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KEY TECHNOLOGIES AND ITS APPLICATION OF GOB-SIDE ENTRY RETAINING BY ROOF CUTTING IN A DEEP MINE

There are many problems associated with the surrounding rocks of the gob-side entry retaining by roof cutting (GERRC) as they are difficult to stabilise in deep mines. The following needs to be studied to understand the problems such as the pressure relief mechanism, evolution law of the surrounding-rock stress and the key technologies of GERRC in deep mines.

Cracks are formed by advanced directional blasting to sever the path of stress transmission from the roof of the goaf to the roof of the entry and reduce the lateral cantilever length of the roof. Therefore the surrounding-rock stress and roof structure are optimised. The broken and expanded gangue formed by the collapse of the strata in the range of roof cutting fills the mining space adequately, which avoids a rapid pressure increase caused by the roof breaking impact and slows down the movement of overlying strata. The deformation of the deep surrounding rocks is transformed from “abrupt” to “slow”, and the surrounding-rock deformation of the retained entry in deep mines is significantly reduced. The average pressure and periodic pressure of the supports near the blasting line can be reduced by the blasting cracks to a certain extent, mainly due to the reduction of the length of the immediate roof cantilever and the effective load of the main roof. The combined support technologies for GERRC in deep mines were proposed, and field tests were performed. The monitoring results show that the coordinated control system can effectively control the deformation of deep rock masses, and all indexes can meet the requirements of the next working face after the retained entry is stabilised.

Keywords: Deep mine; gob-side entry retaining; pressure relief mechanism; cooperative control; engineering application

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1. Introduction

In China, coal is the main energy source and occupies the dominant position in the primary energy structure. To meet the needs of China's rapid economic development, coal will most likely remain the primary energy source in China. Yet the numbers of coal resources are decreasing day by day because of large-scale mining, and with the current low economy of coal, high mining costs and low coal recovery rate caused by the traditional coal pillar mining method are increasingly prominent. In addition, the coal pillars left in the section easily cause stress concentrations [1], which can cause mine disasters such as large deformation of surrounding rocks [2], rockburst [3,4], and coal (rock) and gas outbursts [5,6]. Gob-side entry retaining (GER) is a non-pillar mining method [7,8]. It can reduce mining costs, improve resource recovery rate, alleviate the tension of mining replacement, and avoid mine disasters caused by coal pillars. Gas accumulation is reduced in the upper corner of the working face by a Y-type ventilation system formed by the (GER) mining method. Overall, GER is one of the main directions for the sustainable development of China's coal industry.

Currently, the commonly used technology of GER is to fill the gateroad side along the goaf with the backfilling materials as the advancement of the working face, such as waste rocks, concrete and high water materials, etc. The original gateroad is retained and used as the gateroad of the next working face. Therefore, coal pillars are eliminated. Experts have researched the GER throughout China and have advanced the supporting technology of entry-side and entry-in, determining the movement law of the roof structure, the interaction mechanisms between surrounding rocks and support, etc. [9-15]. Gob-side entry retaining by roof cutting (GERRC) is a new non-pillar mining technology, which was first proposed by He et al. [16]. It is different from the traditional GER technology, as it eliminates the roadside filling process and does not affect the mining speed [17]. Due to its technical advantages, GERRC has been widely used in shallow and medium-depth mines in China. Researchers have determined the mine pressure law, roof movement, key parameters and control technology of GERRC in shallow and medium-depth mines. He and Zhang et al. [16] first applied the technology of GERRC to thin coal seams in the Baijiao Coal Mine, solving the problem of goaf gas accumulation. Sun et al. [18] and Guo et al. [19] researched the key parameters of GERRC in thin coal seams. Given the problem of the GERRC with a hard roof, the mechanism of roof collapse and control measures were analysed by Zhang et al. [20]. He et al. [21] established the "roof-support" mechanical model of GERRC and applied it in a shallow coal seam with a composite roof. He et al. [22] and Ma et al. [23] studied the influence of directional roof blasting on the stress evolution of the surrounding rocks of GERRC, and a control technology system for the surrounding rocks of GERRC was formed. Zhang et al. [24] applied the technology of GERRC in engineering and obtained a satisfactory effect. Dong et al. [25] propose a new analysis method for stability monitoring in practical rock engineering, which is very helpful to the surrounding rock deformation monitoring of GERRC.

The above research has promoted the development and application of GERRC, but most are concentrated in shallow and medium-depth mines with simple geological conditions. However, the original rock stress, tectonic stress and mining stress increase significantly in deep mining [26]. The impact and rheology of the surrounding-rock deformation become prominent [27], and the retained entry is difficult to stabilise, which considerably limits the application and development of GERRC technology in deep mines. Currently, the research on GERRC in deep mines is still in the exploratory stage, there is no systematic theory and technology, and further research is needed. Here, the stress evolution law and surrounding rock deformation mechanism of GERRC in deep mines were comprehensively investigated using theoretical analysis, numerical simulation and field monitoring. Finally, a joint control system that can effectively control the surrounding rock deformation of GERRC in deep mines is proposed and successfully applied in the field. The research results have important theoretical and practical significance for the development of GERRC in China's deep mines.

2. Technical principles of gob-side entry retaining by roof cutting

Based on the technical principles of GERRC combined with field engineering experience, the construction process of GERRC is divided into four steps: support, cutting, protection and sealing. Firstly, from Fig. 1, the roof is reinforced with constant resistance and a large deforma-

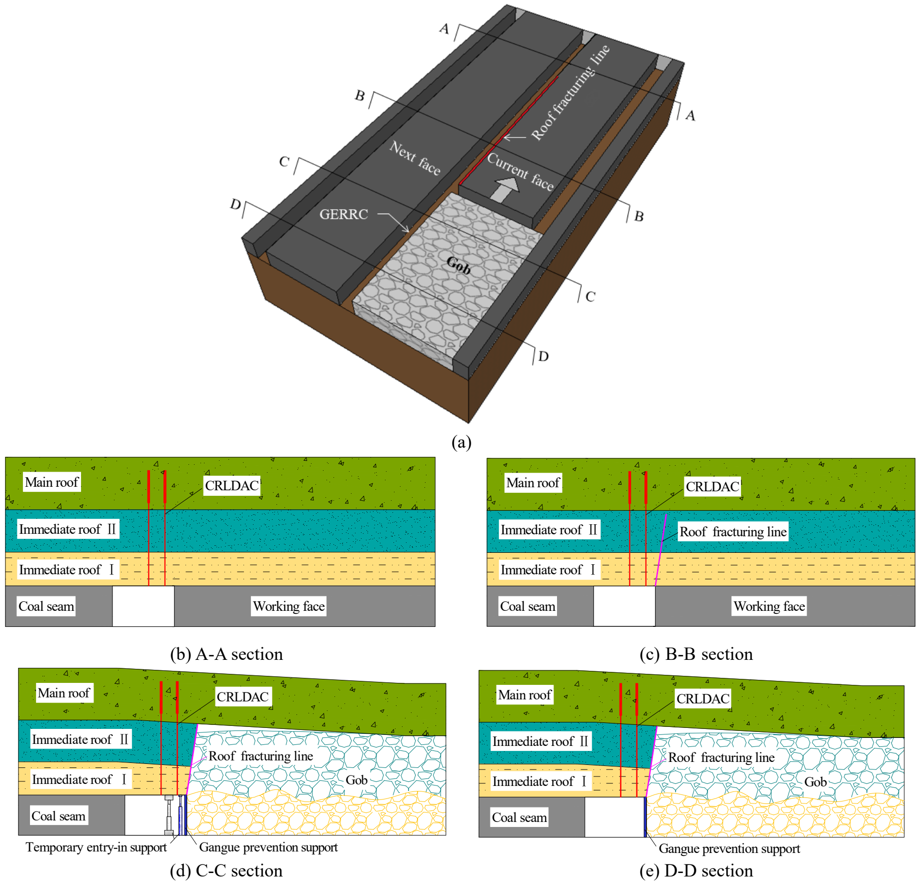


Fig. 1. Technological principle of the GERRC approach: (a) three-dimensional view of GERRC technology; (b) roof support with CRLDAC; (c) energy-gathering blasting to split roof; (d) entry-in support and gangue prevention support; (e) sprayed concrete to close the gangue rib

