

Article

Dome Roof Fall Geohazards of Full-Seam Chamber with Ultra-Large Section in Coal Mine

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Featured Application: In this study, the mechanism of roof fall of full-seam ultra-large section chamber was uncovered and the corresponding control technology was put forward and successfully applied. The study provide a successful case of roof fall control in a special condition, and has a guiding and reference significance for ultra-large section chamber control during extra-thick coal seam mining, which is beneficial to ensure the production efficiency of coal mine.

Abstract: The roof fall hazard is more likely to take place within chamber with ultra-large section, which would not only damage mechanical equipment, but also cause casualties. In this paper, the strap joint chamber of the Tashan coal mine is studied, and finite and discrete element method (FDEM) is used to establish the numerical model of the roof fall of the chamber dome. The simulation results show that the chamber dome mainly undergoes shear failure and forms a large number of cracks. With further development and penetration of cracks, a distinct roof separation is found in the chamber dome. When the crack develops to the dome surface of the chamber, under the effect of the mine pressure, the coal body is separated from the surface of the chamber and the roof fall hazard occurs. Based on the mechanism of roof fall hazard of the chamber dome, it is concluded that improving the shear strength of the surrounding rock and reducing the crack penetration are the main ways to control the roof fall. Therefore, the high-strength anchor bolt and cable support is adopted to fill the cracks and improve the shear strength of the surrounding rock. The result showed that the roof separation of the chamber dome in the field is confined to 0.012 m. The surrounding rock is well controlled and no roof fall occurs.

Keywords: coal mine; ultra-large section; full-seam chamber; roof fall hazards; control technology

1. Introduction

China is a major coal producer, producing more than half of the world's coal annually [1,2]. Among the measured coal reserves in China, thick and extra-thick coal seams account for about 45% [3,4]. With the application and promotion of large mining height and top coal caving technology, thick and extra-thick coal seam become the main mined seam due to its high yield and efficiency [5,6]. Compared with traditional coal mining methods, large mining height and top coal caving methods have higher requirements for the sections of the underground strap joint chamber and washing chamber, etc. [7,8]. With the above two coal mining methods, the area of the chamber section is generally over 70~100 m². When the large-section chamber is arranged in the coal seam, it can not only speed up the excavation, but also increase the coal output and reduce the gangue discharge [9]. However, when excavating

the large-section chamber in the coal seam, the coal body often breaks off from the chamber dome and creates a roof fall hazard due to the long span and high height of the chamber and the relatively low strength of the coal body. There is usually no precursor to the failure of the chamber dome. Once the roof fall hazard occurs, it is often sudden. As shown in Figure 1, the roof fall hazard not only causes casualties, but also brings great psychological pressure to the workers in workplace, damages mechanical equipment and delays production, which adversely affects the normal production of coal mines [10]. Therefore, it is extremely important to control the roof fall hazard of the full seam chamber with ultra-large section. In order to control the roof fall hazard, it is necessary to uncover its mechanism, based on which, the corresponding control technology can then be proposed.

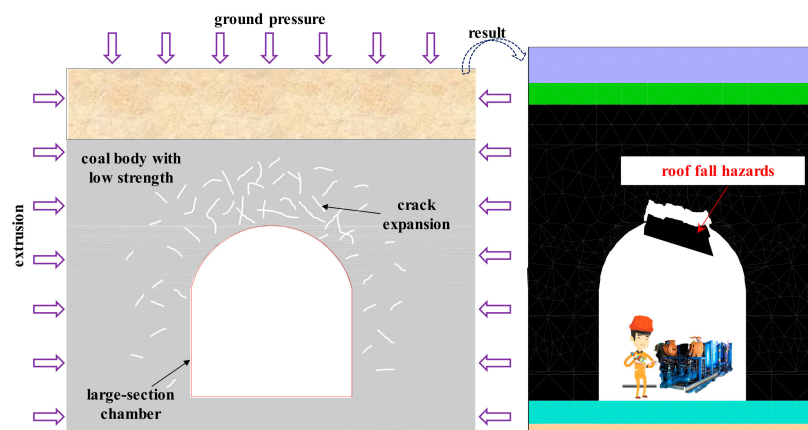


Figure 1. Crack formation and roof fall hazard diagram.

In view of the roof fall hazard of ordinary roadways under different geological conditions, scholars have conducted in-depth research from three aspects: prediction method; roof fall mechanism and control technology. In terms of roof fall prediction, Yan et al. [11] proposed a new method for predicting roof separation from the full-seam roadway based on theoretical analysis. Mahdevari et al. [12] proposed the use of artificial neural networks to predict roof stability of the roadway. Zhao et al. [13] proposed a kind of monitoring system for roof separation. In terms of the leakage mechanism, Bai et al. used UDEC and RS² software to reveal the roof fall mechanism for large span open-off cut under a rich aquifer [14,15]. Coggan et al. [16] used numerical simulation methods to reveal the mechanism of roof fall hazard of the roadway under weak immediate roof conditions. Islam and Shinjo [17] used the boundary element method to study the roof fall induced by gas outburst. Jiang et al. [18] studied the influence of foundation stiffness of the immediate roof on the stability of roadway roof. In terms of roof fall control technology, Jiao et al. [19] proposed a method for strengthening the chamber dome by U-shaped steel support for the roadway in the weak coal seam. Ju et al. [20] proposed an inner stagger layout to control the roadway roof fall for the layout of the lower coal seam roadway. Peng et al. [21] studied the effect of multi-echelon support of deep roadways in a kilometer-deep well based on experiments. By a systematic monitoring and evaluation of the seismic events, Vizintin Goran et al [22] pointed out that rock burst was closely related to the lignite extraction. In addition, the support type was put forward to decrease the deformation.

The above researches mainly focus on the prediction, mechanism and control technology of the roof fall hazard for ordinary roadway under different geological conditions, but none of them investigates the roof fall hazard mechanism of the full seam chamber with ultra-large section. The research on the subject would uncover the mechanism of roof fall hazard of the chamber dome, and provide theoretical basis and technical guidance for the roof fall hazard control.

In the paper, the strap joint chamber of Tashan Coal Mine of Datong Coal Group is taken as the engineering background, and the numerical simulation method is used to uncover the roof fall mechanism of the ultra-large section full-seam chamber. The industrial test results indicated that

the proposed technologies (namely grouting reinforcement, anchor bolt and cable support) have a satisfying controlling effect of roof fall hazard in the chamber at Tashan Coal Mine.

