Research Article

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Deformation and failure mechanism of full seam chamber with extra-large section and its control technology

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Abstract: In order to effectively predict and control the rib spalling and roof leakage, it is necessary to reveal the deformation and failure mechanism of the chamber and propose the corresponding surrounding rock control technology. Based on uniaxial compression experiments and numerical simulations, it is concluded that coal body damage is dominated by shear failure during uniaxial compression, which indicates to some extent the main form of damage of the surrounding rock in the chamber. Then the combined finite and discrete element method is used to establish a numerical model to reveal the evolution law of fracture in the surrounding rock. The simulation results show that after the excavation of the chamber, a large amount of shear failure occurred in the ribs and the roof. Then those cracks further developed, expanded, penetrated, and finally connected with the surface of the chamber. Under the effect of the mine pressure, the coal body is separated from the surface of the chamber, leading to the occurrence of rib spalling and roof leakage. So it was given that support method by controlling crack development. The grouting and high-strength anchor bolt and anchor cable are proposed to improve the shear strength of the surrounding rock, which helps to reduce the occurrence of cracks, and inhibit the cracks from interpenetrating. An industrial test was carried out in the chamber of Tashan Coal Mine, which showed good control effect of the surrounding rock in the chamber.

Keywords: full seam chamber, FDEM, deformation, failure, control technology

1 Introduction

Coal is China's major primary energy source, accounting for more than 70% of the total energy consumption [1,2]. According to the statistics, more than 50% of China's current coal reserves are thick and extra-thick coal seams [3]. For the thick and extra-thick coal seams, domestic and foreign scholars have developed large mining height method and top coal caving method [4,5]. With the maturity of these two technologies, the mine coal production has increased quickly and the section area of the chamber has been increasing [6]. Generally speaking, the width of the chamber is 6–10 m and the height is 8–10 m in the top coal caving method. Therefore, the chamber area is above 70–100 m² [7]. When the strength of the coal seam is high, putting the large-section chamber into seam can not only increase coal production but also significantly accelerate the excavation speed [8]. Due to the complicated stress conditions, large chamber height, span, and disturbance during excavation, the roof and the two ribs of the chamber will be deformed and destroyed. In the worst situation, rib spalling and roof leakage could occur. In order to improve the anti-deformation ability and stability of the large-section full seam chamber, domestic and foreign scholars have adopted different research methods to optimize the cross-sectional shape of the chamber. It is concluded that the shape, straight wall with semi-circular arch, is not only relatively easy to excavate and support, but it also has a stable structure and is easy to maintain. Therefore, it is widely used in coal mines.

At present, the studies on deformation and the failure of the large-section chamber or roadway mainly includes the following. Coggan et al. [7] used the Fast Lagrangian Analysis of Continua (FLAC), a differential numerical simulation software, to reveal the influence of weak roof on the stability of the large-section roadway. Tonon et al. [9] studied the effect of grouting on plastic failure of the 260 m² large-section chamber under the coal mining area. Zheng et al. [10] revealed the failure

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mechanism of anchor resin coils for the large-section roadway based on the theoretical calculations. Xiong et al. [11] established the three-dimensional numerical models of four excavation schemes for the chamber and revealed the characteristics of the plastic zone of the surrounding rock under these excavation methods. Then the excavation method was optimized. Zhang et al. [12] used FLAC to study the dynamic response of coal mine refuge chamber for underground gas explosion. Zhai et al. [13] used the FLAC to study the control of deformation of the surrounding rock of the large-section chamber by means of bolt support and open-roof groove. Islam et al. [14] studied the influence of gas outburst on roadway of the plastic zone. Yan et al. [15] proposed a new method to predict the roof breakage of the large-section coal roadway in the extra-thick coal seam. Jiang et al. [16] studied the influence of foundation rigidity of the direct roof on the stability of the large-section roadway.

