



OPEN The entry retained along gob side with small coal pillar and its surrounding rock control: a case study

Lei Li¹, Yuzhong Yang¹✉, LiYun Wu¹, Weiyu Zhang², Wanli Yang¹ & Yaowei Zhai¹

In order to effectively control the large deformation of coal pillar and roadway during entry retained along gob side with small coal pillar, taking 1311 and 1312 working faces of a coal mine in Shanxi Province as engineering background, the structural characteristics of overlying rock are studied, and the stress source of entry retained along gob side with small coal pillar and the principle of roof cutting and pressure relief are analyzed, then the roof cutting and pressure relief technology of shaped charge blasting with combined and grouped of deep and shallow holes is put forward. According to the geological conditions of 1311 working face, through the calculation of caving zone and crack zone height of the working face, the analysis of roof breaking and instability conditions and the calculation of crack zone blasting radius, the key parameters of roof cutting by deep and shallow hole combined blasting are determined. Through the numerical simulation and analysis of roof cutting and pressure relief in different key strata, the variation law of vertical stress acting on small coal pillar is obtained, as follows: no roof cutting > only cut layer 2 > only cut layer 1 > cut layer 3 + partial layer 2 > cut layer 1 ~ 3; Based on these, a cooperative control scheme of "roof cutting and pressure relief by shaped charge blasting with combined and grouped of deep and shallow holes" + "anchor-cable combined support" + "coal pillar strengthened" is put forward. The field industrial test results show that the control effect of small coal pillar and roadway surrounding rock is good.

Keywords Entry retained along gob side with small coal pillar, Multi-key stratum, Stress source, Shaped charge blasting with combined and grouped of deep and shallow holes, Roof cutting and pressure relief

When the traditional method is used to arrange the section working face in coal mining, it is necessary to set up a large coal pillar between the roadways of the two working faces, so that the roadway in the lower section can avoid the peak area of abutment pressure¹. This method results in a waste of coal resources and is not conducive to improving the stress environment of the roadway. Therefore, many mines in China begin to adopt entry retained along gob or entry driven along gob with small coal pillar. For the entry retained along gob, it is necessary to construct filling bodies or other gangue blocking facilities on the gob side of the retained entry, which bring about some problems, such as the complex construction technology, the heavy auxiliary transport workload, and it is easy to affect the production as well; especially for the thick coal seam, there exists some other problems, such as surrounding rock control and gob isolation, which are not conducive to efficient and safe production²⁻⁵. On the other hand, for the entry driven along gob, while the working face in the upper section has been finished for more than 6 months, and the broken overlying rock in the gob has been stable, the entry along gob could be driven with 4 ~ 8 m coal pillar⁶; as a result, the interval between working face mining and roadway driving is too long, and it is easy to result in the imbalance between driving and mining. Based on these, the researchers put forward the technology of entry retained along gob side with small coal pillar, that is, when driving the roadway in the upper section, the roadway in the adjacent lower section will be excavated in advance, a small coal pillar setting up between the two roadways, then the small coal pillar and the roadway in the lower section will be retained when the upper section is mined, and used when the lower section is mined, as shown in Fig. 1. This technology solves the problems existing in entry retained along gob or entry driven along gob, but the small coal pillar and entry along gob are affected by two mining faces, the stability of surrounding rock is poor, and large deformation is easy to occur. If in the geological environment of large burial depth and high ground stress, it is easy to induce accidents, such as roof fall, coal and gas outburst or rock burst⁷⁻⁹. Therefore,

¹School of Energy Science and Engineering, Henan Polytechnic University, Jiaozuo, Henan 454000, China.

²Zhaozhuang Mining Limited Liability Company, Changzhi 046605, China. ✉email: jityyz@hpu.edu.cn