2. Engineering Background

2.1. Geological Conditions

As shown in Figure 2a, Tashan Coal Mine is located in Datong Coalfield, Datong City, Shanxi Province, China. The upper Jurassic coal seam in the minefield has been basically exploited, and now the main coal seam under extraction is the carboniferous. The thickness of the carboniferous 3–5# coal seam ranges from 14 m to 20 m, the buried depth is 400–800 m, and the dip angle of the coal seam is almost flat ($1\sim4^\circ$). As the 3–5# coal seam is basically stable and the coal reserves are abundant, the geological conditions are relatively simple, and the mining technical conditions are good, so the output of Tashan Coal Mine is over 30 million tons.

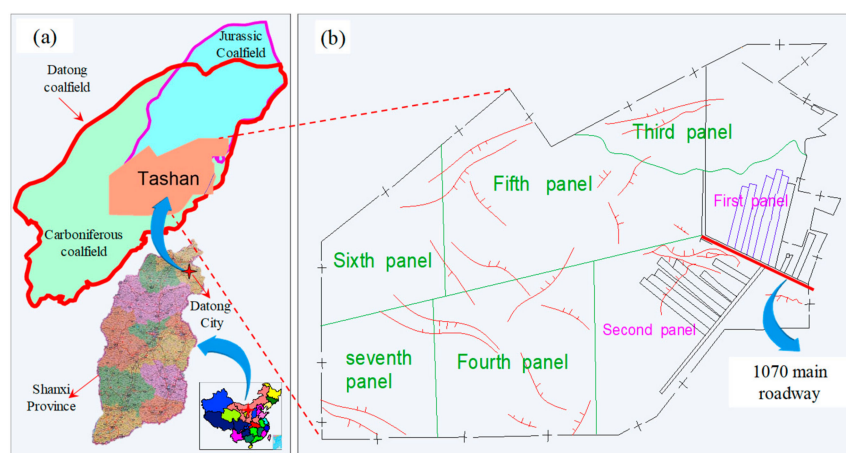


Figure 2. Location of Tashan Coal Mine and 1070 main roadway. (a) location of Tashan coal mine, (b) location of 1070 main roadway.

In order to meet the requirements of coal transportation, Tashan Coal Mine develops the 1070 main roadway in the 3–5# coal seam floor as shown in Figure 2b. According to the borehole near the 1070 main roadway, the thickness of the 3–5# coal seam is 18.14 m, and the occurrence of the rock stratum is shown in Figure 3.

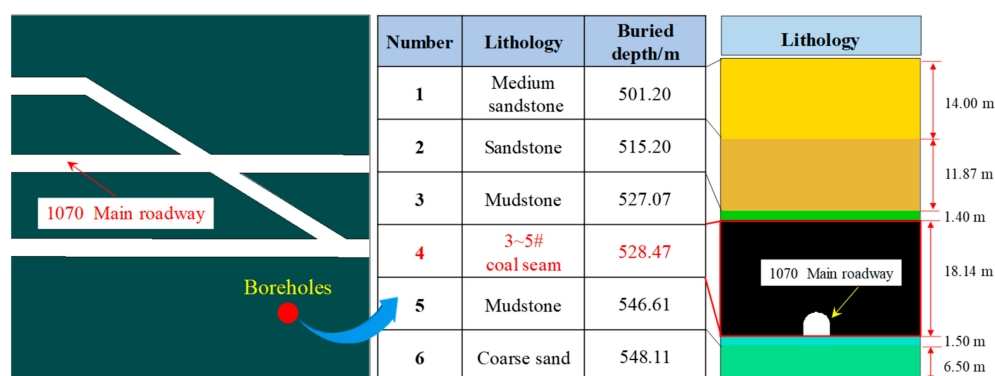


Figure 3. Rock stratum of the borehole near 1070 main roadway.

As shown in Figure 4, the 1070 main roadway has a width of 5350 mm and a height of 4275 mm. The length of 1070 main roadway was 2400m. The roadway support parameters are as follows:

Dome support: The anchor bolt body is 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 2200 mm. The bolt spacing is 800 mm $\times$ 800 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 $\times$ 200 mm $\times$ 10 mm. There is a total of five anchor cables, with a diameter of 17.8 mm, a length of 8300 mm, and a pre-tightening force of 150 kN. The anchoring length is 2500 mm. Four resin rolls are used. The specification of one of the resin rolls is K2335, while the other three are Z2360. W steel strip has a thickness of 4 mm and a width of 250 mm. A diamond-shaped metal mesh is used.

Rib support: High-strength anchor bolt, with the same material performance with that of chamber dome, is used for support. The anchor bolt spacing is 800 mm $\times$ 800 mm. The anchorage is lengthened by resin and the bolt pre-tightening force is 100 kN. W steel strip has a thickness of 4 mm and a width of 250 mm. A diamond-shaped metal mesh is used.

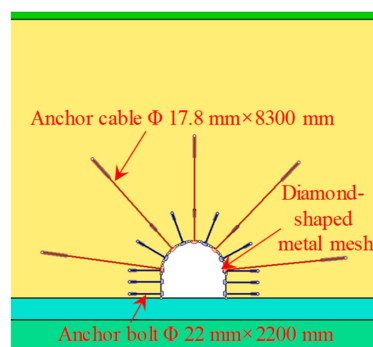


Figure 4. Roadway support plan.

2.2. Chamber Excavation Plan

In order to meet the installation requirements for equipment size, a local position of the 1070 main roadway is expanded to form a full-seam chamber with large section. According to the size of the electromechanical equipment, it is determined that the chamber has a width of 9.5 m and a height of 11.85 m. The specific expansion steps are shown in Figure 5.

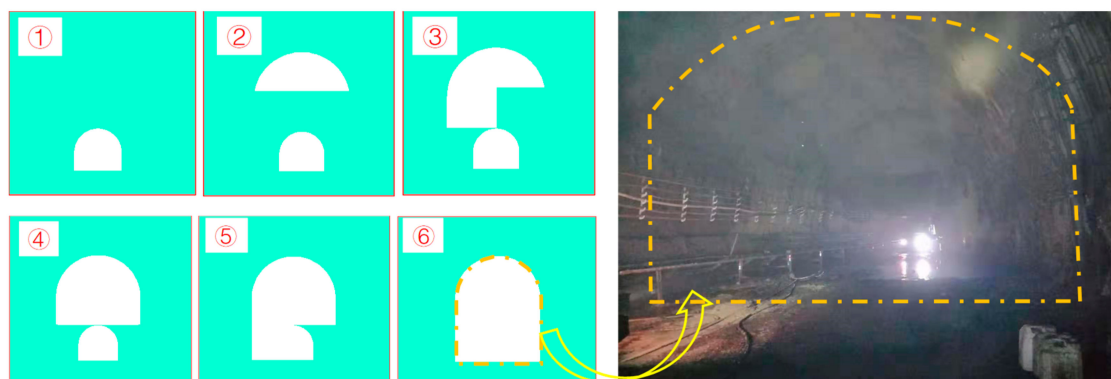


Figure 5. Chamber expansion steps.