The above studies mainly focus on the use of continuous numerical simulation software to reveal the development law of the plastic zone of the surrounding rock of the chamber or roadway or to analyze their stability based on the theoretical calculations. However, the failure of the surrounding rock of the chamber is a process of initiation, extension, and penetration of the crack. The above two methods cannot simulate the crack development of the surrounding rock of the chamber. Therefore, this article proposed the method of combined finite and discrete element method (FDEM). The FDEM is used to simulate the surrounding rock crack development of the chamber, which can accurately reveal the deformation and failure mechanism. Then the

corresponding control measures are proposed to avoid damage to the personnel and equipment caused by rib spalling and roof leakage accidents.

This article takes the chamber of Tashan Coal Mine as the engineering background and reveal the crack development of coal body based on compression test and numerical calculation. Then the FDEM is used to establish the numerical model of the surrounding rock of the chamber. The crack development of the surrounding rock of the chamber is analyzed using the model to reveal the mechanisms of rib spalling and roof leakage. Based on the failure mechanism of the chamber surrounding rock, a support method the combines grouting and high-strength anchor bolt and anchor cable is proposed, and the industrial test is carried out in the chamber of Tashan Coal Mine.

2 Engineering background

2.1 Geological condition of coal mine

As shown in Figure 1, Tashan Coal Mine is in Datong Coalfield, Datong City, Shanxi Province, China, and is mining carboniferous 3–5# coal seam. The thickness of the carboniferous 3–5# coal seam is between 14 and 20 m, the buried depth is 400–800 m, and the dip angle of the coal seam is between 1 and 4°. The 3–5# coal seam is generally stable, with abundant coal reserves and relatively simple geological structure conditions. Therefore, the Tashan coal mine in Datong coal field produces more than 15 million tons for one working face per year.

Figure 1: Location of Tashan Coal Mine.

Figure 2: Rock stratum bar graph.

In order to meet the requirements of coal transportation, Tashan Coal Mine arranges the 1,070 transportation main road in the floor of 3–5# coal seam. According to the borehole near the 1,070 transportation main road, the thickness of the 3–5# coal seam is 18.14 m. The occurrence of rock strata is shown in Figure 2.

2.2 Chamber conditions

In order to meet the cross-sectional area requirements for the installation of mechanical and electrical equipment, it is necessary to expand it at the local position of the original 1,070 transportation main road and form a large-section full seam chamber. According to the design requirements, the width of the chamber is 9.5 m and the height is 11.85 m, as shown in Figure 3. Rib spalling and roof leakage occurred during chamber excavation, which not only has a serious impact on the safety of the personnel and equipment but also makes it difficult to support the chamber. Therefore, it is first necessary to reveal the deformation and failure mechanism of the surrounding rock of the chamber. Then the corresponding surrounding rock control technology should be proposed.

Figure 3: Chamber dimensions.

of the two ends of the coal sample is less than 0.05 mm, and the diameter deviation of the upper and lower ends is less than 0.3 mm. The surface should be smooth and avoid stress concentration due to irregular surfaces. The coal sample is placed in the MTS815 Flex Test rock mechanics test system, and the PCI-II acoustic emission test system is used for the real-time positioning test of the whole process of uniaxial compression.

In order to further analyze the number of cracks caused by shear failure and tensile failure of coal samples during uniaxial compression and reveal the main failure modes in the compression process, a numerical simulation of a set of uniaxial compression is added. The Universal Distinct Element Code (UDEC) numerical calculation software was used to simulate the development characteristics of the fissures of rock blocks during uniaxial compression. The UDEC is a two-dimensional numerical program that simulates the quasi-static or dynamic response to loading of media containing multiple intersecting joint structures.

3 Coal body failure characteristics during uniaxial compression

3.1 Coal sample preparation and experimental equipment

As shown in Figure 4, the coal pieces, selected in the field, are cut into a standard coal sample with a diameter of 50 mm and a height of 100 mm. The non-parallelism

3.2 Coal sample failure characteristics

It can be seen from Figure 5(a) that the uniaxial compressive stress–strain curves of the experiments and numerical simulations are basically consistent, which verify the reliability and accuracy of the simulation. Further, the stress–strain process of the coal sample can be divided into the compaction phase, linear elastic phase, plastic deformation phase, strain-softening phase, and residual strength phases [17], the corresponding strain ranges of which are $0-8.5 \times 10^{-4}$, $8.5 \times 10^{-4}-1.95 \times 10^{-3}$, $1.95 \times 10^{-3}-2.3 \times 10^{-3}$, $2.3 \times 10^{-3}-2.75 \times 10^{-3}$, and $2.75 \times 10^{-3}-3.0 \times 10^{-3}$, respectively [18].

