and the low anchoring strength of ordinary anchor cable cannot meet the production demand. The roof is a coal body and double gobs. The roof coal seam is strongly mined by the upper part. The cracks in the coal body are seriously developed, the strength is extremely low, and the anchoring force of the anchor cable is small. The anchoring force of the anchor bolt is approximately 50 kN, and the anchoring force of the anchor cable is approximately 80 kN. When the roadway pressure increases, the supporting requirements cannot be met. The 11103 haulage roadway is located in the lower part of the double-layer gob, and in the early stage of excavation, the roadway was excavated along the bottom of the #10 + 11 (7.8 m) coal seam, which resulted in the roof of the roadway being the coal seam (5.8 m) for the caving rock of the gob, the weak bearing structure of the upper roof, and the low anchoring strength.
- (3) The thickness of the coal seam is 5.86–9.00 m, including 5–7 layers of 0.02–0.10 m mudstone mixed with gangue. The stability is extremely poor. To meet the needs of low caving coal mining in fully mechanized caving, the excavation width of the mining roadway is 4.5 m, and the height is 2.8 m, with a sectional area of nearly 12.6 m². For the thickness and the low strength of the coal seam, the increase in the cross section will lead to roof tensile failure, rapid expansion of the tension shear failure of the two sides, sharp increase of the degree of breakage of the surrounding rock of the roadway, and inadequate quality control in the construction process, resulting in the obvious deformation of the roadway.



FIGURE 2: Distribution and excavation under double gobs in close coal seams.

2.3. Deformation Failure Mechanism and Support Failure Analysis. The surrounding rock of the 11103 haulage roadway has been affected by the strong mining of the upper coal seam. The cracks in the surrounding rock and the roof of the roadway promote the development and penetration of each other, leading to the weakening of the bearing capacity of the roof. Based on the in-depth investigation of the geological production conditions of the roadway with the characteristics of the close coal seams, strong mining, and lower thick coal seams, the maintenance difficulties are as follows.

Through the borehole peep of the surrounding rock of the 11103 haulage roadway, the results of analysis show that under the close double gobs, the lower coal is affected by the superposition of upper coal mining and stepped coal pillar stress. The borehole observation figure is shown in Figure 5. The borehole observation instrument is used to observe the roof of the roadway and the two sides of the roadway. When the depth of the roadway roof is 2 m, the surrounding rock of the roof is relatively complete, and there is no obvious crack. When the depth of the roadway roof is 4 m, there are small cracks in the surrounding rock of the roof. When the depth of the roof of the roadway is 6 m, the cracks in the surrounding rock of the roof are obvious. The crack is not obvious when the depth of the roadway on the side of the working face is 2 m, but it is more obvious when the depth of the side roadway near the coal pillar is 2 m.

3. Study of the Deformation and Failure Law of the Floor Rock under Double Gobs

3.1. Stress Superposition Mechanical Model of Stepped Coal Pillar. The mechanical model of unsymmetrical mining on both sides of the double-layer stepped coal pillar was established to analyze the failure rule of the floor surrounding the rock when the load, stress superposition, and stress transfer of the double-layer stepped coal pillar occurred. The coal pillar loading is the loading of the overlying strata and the overhanging weight of the gob on both sides.



FIGURE 3: Mining sequence of double gobs in close coal seams.



(a)

(b)







FIGURE 5: The borehole observation.

The collapse height of the overlying strata increases, as shown in Figure 6. This is a preliminary estimation method of the loading of the stepped coal pillars.

The calculation formula for the total load of the doublelayer stepped coal pillar (P_n) is as follows:

$$P_{n} = r \Big[L_{2}H + \frac{1}{2}L_{1}H_{2}\frac{1}{2}L_{3}H_{3} + \frac{1}{2}(H_{1} - H_{2}) \\ \cdot \Big[(H_{1} - H_{2})\tan\alpha + D_{2} \Big] (H - H_{1})^{2}\tan\alpha \qquad (1) \\ + \frac{1}{2}(H_{1} - H_{3}) \Big[(H_{1} - H_{3})\tan\alpha + D_{2} \Big] \Big].$$

$$R_{n1}\left(0.64 + 0.36\frac{L_2}{C_2}\right) = \frac{\gamma}{1000L_2} \left[L_2 H + \frac{1}{2} L_1 H_2 \frac{1}{2} L_3 H_3 + \frac{1}{2} (H_1 - H_2) \left[(H_1 - H_2) \tan \alpha + D_2 \right] \right]$$

$$\cdot (H - H_1)^2 \tan \alpha + \frac{1}{2} (H_1 - H_3) \left[(H_1 - H_3) \tan \alpha + D_2 \right].$$
(2)

Among these parameters, L_2 and D_2 are the widths of the upper and lower pillars, respectively, m; L_1 and L_3 are the lengths of the gob of the extremely close upper coal seam, m; D_1 and D_3 are the lengths of the gob of the extremely close lower coal seam, m; H_1 is the burial depth of the upper coal seam to the first caving height; H_2 is the burial depth of the lower coal seam to the second caving height; α is the caving angle of the overlying strata of the upper coal seam gob; γ is the average volume force of overlying strata, kN/m³; C_1 is the thickness from the upper coal seam to the lower coal seam; C_2 is the thickness of the lower coal seam; C_3 is the thickness of the lower coal seam; and R_{n1} is the ultimate strength of the upper coal pillar.

The floor failure slip field caused by the coal pillar can be calculated by the slip line field theory method, and the simplified mechanical model is shown in Figure 7.

According to the slip line theory of the floor, the formula of the failure depth of the floor is as follows:

$$x_{2} = \frac{m(1 - \sin \varphi)}{2(1 + \sin \varphi)f} \ln \frac{(K_{2}P_{2}/L_{2}) + C \cot \varphi}{C \cot \varphi},$$

$$h_{\max} = \frac{mC_{2}(1 - \sin \varphi)\cos \varphi \ln ((K_{2}P_{2}/L_{2}) + C \cot \varphi/C \cot \varphi)}{4f(1 + \sin \varphi)\cos ((\pi/4) + (\varphi/2))} e^{((\pi/4) + (\varphi/2))\tan \varphi},$$
(3)

$$L_{A_20} = h_{\max} \tan \varphi,$$

where x_2 is the yield strength of the coal pillar; *m* is the mining thickness of the coal seam; h_{max} is the maximum failure depth; φ is the internal friction angle; *C* is the cohesion of the coal; and *f* is the friction coefficient of the contact surface between the coal seam and the roof and the floor.