tion anchor cable (CRLDAC), which ensures the roof's stability during the process of GERRC (Fig. 1b). Secondly, according to the design position, directional blasting holes are constructed in the roof with a special drill, and directional pre-splitting blasting is carried out in advance of the working face to form the roof pre-splitting structural plane (Fig. 1c). Thirdly, after the coal seam of the working face is mined, an I-beam or U-steel props are installed as gangue prevention support along the pre-splitting line in time, the entry-in and entry-side of GERRC are supported by special supports or single-props, the roof of the goaf automatically collapses to form the roadway side due to mine pressure and self-weight, thus eliminating the entry-side filling process of traditional GER (Fig. 1d). Finally, after the gangue is gradually compacted, the retained entry is in a stable state, the temporary entry-in and entry-side support can be gradually withdrawn, and concrete is sprayed to close the gangue rib to prevent air leakage and isolate harmful gases (Fig. 1e). The entry is successfully retained and can be used as a gateroad for the next working face.

3. Mechanism of roof cutting and pressure relief by directional blasting

Under traditional blasting, the energy cannot be gathered well and will cause significant damage to the surrounding rock near the hole and even to the entry support. To achieve accurate blasting, technology for controlling the directional fracture of rock has made substantial progress in recent years [28]. After blasting, the roof rock mass is cut off, and the influence of blasting on roof stability is the lowest. This is very important for the success of entry retaining.

The directional blasting technology is mainly realised by changing the structure of the blasting hole, the slotted charge and the charging structure. The energy-gathering blasting is a type of directional fracture-controlled blasting technology that was invented in the research of controlled blasting and the development of a specially processed PVC pipe (Fig. 2). The PVC pipes have a specific strength, which can be changed according to the lithology and required crack effect. The blasting technology generates an agglomeration effect on explosive products, causing them to form an energy-gathering flow in a set direction. This, in turn, generates a tensile stress concentration, fracturing the roof strata in a set direction which forms a structural plane.

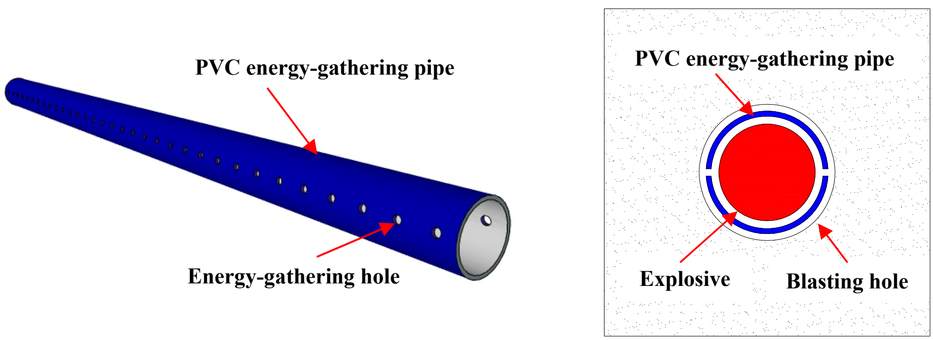


Fig. 2. PVC energy-gathering pipe and charge structure

The principle of energy-gathering blasting is shown in Fig. 3. In addition, because the device is located within the blasting hole, the direct effects of the explosion on the surrounding rocks near the hole are weakened, and crack development is suppressed in the non-set direction, thereby reducing the damage to the rock mass in the non-set direction.

Energy-gathering blasting is performed on the roof in front of the working face. A penetrative crack in a set direction is brought about by the action of shock waves, stress waves, and explosive gas. After the coal seam of the working face is mined, the main roof is fractured along the pre-split plane. The main roof on the side of the goaf can collapse along the pre-split plane in time, reducing the length of the lateral cantilever and severing the mechanical relationship between the goaf roof and the entry roof. Thereby the stress transmission from the goaf roof to the entry roof is weakened, and the additional load of the entry-side support is reduced. In addition, the broken and expanded gangue formed by the collapse of strata in the range of roof cutting can better fill the mining space, which avoids a rapid pressure increase caused by the roof breaking impact and slows the movement of the overlying strata. The deformation of the deep surrounding rocks is transformed from "abrupt" to "slow", and the surrounding-rock deformation of the retained entry in deep mines is significantly reduced.

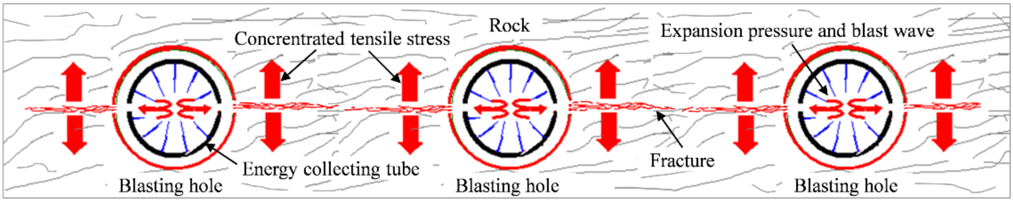


Fig. 3. Principle of energy-gathering blasting

4. Numerical simulation

4.1. Numerical model

To study the laws of overlying strata collapse and the surrounding rock deformation of GERRC, a numerical model was established using the discrete element software 3DEC and according to the geological conditions of the 21304 working face in the Chengjiao Coal Mine, China. As this calculation mainly studies the influence of the pre-splitting structural plane on the collapsing law of the overlying strata and the surrounding rock deformation, it is considered to be a two-dimensional plane problem and ignores the supporting effect of the anchor and cable. The model's fixed bottom and the lateral sides are roller constrained. The model's top is a free boundary, and an equivalent load of 20.04 MPa is applied to simulate the overburden weight of 830 m. Fig. 4 shows the calculation model and loading method. The mechanical parameters of coal and strata are obtained from the mine exploration and laboratory tests, as shown in Table 1. The slit was considered as an empty model, and it was directly excavated during the calculation. The sequence of numerical simulation is as follows: generating the original rock stress field → roadway excavation balance → slit excavation balance → mining coal seam.

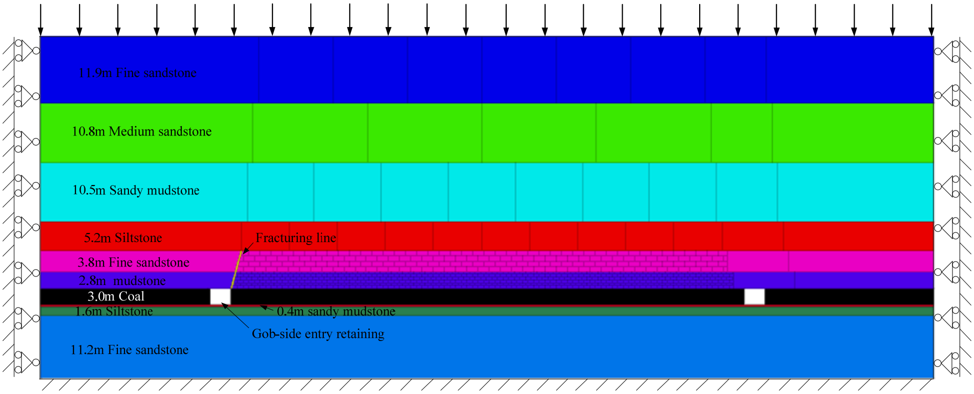


Fig. 4. Numerical model

TABLE 1

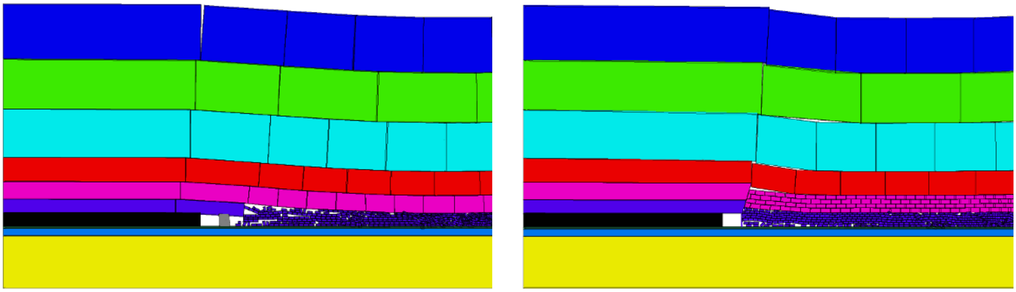
Mechanical parameters of rock mass and coal