In the process of expansion, it is necessary to ensure the normal transportation of coal and the safety of workers. Therefore, a U-shaped steel shed is erected in the 1070 main roadway. The steel shed is designed as a straight wall semi-circular arch with a shed distance of 500 mm. The sheds are linked by a round steel hook.

3. Numerical Model

3.1. Slope Rock Mass Strength

The surrounding rock of the chamber contains joints and cracks. Therefore, the strength of the rock sample measured in the experiment is generally higher than that of the chamber rock at stope. In order to simulate the reduction of the strength of the coal and rock mass, the Hoek-Brown criterion is proposed. Its specific expression is:

$$\sigma_1 = \sigma_3 + \sigma_{ci} \left(m_b \frac{\sigma_3}{\sigma_{ci}} + s \right)^a \quad (1)$$

where σ_1 is the maximum principal stress at failure, σ_3 is the minimum principal stress at failure. m_b is a reduced value (for the rock mass) of the material constant m_i (for the intact rock); s and a are constants which depend upon the characteristics of the rock mass; σ_{ci} is the uniaxial compressive strength (UCS) of the intact rock pieces. The expressions of m_b , s and a are:

$$m_b = m_i \exp\left(\frac{GSI - 100}{28 - 14D}\right) \quad (2)$$

$$s = \exp\left(\frac{GSI - 100}{9 - 3D}\right) \quad (3)$$

$$a = \frac{1}{2} + \frac{1}{6} \left(e^{-GSI/15} - e^{-20/3} \right) \quad (4)$$

where GSI is the Geological Strength Index; the parameter D is a "disturbance factor", which depends upon the degree of disturbance to which the rock mass has been subjected by blast damage and stress relaxation. It varies from 0 for undisturbed in situ rock masses to 1 for very disturbed rock masses. In the tunneling process, the excavator machine was used but not the blasting technology, there is almost no disturbance to the surrounding rocks during the tunneling process. Thus, the D parameter was taken as 0 here. m_i is a material constant for the intact rock.

Hoek et al. provided the calculation formula for the elastic modulus E_{cm} , uniaxial compressive strength σ_{cm} and uniaxial tensile strength σ_t of the coal and rock mass in the stope, which can be expressed as:

$$E_{cm} = E \left(0.02 + \frac{1 - D/2}{1 + e^{(60+15D-GSI)/11}} \right) \quad (5)$$

$$\sigma_{cm} = \sigma_{ci} s^a \quad (6)$$

$$\sigma_t = \frac{\sigma_{ci} s}{m_b} \quad (7)$$

3.2. Hoek-Brown Strain Softening Model

The strain softening phenomenon occurs during the compression of coal and rock mass, that is, after the stress reaches the peak strength, the stress of the coal body rapidly drops to a lower level, as the deformation continues to increase. As shown in Figure 6, a large number of experiments show that the stress-strain curve of coal and rock mass can be divided into elastic phase, strain softening phase and residual stress phase.

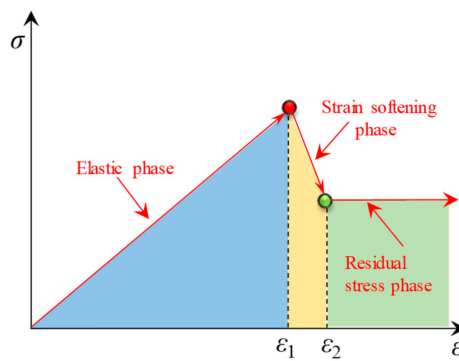


Figure 6. Strain softening model of coal and rock mass.

The strain-soften mechanical behavior of coal and rock mass after the peak can be simulated by reducing the initial GSI_0 value. Assume that GSI decreases linearly with plastic strain, as shown in Figure 7.

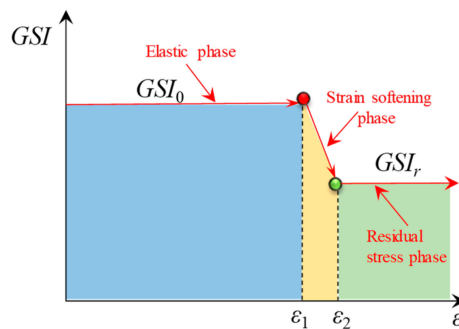


Figure 7. Relationship between Geological Strength Index (GSI) and strain.

Based on this assumption, one has:

$$GSI(\varepsilon) = \begin{cases} GSI_0, & 0 < \varepsilon \leq \varepsilon_1 \\ GSI_0 - \frac{GSI_0 - GSI_r}{\varepsilon_2 - \varepsilon_1} \varepsilon, & \varepsilon_1 < \varepsilon < \varepsilon_2 \\ GSI_r, & \varepsilon \geq \varepsilon_2 \end{cases} \quad (8)$$

M. Cai et al. [23] (2007) gave the calculation formula of the residual GSI_r :

$$GSI_r = GSI \cdot e^{-0.134GSI} \quad (9)$$

Substituting Equation (8)~(9) into Equation (5), it can be obtained that

$$E_{rm}(\varepsilon) = E_{cm} \left(0.02 + \frac{1 - D/2}{1 + e^{(60 + 15D - GSI(\varepsilon))/11}} \right) \quad (10)$$

After the peak, the elastic modulus of the coal and rock mass is dynamically updated by Equation (10) to reflect the strain softening process.

3.3. Fracture Slip and Open Model

In order to simulate the development and evolution of cracks in the surrounding rock of the chamber, the Voronoi elements are arranged in the surrounding rock of the chamber, as shown in Figure 8, which are filled with triangular finite elements. Joint elements are inserted between Voronoi elements to simulate the development and evolution of cracks. The mean side length of 0.15 m of Voronoi elements was selected.

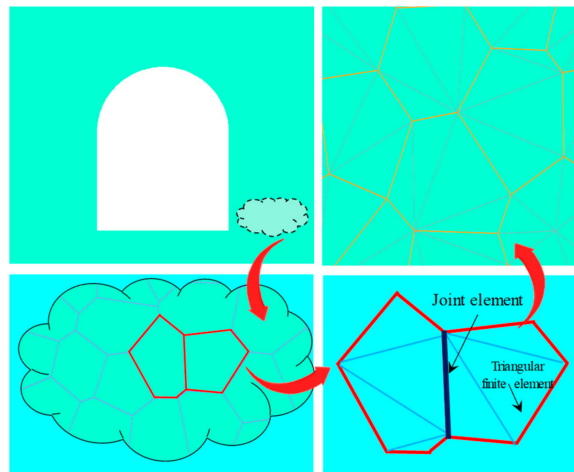


Figure 8. The surrounding rock of the chamber with Voronoi elements.