3.2. Empirical Regression Equations for Floor Failure Depth. In the appendix of the 2016 Edition (China) of coal mine safety regulations, the extremely close coal seam is interpreted as "the coal seam with small distance between coal seam groups and great influence on each other during mining." Literature [22] points out that when the distance between coal seams is less than the failure depth of the rock floor, the coal seam group is called an extremely close coal seam. According to elastic theory, the maximum yield failure depth h_{max} of the floor rock mass under the plane stress state is as follows [23]:

$$h_{\rm max} = \frac{1.57\gamma^2 \rm HL}{4\beta^2 \sigma_c^2},\tag{4}$$

where γ is the average unit weight of the overburden in the stope, kN/m³; *H* is the mining depth of the coal seam, m; *L* is the width of the gob, m; β is the coefficient for the influence of the rock joint fissures; and σ_c is the uniaxial compressive strength of rock, MPa.

According to the formula for the prediction of the failure depth of the horizontal floor, the maximum failure depth h_{max} of the floor is as follows [24]:



FIGURE 6: A mechanical model of the load of an extremely close stepped coal pillar.



I: loading distribution of coal pillars with mining double gobs on one side

II: loading distribution of coal pillars with mining double gobs on both sides

FIGURE 7: Mechanical model of floor failure caused by stress superposition of extremely close double-layer stepped coal pillar.

$$h_{\rm max} = 0.0113H + 6.25In\left(\frac{L}{40}\right) + 2.25In\left(\frac{M}{1.48}\right),$$
 (5)

where M is the mining height.

According to the test report for the physical and mechanical parameters of coal rock in the Zhengwen Coal Mine, the parameters of the factors influencing failure depth are determined as follows: *H* is 231 m, γ is 24 kN/m³, *L* is 146.5 m, β is 0.56, σ_c is 20 MPa, and *M* is 1.5 m and 2 m, respectively. 3.3. Numerical Simulation. To explore the failure mechanism of the thick and soft fully mechanized coal caving roadway under double gobs of a near-distance coal seam, a numerical simulation of its excavation and mining was established according to the geological condition of the 11103 working face by using FIAC3D software. In this simulation, excavation by steps was adopted to monitor the stress superposition and stress transfer of the ladder coal pillar under double gobs and to efficiently evaluate the arrangement in the mining and excavation of the roadway combined with the distribution range of the elastic-plastic area. Additionally, the prestress of the anchor bolt and anchor cable was simulated in the broken roof without grouting and in the modified roof with grouting. Then, the distribution of the prestress field was compared and analyzed, in which the double yield model with two different parameters was adopted to simulate the double gobs, and the strain-softening model was adopted to simulate the ladder coal pillar.

3.3.1. Numerical Model of the Excavation and Mining with Double Gobs in Close Coal Seams. The numerical model with a size of 325*240*105 (length*width*height) was established based on the problem studied and the mesh density shown in Figure 8. In this numerical model, a load of 4.03 MPa was placed on the upper boundary of the model to simulate the weight of the overlying strata according to the geostress measurement. Then, the horizontal pressure coefficient along the x- and y-axes was set to 1.2 times the upper load, and the displacement constraint was adopted in the horizontal direction of the left and right boundaries and the vertical direction of the bottom boundary. Additionally, as shown in Table 1, the rock parameters used in this numerical model were tested in the laboratory. Meanwhile, to determine the rationality of the simulation, the double yield model with two different parameters and the strain-softening model were adopted to simulate the double gobs and the ladder coal pillar, respectively. The structural diagram of the checking calculation in the numerical simulation is shown in Figure 9.

3.3.2. Two-Layer Double Yield Model with Different Parameters Implanted in the Double Gobs. To make the simulation conform to the engineering practice, considering that the overlying strata can be supported by the broken-expanded coal gangue of the gob and the situation of two-time collapse of the gangue under the double gob, the two-layer double yield model with different parameters should be set independently in the gangue model of the gob gangue, and the double yield model is the main model. The calculated formula of stress and strain is as follows [25, 26]:

$$\sigma = \frac{E_0 \varepsilon}{1 - (\varepsilon/\varepsilon_{max})},$$

$$\varepsilon_{max} = \frac{b - 1}{b},$$

$$E_0 = \frac{10.39 \sigma_c^{1.042}}{b^{7.7}},$$

$$b = \frac{h_c + h_m}{h_c},$$
(6)

where σ is the cap pressure, ε is the bulk strain of the coal and rock mass, ε_{max} is the maximum bulk strain, E_0 is the initial elastic modulus (ε_{max} and E_0 can be estimated), σ_c is the

strength of the caved rocks, b is the expansion coefficient of the caved rocks in the gob, and h_c and h_m are the caving zone height and the mining height, respectively.

The input parameters of the double yield model can be divided into the stress-strain relationship and parameters of the material, and the relationship between the stress and strain of double yield model materials is shown in Tables 2 and 3. The values of the material parameters of the double yield model need to be adjusted constantly to make the stress-strain curve of the bulk of the numerical simulation output consistent with the stress-strain curve of the theoretical model shown in Figure 10. Therefore, a bulk modulus of 1.8 GPa, shear modulus of 1.0 GPa, friction angle of 15°, expansion angle of 5°, and density of 1000 kg/m³ were determined as the material parameters of gob gangue in primary mining. Then, the bulk modulus of 1.68 GPa, shear modulus of 0.88 GPa, friction angle of 12°, expansion angle of 3°, and density of 920 kg/m³ were determined in secondary mining.

