

## Research Article

# Study on the Support System Mechanism of Entry Roof with Roof Cutting Pressure Releasing Gob-Side Entry Retaining Technology

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To improve the technical system of the roof cutting pressure releasing gob-side entry retaining technology (RCPRGERT) and provide reference for the adaptive promotion and optimization of the new technology, the entry support system mechanism with RCPRGERT is studied in this paper. Firstly, the stress structure of the roof cutting short beam is analyzed by the mechanical theory analysis method. Then, taking 8304 working face of Tashan Coal Mine as an example, the stress distribution characteristics of the surrounding rock in different entry retaining stages are simulated by numerical simulation, which provides an important reference for the field support design and puts forward an idea of hierarchical support. Finally, an engineering test is carried out in the experimental working face to verify the design of the retaining entry support, which obtains a well entry retaining effect. The research results in this study provide powerful theoretical and practical references for the promotion of RCPRGERT under similar geological conditions.

## 1. Introduction

Coal, as a solid combustible mineral, has been one of the main energy sources used by human beings since the 18<sup>th</sup> century [1]. In the later development process of human energy history, fossil energy, which is mainly composed of coal, has been impacted in the proportion of world energy, however, it is still the main source of human energy consumption [2]. China is a big country of coal production and consumption, and the exploitation and utilization of coal resources play an important role in the development and stability of national economy [3]. However, China's coal reserves have unique characteristics, which put forward higher requirements for the mining technology of coal resources.

First of all, the amount of coal resources suitable for open-pit mining only accounts for 10%–15% of the total resources in China. Most of the coal resources need to be mined by underground mining [4]. Secondly, after a long-term large-scale mining, the reserves of coal resources in the shallow and middle mining depth decrease year by year, more and more mining areas begin to turn into deep mining, and the mining depth still extends to the deep at the rate of about 20 m/year [5]. Well mining itself involves groundwater, gas, rockburst, and other threats. Hence, when the mining depth increases, it will increase mining difficulty and danger [6].

To improve the safety of deep mining and the coal output rate of the mining area, the research team of academician He ManChao puts forward the roof cutting pressure releasing

gob-side entry retaining technology (RCPRGERT) on the basis of traditional gob-side entry retaining technology in 2009 [7, 8]. By cutting off the connection between the entry roof and gob roof along the working face strike direction within a certain height range, the RCPRGERT can promote the gob roof to collapse after coal mining and fill the gob with broken expansion, while the entry roof remains relatively stable, realizing the reuse of entry and mining without coal pillar and filling body in the mining area, which greatly improves the coal yield and roadway retention efficiency in the mining area. After about ten years of development, the RCPRGERT has been successfully tested in many mining areas in China, and many scholars have conducted rich research on this technology: Guo et al., with the help of numerical simulation and theoretical analysis, optimized the design of the key parameters of the roof cutting entry forming technology under the condition of a thin coal seam and obtained the reasonable roof cutting height and angle under the specific geological condition [9]. Sun et al. established the surrounding rock stress model of the retained entry with roof cutting and analyzed the design requirements of roof cutting parameters, such as the roof cutting height, the roof cutting angle, and the distance between the roof cutting holes through mechanical calculation [10]. Zhang et al. obtained the best explosive charge structure of the roof cutting blasting through the field energy concentrated tension blasting test in the Tangshangou Coal Mine, combined with the use of the peering instrument [11]. Yang et al. used the numerical simulation software to study RCPRGERT under shallow buried depth conditions and analyzed the influence of the roof cutting height change on the stress and the migration law of the gob overburden [12]. It can be seen that the existing research about RCPRGERT mainly focuses on the roof cutting link, which is indeed the key to the success or failure of the entry retaining. However, in the field test, the support and control of the entry roof also has a huge impact on the final entry forming effect. To systematically analyze the support mechanism of the entry with RCPRGERT to enrich the theoretical results and provide effective guidance and reference for the field practice, this paper takes an engineering example as the background to study the roof hierarchical support idea and design with RCPRGERT.

## 2. Technology and Process Overview

**2.1. Technical Principle.** As a nonpillar mining method, the gob-side entry retaining technology can cancel coal pillar retention in the working face mining layout and maintain the entry of a working face along the gob after mining for the next adjacent working face to continue use, which can effectively solve the problems of coal resource waste, frequency of tunnel disasters, and so on [13]. This technology can improve the coal mining rate in the mining area, extend the service life of mines, reduce the amount of tunnel excavation work, and simplify the working face connection procedure [14]. However, the traditional gob-side entry retaining is mainly realized by the technology of roadside packing and support. As the problems of fast filling and

retained entry stability control are difficult to be solved effectively, compared with the working face mining advance, the construction of the gob-side entry retaining often lags behind. What's more, the traditional entry retaining methods usually have high cost and complex working procedures, which seriously restrict the application and development of this technology [15].

RCPRGERT, developed on the basis of the above, through the combination of the constant resistance large deformation anchor cable support technology, two-way energy concentrated tension blasting technology, retained entry temporary support technology, gangue wall maintenance support technology, and so on, can realize mining without section coal pillar and additional filling body in the mining area [16]. This technology greatly improves the coal mining rate and reduces the entry excavation rate, and it has a wide application prospect and high research value.

As shown in Figure 1, RCPRGERT can actively change the structural form of entry roof and goaf roof through roof presplitting, and it can realize the gob-side entry retaining and nonpillar mining using the collapse and expansion of goaf roof after mining. The core of the new technology is to carry out directional presplitting on the roof of the entry to be retained along the mining advancing direction of the working face, cut off the partial stress transfer between the entry roof and goaf roof, and at the same time, reinforce the retained entry roof with constant resistance large deformation anchor cable. Then, it forms the roof stress difference on both sides of the roof cutting surface, so that the retained entry roof deformation is controllable after working face mining, while the goaf roof collapses along the roof cutting surface in time and effectively supports the overburden after broken expansion. RCPRGERT effectively solves the two problems of high confining pressure and difficult support of gob-side entry. The entry retaining efficiency is greatly improved, and the cost is greatly reduced, which greatly improves the applicable scope of the mining technology without coal pillar.

**2.2. Technological Process.** Based on the above technical principles, the process flow of RCPRGERT can be summarized into the following six steps (as shown in Figure 2) [8, 17]: (a) excavate the entry to be retained serving two adjacent working faces, and carry out constant resistance anchor cable reinforcement support for the key deformation area of the entry roof to resist the multiple dynamic pressure disturbance during the period of entry retaining. (b) Implement entry roof presplitting cutting within a certain height range on the prior mining working face side to cut off the vertical stress transfer of the roof on both sides. (c) Implement the advance temporary support in the advance of a certain range of the prior mining working face to prevent the large deformation caused by the advance stress concentration of the working face. (d) The goaf roof will collapse after the prior mining face is mined, and at this time, the lagging temporary support and gangue wall maintenance support should be implemented in time. Then, the gangue wall can be formed by the collapse broken expansion of the

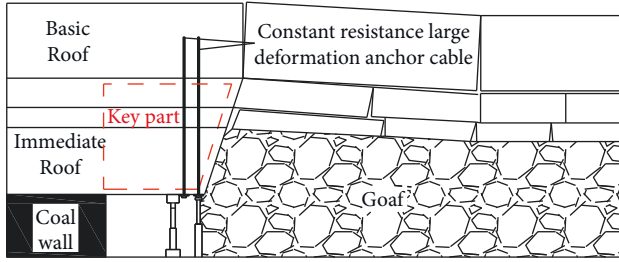


FIGURE 1: Principle of roof cutting and pressure releasing gob-side entry retaining.

goaf roof. (e) After the gangue wall has formed a stable bearing capacity and the deformation area of the retained entry is stable, the lagging temporary support can be removed, leaving the gangue wall maintenance support alone. Then, the entry retaining will be completed. (f) The adjacent working face mining will reuse the retained entry, and the gangue wall maintenance support can be withdrawn synchronously if conditions permit to further increase the economic benefits.

The technology of roof cutting entry retaining includes reinforcement support, roof cutting, advance support, gangue wall support, post support, and other links. Among them, the roof cutting and reinforcement support are the processes of advanced mining, which can be carried out before the mining of the working face. As the primary process, the reinforcement support needs to be at least 20 m ahead of the roof cutting in space to avoid the roof damage caused by the roof cutting blasting.

### 3. Mechanical Properties of Roof Cutting Short Beam

Through the above analysis of technical principle and technological process, it can be seen that the entry roof stability is of decisive significance for the success of the overall entry retaining. To further study the entry roof support mechanism and countermeasures with RCPRGERT, this chapter analyzes the mechanical characteristics of the entry roof cutting short beam. In the process of entry retaining, theoretical analysis and numerical analysis are mainly aimed at the inclined direction of the working face, which mainly takes into account the characteristic of strong compressive capacity and weak tensile capacity of rock, and the mining layout of the working face, namely the main destructive force source of the retained entry roof, is the transverse tension of the goaf roof.

After the entry roof presplitting, with the mining advance of working face and the collapse of the gob roof, the roof of retained entry gradually forms the roof cutting short beam structure (as the key part shown in Figure 1), and its simplified mechanical model is shown in Figure 3 [18, 19]. The rock has the property of strong compressive capacity and weak tensile capacity, and the analysis of the key part of the roof cutting entry retaining focuses on the inclined section of the mining face. Hence, the influence of the strike section is ignored. In addition, considering that the no-

cutting side of entry roof is connected with the roof solid rock stratum, so the connection can be regarded as fixed connection. Above all, the short arm beam model of roof cutting is simplified as Figure 3.

In material mechanics, when the ratio of beam span and section height  $L/H_F$  is more than 5, its internal force can be solved by material mechanics with related derivation. Otherwise, it can be calculated by elastic mechanics. In the structure of roof cutting short beam, usually  $l/H_F$  is less than 5. Hence, elastic mechanics should be used to solve the structure.