Rock Strata	Density (kg/m^3)	Bulk modulus (GPa)	Shear modulus (GPa)	Tensile strength (MPa)	Cohesion (MPa)	Friction Angle (deg)	Normal stiffness of structural plane (GPa/m)	Tangential stiffness of structural plane (GPa/m)
Siltstone	2600	8.2	6.5	1.2	3.3	36	3.3	3.6
fine sandstone	2600	7.6	6.8	1.1	3.1	33	2.8	2.9
mudstone	2200	3.1	1.8	0.5	1.2	22	1.6	1.8
Coal	1600	1.3	0.6	0.2	0.7	18	1.2	1.2
sandy mudstone	2400	3.3	1.5	0.4	1.3	24	2.2	2.3
medium sandstone	2700	5.5	5.7	0.8	2.7	31	2.6	2.5

4.2. Analysis of overlying strata collapse and surrounding rock deformation

The calculation models of entry-side filling and roof cutting were established for comparative analysis. The results of the numerical simulation are shown in Fig. 5 and Fig. 6.

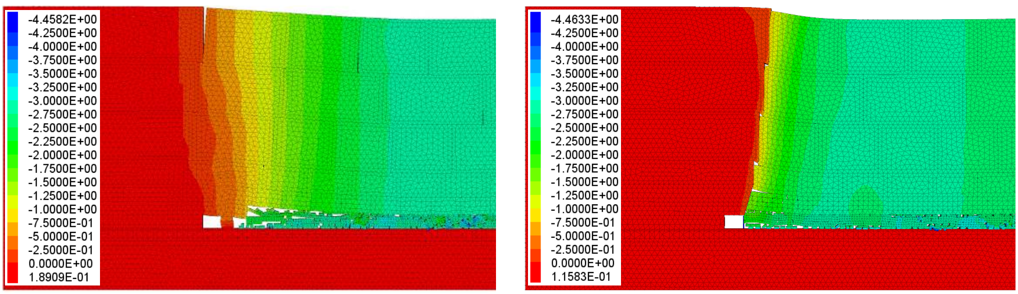
As shown in Fig. 5a, due to the insufficient strength of the entry-side filling body, it is difficult to sever the overlying roof effectively by the support resistance provided, leading to the breakage of the roof on the side of the coal sidewall and the formation of a longer lateral cantilever on the side of the goaf. Moreover, the gangue formed by the collapse of the immediate roof cannot fill the goaf, which creates space for the overlying strata movement. The overlying strata have a large rotation deformation due to gravity stress, which makes the retained entry and filling body bear a large additional load. This causes severe roof subsidence of the retained entry and bulging deformation of the filling body. Finally, the maximum roof subsidence of the retained entry is 500 mm (see Fig. 6a).



(a) (b)

Fig. 5. Overlying strata collapsed form of GER with different methods:
(a) GER by entry-side filling; (b) GER by roof cutting

As shown in Fig. 5b, when GERRC is adopted, the roof of the goaf collapses along the pre-splitting structural plane after mining, a lower short-arm beam and an upper masonry beam are formed. The short-arm beam greatly reduces the lateral cantilever length of the basic roof, and the roof pressure is fully released. The masonry beam can bear most of the overburden pressure, which makes the retained entry r in a low-stress environment. Therefore, the load transfer from the goaf roof to the entry roof is weakened, and the fracture line of the roof is transferred to the side of the goaf, which is more conducive to the maintenance of the retained entry. In addition, after the strata in the range of roof cutting collapse, the broken and expanded gangue can adequately fill the mining space, providing timely support for the overlying roof and limiting its rotary subsidence. This overall significantly reduces the roof subsidence. When the retained entry is stable, the maximum roof subsidence is about 250 mm (see Fig. 6b), which is 50% less than that in the retained entry stabilised by entry-side filling. Therefore, the numerical simulation results verify the advantages of GERRC compared with GER by entry-side filling, such as more optimised surrounding rock structure and smaller surrounding rock deformation.



(a) (b)

Fig. 6. Displacement field distribution of GER with different methods (unit : mm):
(a) GER by entry-side filling; (b) GER by roof cutting

5. Key techniques for controlling surrounding rocks of GERRC in deep mine

According to the deep geomechanical environment and the deformation characteristics of the rock mass, the collaborative control system of the GERRC surrounding rocks in deep mines requires constant resistance for pressure control of the roof. There needs to be a retractable entry-side support device to adapt to the deformation of the roof and floor and resist the lateral gangue pressure and temporary entry-in support with high resistance against dynamic pressure. Finally, the support length needs to increase to protect the coal sidewall.

5.1. Roof support with CRLDAC

Based on the idea of constant resistance and energy absorption, He et al. [29] provided the basic theory for large deformation control of surrounding rocks and successfully developed the constant resistance and large deformation anchor cable (CRLDAC). The CRLDAC is essentially composed of a constant resistance device, stranded steel rope and lock (Fig. 7a). As shown in Fig. 7b, the constant resistance device can be divided into three parts: a steel cylinder, tray, and pyramid. The tray is welded to the steel cylinder, and the pyramid is embedded in the steel cylinder. Through the mutual cooperation between the steel cylinder and the pyramid, different constant

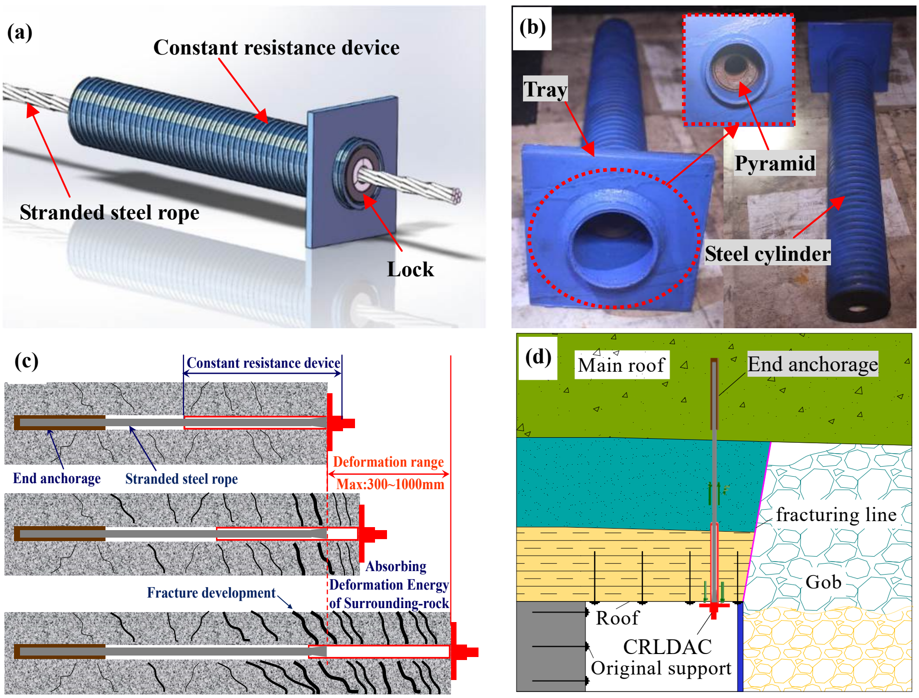


Fig. 7. Structure and work principle of CRLDAC: (a) structure of CRLDAC; (b) structure of constant resistance device; (c) work principle of CRLDAC; (d) support position of CRLDAC