Figure 4: Coal sample preparation and test equipment.

- (1) Compaction phase: in the stress–strain relationship, as the axial stress increases, the original micro-cracks and pores inside the coal sample are closed, and the coal sample is gradually compacted. When there is an increase in the axial stress, a continuous decrease was observed in the rate of strain increase and the stress–strain curve bends downward. While monitoring the acoustic emission, small energy was observed in the micro-cracks and pore closure process of the coal sample. From the simulation it can be observed that the number of cracks in the coal sample that are caused by shear failure and tensile failure is less and increases slowly.
- (2) Linear elastic phase: as the axial stress continues to increase, the coal sample enters the linear elastic phase. The stress–strain curve of the coal sample is almost linear in this phase. While monitoring the acoustic emission, the acoustic emission (AE) signal becomes active and the AE spatial positioning points become obviously denser. New and original micro-cracks extend and connect. Finally, the micro-cracks
- penetrate each other to form macroscopic cracks. In this phase it has been observed from the simulation that the number of fractures caused by shear failure and tensile failure has significantly increased, with the number of fractures caused by shear failure being significantly more than that of tensile failure.
- (3) Plastic deformation phase: after the linear elastic deformation phase, the presence of mineral particles and micro-fracture pores in the coal sample results in the uneven distribution of stress in the sample, causing local stress concentration. The micro-cracks continue to develop and connect with each other, eventually becoming macroscopic cracks.
- (4) Strain-softening phase: the specimen is broken and its bearing capacity is reduced. When monitoring the acoustic emission, we can still observe the acoustic emission events in the coal sample, i.e., significance reduction in energy release, which is mainly caused by micro-rupture events. Considering the number of fractures, the number of cracks caused by shear failure and tensile failure tends to be stable.

Figure 5: Uniaxial compression for coal sample. (a) Uniaxial compressive stress–strain curves; (b) Typical fracture characteristics of coal rock.

- (5) Residual strength phases: when a rock reaches a residual state, the shear cracks and tensile cracks also reach the stable state. At this time, the bearing capacity of the surrounding rock is maintained at a constant value.

A high-power microscope is used to observe the crack structure, as shown in Figure 5(b). The result shows that the crack section is mainly shear failure, while the tensile failure is relatively less.

The above analysis shows that the stress–strain curve of the coal sample, the intensity and quantity of the acoustic emission AE event, and the number of cracks have a good correspondence at different phases. During the compression process of the coal sample, cracks are mainly produced in the linear phase and are mainly due to shear failure. To a certain extent, this indicates that the surrounding rock of the large-section full seam chamber is mainly destroyed by the shearing force under the effect of the stope stress.

4 The combined FDEM

Traditional methods often use continuous numerical methods to analyze the plastic development of the surrounding rock and reveal the damage area of the chamber. However, the destruction of the surrounding rock of the chamber is a dynamic process of fissure generation, expansion, and penetration. Therefore, the continuous numerical method cannot simulate the development process of cracks and reveal the internal mechanism of deformation and failure of the surrounding rock of chamber. Therefore, it is necessary to combine the continuous numerical method with the discrete element method. The combined FDEM is an effective way to solve the above problem. The finite element method can be used for analyzing the stress evolution of the chamber

surrounding the rock, while the discrete element method can be used for analyzing the surrounding rock crack development. These methods are complementary to each other and together they can reveal the complex process of deformation and destruction of the surrounding rock.

4.1 Basic principle of the FDEM

The FDEM is a new simulation method that combines finite element and discrete element methods. As shown in Figure 6, the surrounding rock of the chamber is divided into multiple Voronoi polygons. In each Voronoi polygon, triangular finite elements are used for filling the polygon, while cohesive four-node joint elements are inserted between the Voronoi elements [19]. The stress of continuous medium is approximated using the finite element method before the material is damaged. When the material stress reaches its strength value, the crack propagation is directly reflected by the failure of the joint element. Then the friction of the fracture surface and the displacement of the fracture body are calculated by the discrete element method.