3.3.3. Implantation of Strain-Softening Model. The strainsoftening model is based on the Mohr–Coulomb yield criterion [27]. Therefore, the stress at any point in the postpeak strain-softening phase of the rock material is determined by equation (8) [28]. The softening model curve is shown in Figure 11, and changes in the plastic zone of the softening model are shown in Figure 12.

$$\sigma_{1} = \frac{1 + \sin \varphi(\varepsilon^{\text{ps}})}{1 - \cos \varphi(\varepsilon^{\text{ps}})} \sigma_{3} + \frac{2c(\varepsilon^{\text{ps}})\cos \varphi(\varepsilon^{\text{ps}})}{1 - \sin \varphi(\varepsilon^{\text{ps}})}, \tag{7}$$

where ε^{ps} , $\varphi(\varepsilon^{ps})$, and $c(\varepsilon^{ps})$ are the strain, internal friction angle, and cohesion of the rock material at the yield stage, respectively, which can be expressed as follows:

$$\varepsilon^{\mathrm{ps}} = \frac{\sqrt{2}}{6} \sqrt{\left[\left(2\varepsilon_{1}^{p} - \varepsilon_{3}^{p} \right)^{2} + \left(\varepsilon_{1}^{p} + \varepsilon_{3}^{p} \right)^{2} + \left(2\varepsilon_{3}^{p} - \varepsilon_{1}^{p} \right)^{2} \right]}, \quad (8)$$

$$c\left(\varepsilon^{\mathrm{ps}}\right) = \begin{cases} c_{p}, \varepsilon^{\mathrm{ps}} \le \varepsilon_{p}, \\ \frac{c_{r} - c_{p}}{\varepsilon_{r} - \varepsilon_{p}} \left(\varepsilon^{\mathrm{ps}} - \varepsilon_{p} \right) + c_{p}, \varepsilon_{p} < \varepsilon^{\mathrm{ps}} < \varepsilon_{r}, \\ c_{r}, \varepsilon^{\mathrm{ps}} \ge \varepsilon_{r}, \\ c_{r}, \varepsilon^{\mathrm{ps}} \ge \varepsilon_{p}, \end{cases}$$

$$\varphi\left(\varepsilon^{\mathrm{ps}}\right) = \begin{cases} \varphi_{p}, \varepsilon^{\mathrm{ps}} \le \varepsilon_{p}, \\ \frac{\varphi_{p}, \varepsilon^{\mathrm{ps}} \le \varepsilon_{p}, \\ \varepsilon_{r} - \varepsilon_{p}} \left(\varepsilon^{\mathrm{ps}} - \varepsilon_{p} \right) + \varphi_{p}, \varepsilon_{p} < \varepsilon^{\mathrm{ps}} < \varepsilon_{r}, \\ \frac{\varphi_{r}, \varepsilon^{\mathrm{ps}} \ge \varepsilon_{r}, \\ \varphi_{r}, \varepsilon^{\mathrm{ps}} \ge \varepsilon_{r}, \end{cases}$$

$$(9)$$

where ε_p is the plastic strain at the peak; ε_r is the plastic strain at the beginning of the residual strength; c_p is the cohesion at the peak, MPa; φ_p is the internal friction angle at the peak; c_r is the cohesion at the beginning of the residual strength, MPa; and φ_r is the internal friction angle at the beginning of the residual strength.

In yield peak stage II, the deformation ε_p is 0.006, the cohesion c_p is 0.8 MPa, and the internal friction angle φ_p is



FIGURE 8: Numerical simulation model of mining under double gobs in close coal seams.

Rock strata	$D (\text{kg·m}^{-3})$	K (GPa)	G (GPa)	Φ_m (°)	C_m (MPa)	$\sigma_{\rm tm}$ (MPa)
Overlying strata	2600	8.5	6.3	30	1.8	0.7
Sandy mudstone	1800	5.5	3.3	18	1.1	0.4
Fine sandstone	2650	11.5	7.3	33	2.1	0.8
Mudstone	1600	4.5	2.3	16	1	0.3
Limestone	2850	14.5	9.3	38	2.4	1.2
No. 9 coal seam	1400	2.6	1.5	20	0.8	0.4
Mudstone	1600	4.5	2.3	16	1	0.3
No. 10 + 11 coal seam	1400	2.6	1.5	20	0.8	0.4
Sandy mudstone	1800	5.5	3.3	18	1.1	0.4
No. 12 coal seam	1400	2.6	1.5	20	0.8	0.4
Mudstone	1600	4.5	2.3	16	1	0.3
Fine sandstone	2650	11.5	7.3	33	2.1	0.8
Underlying strata	2600	8.5	6.3	30	1.8	0.7

Гавle 1: Roc	ck strata:	properties	used in	the	numerical	model.
--------------	------------	------------	---------	-----	-----------	--------

D is the density; K is the bulk modulus; G is the shear modulus; Φ_m is the friction angle; C_m is the cohesion; σ_{tm} is the tensile strength.

20°. In residual failure stage III, the deformation ε_r is 0.011, the cohesion c_r is 0.3 MPa, and the internal friction angle φ_r is 13°.

4. Discussion and Analysis of Simulation Results

The step-by-step simulation method was used in the fully mechanized sublevel caving roadway under thick-soft coal in the close-up double mined-out area, and gob 1 was formed by excavating the upper working face of the coal seam. At this time, the high stress value of the coal side is 14 MPa in the single coal seam mining, which is 3 times the original rock stress. The coal on the side of the gob is in a low-stress area, with a low stress value of 6 MPa. Step 1: under gob 1, gob 2 is formed by excavating the lower working face of the coal seam. At this time, the coal seam stress changes from the upper coal seam to the lower coal seam. The low stress of the coal side in the lower coal seam is 6 MPa, and the peak value of the high stress remains unchanged. Step 2: gob 3 is formed by excavating the working face in the upper coal seam, and the 16 m coal pillar is left. The coal pillar shows stress superposition and stress concentration phenomena, and the high stress value is 24 MPa. The stress concentration is

biased towards gob 3, and the low stress on the coal side of the lower coal seam is 6 MPa. Step 3: gob 4 is formed by excavating the working face in the lower coal seam, and a 28 m coal pillar is left. The stress concentration area of the coal pillar appears in the middle, and stress transfer occurs. The stress distribution of the fully mechanized top-coal caving roadway under thick-soft coal in a short-distance gob is shown in Figure 13.