resistance values can be set. When the friction between the steel cylinder and the pyramid reaches the constant resistance value set, the pyramid will slide in the steel cylinder to compensate for the large deformation of the surrounding rocks. The working principle of CRLDAC is shown in Fig. 7c. First, the elastic deformation of the stranded steel rope adapts to the deformation of the surrounding rocks. The stress of the stranded steel rope increases rapidly during the elastic deformation process. When the stress of the stranded steel rope reaches the constant resistance value of CRLDAC, the constant resistance device starts to slip and absorbs the deformation energy of the surrounding rocks. It can maintain a constant support resistance throughout the movement. The stress of the surrounding rocks will be reduced accordingly, and the stress in the stranded steel rope will also be reduced. When the stranded steel rope stress is reduced to less than the constant resistance value, The constant resistance device stops sliding, and the surrounding rocks reach a steady-state again. The CRLDAC allows for large sliding displacement to adapt to the deformation of deep rock masses and maintains a high support resistance during the sliding process [30]. This device can ensure the stability of the surrounding rocks in deep roadways.

The surrounding rock deformation of the roof is mainly due to the high stress in such deep environments due to the direct bearing body of the overlying load. In addition, due to the rheological characteristics of the deep surrounding rocks, the deformation of the rock mass in the anchoring area often exceeds the limit extension of a conventional anchor (cable), resulting in failure of the support. In addition, energy-gathering blasting also causes disturbance to the roof of the retained entry.

By reinforcing the roof of the retained entry with CRLDAC (Fig. 7d), a high pre-tightening force can be applied to CRLDAC during the initial stage, effectively eliminating roof separation and dislocation and improving the bending moment resistance of the roof strata on the side of the coal wall. The CRLDAC can adapt to the deformation of the deep surrounding rocks due to its large extension and maintain a high working resistance during the deformation process [31], ensuring the roof stability of the retained entry. It can also resist the impact disturbance of energy-gathering blasting to the roof of the retained entry and absorb the blasting energy, reducing the damage to the bearing structure [29]. It is also beneficial to the stability of the retained entry.

5.2. Enhanced support length for coal sidewall by anchor cable

After the coal seam is mined, the mining pressure is transferred to the coal sidewall and entry-side supporting body. The coal sidewall and entry-side supporting body become the main load-bearing structures for GER. After the gangue in the goaf is compacted, the retained entry is in a stable state. At this time, the overlying strata pressure is mainly shared by the coal sidewall and the gangue in the goaf. Therefore, the stability control of the coal sidewall is particularly important as it is one of the main load-bearing bodies of the roadway in the process of GERRC.

As this is a high-stress environment in the deep subsurface, the plastic area of the coal sidewall is larger compared to shallow and medium depth mines. Moreover, as an important support body for the roof of the retained entry after the coal seam is mined, the pressure on the coal sidewall is further increased, resulting in serious damage to the shallow section of the coal wall. The stress is transferred to the deep section of the coal body. With the rotary subsidence of the lateral roof, the damage depth of the coal sidewall gradually increases. If the bolt length is insufficient, the anchorage area of the bolt will be in the broken coal body rather than in the stable coal body, leading to the failure of the support for the coal sidewall (Fig. 8a). This is not an effective support for the roof strata and causes instability of the whole load-bearing structure.

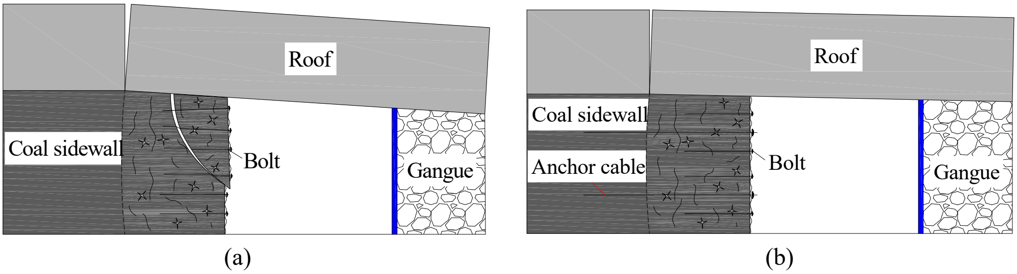


Fig. 8. Action mechanism of coal sidewall anchor cable:
(a) coal sidewall without anchor cable; (b) coal sidewall with anchor cable

Therefore, based on the bolt support, the coal sidewall of the entry in deep mines is reinforced by anchor cables, which increases the support length for the coal sidewall. The shallow load-bearing layer of the bolt can be anchored in the elastic zone of the stabilised coal body by anchor cables (see Fig. 8 for the support mechanism of the anchor cable of the coal wall). This enhances the lateral stress supply for the coal sidewall, reduces the expansion deformation caused by the rock-breaking within the anchorage range, and prevents the coal body from further expansion and loosening at depth. In addition, it improves the mechanical properties of the anchor body and fully mobilises the self-supporting capacity of the coal body, which ensures the stability of the coal sidewall during the entry retaining process. Overall, this forms better support for the roof strata.

5.3. Temporary support with hydraulic supports in the entry

As the method of GERRC cancels the entry-side filling, the entry-in support also plays the role of entry-side support. Before the roof fully collapses after the coal seam has been mined, the overlying strata pressure is mainly shared by the coal wall and the entry-in support. Due to the rapid weighing speed and high strength of the deep mining pressure, the instantaneous impact of the roof fracture is powerful. If the temporary support in the entry adopts low-intensity support equipment such as a single pillar, it will ultimately be impacted and broken (Fig. 9a), which will lead to severe roof subsidence or even collapsing of the retained entry.

The hydraulic support is designed and manufactured for temporary entry-in support of a deep mine with high pressure and to sustain significant impacts and supply high strength to the system. This support system mainly consists of the top beam, column, base, and hydraulic system, as shown in Fig. 9b. The hydraulic support is adopted as temporary entry-in support, which is arranged on the side of the goaf, and the single hydraulic prop is used as auxiliary support. Due to the strong ability to support the roof, the hydraulic support provides an adequate resistance for roof cutting and can resist the strong dynamic pressure of a deep mine. It effectively reduces the impact of the initial roof rotation subsidence on the retained entry, ensuring the stability of the retained entry during the roof movement. After the roof of the mined-out area completely collapses and the gangue is compacted, the overlying strata are effectively supported by the gangue, while the retained entry tends to remain stable. The temporary support in the entry can be gradually withdrawn.