If the overburden load of the roof cutting short beam is  $q_1$ , the beam length is  $l$ , the friction force of the roof cutting surface is  $q_2$ , and the normal support force of the goaf falling gangue is  $q_3$ , then the resulting analysis is shown below:

#### 3.1. Characterization of Stress Components in Function Form.

According to material mechanics, the bending stress  $\sigma_x$  is mainly caused by the bending moment, while the shear stress  $\tau_x$  is mainly caused by shear stress. In addition, the compressive stress  $\sigma_y$  is mainly caused by load  $q_1$ . Since  $q_1$  can be regarded as uniform load,  $\sigma_y$  does not change with  $x$ . Hence,

$$\sigma_y = f(y). \quad (1)$$

3.2. Solution of Stress Function. Substitute (1) in equilibrium differential (2) (where force component  $f_x = 0$ ,  $f_y = \rho g$ ),

$$\begin{cases} \sigma_x = \frac{\partial^2 \Phi}{\partial y^2} - f_x x, \\ \sigma_y = \frac{\partial^2 \Phi}{\partial x^2} - f_y y, \\ \tau_{xy} = -\frac{\partial^2 \Phi}{\partial x \partial y}. \end{cases} \quad (2)$$

Then,

$$\frac{\partial^2 \Phi}{\partial x^2} = f(y) + f_y y. \quad (3)$$

By integrating  $x$  with the above formula twice, we can get the following:

$$\Phi = \frac{x^2}{2} f(y) + \frac{x^2 y}{2} f_y + x f_1(y) + f_2(y), \quad (4)$$

where  $f(y)$ ,  $f_1(y)$ , and  $f_2(y)$  are all unknown functions about  $y$ , and  $f_y$  is the  $y$ -direction force component that does not change with the  $y$  coordinate.

3.3. Solving the Stress Function from the Compatibility Equation. Substitute the results of the above formula in the compatibility equation.

$$\frac{1}{2} \frac{\partial^4 f(y)}{\partial y^4} x^2 + \frac{\partial^4 f_1(y)}{\partial y^4} x + \frac{\partial^4 f_2(y)}{\partial y^4} + 2 \frac{\partial^2 f_2(y)}{\partial y^2} = 0. \quad (5)$$

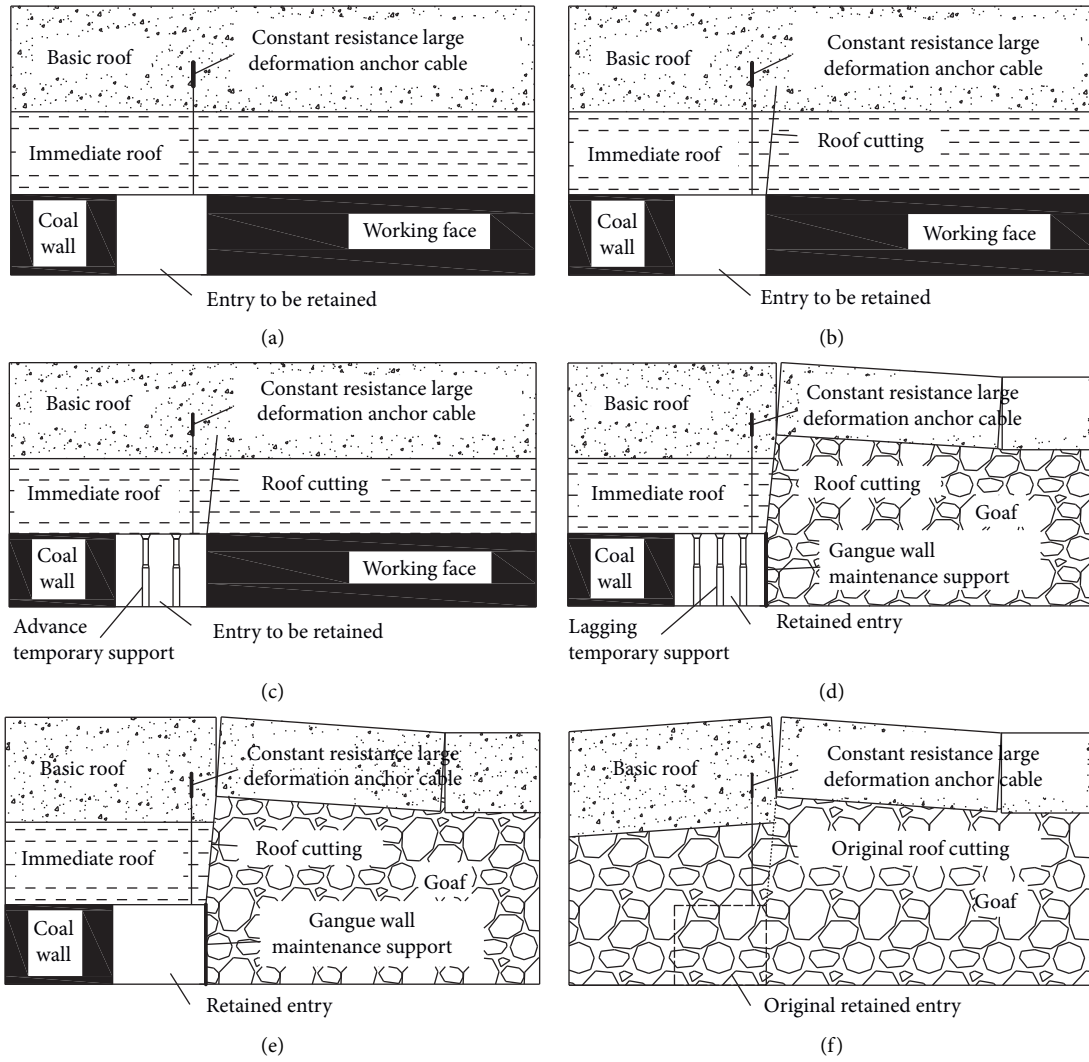


FIGURE 2: Technological process of roof cutting pressure releasing gob-side entry retaining.

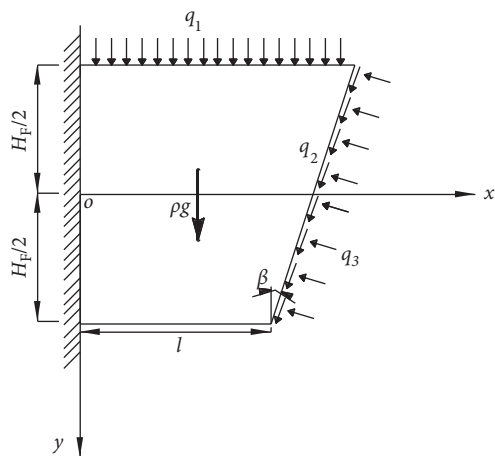


FIGURE 3: Theoretical model stress analysis of short beam under roof cutting.

According to the condition of the compatibility equation with elastic mechanics, the  $x$  value of the whole section should be in accordance with the equation. Hence, the sum of the  $x$  term coefficient and the free term should be 0. Then,

$$\begin{cases} \frac{\partial^4 f(y)}{\partial y^4} = 0, \\ \frac{\partial^4 f_1(y)}{\partial y^4} = 0, \\ \frac{\partial^4 f_2(y)}{\partial y^4} + 2 \frac{\partial^2 f_2(y)}{\partial y^2} = 0. \end{cases} \quad (6)$$

Integral calculate

$$\begin{cases} f(y) = Ay^3 + By^2 + Cy + D, \\ f_1 = Ey^3 + Fy^2 + Gy, \\ f_2(y) = -\frac{Ay^5}{10} - \frac{By^4}{6} + Hy^3 + Ky^2. \end{cases} \quad (7)$$

Since the constant term does not affect the stress component, it has been omitted. Upon substituting the result in (4), we obtain the following:

$$\Phi = \frac{x_2}{2}(Ay^3 + By^2 + Cy + D) + x(Ey^3 + Fy^2 + Gy) - \frac{A}{10}y^5 - \frac{B}{6}y^4 + Hy^3 + Ky^2. \quad (8)$$

Substitute formula (2) to obtain the following:

$$\begin{cases} \sigma_x = \frac{x_2(6Ay+2B)}{2} + x(6Ey+2F) - 2Ay^3 - 2By^2 + 6Hy + 2K, \\ \sigma_y = Ay^3 + By^2 + Cy + D, \\ \tau_{xy} = -x(3Ay+2By+C) - (3Ey^2+2Fy+G). \end{cases} \quad (9)$$

**3.4. Solving Characteristic Solution According to Boundary Condition.**  
The upper and lower boundary conditions are as follows:

$$\begin{cases} (\sigma_y)_{y=\frac{H_F}{2}} = 0.21, \\ (\sigma_y)_{y=-\frac{H_F}{2}} = -q_1, \\ (\tau_{xy})_{y=\pm\frac{H_F}{2}} = 0. \end{cases} \quad (10)$$

Substitute (9) to obtain the following:

$$\begin{cases} A = \frac{2q_1}{H_F^3}, \\ B = F = 0, \\ C = \frac{3q_1}{2H_F}, \\ D = \frac{q_1}{2}, \\ G = -\frac{3}{4}EH_F^2. \end{cases} \quad (11)$$

Then,

$$\begin{cases} \sigma_x = -\frac{6q_1}{H_F^3}x^2y + \frac{4q_1}{H_F^3}y^3 + 6Exy + 6Hy + 2K, \\ \sigma_y = -\frac{2q_1}{H_F^3}y^3 + \frac{3q_1}{2H_F}y + \frac{q_1}{2} - \rho gy, \\ \tau_{xy} = \frac{6q_1}{H_F^3}xy^2 - \frac{3q_1}{2H_F}x - \frac{3}{4}E(4y^2 - H_F^2). \end{cases} \quad (12)$$

Consider the left and right limit conditions of the beam. The right boundary is the stress boundary, while the left boundary is the fixed structure. Hence, the boundary range can be expressed as  $l \leq x \leq l + H_F \tan \beta$ . According to Figure 3, the boundary coordinate relationship can be obtained as follows:

$$x = l + \frac{1}{2}H_F \tan \beta - y \tan \beta, \quad \left(-\frac{H_F}{2} \leq y \leq \frac{H_F}{2}\right). \quad (13)$$

Then, from the right boundary condition, we obtain the following:

$$\begin{cases} \int_{-\frac{H_F}{2}}^{\frac{H_F}{2}} (\sigma_x + \tau_{xy} \sin \beta) dy = P_1, \\ \int_{-\frac{H_F}{2}}^{\frac{H_F}{2}} (\sigma_y + \tau_{xy} \cos \beta) dy = P_2, \\ \int_{-\frac{H_F}{2}}^{\frac{H_F}{2}} (\sigma_x + \tau_{xy} \sin \beta) y dy = 0. \end{cases} \quad (14)$$

$P_1$  and  $P_2$  are the horizontal and vertical components of the resultant force of the right boundary external force on the roof cutting surface, and the expressions are as follows:

$$\begin{cases} P_1 = (q_2 \sin \beta + q_3 \cos \beta)H_F \tan \beta, \\ P_2 = (q_2 \cos \beta + q_3 \sin \beta)H_F. \end{cases} \quad (15)$$

The vector composition diagram is shown in Figure 4.