The core theories of FDEM mainly include the motion control equations, the model of fracture propagation, the contact retrieval method, and the contact force calculation method.

4.2 Governing equations

The governing equation of the FDEM is

$$M \frac{\partial^2 x}{\partial t^2} + C \frac{\partial x}{\partial t} = F \quad (1)$$

where M and C are nodal mass matrix and viscosity damping matrix, respectively. In addition, x is the nodal displacement vector and F is the nodal force vector. The

Figure 6: Discrete process and element composition of the FDEM.

nodal force includes the elastic deformation force of the element, the deformation force of the joint element, and the contact force between the discrete elements.

In order to simulate the kinetic energy consumption in the quasi-static process and to accelerate the convergence rate, the damping force is added. The damping matrix C of the FDEM is [20]:

$$C = \mu I \quad (2)$$

where μ is the viscosity damping coefficient and I is the identity matrix. The μ in numerical simulation can speedup the convergence of numerical calculations. However, a larger value of μ will reduce the accuracy of the simulation results. The μ is usually 1.0×10^{-4} – 1.0×10^{-3} . In numerical simulation, the value of μ is 5.0×10^{-4} , which has been obtained through repeated debugging and comparison with the experiment test.

4.3 Model of fracture propagation

In order to simulate the fissure evolution process of the surrounding rock of the chamber, the surrounding rock is divided into Voronoi elements, as shown in Figure 7(a). The model of fracture propagation for Voronoi elements is shown in Figure 7(b) [21].

The expressions of K_n and K_s in the model have been given as follows [22,23]:

$$K_n = 10 \left[\frac{K + \frac{4}{3}G}{\Delta Z_{\min}} \right] \quad (3)$$

$$K_s = 0.4K_n \quad (4)$$

where $\Delta Z_{\min}$ is the minimum width of adjacent contact elements in the normal direction. K and G are the bulk modulus and shear modulus of the rock, respectively, the expressions of which are given as [24]:

$$K = \frac{E}{3(1 - 2\mu)} \quad (5)$$

$$G = \frac{E}{2(1 + \mu)} \quad (6)$$

In the above equations, E and μ are the elastic modulus and Poisson's ratio of rock, respectively.

To further calibrate K_n and K_s , as shown in Figure 7(c), the FDEM is used to simulate uniaxial compressive, tensile, and shear tests. The stress-strain curves are obtained, which are compared with experiments to obtain more accurate K_n and K_s through multiple adjustments.

4.4 Contact retrieval method

At each time step during the simulation, the position of each element and thus their relative position need to be updated. When the FDEM simulates a non-continuous block system, the contact between the original block and the new rupture block is calculated in real time with the movement between the blocks [25]. This first requires retrieving which blocks may have contact behavior and deleting contacts that have been detached. A non-binary tree (NBS) contact search method was proposed [25]. As shown in Figure 8, the overall idea is to divide the two-dimensional computational domain into square grids of equal size and then map all the discrete elements to the corresponding grid according to their own coordinates. Then the contact search is performed within the elements in each grid and its neighboring grids [26,27]. To gain more knowledge about specific search process, the interested readers can refer to the article "Numerical comparison of some contact detection algorithms" [25].

4.5 Contact force calculation method

The FDEM uses the penalty function method to solve the interaction force between the contact pair. The contact force calculated by the traditional penalty function method belongs to the concentrated force, which only considers the intrusive overlapping displacement of the contact pair. This method is more succinct, but it differs greatly from the real contact. In contrast, the distributed contact force is used in FDEM [28]. It uses the triangular constant strain element as the finite element, which simplifies the contact judgment of the contact pair and the deformation simulation of the element. The two elements of a contact pair are, respectively, referred to as the contact element and the target element, and their overlapping area is denoted as S . Assume that a small force on the contact element that must be produced by the target element in an infinitesimal element dA within the overlapping area equals to the difference in the potential function gradient of dA in these two elements [29], which can be expressed as [29]:

$$df = [\text{grad}\varphi_c(P_c) - \text{grad}\varphi_t(P_t)]dA \quad (7)$$

In the above equation, df is the small force produced by the infinitesimal element dA and P_c is the location of the infinitesimal element dA in the contact element. Similarly, P_t is the location of the infinitesimal element

Figure 7: Model of fracture propagation and parameter calibration. (a) Voronoi elements for the chamber; (b) Constitutive model; (c) Tests for calibrating the parameters.