The Mohr-Coulomb friction law is applied to the failure of the joint elements to simulate slip and open between contacts. As shown in Figure 9, the contact behavior of a contact is simulated by a spring-rider, and the force was divided into normal stress and shear stress. In the normal direction of the contact, the stress-displacement relation is assumed to be linear, and the response is governed by the normal stiffness K_n (Bai et al. 2016) [15]:

$$\Delta\sigma = -K_n\Delta u \quad (11)$$

where $\Delta\sigma$ and Δu_n are the effective normal stress increment and normal displacement increment, respectively.

When the normal stress exceeds the tensile strength of the rock, the normal stress $\sigma_n=0$ and tensile failure occurs.

In the shear direction of contact, the response is governed by the shear stiffness (K_s) and limited by the max shear strength ($\tau_{\max}$). The relationship between stress and displacement can be divided into two stages.

$$\text{If } |\tau_s| \leq c + \sigma_n \tan \varphi = \tau_{\max}, \text{ then } \tau_s = K_s \Delta u_s^e, \quad (12)$$

where τ is the shear force and Δu_s^e is the elastic shear displacement increment.

$$\text{If } |\tau_s| > c + \sigma_n \tan \varphi, \text{ then } \tau_s = \sigma_n \tan \varphi, \quad (13)$$

where φ is internal friction angle.

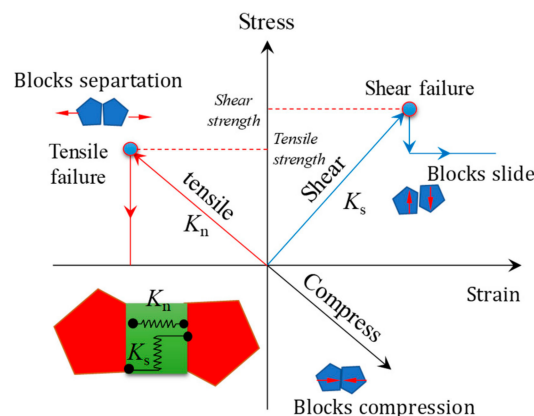


Figure 9. Voronoi element constitutive model and parameter calibration.

K_n and K_s are the key parameters in the above contact model, and Barton [24] (Barton 1972) has provided their expressions as

$$K_n = \frac{EE_{cm}}{L(E - E_{cm})} \quad (14)$$

$$K_s = \frac{GG_{cm}}{L(G - G_{cm})} \quad (15)$$

where L is the average spacing of joints. G_{cm} is the shear modulus of the rock mass, which can be calculated as

$$G_{cm} = \frac{E_{cm}}{2(1 + \mu)} \quad (16)$$

To further calibrate K_n and K_s , the FDEM is used to simulate uniaxial compressive and shear tests to obtain the stress-strain curves. They are then compared with the experimental measurements to obtain more accurate K_n and K_s through multiple adjustments.

3.4. Model Establishment

As shown in Figure 10, a two-dimensional numerical model of 150 m × 54.43 m is established according to the engineering geological background of the strap joint chamber of Tashan Coal Mine. The model mesh size is between 0.5 and 1.5 m, and the Hoek-Brown strain softening model is adopted for the coal and rock mass. The node of the model is 23400. The horizontal displacement on the two sides of the model is constrained, while the vertical displacement on the bottom boundary is constrained. A uniform vertical load of 12.5 MPa is applied on top to replace the rock mass overlying 500 m. The edge of large-section chamber is 70 m away from the two-side boundaries, which is enough to eliminate the boundary effect.

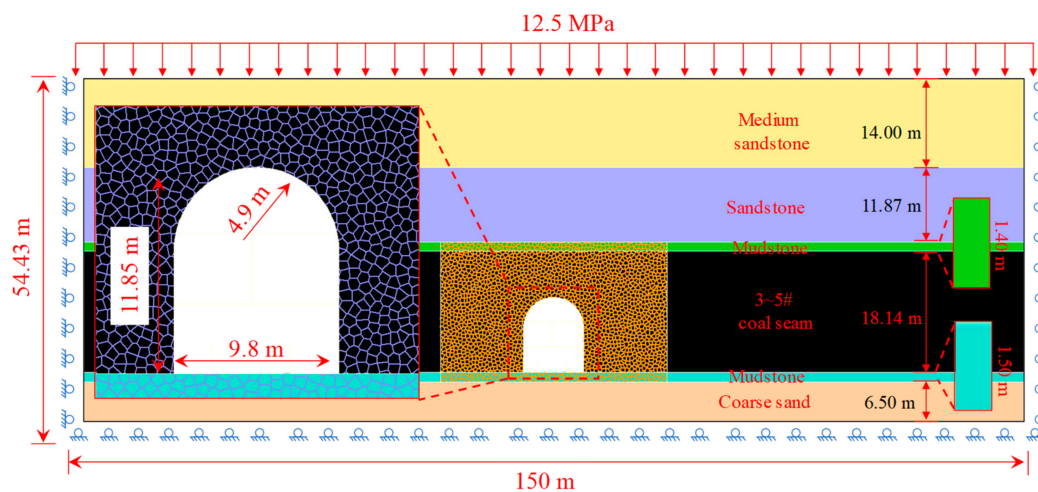


Figure 10. Numerical model.

3.5. Model Verification

From the perspective of crack development characteristics, the constitutive model of the numerical model and the rationality, accuracy of the mechanical parameters is verified. As shown in Figure 11, the completeness of the surrounding rock at 2 m, 5 m, and 8 m is observed via a borehole television (TV). It can be seen that at 2 m, a large number of cracks are generated in the coal body, and the cracks penetrate each other to form a fracture zone. With the depth increase to 5 m, the amount of the cracks reduced, but the fracture zone can still be observed. However, for coal mass at depth of 8 m, only a small amount of cracks can be seen, and the coal mass almost remains intact. In the corresponding position, the crack development in the numerical simulation is basically consistent with the borehole

TV observation, which verifies the reliability and accuracy of the simulation from the perspective of surrounding rock failure.