Upon combining with formulae (4) and (12), it can be obtained as follows:

$$\begin{cases} E = \frac{2P_2 + q_1}{H_F^3 \cos \beta} + \frac{2q_1}{H_F^3} \left(l + \frac{1}{2}H_F \tan \beta\right), \\ G = \frac{6P_2 + 3q_1}{4H_F \cos \beta} - \frac{3q_1}{2H_F} \left(l + \frac{1}{2}h \tan \beta\right), \\ K = \frac{P_1}{2H_F} - \frac{(2P_2 + q_1)(\sin \beta - \tan \beta)}{4H_F \cos \beta}, \\ H = \frac{(9 - \sin \beta)}{60H_F} q_1 l \tan \beta - \frac{q_1}{10H_F} - \frac{1}{H_F^3} q_1 \left(l + \frac{1}{2}H_F \tan \beta\right)^2. \end{cases} \quad (16)$$

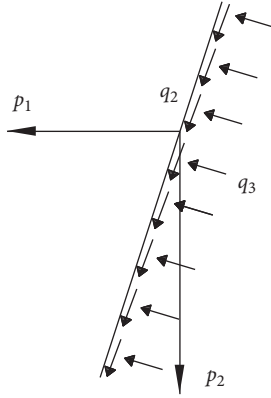


FIGURE 4: .Vector composition of external forces  $P_1$  and  $P_2$  on right boundary.

Therefore, the section stress expression of the short beam is as follows:

$$\sigma_x = -\frac{6q_1}{H_F^3}x^2y + \frac{4q_1}{H_F^3}y^3 + 6xy \left[ \frac{2p_2 + q_1}{H_F^3 \cos\beta} + \frac{2q_1}{H_F^3} \left( l + \frac{1}{2}H_F \tan\beta \right) \right] + 6 \left[ \frac{(9-\sin\beta)}{60H_F} q_1 l \tan\beta - \frac{q_1}{10H_F} \frac{1}{H_F^3} q_1 \left( l + \frac{1}{2}H_F \tan\beta \right)^2 \right] y \quad (17)$$

$$+ 2 \frac{p_1}{2H_F} \frac{(2p_2 + q_1)(\sin\beta - \tan\beta)}{4H_F \cos\beta},$$

$$\sigma_y = -\frac{2q_1}{H_F^3}y^3 + \frac{3q_1}{2H_F}y - \frac{q_1}{2} \rho g y, \quad (18)$$

$$\tau_{xy} = \frac{6q_1}{H_F^3}xy^2 - \frac{3q_1}{2H_F}x - \frac{3}{4}(4y^2 - H_F^2) \left[ \frac{2p_2 + q_1}{H_F^3 \cos\beta} + \frac{2q_1}{H_F^3} \left( l + \frac{1}{2}H_F \tan\beta \right) \right]. \quad (19)$$

Formulas (17)–(19) can be used to calculate the maximum normal stress and shear stress in the roof cutting short beam. The calculation results are usually larger than the tensile and shear strength of the rock stratum. Hence, it is necessary to reinforce and support the roof of the retained entry and implement the roof cutting pressure releasing. By the reasonable design of the roof cutting, the influence of the roof caving in the goaf on the retained entry roof can be greatly weakened, so that the shear stress  $\tau_g$  on the roof cutting surface is less than the shear strength  $\tau_c$  of the roof strata. In addition, as the rock strata usually do not have enough tensile strength, it can be regarded as one of the design principles of retained entry roof support that the surface of the rock beam is not allowed to have tensile stress. At the same time, the reinforcement support design can also be designed according to the deformation control of the short beam.

## 4. Field Examples and Numerical Simulation Analysis

**4.1. Engineering Background.** In this study, the 8304 and 8305 working faces of Datang Tashan Coal Mine's third panel are taken as engineering examples to further study and verify the roof support system mechanism and support countermeasures of RCPRGERT. The designed annual production capacity of Tashan Coal Mine's third panel is 900,000 t, and the minable reserves of the third panel are about 20,105,000 t. The main mining coal seam is block semidark type and dim type coal. It is a complex structure coal seam, which generally contains 2-3 layers of interbedded stone, and the interbedded rock is gray black sandy mudstone and gray brown kaolinite [3].

The 8304 working face of the Tashan Coal Mine is the first mining working face in the third panel eastern flank of the mining area, and the north of the working face is adjacent to the 8305 working face, the south and east of the working face are solid coal areas, and the west of the working face is the panel return air lane, belt lane, and auxiliary transport lane. The upper mining layer of 8304 working face is Jurassic 11# and 14# coal layer, with a layer spacing of 290–320 m, which have been mined. The lower mining layer is Taiyuan Formation 2# coal layer, with a layer spacing of 8–35 m. Specifically, the strike length and inclined length of 8304 working face is 670 m and 127 m, and the strike length and inclined length of 8305 working face are 680 m and 110 m, respectively. The layout plan of the working face is shown in Figure 5(a), the lithologic histogram of the roof is shown in Figure 5(b) (based on the lithologic borehole at 550 m of the working face mining footage), and the basic parameters of the working face are shown in Table 1. The lithology of the roof and bottom is shown in Table 2. The overall geological conditions of the working face are simple, and there is no obvious geological structure affecting the mining operation.

As the test working face, 8304 working face adopts the comprehensive mechanized mining method for mining, and the test entry to be retained is the tail entry. During the mining operation, the head entry is used as the machine rail integration lane, and the tail entry is used for air return, pedestrian, and auxiliary transportation. The test entry is supported by bolt, mesh, and cable, whose section is rectangular with the size of 5.0 m × 3.1 m. After the mining and entry retaining of 8304 working face, the retained entry will be reused for the 8305 working face mining. During the mining period of 8305 working face, the head entry will still be used as the machine rail integration lane, while the retained entry will be used as the tail entry for air return, pedestrian, and auxiliary transportation.

Besides, it should be noted that more than half of the coal seams in China are covered with hard roof. What's more, as the experimental mining area of this study, the hard roof in Datong mining area has the characteristics of many layers, complex structure, and soft interlayer, which is called composite hard roof. In the mining process of the working

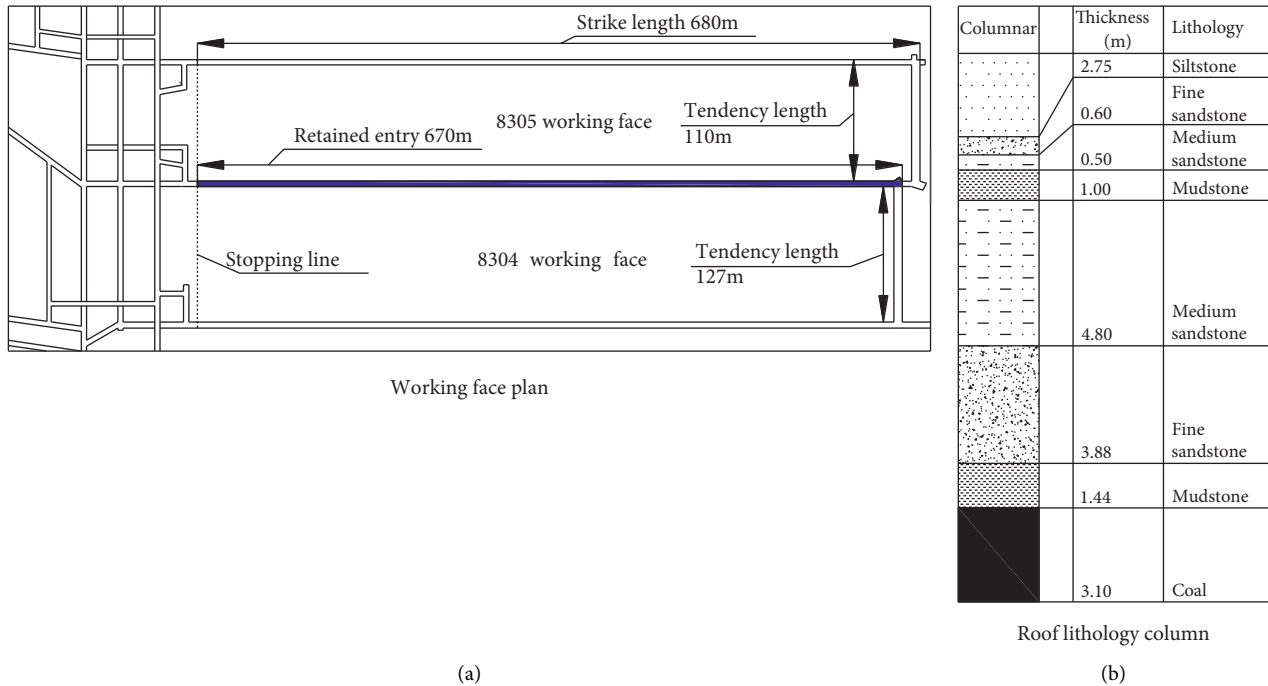


FIGURE 5: Plane and roof lithology column of 8304 and 8305 working face. (a) Working face plan. (b) Roof lithology column.

TABLE 1: Basic parameters of 8304 and 8305 working face.