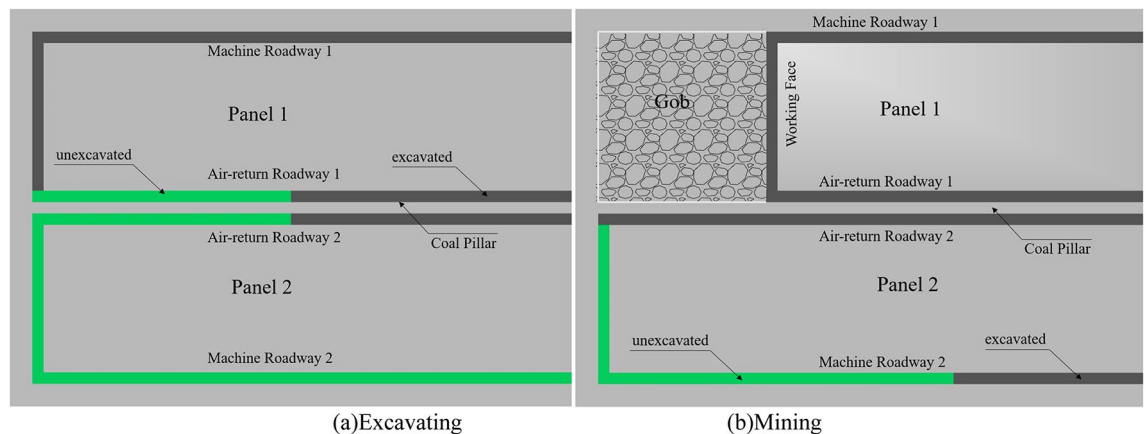


Fig. 1. Common layout of the narrow coal pillar retaining entry.

mine pressure and surrounding rock control is the biggest problem that entry retained along gob side with small coal pillar exists, and many scholars have done a lot of theoretical and technical research on this problem.

Song believes that the abutment pressure can be divided into “internal stress field” and “external stress field”. The internal stress field belongs to the stress reduction area, and the arrangement of small coal pillars and roadways in this area is beneficial to the stability of roadway surrounding rock¹⁰. Hou and Li put forward the principle of large and small structure of surrounding rock of roadway driving along gob, and considered that the arc triangular plate formed during basic roof collapse is the main factor affecting the stability of roadway along gob¹¹. By constructing a boundary mechanical model of double plasticized basic roof structure with elastoplastic deformation of solid coal and weakening of the width and supporting capacity of both sides of the coal pillars, Chen studied the fracture position of the basic roof structure under the engineering condition of both sides of the coal pillars¹². By establishing the mechanical model of the triangular block structure of the roof of the roadway along the gob, Bai determines the stability coefficient of the triangular block structure of the working face, and points out that the stability of the triangular block structure is an important factor affecting the stability of roadway driving along the gob¹³. With the application of damage theory, Wang studies the distribution law of lateral abutment pressure when driving roadway along gob, and puts forward that one of the main factors affecting the stability of roadway along gob is the high lateral abutment pressure in gob¹⁴. By studying the change law of surrounding rock of gob side roadway, Han analyzes the change of coal stress in gob side roadway working face, and thinks that the premise of gob side roadway stability is the existence of a certain range of elastic core in small coal pillar¹⁵. Zhang finds that the fracture, rotation and slip of the basic roof are the root causes of the surrounding rock deformation of the gob side roadway under the unstable strata¹⁶.

Zhang¹⁷ and Huo¹⁸ by studying the distribution and superposition of the roof breaking structure and stress field of multi-layer hard roof, the reasonable coal pillar size for goaf excavation is determined. The pressure behavior of small coal pillar and roadway gob side is affected by roof movement and fracture, in which the fracture location of roof plays a key role. Wang studies the three fracture forms of overlying rock of gob side roadway, and pointed out that when the overburden fault occurs on the outside of gob side roadway, it is most beneficial to the control of gob side roadway¹⁹. By establishing the mechanical model of overlying rock structure of gob side roadway in fully mechanized top-coal caving face, Guo deduces the expression of basic roof breaking position, determines the form of overlying rock breaking structure, and puts forward the surrounding rock control countermeasures of effective support in advance for roof and side near the coal pillar²⁰.

In order to avoid the problems of abnormal mine pressure and surrounding rock control due to the unfavorable fracture position of roof, the researchers put forward the method of artificially controlling the fracture location of roof to actively reduce the surrounding rock pressure of roadway along gob, and put forward a variety of roof cutting and pressure relief methods, such as directional energy concentrated blasting roof cutting technology, directional hydraulic fracturing technology and liquid CO₂ fracturing technology, and achieved good results^{21–23}. According to the theory of energy accumulation in the surrounding rock, Wang proposed the control method of the surrounding rock by roof cutting and absorbing energy, which cut off the transfer of roof stress and released the accumulated energy in the rock, and achieved good results²⁴. Based on the structural analysis of the curved triangular plate of the roof of small coal pillar roadway, Bi determines the fracture location of the roof, and puts forward the method of cutting and relieving pressure by deep-hole blasting, which avoids the influence of high stress on coal pillar and roadway²⁵. Bi puts forward the technology of driving roadway along gob by pre-splitting and pressure relief, improving the mechanical environment of roadway driving along gob and ensuring the overall stability of roadway and coal pillar structure²⁶. Su studies the stress distribution state and roof fracture characteristics of small coal pillar area by means of numerical simulation, theoretical analysis and drilling peep, and points out that the roof pressure relief mechanism of roof cutting by deep-hole blasting effectively controlled the roadway deformation²⁷. According to the characteristics of mine pressure of the coal pillar in Yongdingzhuang Coal Mine, Zhai puts forward the method of roof cutting and pressure relief by hydraulic fracturing. The hydraulic fracturing scheme is determined by analyzing the development characteristics of cracks during hydraulic fracturing, and the stability of roadway is realized²⁸. Sun²⁹ and Wang³⁰