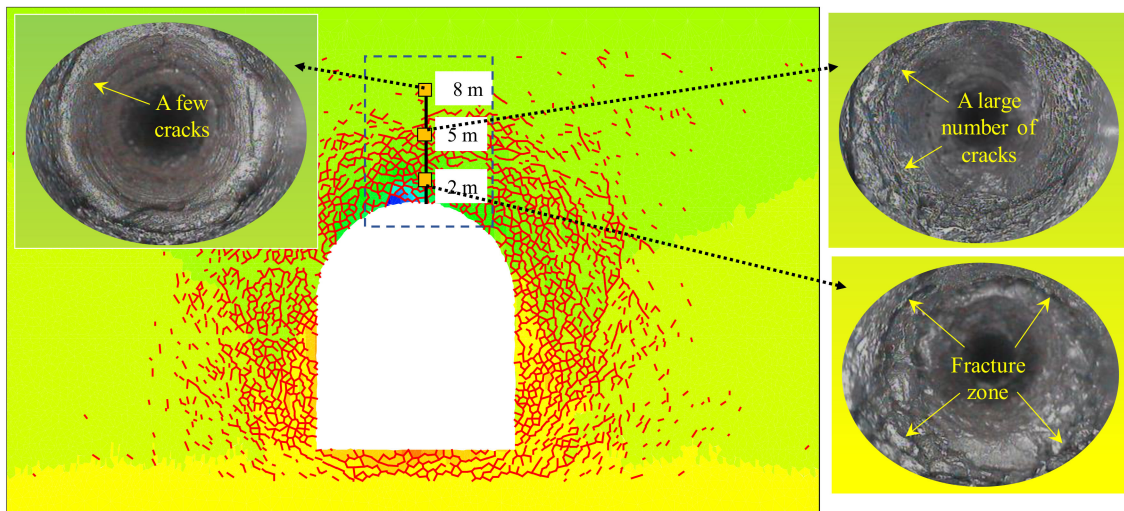


Figure 11. Crack development in the chamber dome.

4. Dome Roof fall Mechanism

Roof fall of the chamber dome is a process of crack development and evolution. During the process, separation occurs in the chamber dome. When the roof separation exceeds a certain value, roof fall hazard would occur in the surrounding rock of the chamber dome. Roof separation in the chamber dome is also related to crack development. Therefore, this chapter first analyzes the relationship between roof separation and the crack development, then the mechanism of roof fall hazard of the chamber dome is uncovered.

4.1. Relationship Between Separation and the Crack Development

As shown in Figure 12, the fracture area of the chamber dome is extracted for the crack development analysis. Also, a measuring line is arranged within 8 m of the chamber dome to monitor the roof separation.

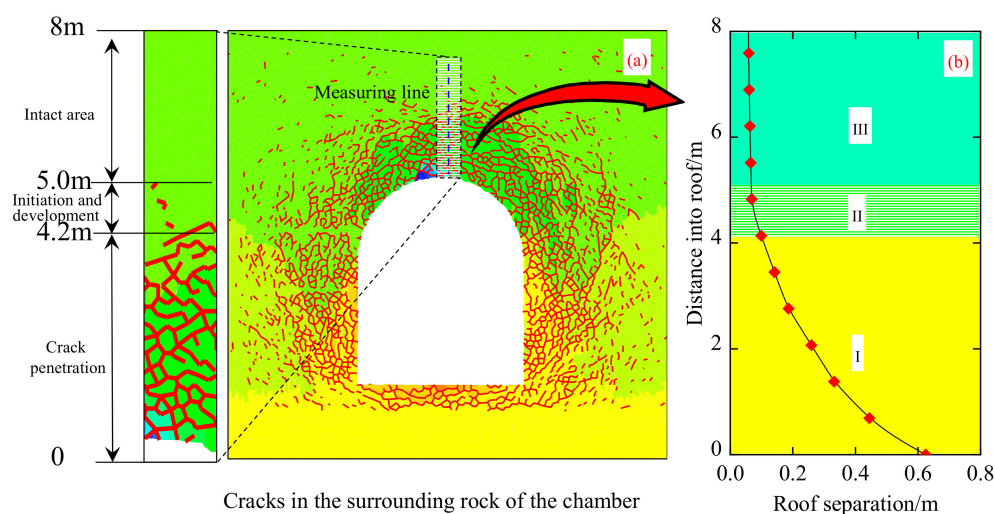


Figure 12. Deformation and damage law of the chamber dome.

As shown in Figure 12a, after excavation, a large number of cracks are generated in the chamber dome, and the development characteristics of the crack in different areas of the chamber dome are various. The closer to the chamber dome surface, the more severely the surrounding rock damaged. Within the range from 0 m to 4.2 m, the cracks generated in the chamber dome are penetrating with each other. In contrast, the chamber dome is basically intact at the depth from 5.0 m to 8.0 m.

It can be seen from Figure 12b that: (1) the roof separation of the chamber dome is rapidly increasing from 0.07 m to 0.63 m in the range of 0 m to 4.2 m (I area). In addition, the closer to the chamber dome surface, the greater the change rate of roof separation. This area corresponds to the crack penetration area in Figure 12a. In the area, the closer to the chamber dome surface, the more developed cracks and more severe of the surrounding rock damaged. Therefore, the bearing capacity is drastically reduced, and the roof separation is large. (2) The changing rate of roof separation significantly reduced within 4.2 m to 5.0 m of the chamber dome (II area). This area corresponds to the crack initiation and development area in Figure 12a. The cracks do not penetrate each other, and the surrounding rock of the chamber dome has a certain bearing capacity. Therefore, the roof separation is significantly reduced. (3) Within 5.0 m to 8.0 m of the chamber dome (III area), the roof separation is basically stable at 0.035 m. This area corresponds to the intact area in Figure 12a. The chamber dome in the area is basically intact, and the amount of dome separation is caused by the overall movement of the chamber dome.

4.2. Roof fall Mechanism of the Chamber

According to the above rules, the roof separation of chamber dome is mainly caused by the development of cracks, and it increases sharply in the crack penetration area. In order to further uncover the mechanical mechanism of crack development in the chamber dome, as shown in Figure 13, the failure forms that lead to crack generation in the chamber dome are given.

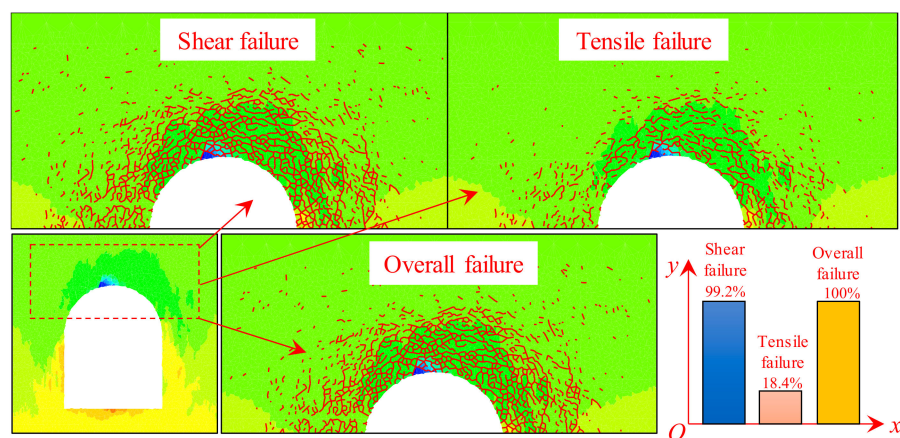


Figure 13. Mechanical mechanism of the crack generation in the chamber dome.