Coal seam thickness/average (m)	1.80–3.55/3.1	Depth (m)	367–411
Mining height (m)	3.1	Coal seam dip/average (°)	2~6/4
Strike length (m)	8304 face: 670 8305 face: 680	Inclination length (m)	8304 face: 127 8305 face: 110
Immediate roof lithology/thickness (m)	Mudstone/1.44	Immediate floor lithology/thickness (m)	Mudstone/3.2
Basic roof lithology/thickness (m)	Fine sandstone/3.88	Basic floor lithology/thickness (m)	Siltstone/3.1

TABLE 2: Roof and floor lithology.

Roof and floor layer	Lithology	Thickness (m)	Lithologic characteristics
Basic roof	Fine sandstone	3.88	Light gray, gray. Subrounded. Horizontal bedding.
Immediate roof	Mudstone	1.44	Grayish black. Nearly uniform structure, fracture shell. Containing a small amount of kaolinite.
Immediate floor	Mudstone	3.20	Grayish black. Nearly uniform structure, fracture shell. Light weight.
Basic floor	Siltstone	3.10	Black gray. Massive, dense, uniform, hard, and complete. Horizontal bedding, containing a small amount of siderite nodules.

face, this kind of roof is easy to produce the compound movement of the two or more layers of hard roof, and the mine pressure is often more and more intense, which seriously restricts the safe and efficient mining of the mine. Specifically, the process of the roof pressure appearance has the following characteristics (roadway deformation in the Datong mining area is shown in Figure 6):

- (1) The roof has a certain degree of self-stability, the roof hanging plate behind the working face hydraulic support is usually longer, i.e., the first pressure step and the periodic pressure step are often longer, and the pressure strength is higher.

- (2) Because of the soft sandwich structure of the composite hard roof, the process of roof pressure is rapid and sudden when the roof breaks, which is easy to damage the working face hydraulic supports and the entry in the area affected by the mining activities.
- (3) Affected by the roof pressure characteristics, the part of mining area entry is prone to two sides and roof deformation, which will increase the risk of rock burst.

4.2. Numerical Simulation Analysis. Based on the analysis of the structure and stress characteristics of the roof cutting short beam, taking the geological conditions of the test

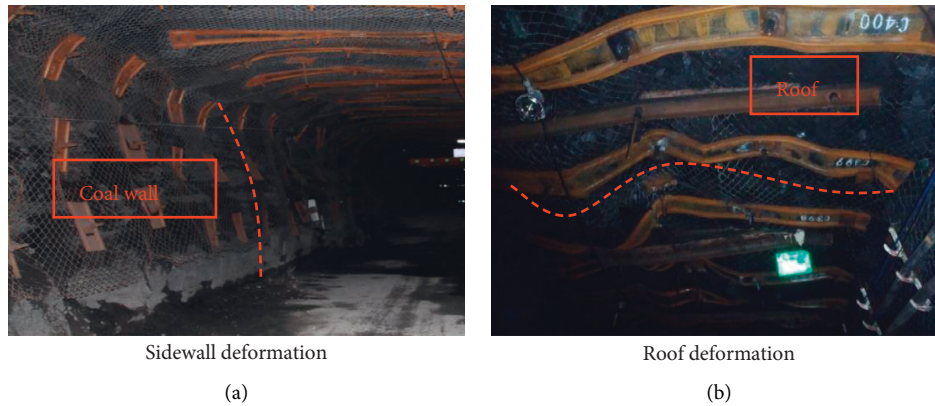


FIGURE 6: Roadway deformation in Datong mining area. (a) Sidewall deformation. (b) Roof deformation.

TABLE 3: Mechanical parameters of model strata.

Lithology	Density (kN/m <sup>3</sup> )	Tensile strength (MPa)	Internal friction angle (°)	Cohesion (MPa)	Bulk modulus (GPa)	Shear modulus (GPa)
Medium sandstone	24	8.4	46	15.2	11.49	7.26
Mudstone	22	3.2	24	10.5	0.20	0.15
Coal	15	2.2	22	8.5	0.12	0.11
Siltstone	22	4.3	28	12.8	2.11	1.86
Fine sandstone	23	7.3	36	15.2	3.81	3.05

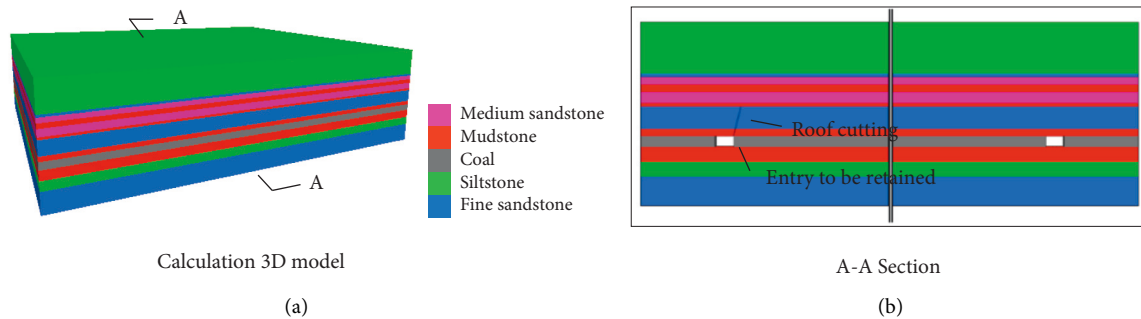


FIGURE 7: Numerical calculation model. (a) Calculation 3D model. (b) A-A section.

working face as an example, the FLAC 3D numerical simulation software is used to simulate the stress distribution of the surrounding rock in each stage of the entry with RCPRGERT, which can provide references for the entry support design [20, 21].

**4.2.1. Model Building.** Based on the actual engineering conditions of the test working face, this study makes some simplification and uses the FLAC 3D numerical simulation software to establish the 3D calculation model based on the Mohr Coulomb constitutive model. The dimensions of the model are length  $\times$  width  $\times$  height = 200 m  $\times$  170 m  $\times$  50 m, the entry excavation dimension is 200 m  $\times$  5 m  $\times$  3 m, and the working face is 200 m  $\times$  120 m  $\times$  3 m, including 30 m of the roof, 17 m of the floor, 1,507,600 grid units, and 1,673,580 nodes. The lithology and thickness of the model from the bottom to the top are as follows: fine sandstone 9 m, siltstone

4 m, mudstone 4 m, coal 3 m, mudstone 2 m, fine sandstone 6 m, mudstone 1 m, medium sandstone 3 m, mudstone 2 m, medium sandstone 2 m, fine sandstone 1 m, and siltstone 13 m. Specific mechanic parameters of the model layers are shown in Table 3, and Figure 7 shows the calculation 3D model.

Before numerical calculation, the left and right boundaries of the model limit the  $x$ -direction displacement, the front and back boundaries limit the  $y$ -direction displacement, and apply the horizontal compressive stress varying with the depth. The lower boundary limits the  $z$ -direction displacement. The upper boundary applies the uniform self-weight stress.

**4.2.2. Simulation Analysis in Each Stage.** In the construction space, there are certain restriction relationships between each construction steps of RCPRGERT, as shown in



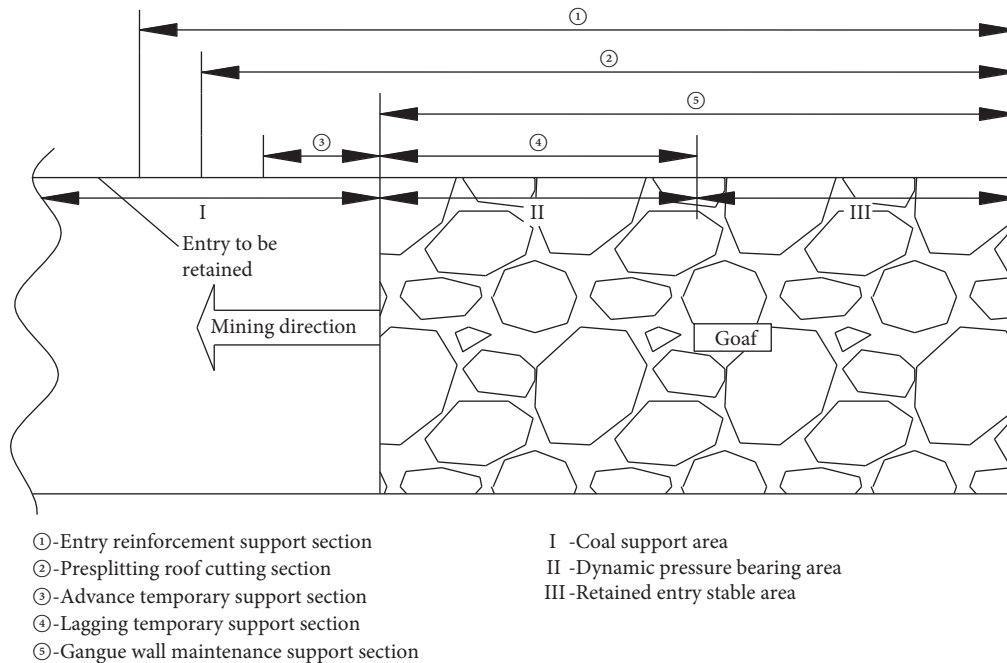


FIGURE 8: Process zoning map of RCPRGERT.

Figure 8. (1) To avoid the influence of the roof presplitting blasting on the entry reinforcement support and the advance support of the working face, the appropriate safety distance should be set between the presplitting section, reinforcement support section, and advance support section in construction. According to the field construction experience, the normal safety distance shall be at least 20 m, namely  $L_{①} - L_{②} \geq 20$  m,  $L_{②} - L_{③} - L_{④} \geq 20$  m. (2) To avoid the entry damage caused by the concentrated stress in the advance of the working face, the advance support section should be set in front of the working face, usually needing 30–40 m, namely  $L_{③} = 30\text{--}40$  m. (3) After the working face mining, the lagging temporary support and gangue wall maintenance support should be implemented in time, in which the gangue wall maintenance support is reserved until the retained entry is fully reused and abandoned, while the lagging temporary support can be withdrawn after the retained entry is stable. Generally, the retained entry tends to be stable about 200 m behind the mining working face, namely  $L_{⑤} \approx 200$  m, however, the specific value should be determined according to the field monitoring data.