analyze the influence of roof and surrounding rock deformation on roof and surrounding rock by constructing mechanical structure model of roof without cutting and unloading.

The above studies mainly focus on the mechanical mechanism of entry driving along gob, influence factors of surrounding rock deformation and stress control technology, but there are few studies on entry retained along gob side with small coal pillar and combined roof cutting and pressure relief under multi-key stratum of the overlying strata. This study takes entry retained along gob side with the small coal pillar in a mine as the research background, analyzes the roof overburden structure of the entry retained along gob side with the small coal pillar, and determines its stress source. By combining the movement law of roof overburden of entry retained along gob side with the small coal pillar with the theory of key stratum, the key stratum affecting the stability of surrounding rock of the small coal pillar and roadway are determined, and the advanced roof cutting and pressure relief technology of shaped charge blasting with combined and grouped of deep and shallow holes is put forward. The main parameters of roof cutting are determined by theoretical calculation and numerical simulation and applied in the field.

Model and theory analysis

Stress source of entry retained along gob side with small coal pillar

During the excavation of entry retained along gob side with the small coal pillar, the stress of roadway 1, roadway 2 and small coal pillar are mainly the self-weight stress of coal and rock and the mining stress caused by two roadways, as shown in Fig. 1a. During the mining period of working face 1, the stress of roadway 2 and small coal pillar is the self-weight stress of coal and rock and the more complex dynamic load stress caused by mining work face.

According to the theory of “voussoir beam”, after mining, overlying rock above the gob bends, breaks and collapses, forming caving zone, crack zone and bending subsidence zone from bottom to bottom. The strata in the caving zone appear irregular caving and disordered piling up. After the rock strata in the crack zone are broken, the rock mass is still neatly arranged^{1,31}, as shown in Fig. 2. When the rock strata in the caving zone collapse directly, the compression of the small coal pillar and roadway 2 presents the characteristics of immediately loading, that is, after the caving zone appears, the stress generated by the surrounding rock structure formed at the end of the small coal pillar will immediately act on the coal pillar and surrounding rock of roadway 2. There is a certain latency in the fracture of the rock strata in the crack zone, so the stress acting on the small coal pillar and the surrounding rock of roadway 2 in this zone shows the characteristics of delayed loading, and the stress increases gradually with time. However, due to the gangue piling up in the caving zone, the broken rock in the crack zone will be supported to a certain extent after contacting the gangue, and the stress increases and slows down gradually.

According to the above analysis, from the driving to the mining stage, the stress source of the small coal pillar and roadway 2 is the combination (the “Triangular combination” in Fig. 2) formed on the side of the coal pillar after the fracture of the multiple hard rock strata (key strata) which act as the “skeleton” of the rock mass structure in the caving zone and crack zone. By the key strata that form the hinged structure after rock strata breaking and the “cantilever beam” on the side of the coal body, the combination applies the weight and the rotary extrusion stress of the rock strata to the small coal pillar and the surrounding rock of roadway 2, therefore, in addition to the cantilever beam on the side of the coal body, the key strata in the caving zone and crack zone are also the main research objects of mine pressure control during the entry retained along gob side with the small coal pillar.

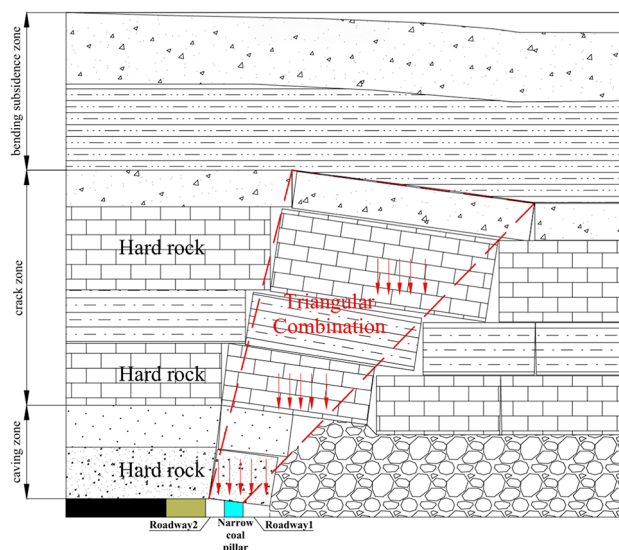


Fig. 2. Stress source model diagram of entry retained along gob side with small coal pillar in mining stage.