Figure 8: Element and grid mapping and the element contact relationship.

dA in the target element. φ_c is the potential at the midpoint of the contact element.

The total contact force can be obtained by integrating the difference in potential function gradient with the total overlapping area S such that [30]:

$$f_c = \int_{S=\beta_c \cap \beta_t} [\text{grad} \varphi_c - \text{grad} \varphi_t] dA \quad (8)$$

For more details about the calculation of Voronoi element contact force, please refer to the article “A new penalty method for nonlinear programming” [31].

5 Deformation and failure mechanism of surrounding rock of full seam chamber with extra-large section

5.1 Numerical simulation

5.1.1 Mechanical properties of stope rock

The mechanical parameters of the intact coal rock near the chamber are obtained through experiments. See Table 1 for details. Due to the large number of irregular joints and cracks in the rock, the experimentally determined rock mechanic parameters are usually higher than those of the stope rock mass. In order to further obtain the mechanical parameters of stope rock mass, the rock quality designation (RQD) value is usually used to reduce the rock strength in engineering. In other

words, the elastic modulus E of the laboratory rock and the elastic modulus E_m of the stope rock mass have the following relationship [32]:

$$\frac{E_m}{E} = 10^{0.0186\text{RQD}-1.91} \quad (9)$$

Similarly, the ratio of the compressive strength σ_c of the laboratory rock to the compressive strength σ_{cm} of the stope rock mass σ_{cm}/σ_c has the following functional relationship with E_m/E [33]:

$$\frac{\sigma_{cm}}{\sigma_c} = \left(\frac{E_m}{E} \right)^q \quad (10)$$

where q is the empirical parameter and is taken as 0.7 in this study [34]. The uniaxial tensile strength is approximately 0.1 times of the uniaxial compressive strength [35].

Based on the experimentally obtained rock parameters, the mechanical parameters of the stope rock mass are calculated by equations (9) and (10). See Table 1 for details.

5.1.2 Strain softening model of coal and rock

The strain softening phenomenon occurs in the coal and rock body during the compression process, that is, after the stress reaches the peak strength, the strength of the coal body rapidly drops to a lower level as the deformation continues to increase [36]. Based on compaction experiments, the strain softening model of coal can be divided into the elastic phase, softening phase, and residual stress phase [37,38], as shown in Figure 9. The residual elastic modulus, cohesion, and internal friction angle required for the strain softening model are obtained experimentally. As shown in Figure 9, the accuracy of the strain softening model is verified by comparing the uniaxial compressive stress–strain curves of the coal sample in numerical simulation and compaction experiments.

5.1.3 Model establishment

The specific modeling process is as follows: as shown in Figure 10, according to the engineering background, a two-dimensional plane numerical model with a length of 150 m and a height of 54.43 m is established using the FDEM numerical method. The mesh size of the model is between 0.2 and 1.0 m [39]. The Mohr–Coulomb strain softening model is used for the coal body and rock mass. The horizontal displacement of the left and right boundaries of the model is constrained, while the vertical displacement of the bottom boundary is constrained. A uniform vertical load of 12.5 MPa is applied on the top boundary to replace the rock mass overlying 500 m [40].

5.2 Deformation and failure mechanism of surrounding rock

5.2.1 Ribs of the chamber

With the excavation of the full seam chamber with a large section, the surrounding rock of the chamber changes from the original three-direction stress state to the two-direction stress state and even a unidirectional stress state occurs. The compressive stress of the coal in the ribs of the chamber is concentrated under the effect of the floor and roof, where deformation and damage occur. It could also lead to the occurrence of rib spalling. The instability of the coal body in the coal roadway ribs will lead to an increase in the actual suspended area of the roof, which increases the probability of roof failure in the roadway. At the same time, the roadway rib spalling will also hurt the staff and damage the underground equipment.