Considering the dynamic advancement of the working face, according to the connection relationship between the time and space of the entry retaining process, the surrounding rock structure of the entry can be divided into three areas: the coal support area, the dynamic pressure bearing area, and the retained entry stable area (as shown in areas I, II, and III in Figure 8). Among them, the coal support area is the entry section in front of the working face, and the section is as shown in Figures 2(a)–2(c). At this time, the two sides of the entry are supported by the coal wall of the working face and the coal wall of the next working face, respectively. The stress of the entry surrounding rock in this

area along the advancing direction of the working face mining can be divided into three states: the advance support stress state, the reinforcement support stress state, and the original support stress state. The dynamic pressure bearing area is the retained entry section behind the working face, and the section is not stable yet, as shown in Figure 2(d). At this time, the two sides of the entry are, respectively, supported by the gangue wall and the coal wall, however, the gangue wall is not stable and is still affected by the dynamic pressure of the overburden. The retained entry stable area is the stable entry section behind the working face, and the section is shown in Figure 2(e). Two sides of the entry are still supported by the gangue wall and the coal wall, however, at this time, the gangue wall and the overburden of the goaf are in a relatively stable state, i.e., the gangue wall has been compacted and the overburden has no obvious dynamic pressure on it.

The numerical simulation results of the entry retaining each stage are analyzed as follows: the mechanical effect of roof cutting is mainly reflected in the change of the vertical stress field of the surrounding rock. Therefore, this study focuses on the analysis of the change of the vertical stress field.

- ① Entry excavation state: Figure 9(a) shows the numerical simulation result of the original rock stress state, and Figure 9(b) shows the stress state of the surrounding rock after the entry excavation. It can be seen that the stress distribution near the entry in this stage is basically symmetrical, and there are obvious stress concentration areas on both sides of the entry.
- ② Roof presplitting cutting state: Figure 9(c) shows the pressure releasing stress state of the surrounding

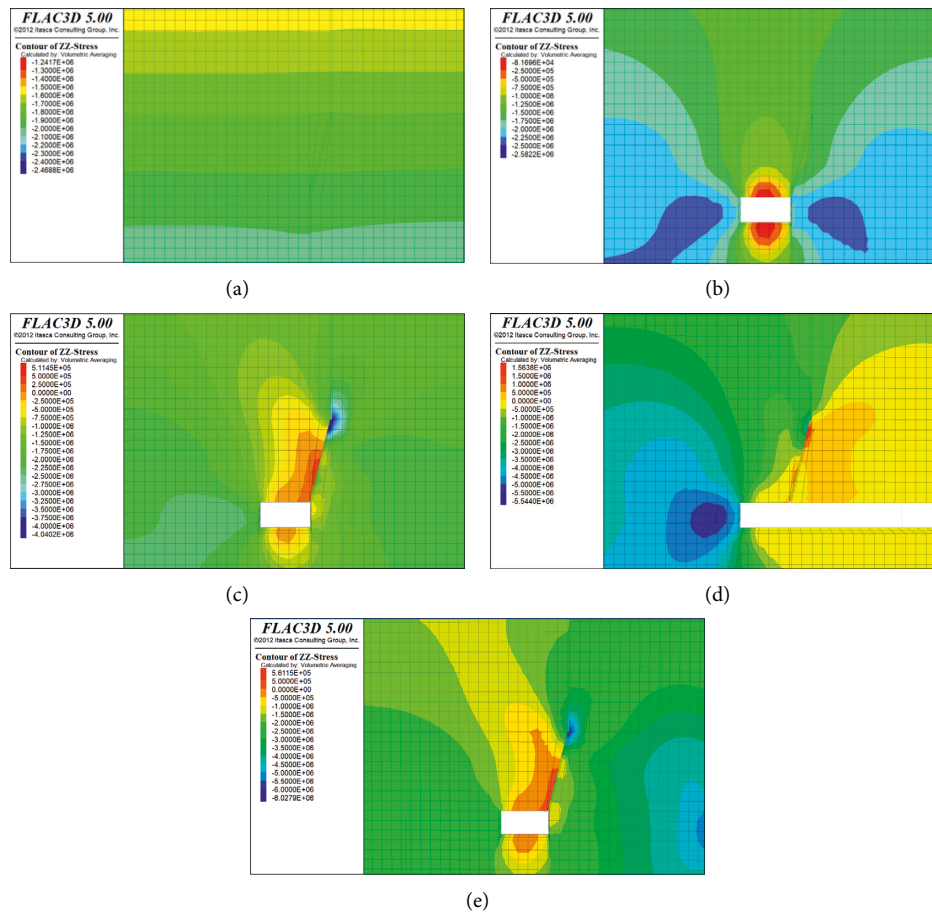


FIGURE 9: .Simulation results of RCPRGERT.

rock after the roof presplitting cutting. At this time, two obvious changes can be seen: firstly, the roof has obvious tensile stress area, and the roof on the roof cutting side is unstable without support. Secondly, the stress distribution on both sides of the entry is asymmetric, and the side pressure stress of the working face is less than the side pressure stress of the coal wall.

- ③ Working face mining state: Figure 9(d) shows the surrounding rock stress state after the working face mining and before the goaf is filled up. It can be seen that the roof cutting can significantly block the stress transmission of the goaf roof, and at this time, the coal wall side of the entry again has a significant stress concentration area, indicating that the surrounding rock of the entry tends to be unstable in this state. This stage is the key stage of reinforcement support and mining pressure monitoring, and it is also the most important stage of entry retaining.
- ④ Goaf-filled entry retained state: Figure 9(e) shows the stress state of the goaf roof collapse, and a stable filling in the goaf has been formed. At this time, there is a stress concentration area far away from the

entry, and the surrounding rock stress state of the retained entry returns to a relatively stable state.

**4.2.3. Idea of Hierarchical Support.** According to the above analysis of the technical process, mechanical mechanism, and numerical simulation, the following guidance and reference can be provided for the entry support design. In the entry excavation state, the entry is relatively stable as a whole. Only the compressive capacity of the coal pillars on both sides of the entry and the tensile capacity of the roof need to be strengthened. In the roof presplitting cutting state, the entry roof appears in an obvious tensile stress area, and the roof support needs to be strengthened at this time. Especially, the entry roof on the roof cutting side needs higher reinforcement support in the vertical direction to balance the cut off of horizontal stress transmission caused by roof cutting. In the process of working face mining, the entry successively experiences the advance stress disturbance, the mining disturbance, and the goaf roof collapse disturbance. However, before the gangue reaches the goaf roof, the stress state of the entry surrounding rock is consistent, the main stress source is the overall movement of the high layer roof, and the demand for the vertical support of

the retained entry roof at the roof cutting side reaches the peak value. In the goaf-filled entry retained state after the gangue reaching the goaf roof, the broken expansion gangue in the goaf bears a lot of support pressure, the retained entry tends to be stable, and the entry support demand decreases again.

In the process of RCPRGERT, the commonly used supporting materials are screw steel bolt, steel strand anchor cable, constant resistance anchor cable, single hydraulic prop, I steel beam, etc. Through the combination use of these supporting materials, the retained entry hierarchical support with roof cutting can be realized. First of all, in the entry excavation state, considering that the surrounding rock of entry is relatively stable at this time, it is proposed to adopt the screw steel bolt and steel strand anchor cable to realize the surrounding rock support, namely the first level support. Before the roof presplitting cutting, to prevent the entry roof large deformation on roof cutting side, it is proposed to increase the constant resistance anchor cable support on the basis of the original support, which can not only provide stable support capacity but also adapt to the roof subsidence deformation, namely the second level support. Then, turn into the working face mining process. This stage is the most unstable stage of the retained entry surrounding rock. In addition to the constant resistance anchor cable, it is proposed to add the single hydraulic prop as the third level support. The drainage mechanism of the single hydraulic prop can effectively and continuously limit the entry deformation and provide support. Hence, cooperate with the constant resistance anchor cable to realize the energy unloading of the entry surrounding rock and increase the entry support effect. Finally, in the goaf-filled entry retained state after the gangue reaches the goaf roof, the retained entry surrounding rock tends to be stable. At this time, the single hydraulic prop can be removed according to the actual situation to improve the economic benefits of entry retaining.

In the above three-level support system, each level of support has different characteristic, which can play their respective role in different entry retaining stages, specifically. (1) The first level support is rigid support, and the allowable shape variable of the support is small, which can play an effective role in protecting the surrounding rock in a relatively good condition. (2) The second level support is the plastic deformation support. When the deformation of the roof causes the failure or partial failure of the first level support, the second level support can provide continuous high energy support within a certain deformation range. (3) The third level support is hydraulic support. When the surrounding rock stress state of the entry is poor and the roof is severely disturbed and deformed, the third level support can provide supplement for the second level support, and it

enhances the overall energy absorption and support capacity of the support system through the pressure relief mechanism.

## 5. Support Strategy and Effect

Through the mechanical model calculation and numerical simulation analysis in the above section, the concrete support methods in each stage are developed in the test entry, and the scheme is verified by the implementation of the field support. In the field practice, the entry retention effect is still repeatedly tested by some detection means, and the entry retention design is modified accordingly. The engineering rock mass has strong anisotropy. Hence, in engineering practice, sufficient safety factors need to be reserved. In this study, it is mainly reflected in the reinforcement support, advance support, and post support links of the entry retaining, especially the withdrawal of post support, which must be carried out in strict accordance with the mine pressure monitoring results to ensure the safe implementation of the project.