The theory of roof cutting and pressure relief

During the mining period of working face 1, with the increase of mining space, the exposed area of roof increases, and the “O-X” type fracture occurs on the hard roof above the working face, and the hinged structure of rock block A, rock block B and rock block C is formed at the end of the working face (shown in Fig. 3). Roadway 2 and small coal pillar are located below rock block A and rock block B, where rock block An is above solid coal and is complete, while rock block B is above the coal wall and one end is in the gob and sinks with mining. Therefore, although it forms a hinged structure with rock block An and rock block C, it has the greatest influence on the stability of coal pillar roadway, so rock block B is the key block affecting the surrounding rock^{1,32,33}.

On the other hand, if the roof cutting measures are taken in advance on the side of the coal pillar in roadway 1, it can ensure that the roof breaks along the roof cutting line after mining, and the key block B cannot be formed on the side of the coal pillar (shown in Fig. 4), which shortens the length of the force arm above roadway 2 and small coal pillar, actively changes the breaking structure of overlying rock, reduces the stress acting on the small coal pillar and roadway 2, and cuts off the mechanical connection of the roof between roadway and working face, so that the stress can not be transferred to the small coal pillar and surrounding rock of the roadway 2 through rock block B, which improves the stability of roadway 2 and small coal pillar.

According to the analysis of the collapse of the hard roof above the coal seam, when the roof is not cut and the first break has not occurred, it is a plate fixed on four sides; after the first break, when the periodic break occurs, it becomes a plate with three sides fixed and one side free (shown in Fig. 5a). Once the roof is cut in advance, it becomes a plate fixed on three sides and free on one side before the first break, and when periodic breaking occurs after the first break, it becomes a plate fixed on two sides and free on the other sides (shown in Fig. 5b). The fracture distance of the roof before and after roof cutting can be calculated according to the following equation³⁴.

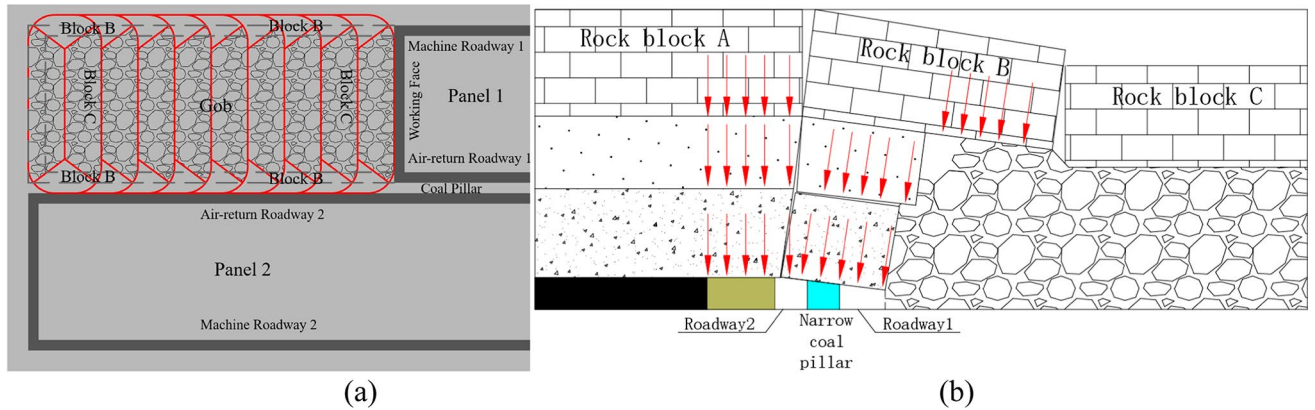


Fig. 3. Fracture morphology of overlying rock structure of entry retained with small coal pillar without roof cutting.