Table 1: Mechanical parameters of rock and stope rock mass in Tashan Coal Mine

Lithology	Intact rock					RQD	Rock mass		
	E/GPa	σ_c/MPa	Poisson's ratio μ	Cohesion c/MPa	Internal friction angle $\varphi/^\circ$		E_m/GPa	σ_{cm}/MPa	σ_{tm}/MPa
Medium sandstone	45.5	69.4	0.26	10.5	23	90	26.4	47.4	4.7
Sandstone	36.2	23.6	0.32	8.2	21	89	20.1	15.7	1.6
Mudstone	15.8	17.4	0.29	5.5	19	57	2.2	4.4	0.4
3–5# coal seam	6.8	13.2	0.26	6.1	17	65	1.4	4.2	0.4
Mudstone	15.2	7.2	0.31	4.3	19	63	2.8	2.2	0.2
Coarse sand	35.1	75.6	0.25	9.2	24	91	21.3	53.3	5.3

Figure 9: Strain softening model of coal.

In order to further reveal the deformation and failure mechanism of the surrounding rock in the chamber ribs, the crack development in the local position of the chamber ribs is extracted as shown in Figure 11.

As shown in Figure 11(a), many cracks are generated in the chamber ribs under the effect of mine pressure and excavation disturbance. As shown in Figure 11(b), the main cause of the crack is that the shear stress on the rib exceeds its shear strength and thus shear failure occurs. Compared with the shear failure, the tensile failure is relatively less. As shown in Figure 11(c), under the effect of mine pressure, the cracks generated in the failure chamber ribs further expand and penetrate each other and finally connect with the free surface of the coal wall. As shown in Figure 11(d), under the effect of lateral stress, the coal body is separated from the ribs, leading to the occurrence of rib spalling.

Based on the crack development in the chamber ribs, the mechanism of rib spalling in chamber ribs is as follows: after the excavation of the chamber, under the effect of mine pressure and excavation disturbance, many

shear failures are produced in the surrounding rock of the ribs, which leads to the formation of a large number of cracks. The cracks then further develop, expand, and penetrate. When the crack is connected to the surface of the ribs, under the effect of the lateral pressing force, the coal body is separated from the ribs and thus rib spalling occurs.

5.2.2 Roof of the chamber

The destruction and instability of the chamber roof generally involve no precursor. Once the roof failure occurs, it is often sudden, and in most cases the failed roof has a large area, which can greatly harm the safety of equipment and personnel. Revealing the deformation and failure mechanism of the chamber roof can provide a theoretical basis for the prediction and control of roof leakage.

In order to further reveal the mechanism of crack development in the roof area, as shown in Figure 12, the process of the roof separation in the chamber is analyzed.

As shown in Figure 12(a), many cracks are produced in the chamber roof under the effect of mine pressure and excavation disturbance. As shown in Figure 12(b), the main cause of crack development is also the shear stress on the roof exceeding its shear strength, which leads to shear failure. Compared with shear failure, tensile failure generates relatively less cracks. As shown in Figure 12(c), after the internal cracks of the roof are initiated, they are further expanded under the effect of mine pressure. As a result, they penetrate each other and are finally connected to the roof surface. As shown in Figure 12(d), the tensile strength and cohesion of the surrounding rock in the crack zone of the roof are drastically reduced. Under the effect of mine pressure and self-weight, the coal body is separated from the dome, leading to the occurrence of roof leakage.

Figure 10: Numerical model.

Figure 11: Rib spalling mechanism in the two ribs. (a) Cracks in the ribs; (b) Failure model; (c) Development characteristics of the cracks; (d) Rib spalling.

Based on the evolution law for cracks in the roof, the mechanism of roof separation in the chamber is as follows: after the excavation of the chamber, under the effect of mine pressure and excavation disturbance, many shear failures are produced in the roof of the surrounding rock, which leads to the formation of cracks. The cracks further develop, expand, and penetrate. When the crack is connected to the roof surface, under the effect of mine pressure and self-weight, the coal body is separated from the roof and thus roof leakage occurs.