### 5.1. Analysis of Support Characteristics

*5.1.1. Analysis of the Entry Original Support.* The section of 8304 working face air return entry is rectangular with a size of 5000 mm × 3100 mm. The entry is driven along the coal seam roof with equal height, and its original support form is the bolt-mesh-cable support. Besides, the support construction strictly follows the sequence of roof first and then sides. The main support design and parameters are as follows (as shown in Figure 10) [22, 23]:

- (1) Bolt support of roof: The entry roof is supported by four left-hand screw steel bolts in each row, with a diameter of 22 mm and a length of 2500 mm, and the row spacing between the roof bolts is 1100 × 1000 mm. Among them, the bolts on the side of the roof close to the solid coal incline to the solid coal side at 70° to the horizontal line, and other bolts are arranged vertically.
- (2) Bolt support of two sides: Three column left-hand screw nonlongitudinal rebar steel bolts are used for each side of the entry, with a diameter of 22 mm, a length of 2000 mm, and a row spacing of 900 × 1000 mm. The top and bottom four column bolts of the two sides incline to the roof and floor, respectively, at an angle of 10° with the horizontal line.
- (3) Anchor cable support of roof: The roof is uniformly distributed with 3 columns of steel strand anchor cables, with a diameter of 21.8 mm, length of 9000 mm, and spacing of 1600 × 3000 mm.

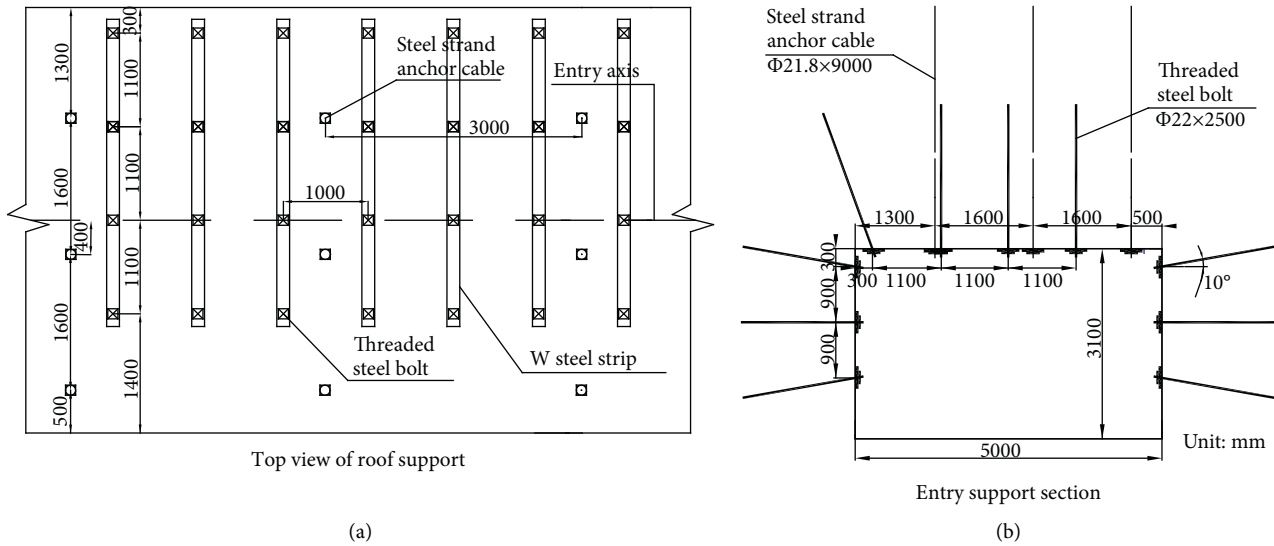


FIGURE 10: Original design of entry support. (a) Top view of roof support. (b) Entry support section.

### 5.1.2. Analysis of the Entry Retaining Support

(1) *Anchor Cable Support Design.* The key parts of anchor cable support under the condition of the roof cutting entry retaining are shown in Figure 1. To reserve enough anchor sections, the length of the anchor cable shall be designed to be 1-2m longer than the height of the roof cutting. According to the roof histogram of 8304 working face in the Tashan Coal Mine, it can be seen that there is a stable medium sandstone layer above the roof cutting top. Hence, the constant resistance anchor cable + steel strand anchor cable can be used for reinforcement and support. The design roof cutting height is 9.0 m from the roof rock broken expansion coefficient, and according to the suspension theory, the anchor cable support strength can be preliminarily calculated according to the following formula:

$$P_m \geq H_F \rho_d \left( L_h + \frac{H_F \tan \beta}{2} \right), \quad (20)$$

where  $P_m$  is the strength of the anchor cable support,  $\text{kN/m}^2$ .  $L_h$  is the width of the retained entry, m.  $\rho_d$  is the roof bulk density within the range of roof cutting. When the roof is composed of multiple layers, it can be calculated with weighted according to the layer thickness.

$$\rho_d = \sum_{i=1}^n \rho_i D_i / H_F, \quad (21)$$

where  $n$  is the number of roof layers,  $\rho_i$  is the unit weight of single roof,  $\text{kN/m}^3$ , and  $D_i$  is the thickness of the single roof, m. As the goaf roof can effectively support the overlying strata after falling, crushing, and expanding, the subsidence deformation of the overlying strata above the theoretical cutting height is small and the rock layer structure is relatively stable. Therefore,  $H_F$  is taken as the theoretical cutting height for calculation. According to the roof lithology of 8304 working face, it is calculated that  $\rho_d = 15.0 \text{ kN/m}^3$  and  $P_m \geq 112.5 \text{ kN/m}^2$ , that is to say, the anchor cable support

density is at least  $0.25/\text{m}^2$ . Hence, it is designed such that the anchor cable support column spacing is 3 m, and each row contains 3 anchor cables. Among them, the roof cutting side is disturbed by blasting and roof collapse most seriously. Hence, the two columns of the anchor cables at the cutting side are all constant resistance anchor cables. A constant resistance anchor cable is added to each two of the column closest to the roof cutting side for reinforcement, and the W steel strip is used to connect this column of anchor cables. The coal wall side is less disturbed. Hence, the steel strand anchor cable is used for support (shown in Figure 11).

(2) *Temporary Support Design.* In RCPRGERT, temporary support includes two parts: advance temporary support and lagging temporary support. Among them, the purpose of the advance temporary support is to prevent the disturbance and damage to the entry caused by the advance stress concentration of the working face. The advance temporary support of the 8304 working face adopts the support mode of adjacent working face under the condition that mining with coal pillars (as shown in Figure 12(a)), the DW40-300/110x single hydraulic prop are selected for support with the 1 m support row spacing and 3 props in each row, and the support range is an advance working face within 30 m.

The lagging temporary support is the special link of RCPRGERT. According to the analysis of entry retaining process, the key area of the temporary support behind the working face is the dynamic pressure bearing area. Hence, the support parameters can be calculated according to the surrounding rock structure of the dynamic pressure bearing area. As shown in Figure 13, the entry two sides of the dynamic pressure bearing area are, respectively, supported by the coal wall of the next working face and the gangue wall, and the gangue wall is not stable at this time [24, 25]. Among them,  $T_A$  is the horizontal thrust of rock block A,  $N_A$  is the shear force of rock block A,  $M_A$  is the bending moment of rock block A at point A',  $T_B$ ,  $N_B$ , and  $M_B$  are the same to rock

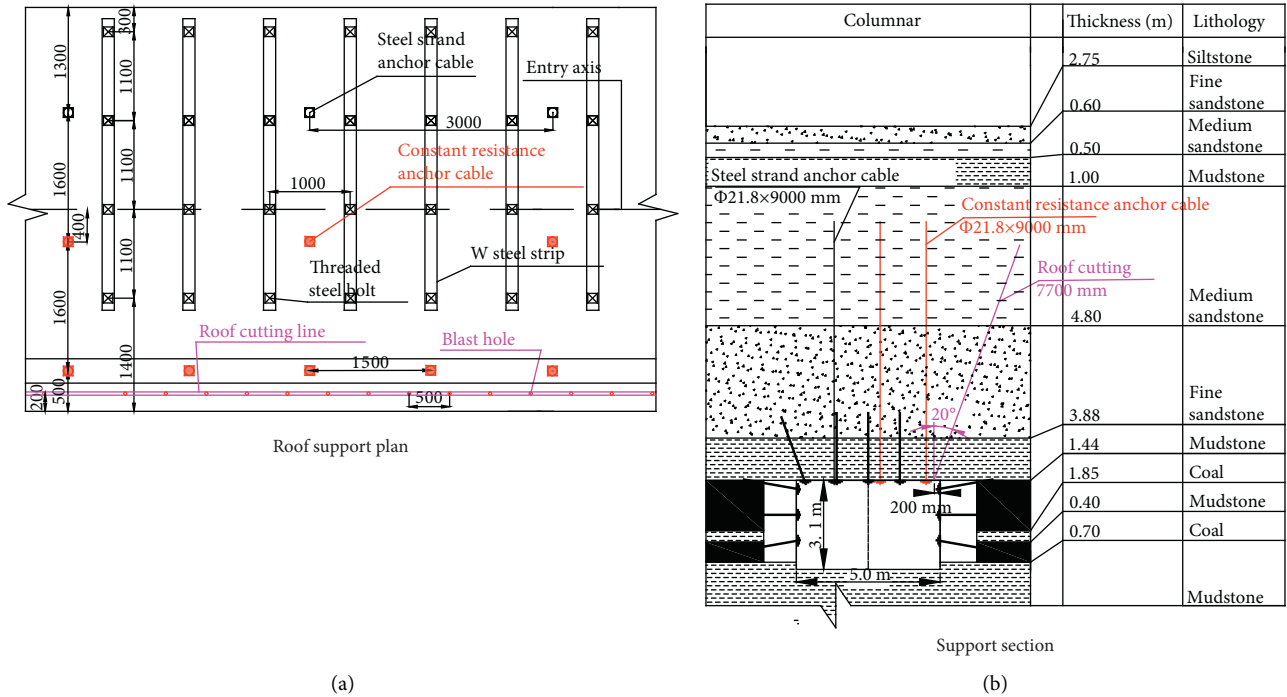


FIGURE 11: Design of anchor cable support. (a) Roof support plan. (b) Support section.