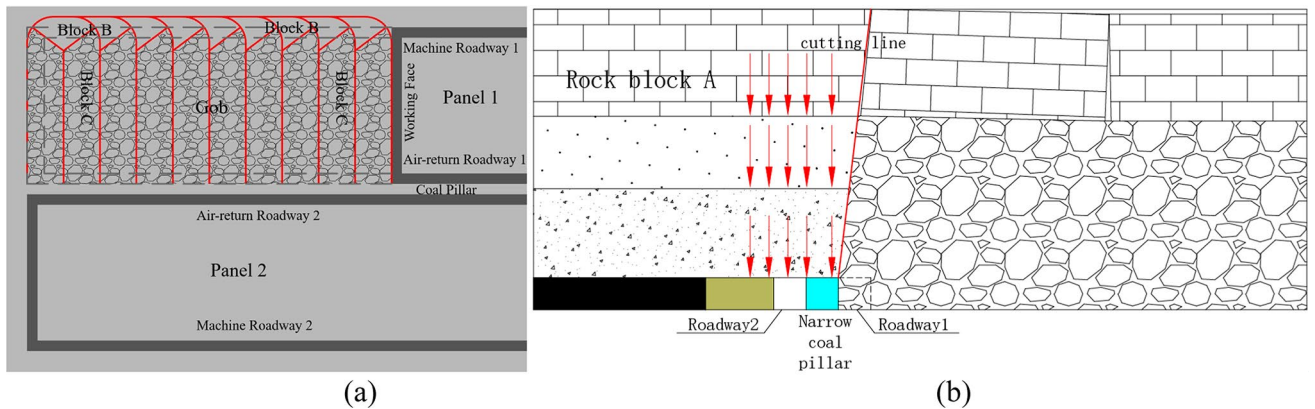


Fig. 4. Fracture morphology of overlying rock structure of entry retained with small coal pillar after roof cutting.

$$L_1 = \frac{2h_m}{1 - \nu^2} \sqrt{\frac{\sigma_t}{q} \cdot \frac{2 + \varepsilon^4}{4 + 3\nu\varepsilon^2}} \tag{1}$$

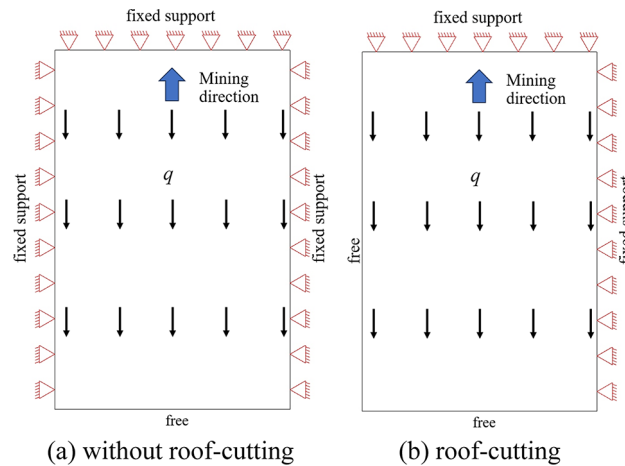


Fig. 5. The breaking models of hard roof.

$$L_2 = \frac{2h_m}{1-\nu^2} \sqrt{\frac{\sigma_t}{3q} \cdot \frac{1+\varepsilon^4}{4+\nu\varepsilon^2}} \quad (2)$$

where, h_m is the thickness of hard roof; q is the load of hard roof; σ_t is the tensile strength of hard roof; ν is Poisson's ratio; ε is the geometric shape coefficient of working face.

Subtract Eq. (2) from Eq. (1), we have Eq. (3)

$$L_1 - L_2 = \frac{2h_m}{1-\nu^2} \sqrt{\frac{\sigma_t}{q}} \left(\sqrt{\frac{2+\varepsilon^4}{4+3\nu\varepsilon^2}} - \sqrt{\frac{1+\varepsilon^4}{4+\nu\varepsilon^2}} \right) > 0 \quad (3)$$

It can be seen that the collapse distance of the hard roof decreases after roof cutting. Therefore, after taking the measure of roof cutting, the collapse distance of the roof decreases obviously, and the surrounding rock stress also decreases with the decrease of the collapse step distance, which plays the role of pressure relief.

Principle of shaped charge blasting with combined and grouped of deep and shallow holes

According to the above, during entry retained along gob side with the small coal pillar, the pressure source of the small coal pillar and roadway 2 is mainly the “triangular combination” composed of the key strata and its cushion in the caving zone and the crack zone. At present, most mines in China use the method of energy-concentrated blasting to deal with the rock strata in the zone. If there is only a single key stratum in the zone, only the key stratum needs to be cut off, but as usual, there exist multiple key strata (shown in Fig. 6a,b). If only a single length of hole is arranged for the high key stratum, the low key stratum has not to be cut off (shown in Fig. 6a,b) and it will still act on the small coal pillar and roadway 2 in the form of “cantilever beam” in the early stage of mining, and the collapse of the middle and high key strata is also hindered in the later stage. In order to completely cut the key strata in caving zone and crack zone which have influence on the entry retained along gob side with the small coal pillar, the technology of shaped charge blasting with combined and grouped of deep and shallow holes is put forward (shown in Fig. 6c,d).