The distribution of the plastic zone of the fully mechanized caving roadway under thick-soft coal in the close-up double mined-out area is shown in Figure 14. The shape of the plastic zone of the coal side and the bottom rock layer is generally a semislip surface. When excavating the working face of the upper coal seam to form gob 1, the depth of the influence of the plastic zone on the coal side is 7.5 m. When excavating the lower coal seam working face to form gob 2, the depth of the influence of the plastic zone on the side of the lower coal gangue is 7 m, and the total depth of the upper and lower coal gangue is 11.5 m. When the upper coal seam is excavated to form gob 3 near the working face, a small trapezoidal elastic area is formed in the middle of the upper coal pillar. A large trapezoidal elastic region is formed in the lower part of the lower coal seam, and the



FIGURE 9: The structural diagram of the checking calculation.

TABLE 2: Cap pressures used in the double yield model.

Strain	Stress (MPa)
0.02	0.26
0.04	0.53
0.06	0.81
0.08	1.08
0.10	1.60
0.12	2.09
0.14	3.45
0.16	4.81
0.18	6.38
0.20	8.68
0.22	13.30
0.24	23.14

depth of the influence of the plastic zone on the side of the lower coalbed does not change. When the working face of the lower coal seam is excavated to form gob 4, the elastic

TABLE 3: Cap pressures used in the double yield model.

Strain	Stress (MPa)
0.02	0.21
0.04	0.42
0.06	0.65
0.08	0.86
0.10	1.28
0.12	1.67
0.14	2.76
0.16	3.85
0.18	5.10
0.20	6.94
0.22	10.64
0.24	18.51

area formed in the middle of the upper coal pillar disappears. The upper coal pillars are all plastic, and the lower coal pillars have a triangular elastic area.



FIGURE 10: Stress-strain in the double yield model.







FIGURE 12: Changes in the plastic zone of the softening model.

4.1. Stress Monitoring. Five survey lines are arranged in the model: line 1 is arranged in the middle of the upper coal seam pillar, and line 2 is arranged in the middle of the lower coal seam pillar. Survey line 3 is arranged in the middle of the lower coal seam, survey line 4 is arranged between the lower coal seam and the sandy mudstone of the floor, and survey line 5 is arranged on the sandy mudstone of the floor. The layout of the measuring lines is shown in Figure 15, and the measurement results are shown in Figures 16–19. The stress

curves in Figure 16 show that under the double mined-out area, the vertical stress peaks have not changed substantially. However, the depth of the plastic zone increases from 7.5 m to 11.5 m. The stress curves in Figure 17 show the stress superimposition and stress transfer of the stepped coal pillar, and its peak stress increases from 20.5.

ε

As shown in Figure 18, when the thick-soft fully mechanized coal roadway is excavated under the close-up double mined-out area, the coal gangue and coal pillar are in



FIGURE 13: Stress distribution diagram of double gobs in close coal seams.



FIGURE 14: Distribution map of plastic zone under double gobs in close coal seams.



FIGURE 15: Monitoring curve of coal seam and floor rock.



FIGURE 16: Double gobs in close coal seams.

the high-stressed area, and the mined-out area is in the lowstressed area. The stress distribution law first decreases and then increases. As shown in Figure 19, the survey lines arranged in the surrounding rock of 7.5 m, 10.5 m, and 13.5 m in the lower coal seam can be divided into the stress stability zone and the stress instability zone according to the rising and falling trends of the survey lines. The stress stabilization area is at a distance of 4 m from the left side of the stepped coal pillar. All three lines are in a smooth transition stage, and the 10.5 m line tends to be stable here. Based on the above, the location of the roadway in the lower coal seam is evaluated according to the stress distribution of



FIGURE 17: Stepped coal pillars in close coal seams.

the double-layer stepped coal pillar, the distribution of the plastic zone, and the degree of damage of the bottom line to the surrounding rock of the lower coal seam. The soft coal roadway under the short-distance double gobs is considered to be arranged horizontally at a distance of 4 m from the stepped coal pillar and vertically at a distance of 10.5 m from the upper coal seam, which is stable for the surrounding rock of the roadway and avoids the range of influence of the stepped coal pillar stress.

4.2. Prestressed Field Distribution of Grouting Anchor Cable in the Broken Roof. Considering the grouting support effect of broken roofs, the prestressed field of grouting and ungrouting roof anchor cables was simulated and analyzed. The "cable" structural element embedded in FLAC3D is used to simulate the anchor cable. The mechanical and geometric parameters of the "cable" structural element are shown in Table 4. Among these parameters, the length, diameter, elastic modulus, and tensile yield strength can all be obtained manually. The parameters C_g and K_g are expressed by formula (10). The support mode in the simulation is shown in Figure 20.

$$C_g = \pi (D + 2t)\tau_{\text{peak}},$$

$$K_g \approx \frac{2\pi G}{10In(1 + 2t/D)}.$$
(10)

In the broken roof, the anchor bolt (cable) uses end anchors, lengthened anchors, and full-length anchors, which cannot achieve complete prestress diffusion. However, the use of grouting anchor cables to strengthen surrounding rock can achieve full-length prestressed anchoring.