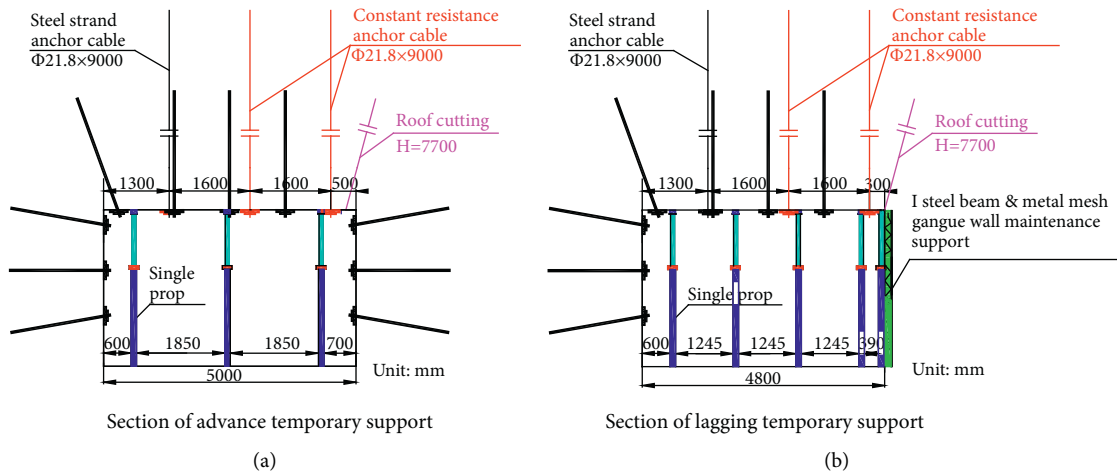


FIGURE 12: Forward and rear support design of 8304 working face. (a) Section of advance temporary support. (b) Section of lagging temporary support.

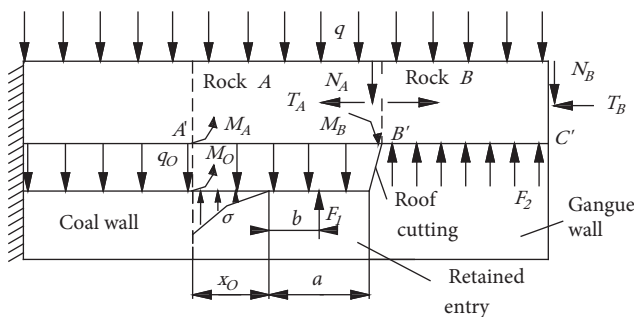


FIGURE 13: Simplified mechanical model of dynamic pressure area.

block B,  $M_0$  is the bending moment of the immediate roof,  $x_0$  is the width of the limit equilibrium zone in the lateral coal wall,  $\sigma$  is the roof support force of the coal wall in the plastic zone, and  $F_2$  is the support force of the gangue wall to overburden. As the gangue wall in this area is not stable, the support force  $F_2$  is small, especially when the retained entry enters into the dynamic pressure bearing section initially behind the working face, and  $F_2$  is approximately 0. When calculating the support force  $F_1$ , first of all, it is necessary to consider the rotary deformation to the goaf of the overlying rock block A with the axis of point  $A'$  at the elastic-plastic interface. The width of the coal wall limit balance area and

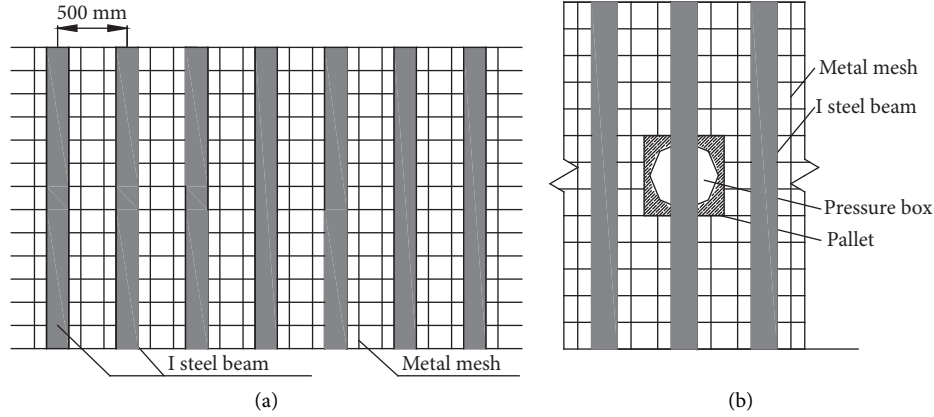


FIGURE 14: Design of gangue retaining support and pressure detection. (a) Gangue wall maintenance support design. (b) Gangue wall pressure monitoring scheme.

the support force of the coal wall in the plastic area to the roof are as follows:

$$x_0 = \frac{mk_a}{x} \ln \left( \frac{k_a \gamma H_c + c' / \tan \varphi}{c' / \tan \varphi + p_x / k_a} \right), \quad (22)$$

$$\sigma = \left( \frac{c'}{\tan \varphi} + \frac{p_x}{k_a} \right) e^{2x \tan \varphi / mk_a} - \frac{c'}{\tan \varphi}.$$

In the above formulas,  $c'$ ,  $\varphi'$  are the cohesion and internal friction angles of the interface between the coal seam and the roof and floor.  $Ka$  is the lateral pressure coefficient,  $\gamma$  is the average unit weight of the coal seam,  $HC$  is the mining depth, and  $PX$  is the support strength of the coal wall. The static equilibrium method is used to analyze the stress of rock blocks A and B.

$$\begin{cases} \sum F_x = 0 \\ \sum F_y = 0 \\ \sum M_B = 0 \end{cases}, \text{ we can get}$$

$$\begin{cases} T_A = T_B \\ N_A = N_B + qL \end{cases}. \text{ Then,}$$

$$M_B + T_B(H - \Delta S_B) - N_B L - qL^2/2 = 0$$

$$N_B = \frac{M_B + T_A(H - \Delta S_B) - qL^2/2}{L},$$

$$N_A = \frac{M_B + T_A(H - \Delta S_B) + qL^2/2}{L}. \quad (23)$$

To block A, according to  $\sum M_A = 0$ , it can be calculated as follows:

$$M_A + M_0 + F_1(x_0 + b) + \int_0^{x_0} \sigma(x_0 - x)dx + T_A(H - \Delta S_A) - M_B - qL^2/2 - q_0(x_0 + L_h)^2/2 - N_A(x_0 + L_h) = 0. \quad (24)$$

The lateral horizontal thrust is as follows:

$$T_A = T_B = \frac{qL}{2(H - \Delta S_B)}, \quad (25)$$

where  $\Delta SA$  is the subsidence of block A at point  $B'$ , and  $\Delta Sb$  is the subsidence of block B at point  $C'$ . Then,  $F_1$  is calculated as follows:

$$F_1 = \frac{[M_B(L + x_0 + L_h) + q(x_0 + L_h)^2] + \frac{q_0(x_0 + L_h)^2}{2}}{2 + qL(x_0 + L_h)/2} + \frac{q(x_0 + L_h)}{2} - M_A - M_0 - \frac{qL}{4} - \int_0^{x_0} \sigma(x_0 - x)dx / (x_0 + b). \quad (26)$$

By substituting  $C' = 0.1$  MPa,  $\varphi' = 18^\circ$ ,  $Ka = 2$ ,  $\gamma = 25$  kN/m<sup>3</sup>, and  $PX = 0.04$  MPa into the calculation, the support force requirement of per meter entry is 1650 kN, and as the working resistance of the adopted single prop is 300 kN, the prop support density of the lagging temporary support is at least 1.1/m<sup>2</sup>. Therefore, in the 8304 working face entry retaining test, each row of the lagging support is arranged with 5 single props, among which the column spacing of the single props at the roof cutting side is 0.5 m, the other four columns spacing is 1 m, and the support section is shown in Figure 12(b). Then, according to the deformation monitoring of the retained entry, it can be judged that whether the entry is in the stable stage. After the retained entry is stable, the lagging temporary support can be withdrawn gradually according to the specific situation.

(3) *Gangue Wall Maintenance Support Design.* At the entry retaining initial stage of 8304 working face, according to the previous entry retaining experience, No. 11 mining I steel beam is selected to support the gangue wall with a metal mesh, and the support spacing of I steel beam is 500 mm (as shown in Figure 14(a)). At the same time, the real-time measurement of the gangue wall maintenance lateral pressure is carried out using the pressure box (as shown in Figure 14(b)). The measurement results, combined with the entry deformation monitoring, the stress monitoring of the anchor cable at the roof cutting side, and so on, can be used to judge the stability of the gangue wall, as the basis for the reinforcement, optimization, or withdrawal of the gangue maintenance support.