Principle of shaped charge blasting with combined of deep and shallow holes

Shaped charge blasting is a blasting method in which explosive is detonated in a device with energy accumulation effect, and after the blasting energy is centrally released along the set direction, a directional crack is formed in the rock mass³⁵. In the deep hole shaped charge blasting, the blasting energy (BE1) is concentrated in the middle and deep part of the hole and released directionally by the device with energy accumulation effect, meanwhile the blasting gas produced by the blasting is wedge into the rock mass along the energy release direction, which makes the rock mass subjected to concentrated tension on both sides of the specified direction and crack along this direction, finally form a directional crack in the middle and deep part of the deep hole. In shallow hole blasting, the blasting energy (BE2) mainly forms the crack surface in the middle and deep part of the shallow hole (the sealing range of the deep hole). by the shaped charge blasting with combined of deep and shallow holes, the superposition of deep and shallow blasting energy can be realized, and finally the key strata in the caving zone and crack zone can be completely cut off from low to high (shown in Fig. 7).

Shaped charge blasting with grouped of deep and shallow holes

With the mining of the working face, the low key stratum in the caving zone breaks firstly and loads immediately, and the middle and high key strata in the crack zone break until the working face is mined to a certain length,

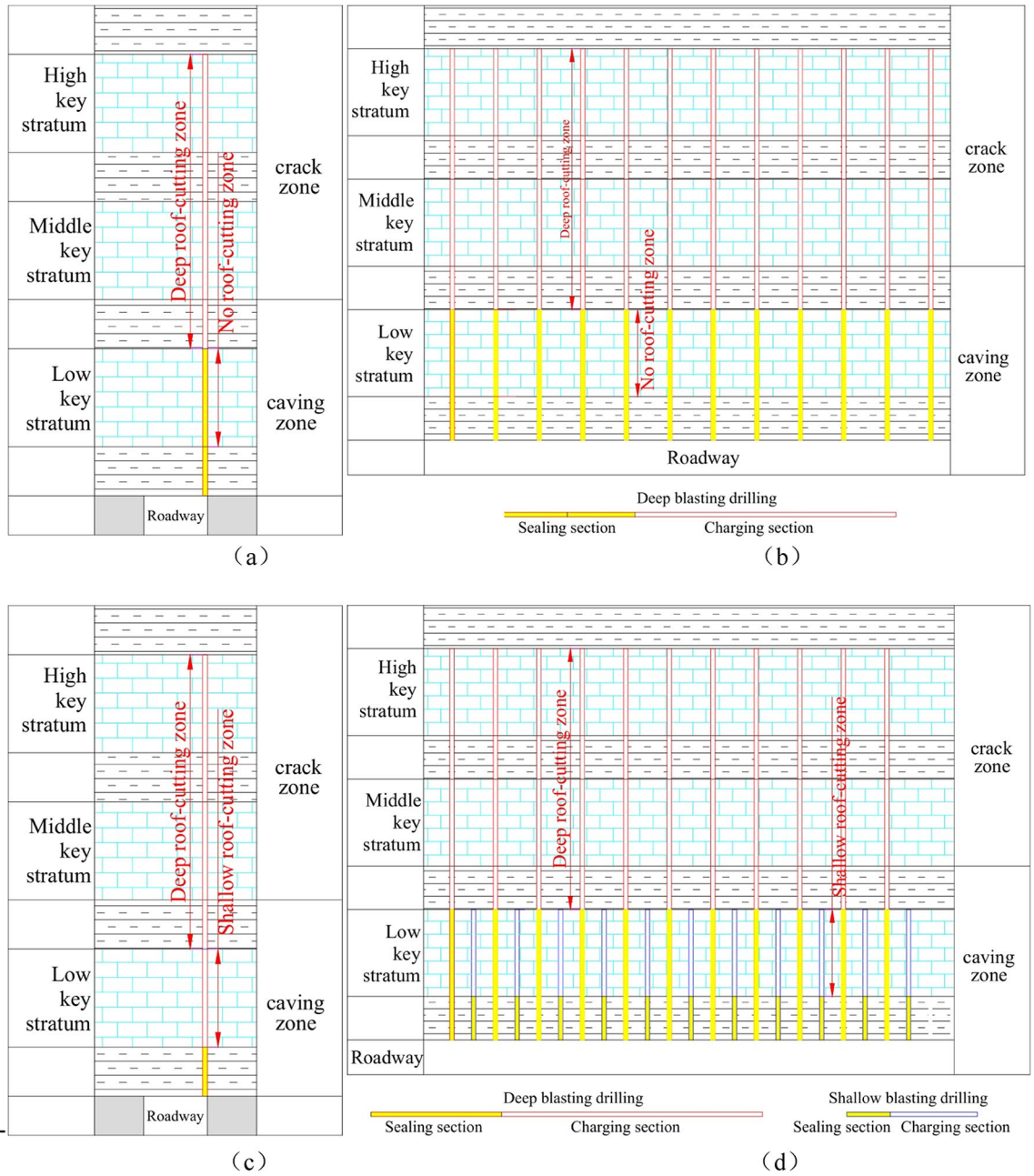


Fig. 6. Blasting roof-cutting diagram: (a, b) blasting roof-cutting diagram of single deep drilling; (c, d) blasting roof-cutting diagram of combination with deep and shallow drillings.

showing the characteristic of delayed load, so the low key stratum is the primary target layer of roof cutting, it is also the object of shallow hole blasting. In the staggered combination arrangement of deep and shallow holes, the sealing section of deep holes is used as the guide hole of shallow holes by blasting in different time periods, which increases the free surface of shallow holes and promotes the formation of directional cracks.