As shown in Figure 13, under the effects of ground pressure and excavation disturbance, a large amount of cracks is generated in the chamber dome. The main reason contributing to the crack generation is that the shear stress of the chamber dome exceeds its shear strength, thus shear failure occurs. However, the cracks caused by tensile failure are relatively less. The proportion of cracks of chamber dome caused by shear failure and tensile failure is obtained by statistics, which are 99.2% and 18.4%, respectively. The shear failure leads to a sharp decrease in cohesion of the surrounding rock, and the tensile strength of the surrounding rock is also reduced. Therefore, the tensile failure happens.

As shown in Figure 14, the roof fall mechanism of the chamber dome is that after the excavation, the chamber dome mainly undergoes shear failure, and a large number of cracks are initiated. After the cracks are initiated, under the action of ground pressure, the cracks further expand and penetrates each other, and finally developed to the surface of the chamber dome. The strength of the surrounding

rock in the chamber dome fracture zone are drastically reduced. Under the effects of ground pressure, the coal body is broken and thus the roof fall hazard occurs.

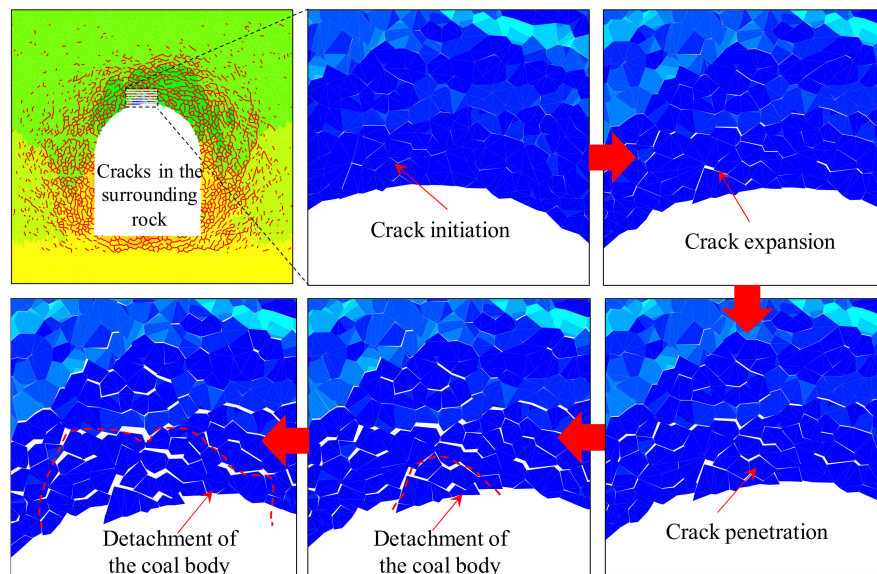


Figure 14. Roof fall mechanism of the chamber dome with time.

5. Control Technology of Roof Fall

According to the roof fall hazard mechanism of the chamber dome, it is known that the chamber dome mainly undergoes shear failure, which leads to the formation of a large number of cracks. As a result, the deformation resistance of the surrounding rock in the crack penetration area is drastically reduced. Therefore, improving the shear strength of the chamber dome and inhibiting the penetration of the crack is the main way to control the roof fall hazard of the chamber dome. Therefore, the grouting reinforcement plus anchor bolt and cable are used as the support measures. The grouting method can significantly improve the shear strength of the surrounding rock of the chamber, while the anchor bolt and anchor cable strengthens the rock mass, improves the cohesion of the surrounding rock and thus increases the shear strength of the surrounding rock.

5.1. Parameters of Anchor Bolt and Cable

The selection of anchor bolt and cable parameters is the key problem of support for the chamber, which includes spacing, length and the pre-tightening force of the anchor bolt and cable. Appropriate anchor bolt and cable parameters can inhibit the shear failure of the surrounding rock of the chamber and avoid the mutual penetration of the cracks. Taking the spacing between bolts as an example, the numerical simulation method is used to study the crack development of the surrounding rock of the chamber when the spacing is 0.6 m, 0.9 m, 1.2 m and 1.5 m, as shown in Figure 15. The parameters of these models are the same as the model in Section 3, and only the supporting anchor bolt are added. In each model, all the parameters are the same except the change of bolt space in the criteria of single factor variation.

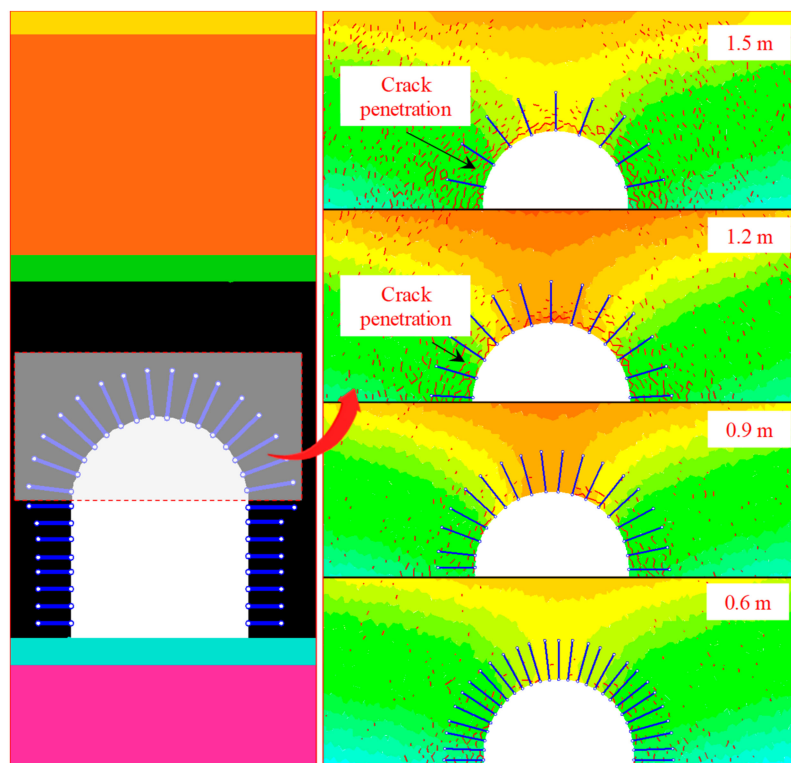


Figure 15. Effect of bolt spacing on the crack development in the chamber dome.