The roadway in the 11103 working face is a broken roof roadway. Due to the low strength of the roof coal, it is easy to break, and the roadway span is large. The roof is easy to separate from the layer and fall. By numerically simulating the distribution of the prestressed field of the anchor cable in the broken roof in the two states of ungrouted and grouted, the influence of the broken roof grouting on the prestressed field of the anchor cable was analyzed. The distribution of the prestressed field in the roadway is shown in Figure 20. When the straight line of 0.02 MPa is used as the effective compressive stress limit, the upper part is the range of the tensile stress and the ineffective compressive stress of the



FIGURE 18: Mining stress change curve under double gobs in close coal seams.

anchoring. During the downward extension process, the pressure stress continues to increase until the pressure stress of 0.14 MPa. The stress contours of 0.06 MPa are almost straight lines, and the lowest part is the core stress compression area of the anchor cable. The pressure value increases greatly, the growth rate is fast, and the contour lines are densely distributed. The diffusion range of the prestressed field of the anchor cable and the value of the prestress value decrease with the increase in the length of the anchor cable and increase with the increase in the pretension of the anchor cable [29–32]. Reasonable grouting bolts and cables in combined support parameters are the key to controlling the surrounding rock of broken coal roofs under short-distance mining. The stability of the roof can be maintained by the action of the prestressed field of the anchor cable alone, but the maintenance of the surrounding rock of the roof failure is weak. The failure of the surrounding rock not only increases the free surface of the roof but also causes the prestressed release of the anchor cable and reduces the support strength of the anchor cable. Therefore, grouting anchor cable combined support and grouting to strengthen the broken surrounding rock make

the prestressed field of the anchor cable diffuse, but the cable does not release in the deep surrounding rock. After grouting, the effective prestress of the anchor cable increases, and the width and height of the prestressed field increase, as shown in Figure 20.

5. On-Site Monitoring and Grouting Bolt (Cable) Combined Support

5.1. Roadway Layout. After the upper coal seam of the close coal seams is mined, the vertical stress is redistributed, and the whole floor exhibits shear failure, local position shear failure, and vertical failure. Therefore, for the selection of the location of the roadway in the lower coal seam, the following factors should be considered. (1) Stress transfer in the surrounding rock shows that the load on the side of the coal is higher and the stress influence range is far, so the location of the driving roadway should avoid these areas to reduce the damage of high stress to the surrounding rock of the roadway. (2) When the floor is a compound rock stratum, according to the yield strength of the rock, the surrounding rock deformation of the roadway can be controlled by



→ 13.5 m_rock-stratum

FIGURE 19: Stress distribution curve of different layers of floor.

TABLE 4: Bolt/cable properties used in the nume	erical mo	del.
---	-----------	------

Туре	<i>D</i> (mm)	<i>L</i> (mm)	E (GPa)	<i>F</i> (N)	$C_g (\text{N.m}^{-1})$	$K_g (\mathrm{N.m}^{-2})$
Roof bolt	22	2700	2.00 E + + 02	1.34 E + 05	3.23 <i>E</i> + 05	2.02 E + 08
Rib bolt	16	1800	2.00 E + 02	1.34 E + 05	3.23 E + 05	2.02 E + 08
Roof cable	17.8	7000	2.00 E + 02	2.54 E + 05	3.61 E + 05	2.53 E + 08







FIGURE 20: Distribution of the prestressed field of the grouting anchor cable. (a) Bolts with prestressed 60 kN and anchor cables with prestressed 120 kN. (b) Bolts with prestressed 90 kN and anchor cables with prestressed 150 kN. (c) Bolts with prestressed 120 kN and anchor cables with prestressed 180 kN.

coal-rock drift or rock roadway. (3) When the broken roof is supported by the anchor cable, an effective large-scale prestress field cannot be formed due to the release of the prestress. After mining of the upper coal seam, the floor strata under the gob are mainly destroyed by horizontal stress and vertical stress. The floor rock exhibits obvious crack damage, and the strength decreases, so the anchor cable cannot apply effective prestress. The roadway adopts coal-rock drift or rock roadway to avoid the area of maximum failure depth. The lithology of the two sides of roadway is standard integrity. At the same time, the coal pillar in the lower coal seam is reserved at 34 m to reduce the concentrated stress of the coal pillar and keep the stress of the surrounding rock lower than or equal to the original rock stress to ensure that the surrounding rock of the roadway is in a low stress state.

5.2. Grouting Anchor Cable + Single Prop + 29U Type Steel Combined Support. The 11103 haulage roadway in the Zhengwen Coal Mine has a design section of $4.5 \text{ m} \times 2.8 \text{ m}$ (width \times height), and the length of the roadway is 1340 m. To improve the stability of the surrounding rock of the roadway, the combined support of "bolt (cable) + 29U type steel" is adopted for the 11103 haulage roadway. $\Phi 22 \times 2400$ mm lefthanded nonlongitudinal rebar threaded steel bolts and Φ 17.8 × 4500 mm high-strength hollow grouting anchor cables with an elongation of 5% were installed on the roof with row spacing and column spacing of 865 mm × 800 mm and 1500 mm × 1600 mm, respectively. The matching steel bolt plate is 120 mm ×120 mm ×8 mm. The anchor cable is a highstrength and low-relaxation steel strand with a prestress of 150 kN, which greatly enhances the stiffness of the bolt support system and effectively controls roadway deformation. To solve the problem of breaking and the failure of the side bolt, three rows of $\Phi 16 \times 1800 \text{ mm}$ A3 steel bolts were installed on the solid coal side with row spacing and column spacing of $900 \text{ mm} \times 900 \text{ mm}$, and the distance from the bolt to the top and bottom plate was 500 mm. The matching steel bolt plate size is 300 mm ×275 mm × 3 mm. The roadway support parameters are shown in Figure 21.



FIGURE 21: Roadway combined support scheme.

5.3. Control Mechanism of Combined Support and Laboratory Monitoring

5.3.1. Grouting Anchor Cable Mechanism. For the broken surrounding rock, the ordinary anchor cable easily breaks off and cannot apply high prestress. It is possible to repair the broken part with grouting technology, improve the mechanical properties of the weak surface, improve the cohesion and internal friction angle of the cracks, and enhance the overall strength of the surrounding rock to form an effective prestressed bearing structure for the anchor cable [33–36]. Figure 22 shows a technical diagram of the grouting anchor cable. The mechanism of the grouting anchor cable is as follows.