In shallow hole blasting, the stress wave produced by blasting propagates outward to the guide hole in the form of compression wave under the action of energy accumulation device, and forms tensile wave after reflection on the free surface of the guide hole; the stress concentration effect is formed in the surrounding rock around the guide hole after the interaction of compression wave and tensile wave.

According to the theory of elasticity mechanics, the peak stress state of concentrated stress near the guide hole can be expressed by the following equation³⁶.

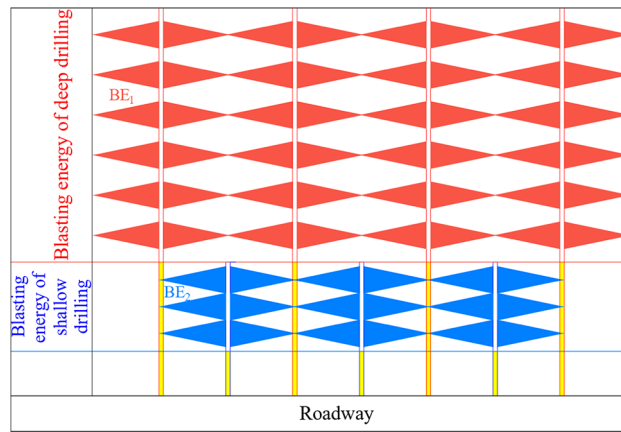


Fig. 7. Principle of combination blasting with deep and shallow drillings.

$$\begin{cases} \sigma_{rr} = \frac{1}{2} [(1 - k_1^2) (\sigma_\theta - \sigma_r) + (1 - 4k_1^2 + 3k_1^4) (\sigma_r + \sigma_\theta) \cos 2\theta] \\ \sigma_{\theta\theta} = \frac{1}{2} [(1 - k_1^2) (\sigma_\theta - \sigma_r) + (1 + 3k_1^4) (\sigma_r + \sigma_\theta) \cos 2\theta] \\ \tau_{r\theta} = \frac{1}{2} [(1 + 2k_1^2 - 3k_1^4) (\sigma_r + \sigma_\theta) \cos 2\theta] \\ k_1 = \frac{r_0}{r_B} \end{cases} \quad (4)$$

where σ_{rr} , $\sigma_{\theta\theta}$ are tangential stress and radial stress in guide hole rock under stress concentration; $\tau_{r\theta}$ is shear stress in guide hole rock under stress concentration; σ_r , σ_θ are radial stress and tangential stress in the rock; r_0 is the radius of the guide hole; r_B is the distance from any point in the rock to the center of the guide hole; θ is the angle between the line between the blasting hole and any direction.

In the Eq. (4), when $k_1 = 1$, $\sigma_{rr} = 0$, $\tau_{r\theta} = 0$

$$\sigma_{\theta\theta} = (\sigma_\theta - \sigma_r) + 2(\sigma_\theta + \sigma_r) \cos 2\theta \quad (5)$$

Derivation of θ on both sides of Eq. (5), and make $d\sigma_{\theta\theta}/d\theta = 0$;

When $\theta = 0$, $\pm\pi$, $\sigma_{\theta\theta} = 3\sigma_\theta + \sigma_r$, $\sigma_{\theta\theta}$ is the maximum value;

When $\theta = \pm\pi/2$, $\sigma_{\theta\theta} = -\sigma_\theta - 3\sigma_r$, $\sigma_{\theta\theta}$ is the minimum value.