It can be seen from Figure 15 that when the spacing of the anchor bolts is 1.2 m and 1.5 m, there are a lot of cracks in the chamber dome. The cracks initiate, develop, and penetrate each other. At this time, the chamber dome is seriously damaged. When the bolt spacing decreases to 0.9 m, the number of shear cracks in the chamber dome is significantly reduced. Besides, the cracks only initiate and develop without penetration between each other. The deformation and failure of the chamber dome can be well controlled. When the bolt spacing is reduced to 0.6 m, the cracks in the chamber dome do not change much, and are basically similar with that of 0.9 m bolt spacing.

Based on the above rules, when the anchor spacing is 0.9 m, the crack generated due to the shearing of the chamber dome can be controlled very well, and the deformation of the chamber dome is not obvious. Therefore, the spacing of the anchor bolts is to be 0.9 m.

A similar process is used to study other parameters of the anchor bolt and cable, the final optimized support parameters are followed as: The bolt spacing is 900 mm × 900 mm, the pre-tightening force is 100 kN. The length of dome bolt is 2500 mm and the length of ribs' bolt is 1800 mm. The anchor cable spacing is 2700 × 1800 mm, the pre-tightening force is 150 kN, and the length is 8300 mm.

5.2. Grouting Reinforcement Technology

5.2.1. Grouting Fluid Permeation Diffusion Equation

The chamber dome is a typical pore-fracture dual medium that can be assumed to be a continuous medium. Slurry diffusion is an unsteady Darcy seepage motion with grouting pressure as the main driving force, which satisfies the following osmotic continuous differential equation:

$$-\left[\frac{\partial(\rho v_x)}{\partial x} + \frac{\partial(\rho v_y)}{\partial y} + \frac{\partial(\rho v_z)}{\partial z}\right] = \rho(\alpha + n\beta) \frac{\partial p}{\partial t} \quad (17)$$

where ρ is the density of the grouting fluid, α and β represent the compressibility of the chamber dome and slurry, respectively. n is the porosity of the chamber dome and v_i ($i=x, y, z$) is the equivalent

permeation velocity of the slurry in the chamber dome. p is the grouting pressure. Since the flow of the slurry in the chamber dome cracks belongs to the Darcy flow, the equivalent permeation velocity of the slurry can be expressed as the differential form of the grouting pressure p

$$v_x = -\frac{1}{\gamma} \frac{\partial(K_e p)}{\partial x}, v_y = -\frac{1}{\gamma} \frac{\partial(K_e p)}{\partial y}, v_z = -\frac{1}{\gamma} \frac{\partial(K_e p)}{\partial z} \quad (18)$$

where K_e is the equivalent slurry permeability coefficient, indicating the permeability of the slurry in the chamber dome cracks. $\gamma = \rho g$ is the bulk density of the slurry where g is the acceleration of gravity.

Assuming that the grouting slurry is an incompressible fluid, that is, ρ is a constant, the basic differential equation for the penetration and diffusion of the grouting slurry in the chamber dome can be obtained by substituting the Equation (17) into the Equation (18):

$$-\left[\frac{\partial^2(K_e p)}{\partial x^2} + \frac{\partial^2(K_e p)}{\partial y^2} + \frac{\partial^2(K_e p)}{\partial z^2} \right] = S \frac{\partial p}{\partial t} \quad (19)$$

In the above equation, $S = \rho g(\alpha + n\beta)$ is the chamber dome crack slurry storage coefficient, which represents the amount of stored slurry released due to the compression of chamber dome cracks and slurry expansion when the slurry hydraulic head is lowered by 1 unit per unit volume of saturated surrounding rock.

Equation (19) requires the determination of the equivalent slurry permeability coefficient K_e and the equivalent slurry storage coefficient S of the chamber dome. Then it can be solved by combining appropriate boundary conditions and initial conditions. The equivalent slurry storage coefficient S is a constant obtained in the test, while the chamber dome stress and the grouting pressure affect the equivalent slurry permeability coefficient K_e of the chamber dome. A large number of engineering experiments have shown that disturbances, such as chamber excavation, cause the chamber dome to form a stress relaxation zone, a stress increase zone and an initial stress zone from the shallow to the deep. These zones greatly change the permeability characteristics of the chamber dome and influence the penetration and diffusion of slurry in the chamber dome. Therefore, in order to consider the influence of the stress redistribution and the grouting pressure on the slurry diffusion law, Huang et al. [25] (2015) proposed the negative exponential relationship between the material stress and the permeability coefficient based on a large number of experimental results. This relationship can express the equivalent slurry permeability coefficient of the grouting slurry in the chamber dome as:

$$K_e = K_0 e^{-\lambda(\sigma_m - \sigma_{m0} - p)} \quad (20)$$

where $\sigma_m = (\sigma_1 + \sigma_2 + \sigma_3)/3$ is the average stress of the chamber dome after excavation. σ_{m0} is the average stress of the chamber dome before excavation. K_0 is the initial permeability coefficient of the chamber dome and λ is a macroscopic test parameter.

The combination of Equation (19) and Equation (20) yields the basic governing equations for the permeation and diffusion of grouting slurry under the coupling of the chamber dome disturbance stress and the grouting pressure.

5.2.2. Grouting Slurry Permeation and Diffusion Law

When the stress is released after the excavation of the chamber, the shear strength of the surrounding rock can be significantly improved by grouting. The injected slurry should fill the fracture area of the chamber dome. In order to achieve this, it is necessary to study the diffusion radius of the slurry. There are many factors affecting the diffusion radius of the slurry, such as the grouting pressure, the mechanical properties of the surrounding rock. In order to optimize the parameters, as shown in Figure 16, the numerical method is used to simulate the diffusion range of the slurry under different grouting pressures.

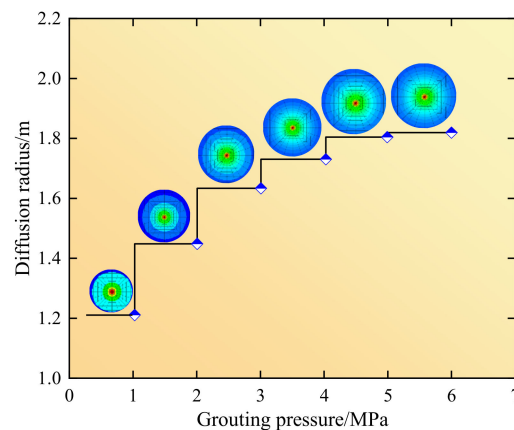


Figure 16. Effect of grouting pressure on diffusion radius.