The essential function of grouting anchor cable is to recement the broken rock mass into a whole to form a bearing structure, which controls the discontinuous and uncoordinated expansion and deformation of surrounding rock separation, sliding and tensile properties, shear cracks, etc., maintains the integrity and self-supporting capacity of surrounding rock, and reduces the strength of the surrounding rock.



FIGURE 22: Grouting technology.

The stiffness of the grouting anchor cable support is a key factor. Its prestress and effective diffusion play a decisive role in strengthening the surrounding rock. Reasonable prestressing can make the roof plate under compressive stress after the grouting anchor cable is installed, forming a prestressed bearing structure. The anchor cable support system should have sufficient elongation and impact toughness, on the one hand to continuously deform and release the surrounding rock and on the other hand to prevent the anchor cable from being damaged due to excessive force [37].

High-prestressed, high-strength, and high-elongation cables are active supports in the surrounding rock. The fractured surrounding rock with grouting anchor cable is essential to exert the radial and tangential directions of the anchor cable on the fractured surrounding rock. Supporting resistance strictly limits the shear deformation of the surrounding rock along the primary fissures and the secondary rupture slip surface, as well as the deformation and destruction of the surrounding rock in the anchorage zone, separation, slippage, generation of new fissures, and maintenance of the integrity of the surrounding rock.

5.3.2. 29U Type Steel + Single Prop Combined Support Mechanism in Roadway. The 29U type steel + a single support is used to support the surrounding rock surface in the broken roof roadway, and the 29U type steel support is used to support the roadway as a whole to maintain the stability of the surrounding rock. However, the concentrated load in the middle of the roadway easily crushes and bends the 29U type steel. For this reason, a single prop is used in conjunction with the 29U type steel. The single hydraulic prop has the advantages of light weight, convenient transportation, convenient installation and maintenance, long service life, low cost, etc., and the hydraulic design of the single hydraulic prop provides a strong bearing capacity for the single prop, while the single hydraulic prop has a good cushioning effect on pressure and impact force and can act on the roof for a long time, provide stable support, and prevent the roof from deformation, offsetting, and collapse. The combination of single hydraulic props and 29U type steel equipment can exert an effective force on the roof to achieve the "divide pressure and reduce span" roof rock load, reduce the bending stress and deflection of the roof rock, and reduce the roof rock to the two sides of the roadway. The effect of the load is to realize the weak bearing structure of the shallow surrounding rock of the roof, limit the deformation of the surrounding rock of the roof, and effectively support the surrounding rock of the roof. In summary, the combination of 29U type steel + a single prop support can maintain the integrity of the surrounding rock, thereby effectively limiting the deformation and destruction of the surrounding rock of the roadway and improving the stability of the surrounding rock of the roadway.

5.3.3. Laboratory Monitoring. The grouting anchor cable technology is shown in Figure 22. The technology starts to turn holes and then uses resin anchoring agent to anchor, install grouting anchor cable, and then inject cement slurry into the broken rock body through the anchor cable hollow pipe. The grouting pressure is 0.8–0.9 MPa. Directional grouting is adopted, which not only eliminates the cavitation generated during grouting but also ensures that the

anchoring slurry is filled with rotating holes and cracks. After grouting and waiting for the initial setting of the slurry, the outer stopper plug is opened to block the outflow of the slurry and complete the grouting.

In the initial stage of grouting the surrounding rock, at 1 day, the grouting liquid was not solidified, and the strength of the grouting surrounding the rock increased slowly; the strength was 3.8 MPa, and the overall performance was low strength and high viscosity. In the middle stage of grouting the surrounding rock, the strength of the grouting surrounding the rock increased significantly, and the deformation of the surrounding rock decreased. At 3 days, the grouting could basically reach 62.5% of the maximum strength, and at 7 days, the grouting could basically reach 87.5% of the maximum strength. In the later stages of grouting the surrounding rock, the strength of the grouting surrounding the rock increased, and the strength of the grouting surrounding the rock reached approximately 95% of the maximum strength at 21 days. After 28 days, the strength of the grouting surrounding the rock reached the maximum, and the strength of the surrounding rock stabilized at the maximum of 16 MPa, as shown in Figure 23.

5.4. Field Measurement of Displacement and Deformation of Roadway. To further master the supporting effect of the roadway and the mining roadway in the 11103 working face, the drawing force of the anchor cable in grouting and ungrouted sections was tested. The drawing force of the grouting anchor cable is shown in Figure 24. The cross point method was used to monitor the surface displacement between the roof and the bottom plate and the displacement between the coal sides. The actual drawing force of the anchor is only approximately 50 kN, and the drawing force of the anchor bolt is only approximately 80 kN. After the grouting anchor cable is used, the drawing force of the anchor bolt can reach approximately 160 kN, and the drawing force of the anchor cable can reach approximately 350 kN. The displacement monitoring curve of the driving roadway is shown in Figure 21. During this period, due to the continuous adjustment and movement of the roof strata of the drawing roadway, the pressure changes of the pillar side and the roof are more intense, which has a great influence on the relative moving speed of the roadway. The relative displacement of the roof is greater than the relative displacement of the floor, the maximum relative moving displacement of the roof is 142 mm, the relative displacement of the pillar side is larger than the relative displacement of the solid coal side, the maximum relative displacement of the pillar side is 159 mm, and the self-stabilization time of the roadway is approximately 28 days. There is no failure of bolt (cable), single prop and 29U type steel pressure loss, and large deformation of roadway surrounding rock in the application site of the mining roadway. The deformation of roadway is small, which realizes the stability control of the surrounding rock of the coal seam roadway under the extremely close coal seam.



FIGURE 23: Uniaxial compression test of laboratory grouting block strength.



FIGURE 24: The drawing force of the grouting anchor cable.