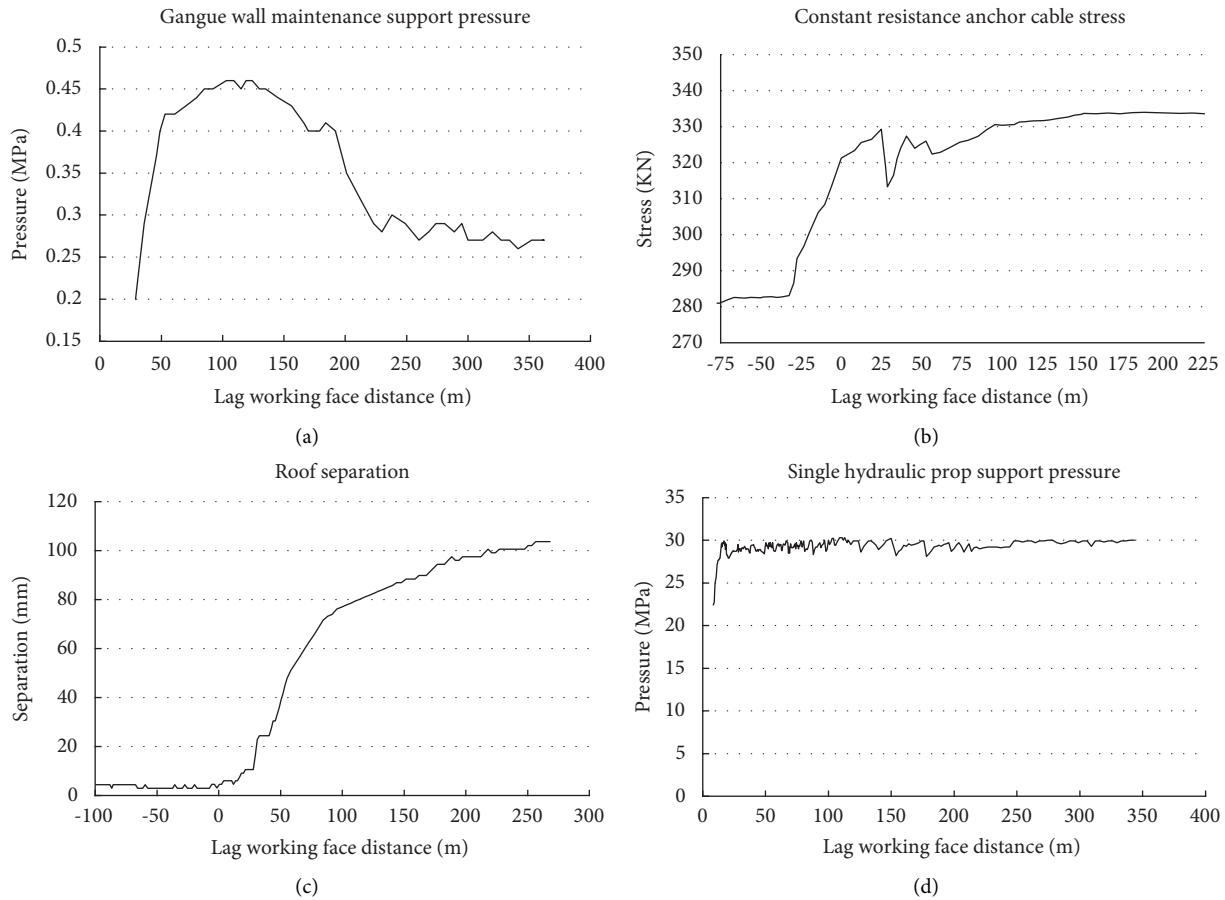


FIGURE 15: Monitoring results. (a) Gangue wall maintenance support pressure. (b) Constant resistance anchor cable stress. (c) Roof separation. (d) Single hydraulic prop support pressure.

**5.2. Example Support Effect.** During the mining process of the test working face, the stress of I steel beam maintaining gangue wall, the constant resistance anchor cable at the roof cutting side and single hydraulic prop and roof separation amount are monitored, respectively, so as to judge the deformation of the retained entry and whether the entry enters the stable state.

The monitoring results are shown in Figure 15, in which the initial value of the gangue wall maintenance support pressure is 0.2 MPa when it is 29 m behind the working face, and then the pressure gradually rises until it reaches the maximum value of 0.46 MPa when it is 103 m behind the working face. Then, the pressure decreases again with the mining advance. At last, the pressure gradually stabilizes at 0.28 MPa when it is 223 m behind the working face. The stress value of the constant resistance anchor cable at the roof cutting side also experiences the process of rising first, then falling down, and at last tends to be stable. The anchor cable stress starts to rise when it is 19 m ahead of the working face, reaches the constant resistance value when it is 3 m behind the working face, starts to decline when it is 25 m behind of the working face, and then slightly rises again, until it is 112 m behind the working face. The constant resistance value tends to be stable, and the displacement of cable lock can be obviously observed. The average move up

value of the cable lock is about 37 mm. The length of the hydraulic support of the working face is 8 m. Hence, the single hydraulic prop starts to be erected when it is 8 m behind the working face. When the single hydraulic prop at the roof cutting side is 18 m behind the working face, the working resistance reaches the peak value of 29.6 MPa, and the leakage phenomenon of the hydraulic prop begins to appear. When it is 214 m behind the working face, the pressure value tends to be stable, about 29 MPa.

From the monitoring data, it can be seen that the effect of roof cutting pressure releasing is obvious. When the retained entry is behind the working face by more than 223 m, the gangue wall enters into a stable state. When it is behind the working face by more than 112 m, the constant resistance anchor cable enters into a stable state. When it is behind the working face by more than 218 m, the convergence between the roof and floor enters into a stable state. When it is behind the working face by more than 214 m, the stress of the single hydraulic prop enters into a stable state. Therefore, it can be judged that the retained entry enters into the stable state as a whole when it is behind the working face by more than 230 m. Then, the hydraulic props can be partly withdrawn, retaining the gangue wall maintenance support and the hydraulic props at the roof cutting side. The entry retaining effect is as shown in Figure 16. After the hydraulic props are



FIGURE 16: Field retaining effects. (a) Entry retaining section. (b) Gangue wall forming effect.

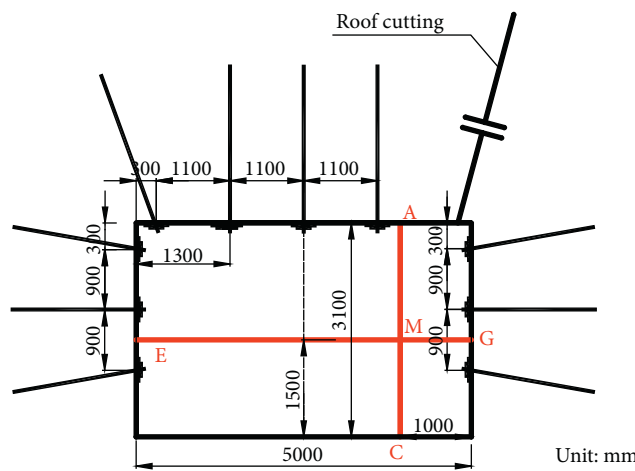


FIGURE 17: Deformation monitoring scheme of retained entry.

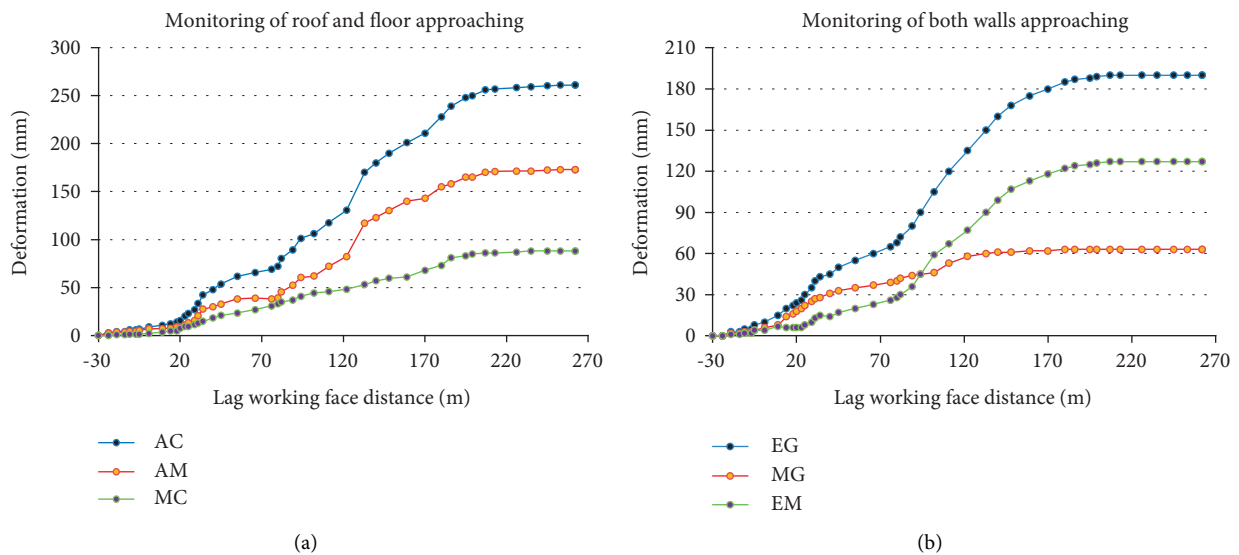


FIGURE 18: Typical monitoring results at footage 500 m. (a) Monitoring of roof and floor approaching. (b) Monitoring of both walls approaching.

withdrawn, the roof separation amount and the convergence amount between the roof and the floor increase slightly. The amount of roof separation is finally stable at 103.7 mm. The

approach of roof and floor is stable at 303.4 mm, and the average height of the retained entry is 2796.6 mm, which are enough to meet the reuse requirements.



The scheme of the retained entry deformation monitoring is shown in Figure 17, namely monitoring the length variation of line AM, MC, EM, and MG, to determine the specific degree and source proportion of entry deformation.

The entry deformation monitoring results at footage 500 m of retained entry are as shown in Figure 18. It can be seen that (1) in the early stage, the approach of entry two sides is mainly characterized by the deformation of the gangue wall, however, in the later stage, the approach is mainly caused by the coal wall deformation. At 500 m monitoring section of the retained entry, the gangue wall approach amount is 63 mm, and the coal wall approach amount is 127 mm. (2) The final deformation of the monitoring section is 88 mm of floor heaving and 173 mm of roof subsidence, and the effect of entry retaining is good.

## 6. Conclusions

- (1) Through the technical summary and mechanical analysis, the mechanical characteristics of the roof cutting short beam with RCPRGERT are analyzed and summarized, and the key points of the retained entry roof support are defined as preventing the occurrence of tensile stress and subsidence deformation on the roof cutting side of the entry roof.
- (2) The stress distribution characteristics of the surrounding rock in different entry retaining stages are simulated by numerical simulation, and the hierarchical support idea is put forward. According to the stress state evolution of the surrounding rock in each stage of entry retaining, different supporting materials are used to reinforce the support of the surrounding rock in different stages, namely the three-level support system with RCPRGERT.
- (3) The supporting design is carried out for the test working face, and the supporting effect of the retained entry is verified by the engineering test. Taking the 8304 working face of Tashan Coal Mine as an example, the support design and field test are carried out based on the three-level support idea, and the roof support and entry retaining effects are well.

## Data Availability

All the data used to support the findings of this study are available from Xingen Ma upon request.

## Conflicts of Interest

The authors declare that they have no conflicts of interest.

## Acknowledgments

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