It can be seen from Figure 16 that the slurry diffusion radius increases rapidly with the increase of the grouting pressure. When the grouting pressure is 5~6 MPa, the slurry diffusion radius is basically stable at about 1.8 m. Therefore, the grouting pressure is chosen to be 5 MPa, which corresponds to a diffusion radius is 1.82 m. Since the spacing of the anchor bolts is 0.9 m, a grouting pipe is placed every two anchor bolts.

5.3. Chamber Dome Control Effect

Based on the above analysis, it can be concluded that appropriate anchor bolt, cable support parameters, and grouting parameters can improve the shear strength of surrounding rock and inhibit the cracks penetration. As shown in Figure 17, the expansion law of the cracks in surrounding rock after combining the two kinds of support technologies is further analyzed to evaluate the controlling effect of surrounding rock.

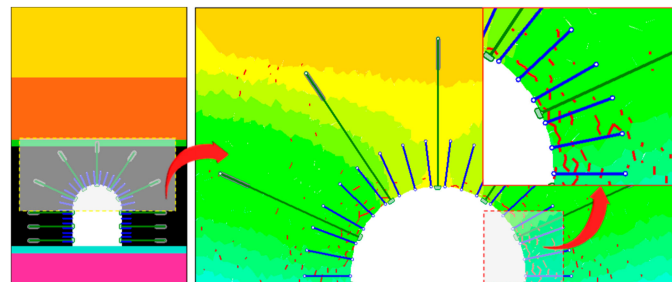


Figure 17. Control effect of cracks in the chamber dome.

It can be seen from Figure 17 that after the combined support, there is only a small amount of cracks at the top of the chamber dome. Also, the cracks only initiate and expand, while more cracks initiate on the two sides of the chamber dome. However, in general, the number of cracks is significantly reduced and there is no crack penetration. Therefore, the cracks are well controlled. Based on the mechanism of the dome roof fall hazard, it can be concluded that the deformation of the surrounding rock of the chamber can be well controlled and no roof fall hazard would happen.

6. Engineering Application

6.1. Chamber Support Scheme

According to the above analysis, the grouting pressure is 5 MPa. After the grouting, the anchor bolt and cable support is also implemented, as shown in

Dome support: The anchor bolt body is 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 2500 mm. The bolt spacing is 900 mm $\times$ 900 mm. The anchorage is lengthened by resin and the bolt pre-tightening force is 100 kN. The tray is made of high-strength steel, the measures of which are 200 mm $\times$ 200 mm $\times$ 10 mm. There are a total of 5 anchor cables, with a diameter of 17.8 mm, a length of 8300 mm, a spacing of 2700 $\times$ 1600 mm, and a pre-tightening force of 150 kN. The anchoring length is 1500 mm. There were 3 resin rolls used. The specification of one of the resin rolls is K2335, while the other two are Z2360. W steel strip has a thickness of 4 mm and a width of 250 mm.

Rib support: The length of anchor bolts is 1800 mm. In total, 2 resin rolls were used, with one being K2335 and another being Z2360. Other parameters are the same as those of the chamber dome support. Anchor cable support parameters are also the same as those of the chamber dome support. Figure 18. The specific support parameters are as follows:

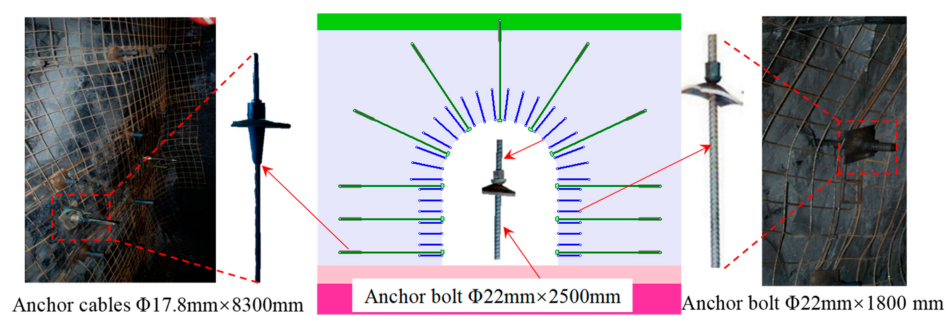


Figure 18. Chamber support scheme.

6.2. Chamber Deformation Measurement

In order to monitor the surrounding rock deformation at different depths, the measuring points of the multi-point displacement meter are arranged at 2 m, 4 m, 6 m, 8 m and 10 m depth of the chamber dome surrounding rock, as shown in Figure 19.

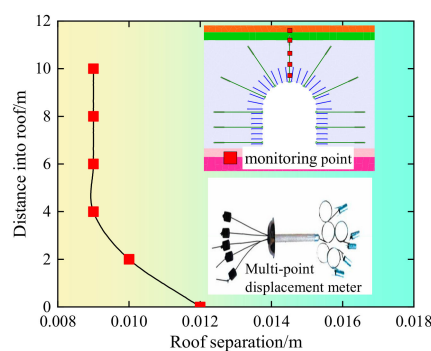


Figure 19. Deformation of the chamber surrounding rock.

It can be seen from Figure 19 that the roof separation of the chamber dome in the range of 0 m to 4 m is relatively large. The maximum roof separation is 0.012 m, corresponding to the crack initiation and expansion area. The amount of separation is basically maintained at 0.009 m within 4 m to 10 m, corresponding to the intact area. Based on the above rules, it can be concluded that the deformation of the surrounding rock of the chamber dome is well controlled and roof fall hazard would not occur.

7. Conclusions

In this paper, the strap joint chamber of Tashan Coal Mine was selected as the engineering background. The FDEM was used to uncover the roof fall mechanism of the chamber dome. The following conclusions can be drawn:

- (1) After excavation, the chamber dome suffers from shear failure, and forming a large number of cracks. In the crack penetration area, the bearing capacity of the chamber dome is significantly reduced and the roof separation is sharply increased. In the crack penetration area, when the crack is connected with the chamber surface, under the effect of the ground pressure, the coal body is separated from the chamber dome and roof fall hazard occurs.
- (2) According to the roof fall mechanism of the chamber dome, it was concluded that improving the shear strength of surrounding rock and reducing the penetration of cracks are the main ways to control the roof fall hazard. With the concepts, the pre-tightening force, spacing and length of the anchor bolt and cable, grouting pressure and diffusion radius of the grouting slurry were optimized.
- (3) The optimized parameters of anchor bolt, anchor cable and grouting were applied to the strap joint chamber of Tashan Coal Mine. The maximum roof separation of chamber dome is only 0.012 m. The surrounding rock control effect is great, and no roof fall occurs.

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