6. Conclusion

(1) In the mining under the double gobs in close coal seams, when mining double gobs on one side of the stepped coal pillars, there is a "slip line" plastic zone distribution. The depth of the plastic zone increases from 7.5 m to 11.5 m, and the peak supporting stress of the upper coal pillar is 14 MPa. When mining double gobs on both sides of the stepped coal pillars, the upper coal pillar completely enters the plastic zone, the center of the lower coal pillar appears to be a triangular elastic zone, and the peak support stress increases to 24 MPa. The position of the stabilizing zone is 4 m horizontally from the stepped coal pillar and vertically 10.5 m away from the upper coal seam, and the depth of the stabilizing zone is 7.5 m, 10.5 m,

and 13.5 m in the floor. The reasonable layout position of the roadway is in the low stress stabilizing zone, and the stress concentration of the coal pillar will be avoided.

- (2) The actual drawing force of the anchor bolt and the anchor cable is only approximately 50 kN and 80 kN, respectively, in the thick soft coal fully mechanized caving roadway under double gobs in the lower coal seams. After the grouting anchor cable is used, the drawing force of the anchor bolt can reach approximately 160 kN, and the drawing force of the anchor cable can reach approximately 350 kN. In the numerical simulation, when the grouting is not performed, the broken surrounding rock causes the prestress of the anchor cable to be released, which cannot form a straight-line stress limit of 0.02 MPa. However, after the grouting, the width and height of the stress field formed by the prestress of the anchor cable in the surrounding rock increase, and an effective prestress field with the straight-line stress limit of 0.02 MPa can be formed.
- (3) Laboratory tests on the strength of the crushed coal body with grouting: after 3 days of grouting, the strength of the grout can reach 62.5% of its maximum strength, and after 7 days of grouting, its strength can reach 87.5% of its maximum strength. The grouting anchor cable + single prop + 29U type steel joint support technology was proposed to be used to improve the overall support strength and achieve the effect of surface protection and deep surrounding rock reinforcement in roadway support. Grouting anchor cables can repair and strengthen the damaged surrounding rock and form an effective prestressed bearing structure in the shallow and deep roof of the roadway. The surrounding rock of the roadway under double gobs in close coal seams has been stably controlled.

Data Availability

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

Conflicts of Interest

The authors declare that they have no conflicts of interest.

Acknowledgments

This study was financially supported by the National Natural Science Foundation of China (grant nos. 52074296, 52004286, and 51674263), the China Postdoctoral Science Foundation (grant nos. 2020T130701 and 2019M650895), the Fundamental Research Funds for the Central Universities (grant no. 2020YJSNY06), and the Yue Qi Young Scholar Project, China University of Mining and Technology, Beijing (grant no. 800015Z1104), as well as "Strong Strata Behaviors Mechanism and Control of the Longwall Face Mining in a Shallow Depth Seam in Proximity beneath a Room Pillar Mining Goaf" (grant no. 2018LH05009). The authors would like to thank the continuous support from the Zhengwen Coal Mine during the on-site sampling and monitoring.

References

- W. Zhang, D. Zhang, D. Qi, W. Hu, Z. He, and W. Zhang, "Floor failure depth of upper coal seam during close coal seams mining and its novel detection method," *Energy Exploration & Exploitation*, vol. 36, no. 5, pp. 1265–1278, 2018.
- [2] G. Wu, X. Fang, H. Bai, M. Liang, and X. Hu, "Optimization of roadway layout in ultra-close coal seams: a case study," *PLoS ONE*, vol. 13, no. 11, p. e0207447, 2018.
- [3] J. Xue, Z. Fan, and Z. Huang, "Determination on rational location of two gateways in coal mining face under gob with ultra short distance to mining seam," *Coal Science and Technology*, vol. 40, pp. 37–41, 2012, in Chinese.
- [4] G. S. Cai, Study on Roadway Layout & Supporting in Ultraclose Distance Coal Seams Mining of Zhongxing Mine, PhD Thesis, China University of Mining and Technology, (Beijing), China, 2013.
- [5] C. Sun, B. Yang, and C. Liu, "Strata behavior regularity of 1.0 m very contiguous seams combined mining," *Journal of China Coal Society*, vol. 36, pp. 1423–1428, 2011.
- [6] Y. Xu, S. Liu, Z. Liu et al., "Overburden failure laws in working face of short distance thick coal seams group," *Journal of mining and safety engineering*, vol. 30, pp. 506–511, 2013.
- [7] X. Yang, G. Wen, L. Dai, H. Sun, and X. Li, "Ground subsidence and surface cracks evolution from shallow-buried close-distance multi-seam mining: a case study in bulianta coal mine," *Rock Mechanics and Rock Engineering*, vol. 52, no. 8, pp. 2835–2852, 2019.
- [8] J. Bai, "Experimental investigations on the progressive failure characteristics of a sandwiched coal-rock system under uniaxial compression," *Applied Sciences (Switzerland)*, vol. 9, 2019.
- [9] J. Zhang, Q. Sun, A. Fourie, F. Ju, and X. Dong, "Risk assessment and prevention of surface subsidence in deep multiple coal seam mining under dense above-ground

buildings: case study," Human and Ecological Risk Assessment: An International Journal, vol. 25, no. 6, pp. 1579–1593, 2019.