6 Control method for the surrounding rock based on crack development characteristics

It can be seen from the deformation and failure mechanism of the chamber with large section that the cause of the surrounding rock failure is mainly shear failure. Therefore, improving the shear strength of the surrounding rock and inhibiting the penetration of cracks are the main ways to control the deformation and failure of the surrounding rock

of the chamber. After the excavation of the chamber, the stress of the surrounding rock is released. It is more effective for controlling the deformation and destruction of the surrounding rock of the chamber via increasing the shear strength of the surrounding rock and inhibiting crack penetration. At present, the main reinforcement methods for the broken surrounding rock include grouting reinforcement, high-strength anchor bolt, and anchor cable. Under the effect of the pumping pressure, the slurry can penetrate, fill and close the cracks in the surrounding rock, improve the integrity of the surrounding rock mass, and thus increase the shear strength of the surrounding rock. On the other hand, the action of the anchor on the surrounding rock can improve the shear strength of the fracture surface of the surrounding rock [41,42].

6.1 Anchor bolt and anchor cable parameters

The selection of anchor bolt and anchor cable parameters is the core problem of support, which includes the spacing, length, and pre-tightening force of anchor bolt and anchor cable. Reasonable anchor bolt and anchor cable parameters

Figure 12: Mechanism of roof separation. (a) Cracks in the roof; (b) Failure model; (c) Development characteristics of the cracks; (d) Rib separation.

can inhibit the shear failure of the chamber surrounding the rock and avoid the mutual penetration of the cracks. Taking the spacing between anchor bolts as an example, numerical simulation is used to study the development of shear failure cracks in the surrounding rock of the chamber when the spacing is 1.5, 1.2, 0.9, and 0.6 m. The results are shown in Figure 13.

It can be seen from Figure 13 that when the spacing between the anchor bolts is 1.5 and 1.2 m, many cracks can be observed in the surrounding rock of the chamber. The cracks initiate, develop, and penetrate each other. For these cases, the surrounding rock is seriously damaged. When the bolt spacing is 0.9 m, the shear cracks of the surrounding rock of the chamber are obviously reduced. The cracks only initiate and develop. Therefore, there is basically no penetration between them. For this case, the shear failure of the chamber surrounding the rock is relatively less. The deformation and failure of the surrounding rock are well controlled. When the bolt spacing is reduced to 0.6 m, the cracks in the surrounding rock of the chamber do not change much, basically similar to the case with a spacing of 0.9 m.

Based on the above rules, when the bolt spacing is 0.9 m, the cracks generated by the shearing of the surrounding rock of the chamber are well controlled. As a result, the deformation and failure of the surrounding rock are not obvious. Therefore, the spacing of the bolts is determined to be 0.9 m.

Similar methods are used to study other parameters of anchor bolt and anchor cable. The final optimized support parameters are as follows: The bolt spacing of is 900 mm $\times$ 900 mm, pre-tightening force is 100 kN. The length of dome anchor cable is 2,500 mm, while the length of the rib anchor cable is 1,800 mm. The cable spacing is 2,700 $\times$ 1,800 mm. Its pre-tightening force and length are 150 kN and 8,300 mm, respectively.

6.2 Grouting parameters

Many factors affect grouting, such as the grouting pressure, the mechanical properties of the surrounding rock, the degree of crack development, the flow kinetic

Figure 13: Effect of bolt spacing on cracks in the surrounding rock of the chamber. (a) 1.5 m; (b) 1.2 m; (c) 0.9 m; (d) 0.6 m.

Figure 14: Grouting diffusion range.

Figure 15: Development of cracks in the surrounding rock of the chamber after grouting.

parameters of the slurry, and the initial setting time. Therefore, a numerical simulation method was adapted. After grouting, the grouting range of the surrounding rock of the chamber is shown in Figure 14.

It can be seen from Figure 14 that the slurry diffusion range basically covers the crack development area of the surrounding rock of the chamber. Therefore, grouting achieved its expected effect.

Figure 15 further shows the crack development of the surrounding rock of the chamber after grouting. It shows that after grouting, the number of cracks in the roof and the ribs of the chamber are obviously reduced. Also, the cracks only exist in the initiation and development stages, and no mutual penetration is observed. This indicates that grouting reinforcement improves the shear strength of the surrounding rock and effectively suppresses the occurrence of cracks in the surrounding rock.