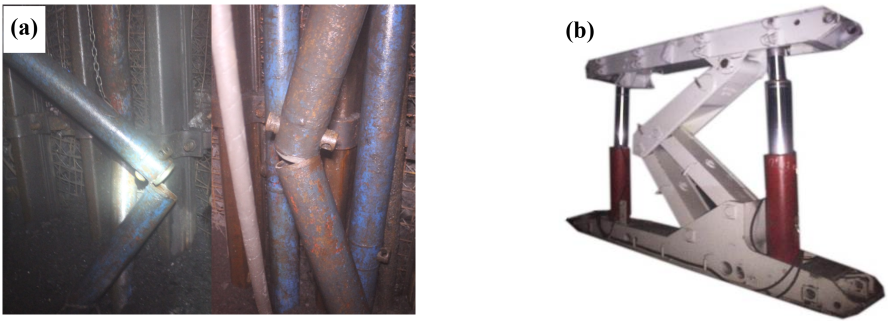


Fig. 9. Temporary strengthening support in the entry: (a) impact damage of entry-in single hydraulic props in deep mine; (b) Hydraulic support for temporary entry-in support

5.4. Contractible U-shaped steel for entry-side gangue prevention

After the gangue in the goaf is fully compacted, the GERRC is in a stable state. At this time, the overlying strata pressure of the entry is mainly borne by the coal wall and the gangue in the goaf. The final stability of GERRC is determined by the coal wall and the material and volume of the gangue in the goaf, so the entry-side support (gangue prevention support) is also very important. The gangue rib is mainly controlled by the gangue prevention device. Due to the high-stress environment in the deep mine, the rigid gangue prevention device often bends and breaks because it cannot adapt to large deformation (Fig. 10a), thus losing the capacity of gangue prevention. The bending and damage of the rigid gangue prevention support are mainly caused by the vertical deformation of the deep surrounding rocks, not the lateral horizontal thrust of the gangue on the prevention structure. Therefore, in order to adapt to the deformation of the deep surrounding rocks, the gangue prevention device needs to have the capability of some vertical displacement and pressure relief to maintain good lateral pressure resistance and stability of the gangue rib.

Based on the deformation characteristics of the surrounding rocks, a contractible gangue prevention structure with the ability of vertical displacement and pressure relief is proposed. As shown in Fig. 10b, the structure is mainly composed of two sections of U-shaped steel units

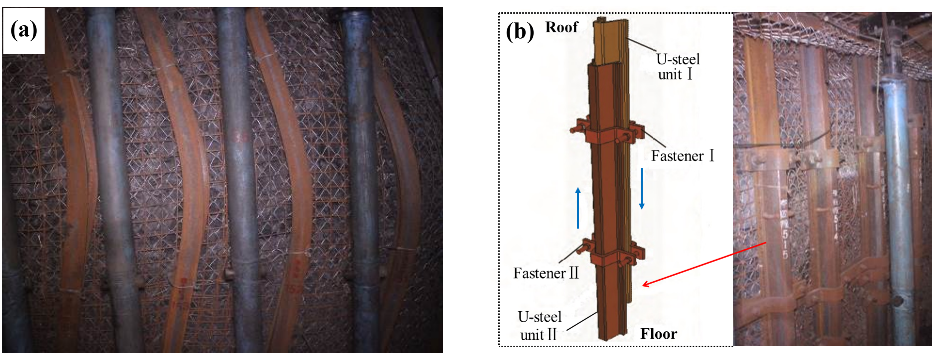


Fig. 10. entry-side support structure for GERRC in deep mine: (a) bending failure of rigid gangue prevention device in deep mine; (b) contractible gangue prevention structure for GERRC in deep mine

connected by fasteners. With the roof weighting of the entry, the vertical stress that the U-steel structure bears gradually increases. When the vertical stress reaches its sliding friction, relative sliding occurs between the two sections of the U-steel structure, which results in the release of the pressure of the U-steel structure and the vertical compression bending damage of the U-steel structure is avoided. The U-steel structure can adequately control the deformation of the gangue rib and allows for coordinated deformation with the roof and floor.

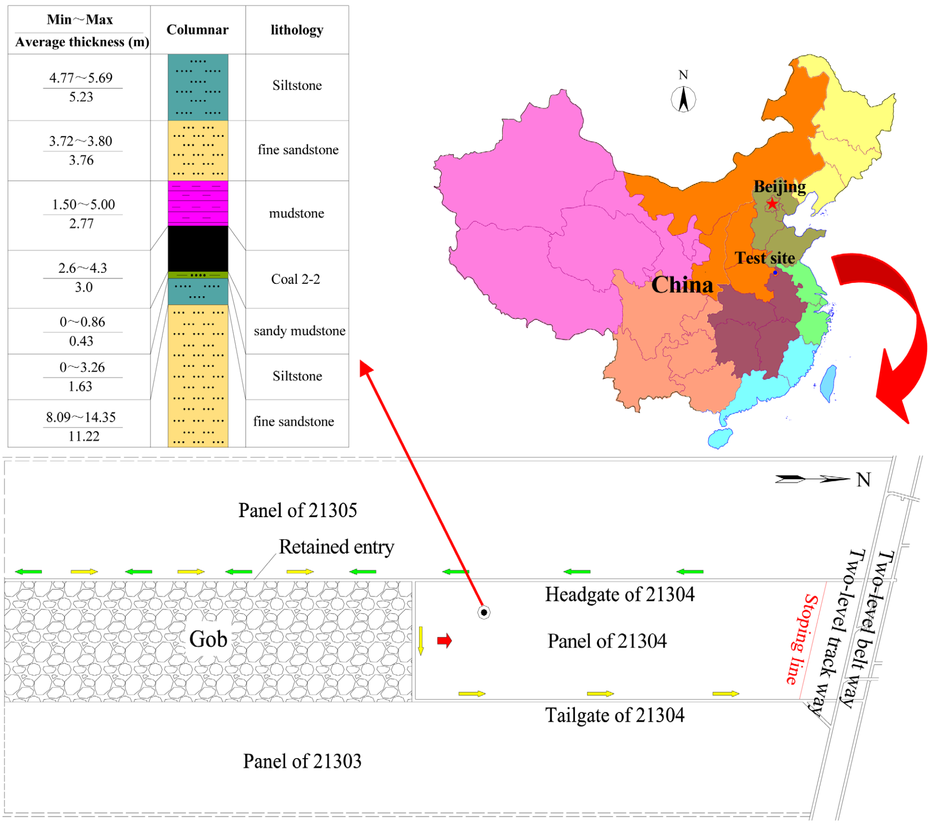


Fig. 11. The plane layout of the test ite and the lithology of the strata near the coal

6. Field Test

6.1. Geological Details

The Chengjiao Coal Mine is located in the east of the city of Yongcheng, Henan Province, China. The 21304 working face is used here as the testing working face with an average mining depth of 875 m. The coal seam is 2-2 coal with an average thickness of 3.0 m and an average inclination of 4°. The length of the gateroad is 1460 m, and the length of the working face is

180m. The immediate roof of the coal seam is mudstone with an average thickness of 2.77 m. The main roof consists of fine sandstone with an average thickness of 3.76 m and siltstone with an average thickness of 5.23 m. The immediate floor is siltstone with an average thickness of 0.43 m. The main floor is composed of siltstone with an average thickness of 1.63 m and fine sandstone with an average thickness of 11.22 m. The test site, the lithology of the roof & floor, and the layout of the roadways are shown in Fig. 11.

The 21304 working face is the first mining face in the 13th mining area. The 21305 working face is in the west, and roadway driving is performed along this working face. The 21303 working face is in the east. The protection coal pillar for the F20 fault is in the south, and the two-level trackway and the two-level belt-way are in the north. The headgate of the 21304 working face is the retained entry formed by roof cutting. Its design section is rectangular, with a clear height of 2.8 m and a clear width of 4.2 m. As the open-off cut of the 21305 working face is not formed, only a ventilator is used as the ventilation method in the retained entry.

6.2. Key parameters design

Based on the deformation characteristics of the deep surrounding rocks, the final design parameters are shown in Figure 14, which are described as follows:

(1) CRLDAC
Based upon the original support of the roadway (see Fig. 12), the roof was reinforced with CRLDAC, which had a diameter of 21.8 mm and a length of 10,000 mm. Two rows were arranged with a spacing of 400 mm between the rows. One had a spacing of 700 mm, and the other had a spacing of 1400 mm. The constant resistance and maximum constant resistance deformation were 350 kN and 500 mm, respectively. The adjacent anchor cables were connected by a W-shaped steel strip.

(2) Anchor cable for supporting coal wall
High-strength bolts with a diameter of 20 mm and a length of 2500 mm were used for supporting the coal wall, with a spacing of 800mm and an array pitch of 700 mm. The second or third bolt of each row was replaced by an anchor cable with a diameter of 18.9 mm and a length of 4800 mm. The anchor cables were arranged in a "W" manner, and the pretension force was not less than 120 kN.