- [10] H. Yan, M.-y. Weng, R.-m. Feng, and W.-K. Li, "Layout and support design of a coal roadway in ultra-close multipleseams," *Journal of Central South University*, vol. 22, no. 11, pp. 4385–4395, 2015.
- [11] J. F. Ju and J. L. Xu, "Prevention measures for support crushing while mining out the upper coal pillar in close distance shallow seams," *Caikuang Yu Anquan Gongcheng Xuebao/Journal of Mining and Safety Engineering*, vol. 30, pp. 323-330, 2013.
- [12] J. Ju, J. Xu, W. Zhu et al., "Influence of cut-hole position on support crushing during mining out coal pillar at the side of upper goaf in close distance of shallow seams," *Yanshilixue Yu Gongcheng Xuebao/Chinese Journal of Rock Mechanics and Engineering*, vol. 33, pp. 2018–2029, 2014.
- [13] J. Ju, J. Xu, and W. Zhu, "Influence of overlying key strata structure pre-sliding on support failure disaster while mining in the lower coal seam cut across below the upper adjacent coal pillar under shallow cover," *Journal of China Coal Society*, vol. 9, pp. 2033–2039, 2015.
- [14] S. Gu, B. Jiang, G. Wang, H. Dai, and M. Zhang, "Occurrence mechanism of roof-fall accidents in large-section coal seam roadways and related support design for bayangaole coal mine, China," *Advances in Civil Engineering*, vol. 2018, p. 1, Article ID 6831731, 2018.
- [15] S. Zhang, D. Zhang, H. Wang, and S. Liang, "Discrete element simulation of the control technology of large section roadway along a fault to drivage under strong mining," *Journal of Geophysics and Engineering*, vol. 15, no. 6, pp. 2642–2657, 2018.
- [16] G. an, "Zhu "case study of passive seismic velocity tomography in rock burst hazard assessment during underground coal entry excavation," *Rock Mechanics and Rock Engineering*, vol. 49, pp. 4945–4955, 2016.
- [17] H. Jia, L. Wang, K. Fan, B. Peng, and K. Pan, "Control technology of soft rock floor in mining roadway with coal pillar protection: a case study," *Energies*, vol. 12, no. 15, p. 3009, 2019.
- [18] Y. Gao, Y. Wang, J. Yang, X. Zhang, and M. He, "Meso- and macroeffects of roof split blasting on the stability of gateroad surroundings in an innovative nonpillar mining method," *Tunnelling and Underground Space Technology*, vol. 90, pp. 99–118, 2019.
- [19] Y. Zhao, C. Liu, Y. Zhang, J. Yang, and T. Feng, "Damaging behavior investigation of an operational tunnel structure induced by cavities around surrounding rocks," *Engineering Failure Analysis*, vol. 99, pp. 203–209, 2019.
- [20] Z. Zhang, F. Chen, N. Li, and M. He, "Influence of fault on the surrounding rock stability for a mining tunnel: distance and tectonic stress," *Advances in Civil Engineering*, vol. 2019, p. 1, Article ID 2054938, 2019.
- [21] 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, no. 7, p. 1375, 2019.
- [22] S. Zhang, W. Meng, Z. Song, Z. Liu, and M. Guo, "Roof control technologies of lower layer mining in close distance coal seams," in *Proceedings of the 38th International Conference on Ground Control in Mining, ICGCM 2019 (Society for Mining, Metallurgy and Exploration*, pp. 319–3, Morgantown, WV, USA, July 2019.

- [23] B. Zhang, "Study on the surrounding rock control theory and technology of ultra-close multiple-seams min," *Taiyuan University of Technology*, vol. 272008, in Chinese.
- [24] L. Q. Shi, D. J. Xu, M. Qiu, X. Jing, and H. H. Sun, "Improved on the formula about the depth of damaged floor in working area," *Meitan Xuebao/Journal of the China Coal Society*, vol. 38, pp. 299–303, 2013.
- [25] Z. Guo, L. Zhang, Z. Ma, F. Zhong, J. Yu, and S. Wang, "Numerical investigation of the influence of roof fracturing angle on the stability of gob-side entry subjected to dynamic loading," *Shock and Vibration*, vol. 2019, p. 1, Article ID 1434135, 2019.
- [26] M. Shabanimashcool and C. C. Li, "Numerical modelling of longwall mining and stability analysis of the gates in a coal mine," *International Journal of Rock Mechanics and Mining Sciences*, vol. 51, pp. 24–34, 2012.
- [27] G. Li, F. Ma, G. Liu, H. Zhao, and J. Guo, "A strain-softening constitutive model of heterogeneous rock mass considering statistical damage and its application in numerical modeling of deep roadways," *Sustainability*, vol. 11, no. 8, p. 2399, 2019.
- [28] W. He, F. He, and Y. Zhao, "Field and simulation study of the rational coal pillar width in extra-thick coal seams," *Energy Science & Engineering*, vol. 8, no. 3, pp. 627–646, 2020.
- [29] W. Yu, G. Wu, B. An, and P. Wang, "Experimental study on the brittle-ductile response of a heterogeneous soft coal rock mass under multifactor coupling," *Geofluids*, vol. 2019, p. 1, Article ID 5316149, 2019.
- [30] W. Yu and F. Liu, "Stability of close chambers surrounding rock in deep and comprehensive control technology," *Advances in Civil Engineering*, vol. 2018, p. 1, Article ID 6275941, 2018.
- [31] 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, pp. 1195–1204, 2018.
- [32] J. Wang, J. Ning, P. Qiu, S. Yang, and H. Shang, "Microseismic monitoring and its precursory parameter of hard roof collapse in longwall faces: a case study," *And Engineering*, vol. 17, pp. 375–383, 2019.
- [33] Z.-L. Zhong, X.-X. Hu, W.-K. Luo, and Y. Ren, "Continuousdiscrete coupling-based stability study of surrounding rock of limestone roadway in acid environment," *Arabian Journal of Geosciences*, vol. 12, no. 9, 2019.
- [34] Z. Cheng, "Study on the reasonable layout of gob-side remained gateway of lower coal seam close to coal seam group," *Caikuang Yu Anquan Gongcheng Xuebao/Journal of Mining and Safety Engineering*, vol. 32, pp. 453–458, 2015.
- [35] G. Liu, F. Ma, H. Zhao, G. Li, J. Cao, and J. Guo, "Study on the fracture distribution law and the influence of discrete fractures on the stability of roadway surrounding rock in the Sanshandao coastal Gold Mine, China," *Sustainability*, vol. 11, no. 10, p. 2758, 2019.
- [36] Y. K. Liu, "Study on the depressurization effect produced by exploitation of lower distant protective coal seam and elimination of outburst hazard of the protected seams by applying gas drainage with surface boreholes," *Meitan Xuebao/Journal* of the China Coal Society, vol. 37, pp. 1067-1068, 2012.
- [37] W. Yu, B. Pan, F. Zhang, S. Yao, and F. Liu, "Deformation characteristics and determination of optimum supporting time of alteration rock mass in deep mine," *KSCE Journal of Civil Engineering*, vol. 23, no. 11, pp. 4921–4932, 2019.