7 Engineering applications

7.1 Chamber support scheme

According to the above analysis, the grouting pipe is used for grouting with a grouting pressure of 5 MPa. After grouting, the anchor bolt and cable support are carried out. The specific support parameters are as follows:

Dome support: the anchor bolt body consists of a left-handed steel without longitudinal reinforcement, with a yield strength of 600 MPa, a tensile strength of 800 MPa, a diameter of 22 mm, and a length of 2,500 mm. The bolt spacing is 900 mm × 900 mm. The anchorage is lengthened by resin and the bolt pre-tightening force is 100 kN. The tray is made of high-strength tray, the measures of which are 200 mm × 200 mm × 10 mm. A total of five anchor cables are involved, with a diameter of 17.8 mm, a length of 8,300 mm, a spacing of 2,700 × 1,600 mm, and a pre-tightening force of 150 kN. The anchoring length is 1,500 mm. Three resin rolls are used. The specification of one of the resin rolls is K2335, while the specification of the other two is Z2360. The

Table 2: Mechanical properties of the supporting materials

Properties	Values
Bolt/anchor cable	
Elastic modulus, GPa	210
Yield strength, kN	250/470
Pretightening force, kN	100/150
Resin roll stiffness, N/m/m	2e9
Resin roll cohesive force	4e5
W-shaped steel strip and metal mesh	
Elastic modulus, GPa	210
Tensile strength, MPa	500
Compressive strength, MPa	500
Normal contact stiffness, GPa/m	10
Contact shear stiffness, GPa/m	10

Figure 16: Deformation of the surrounding rock of the chamber.

W-shaped steel strip has a thickness of 4 mm and a width of 250 mm. Rib support: The anchoring length is 1,800 mm. Two resin rolls are used, whose specifications are K2335 and Z2360. Other parameters are the same as the dome. The mechanical parameters of anchor cable are shown in Table 2.

7.2 Field measurement of chamber deformation

The deformation of the surrounding rock of the chamber is monitored using the cross method [43]. As shown in Figure 16, four measuring points are arranged in the surrounding rock of the chamber, which are located at the two ribs, the dome, and the roof, respectively [44,45]. The monitoring results provide the magnitudes of the rib convergence and the roof-to-floor convergence.

It can be seen from Figure 16 that during the period of 0–16 days of support, the magnitudes of the rib convergence and the roof-to-floor convergence are increasing but the rate of increase is decreasing. After 16 days of support, the deformation of the surrounding rock of the chamber remained basically the same, with the magnitudes of the rib convergence and the roof-to-floor convergence stabilized at 12 and 38 mm, respectively.

Hence, based on the above discussion on deformation, it is concluded that the deformation and failure of the surrounding rock of the chamber is well controlled.

8 Conclusion

(1) Through uniaxial compression experiments and numerical calculations, the law of crack development and

evolution during the compression process of coal samples were revealed. It was concluded that shear failure is one of the causes of cracks, which mainly occur in the elastic phase.

- (2) The cracks generated in the surrounding rock of chamber are mainly caused by shear failure, while the tensile failure produces relatively few cracks. The roof separation and rib lateral displacement of the surrounding rock of chamber are closely related to the crack development. Both are mainly caused by the development and eventually the penetration of the cracks.
- (3) After the excavation of the chamber, the effect of the mine pressure and excavation disturbance causes many shearing damage to the ribs and the roof. As a result, many cracks are formed. The cracks further extend, expand, and penetrate with each other. When the crack is connected to the surface of the chamber, under the effect of the mining pressure, the coal body is separated from the surface of the chamber, leading to the occurrence of rib spalling and roof leakage.
- (4) Based on the failure mechanism of the surrounding rock in the chamber, a support method combining grouting and high-strength anchor bolt and cable was proposed to improve the shear strength of the surrounding rock, which helped to reduce the occurrence of cracks and inhibit the cracks from interpenetrating. Also, an industrial test was carried out in the chamber of Tashan Coal Mine, which showed that deformation and damage of the surrounding rock of the chamber is well controlled.

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