(3) Energy-gathering blasting
The determined height and angle of the roof cutting were 8 m and 15°, respectively. The blasting holes were 200 mm away from the coal wall on the mining side, with a diameter of 48 mm and a spacing of 500 mm. The inner diameter of the energy gathering tube was 36.5 mm, the outer diameter was 42 mm, and a length of 1500 mm. The explosive used a secondary water gel explosive with a diameter of 32 mm and a 200 mm length. The blasting method adopts continuous blasting with joint holes, forward charging, and an uncoupling coefficient of 1.5, with four energy gathering tubes installed in a single hole. A charging structure of "4 + 3 + 2 + 0" pattern was used with stemming length of 2000 mm.

(4) Temporary entry-in support
Hydraulic supports with a rated working resistance of 3200 kN were used for temporary entry-in support, and single hydraulic props were used as auxiliary support. The support length was 250 m. After the retained entry is stabilised, the temporary entry-in support can be removed gradually.

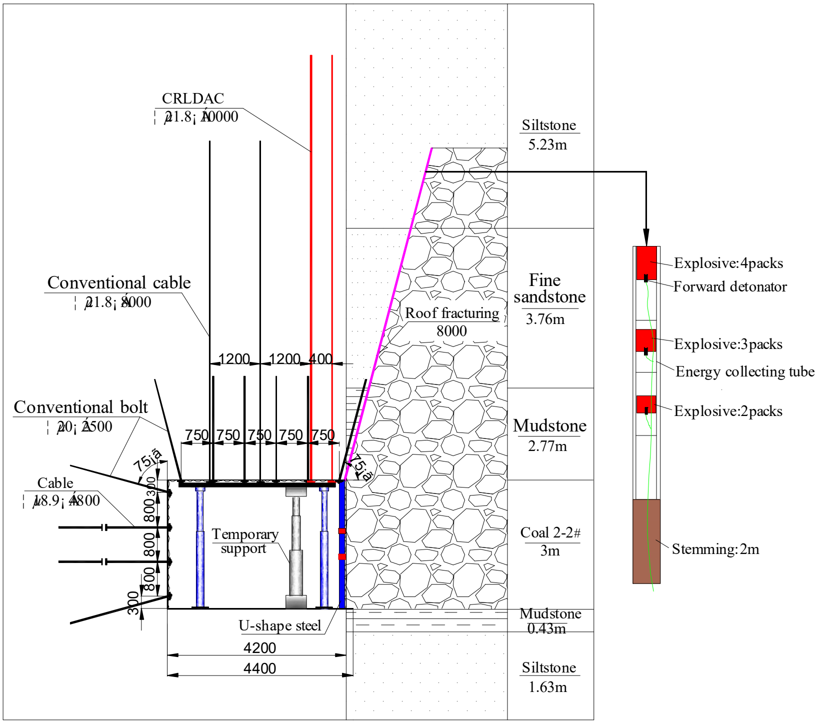


Fig. 12. Design parameters of GERRC in deep mine (mm)

(5) Entry-side gangue prevention support
The contractible 29 U-shaped steel with a spacing of 500 mm was adopted, and the bottom end was inserted into the floor of a depth of 200 mm and affixed within the hole. The inside of the U-shaped steel was arranged with a steel and diamond mesh to prevent the gangue in the goaf from entering the entry.

6.3. The influence of blasting cracks on the mine pressure on the working face

Overall, the roof structure of the roadway changed because of the advanced energy-gathering blasting, and the stress transmission path between the entry roof and the goaf roof was severed. This changes the mining pressure and stress distribution of the working face. Understanding the influence of the blasting cracks on the mining pressure on the working face can provide an important scientific basis for the optimisation of roadway support and safety in production.

6.3.1. Measuring points arrangement

The inclined length of the 21304 working face in the Chengjiao Coal Mine is 180 m, with 121 shield-type hydraulic supports (Fig. 13). A pressure monitoring point is set for every six hydraulic supports, and a total of 21 pressure monitoring stations are arranged evenly. The pressure data can be transmitted wirelessly between the monitoring stations. The data is transmitted to the head-master station and then to the ground machine room for data processing and pressure analysis.

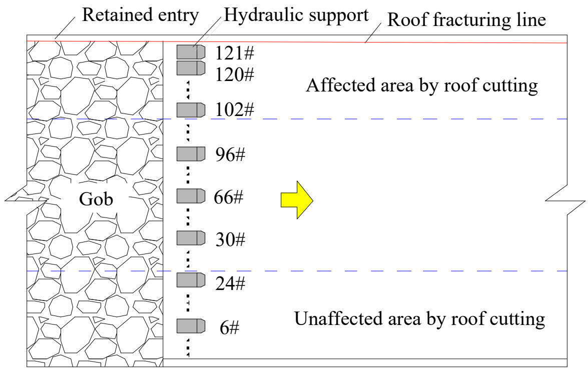


Fig. 13. Monitoring points of supports pressure in the working face

6.3.2. Influence of the blasting cracks on pressure

In order to study the influence of the blasting cracks on the mining pressure of the working face, two measurement points were selected for pressure analysis on the side of the headgate (cutting roof side) and tailgate (no cutting roof side), respectively. These points were selected while considering the symmetry of the surrounding rock structure movement. On the side of the headgate, 120 # (1.5 m away from the roof fracturing line) and 102 # (30 m away from the roof fracturing line) hydraulic supports were used as pressure analysis points. On the side of the tailgate, 6 # (4.5 m away from the coal side of the tailgate) and 24 # (33 m away from the coal side of the tailgate) hydraulic supports were used as pressure analysis points. The changes in loads of the hydraulic supports with the advancement of the working face are shown in Fig. 14, and the pressure statistics of the hydraulic supports in the working face are shown in Table 2.

TABLE 2

Pressure statistics of hydraulic supports in the working face

Position	Number	Pressure of the supports/MPa	
		Maximum load	Average load
Affected area by roof cutting	$120^{\#}$	30.4	10.8
	$102^{\#}$	42.4	19.1
Unaffected area by roof cutting	$6^{\#}$	44.3	18.5
	$24^{\#}$	48.3	25.7

By analysing the pressure data of the supports, the following rules were found: within the influence range, pre-splitting roof cracks effectively reduced the periodic weighting loads and average weighting loads of the hydraulic supports. Using the 120 # and 6 # hydraulic supports as examples, the maximum periodic weighting load of the support on the roof cutting side (120 #) was 30.4 MPa, and the maximum periodic weighting load of the support on the side without roof cutting (6 #) was 44.3 MPa. The maximum periodic weighting load of the support on the roof cutting side was reduced by 13.9 MPa and 31.4% compared to the side without roof cutting, and the average weighting load was reduced from 18.5 MPa to 10.8 MPa, a total reduction of 41.6%. The monitoring data indicated that the weighting loads were only affected by blasting cracks within a limited area. As the distance from the roof fracturing line gradually increased, the influence degree gradually decreased. From the monitoring data, the influence range of the roof cracks on the pressure of the supports in the working face was about 30 m.

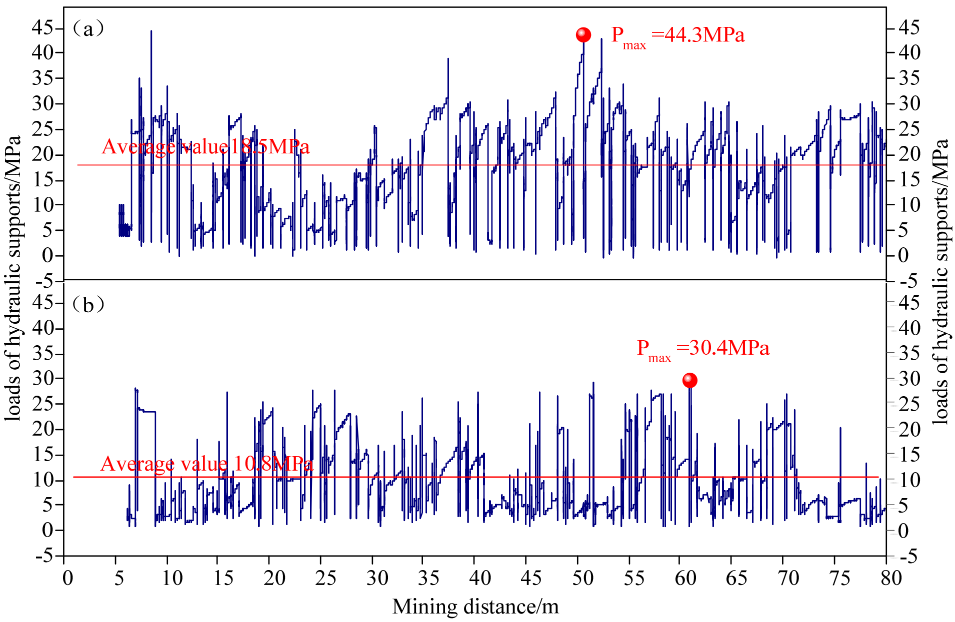


Fig. 14. Pressure curves of hydraulic supports on two sides of the working face:
(a) pressure curve of 6 # hydraulic support; (b) pressure curve of 120 # hydraulic support

6.3.3. Influence mechanism of blasting cracks on the mine pressure of the working face

By analysing the distribution law of the pressure on the supports in the working face, the roof blasting cracks reduced the average pressure and periodic pressure of the supports in the affected area. The influence mechanism is analysed as follows.

(1) Normal mining phase

During the normal mining phase, the rock blocks of the main roof are hinged to form a stable masonry beam structure, and the supports only need to bear the weight of the cantilever beam of the immediate roof, as shown in Fig. 15.

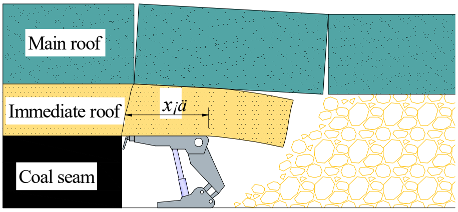


Fig. 15. Roof structure of the working face

For the cantilever beam structure of the immediate roof:

$$L_2 = \Sigma h \sqrt{R_T / 3q_0} \tag{10}$$

$$G = \gamma' L_2 b' \Sigma h \tag{11}$$

$$x' = L_2 / 2 \tag{12}$$

$$F_1 = kGx / l_r = k\gamma' L_2^{\,2} b' \Sigma h / 2l_r \tag{13}$$

where L_2 is the length of the cantilever beam of the immediate roof, $\sum h$ is the thickness of the immediate roof, R_T is the tensile strength of the immediate roof rock, q_0 is the load on the immediate roof cantilever beam, G is the self-weight of the immediate roof cantilever beam, γ' is the bulk density of the immediate roof strata, b' is the width of the supports, x' is the horizontal distance from the coal rib to the gravity centre of the immediate roof cantilever beam, k is the design coefficient considering the advancement of the adjacent supports (taken as 1.10 ~ 1.25), and l_r is the distance from the coal rib to the action centre line of the support columns.

(2) Periodic weighting phase

With the continuous advancement of the working face, after the length of the main roof cantilever reaches the ultimate fracture length, a break will also occur, i.e., it comes to the periodic weighting phase. At this time, a stable main roof masonry structure has not yet been formed, and the supports are mainly in place to control the stability of the main roof masonry beam. In addition, to support the immediate roof cantilever beam, the supports also need to provide some support for the main roof masonry beam structure to ensure the stability of the overlying strata in the stope. Therefore, the working resistance of the supports is required to effectively control the roof. The supports are divided into two sections: one is used to support the immediate roof cantilever beam, and the other prevents the instability of the main roof masonry beam structure.

The force required to prevent the main roof masonry beam structure from sliding instability is:

$$F_2 = \left[2 - \frac{L_1 \tan(\varphi' - \theta')}{2(h - \Delta S)} \right] Qb' \tag{14}$$

where F_2 is the force required to prevent the instability of the main roof masonry beam structure, θ' is the breaking angle of the main roof rock block, and Q is the self-weight and load of the main roof.

Therefore, the working resistance F of the supports in the periodic weighting phase is:

$$F = F_1 + F_2 = \frac{k\gamma' L_2^{\,2} b' \Sigma h}{2l_r} + \left[2 - \frac{L_1 \tan(\varphi' - \theta')}{2(h - \Delta S)}\right] Qb' \tag{15}$$

The energy-gathering blasting weakens the mechanical properties of the immediate roof, which causes the length of the immediate roof cantilever beam to shorten. Moreover, the energy-gathering blasting cracks change the roof from fixed support to a simple support structure, which reduces the length of the immediate roof cantilever beam on the roof cutting side. From equation (13), the reduced immediate roof cantilever beam decreases the support pressure F1 in the normal mining phase.

The periodic weighting distance of the main roof L_1 is:

$$L_1 = h\sqrt{R_T' / 3q} \tag{16}$$

The self-weight and load of the main roof Q are:

$$Q = qL_1 = qh\sqrt{R_T' / 3q} = h\sqrt{qR_T' / 3} \tag{17}$$

where R_T' is the tensile strength of the main roof rock mass.

After the roof is severed, the strata collapse more quickly and to a greater extent within the range of the roof cutting. The crushed and expanded gangue can quickly support the main roof. Assuming that the supporting force of the crushed gangue to the main roof is q', and the effective load acting on the main roof after roof cutting is reduced to $q - q'$, then the periodic weighting distance is increased to:

$$L_1' = h\sqrt{R_T' / 3(q - q')} \tag{18}$$

Similarly, the main roof self-weight and load are reduced to:

$$Q' = h\sqrt{(q - q')R_T' / 3} \tag{19}$$

From equation (14), when the periodic weighting distance increases or the main roof self-weight and load are reduced, the force F2 required to prevent the instability of the main roof masonry beam structure decreases. Therefore, both F1 and F2 decrease after roof cutting, so the pressure on the supports F also decreases during the periodic weighting phase.

In summary, it can be seen that after roof cutting, the pressure on the supports affected by the blasting cracks is reduced during the normal mining and periodic weighting phases, but the influence of the blasting crack on the mining pressure on the working face has a limited area, and the influence gradually decreases further from the blasting fracturing line.

6.4. Application effect

The field engineering tests were performed in the headgate of the 21304 working face of the Chengjiao Coal Mine. In the process of mining and retaining the entry, the method of the crisscross point arrangement was used to monitor the surface displacement of the entry. The deformation curve of the surrounding rocks at a monitoring point is shown in Fig. 16.

The surrounding-rock deformation can be divided into four stages: (I) advanced influence zone, (II) severe deformation zone, (III) slow deformation zone, and (IV) stable zone. In front of the working face, the surface displacement of the roadway was small, and the deformation of the surrounding rocks was not significant, which lays a foundation for retaining the entry in a later stage. At about 30 m in front of the working face, the surrounding rocks of the roadway began to deform, and the deformation rate of the surface displacement was low. The maximum rate of the roof subsidence was 4 mm per day, the maximum rate of the floor heave was 7.6 mm per day, and the maximum rate of the rib-to-rib contraction was 5.6 mm per day. In the advanced influence stage, the roof subsidence, floor heave and rib-to-rib contractions were 31 mm, 71 mm and 42 mm, respectively. Overall, the strengthening support measures enacted before mining effectively controlled the surrounding-rock deformation. The surrounding-rock deformation of the retained entry is the most severely from 0 to 96 m behind the working face. During this period, the maximum rates of the roof-to-floor convergence and rib-to-rib contractions were 32 mm per day and 14.7 mm per day, respectively. The deformation of the surrounding rocks in this stage accounts for about 70% of the total deformation. The deformation rate of the surrounding rocks decreased gradually from 96 ~ 291 m behind the working face, indicating that the deformation of the surrounding rocks of GERRC in deep mines is long-term and continuous. Beyond 291 m behind the working face, the surrounding rock was stable. In the end, the roof-to-floor convergence was 672 mm, in which the roof subsidence was 281 mm, and the floor heave was 389 mm. The rib-to-rib contraction was 343 mm, in which the deformation of the coal side was 186 mm, and the deformation of the gangue rib was 157 mm. All deformation was within the allowable range.

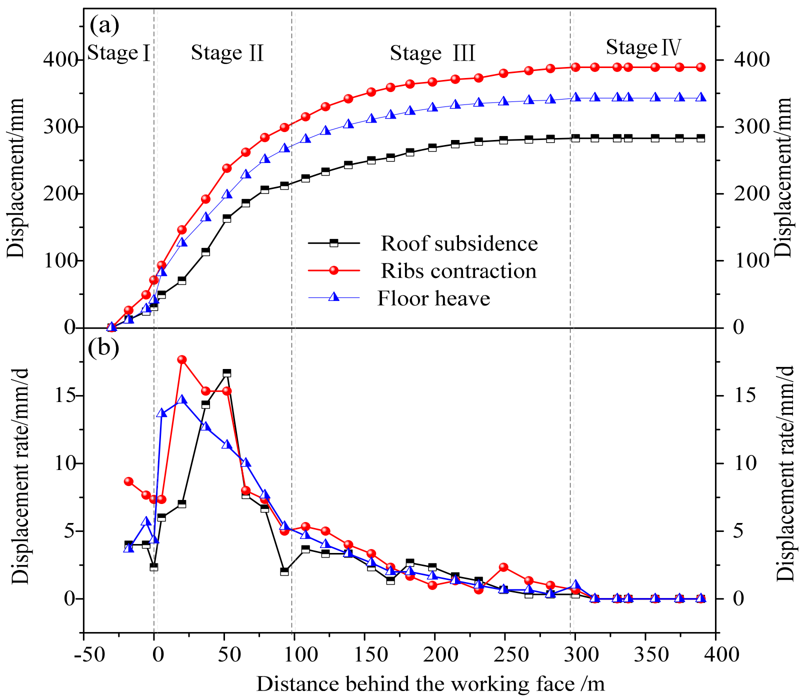


Fig. 16. Curves of surrounding rock deformation: (a) curve of displacement; (b) curve of displacement rate

The effect of the field application is shown in Fig. 17. The roof was reinforced by the CRLDAC in advance of the working face (Fig. 17a), which improved the integrity and optimised the mechanical properties of the rock mass in the anchorage area of the roof. According to the designed blasting parameters, the roof was subjected to energy-gathering blasting. The field observation results showed that the crack rate of the blasting holes reached 90.3%, and the crack's development from the blasting hole was shown in Figure 19a. This successfully severed the mechanical relationship between the roof of the working surface and the roof of the retained entry. After the coal seam was mined, the temporary entry-in support (Fig. 17b) and the gangue prevention support were used in the entry in time. The roof of the goaf quickly fell along the blasting structure plane to form the entry side (Fig. 17d). After the gangue in the goaf completely collapsed and was compacted, the hydraulic supports and several single hydraulic props in the entry were gradually removed, effectively maintaining the entry (Fig. 17c). This retained entry was also used as the return air gateroad of the next working face.

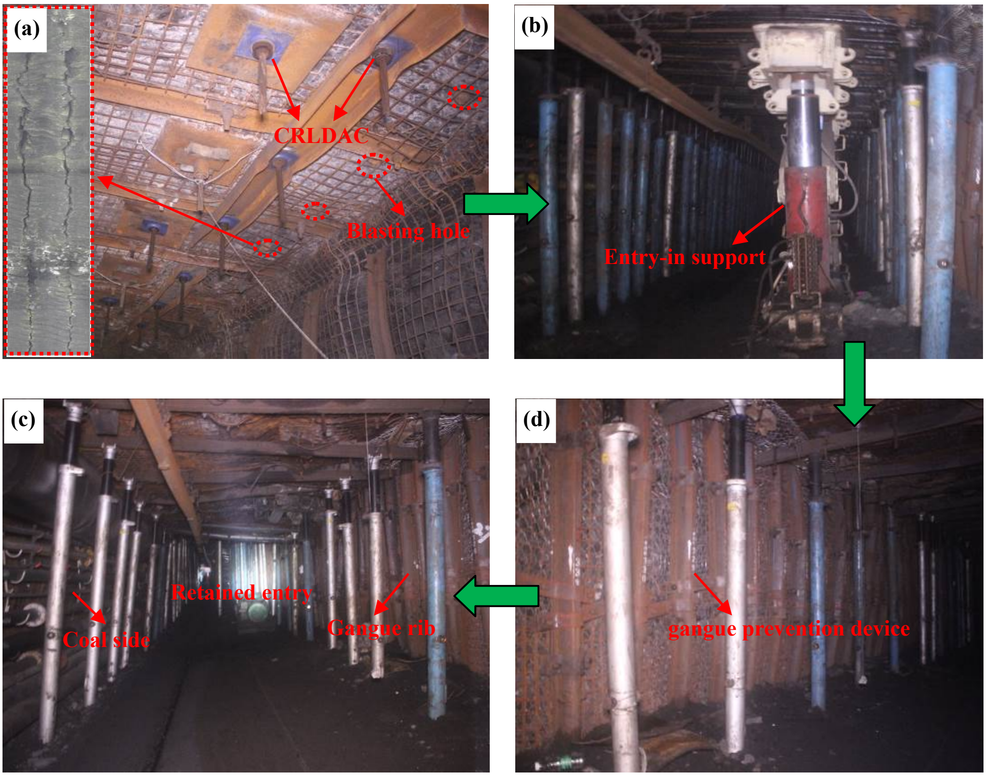


Fig.17. Field effect of deep GERRC: (a) Roof support with CRLDAC and energy-gathering blasting; (b) Temporary entry-in support; (c) effect of deep GERRC; (d) gangue prevention support to form gangue rib

The headgate with a total length of 1200 m was successfully reserved in the 21304 working face, and the roadway section was reduced by about 10%, which can meet the requirements for use. The project has achieved economic benefits of 21.66 million RMB, and a new pattern of

GER was explored throughout to prove its feasibility in deep mines. This study also provides a reference for similar deep mines. However, further exploration and verification are needed for GERRC in deep mines with different geological conditions.

7. Conclusion

This paper introduces an innovative method of GER, which was first applied to a deep mine in China. The research results show that the advanced blasting cracks in the roof weaken the stress transfer from the goaf roof to the entry roof, and the broken and expanded gangue formed by the collapse of the strata within the roof cutting can better fill the mining space. This avoids the rapid pressure increase caused by the roof breaking impact and slows down the movement of the overlying strata. Deformation of the deep surrounding rocks is transformed from “abrupt” to “slow” deformation and is overall significantly reduced. Therefore, this method of GERRC has good adaptability to a deep mechanical environment.

According to the deep geological environment and the deformation characteristics of the rock mass, the collaborative control system of the surrounding rocks of GERRC in deep mines allows for a constant resistance for pressure to control the roof, flexible retraction to resist lateral pressure, high-resistance temporary support against dynamic pressure, and increased support length to protect coal sidewall. The measures were applied in the field, and the monitoring results show that the blasting cracks reduced the average pressure and periodic pressure of the supports within a limited range, which diminishes with increased distance. This is mainly due to the reduction of the length of the immediate roof cantilever beam and the effective load of the main roof. Approximately 290 m behind the working face, the retained entry is stable. The collaborative control system can effectively control the deformation of the surrounding rocks of GERRC in deep mines. After the entry was stabilised, all indicators met the requirements for use. This study provides guidance and reference for the non-pillar mining of deep mines in China.

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