It can be seen from the above that the maximum tensile stress during blasting occurs in the direction of the center line of the adjacent blasting hole; if the tensile stress is greater than the tensile strength of the surrounding rock of the drilling hole, the rock will fracture along the direction of the maximum tensile stress. In addition, with the decrease of the distance between the guide hole and the adjacent hole, the maximum tensile stress increases gradually. Therefore, in the group blasting of deep and shallow holes, when the sealing section of the deep hole is on the center line of the two shallow holes, the blasting energy can be used to realize the pre-splitting of the low key layer to the maximum extent. Therefore, in the group blasting of deep and shallow holes, when the deep hole sealing section is used as the guiding hole and on the connecting line of two shallow holes, the blasting energy can be used to cut the low key stratum to the maximum extent.

Theoretical calculation Field background

The test mine is located in Qinshui coalfield, Jincheng City, Shanxi Province, China, and the mine mainly mines No.3 coal seam. There are some problems in the mine, such as large mining depth, high ground stress, coal and gas outburst, strong mine pressure and so on, resulting in slow driving speed of the roadway. Therefore, in order to ensure the safe production and continuous mining, the test mine adopts the method of double roadway driving. According to the previous engineering experience of the mine, a 50 m coal pillar is left between the roadways, which results in a serious waste of coal resources; if only a small coal pillar is left, it is easy to cause the imbalance of mining replacement in the mine, and high ground stress is also easy to cause the overall instability of small coal pillars and the serious deformation of the roadway. Considering the above problems, it is planned to set up an 8 m small coal pillar between the 13,112 air-return roadway of the 1311 working face and the 13,122 air-return roadway of the 1312 working face. During the mining period of the 1311 working face, by roof cutting and pressure relief technology of shaped charge blasting with combined and grouped of deep and shallow holes and the corresponding surrounding rock control scheme to implement entry retained along gob side with the small coal pillar, to retain the 13,122 roadway.

The 1311 working face is located in the east wing of the No.1 mining district of the mine, the solid coal area in the east, the protective coal pillar in the south and southeast, and the main roadway in the west, the 13,122

air-return roadway (test roadway) of 1312 working face in the north. 1311 working face and 13,122 roadway are shown in Fig. 8. The relevant parameters of 1311 working face are as follows: the buried depth of the face is 623~681 m, with a strike length of 539 m, a dip length of 218 m; The thickness of the mining coal seam is 4.2~5.3 m, with an average of 4.5 m; the dip angle is 0°~5°, with an average of 3°; the hardness coefficient of the coal is 0.6~0.8; and the designed length of 13,122 air-return roadway is 600 m, of which the length of test section is 590. The roof and floor of No.3 coal seam are mostly fine sandstone, siltstone, sandy mudstone and mudstone, and some are medium sandstone. According to the test results of rock physical and mechanical parameters, the overlying rock in 1311 working face is characterized by more hard rock and multi-strata composite overburden, and the lithology of coal seam roof and floor is shown in Fig. 9.

Calculation of key parameters of roof cutting and pressure relief

Determination of target strata for roof cutting and pressure relief

From the previous analysis, the pressure of small coal pillar and roadway along gob comes from the “triangular combination” composed of key strata and cushion in the caving zone and crack zone at the end of coal pillar, so the target strata of the roof cutting are the key strata in the caving zone and the crack zone, and the roof cutting height should be the upper key stratum in the crack zone. Therefore, it is necessary to

calculate the height of caving zone and crack zone during the mining of 1311 working face, then distinguish the key strata in the caving zone and crack zone, and finally determine the cutting height.

(1) Calculation of height of caving zone and crack zone.

As shown in Fig. 9, the overburden of work face 1311 is mostly mudstone, sandy mudstone and sandstone, which belongs to medium-hard rock. The theoretical developing height of caving zone and crack zone can be calculated from the following equations³⁷.

$$\text{Caving zone : } H_m = \frac{100 \sum M}{4.7 \sum M + 19} \pm 2.2 \tag{6}$$

$$\text{Crack zone : } H_L = \frac{100 \sum M}{1.6 \sum M + 3.6} \pm 5.6 \tag{7}$$

where, H_m is the height of caving zone; H_L is the height of the crack zone; $\sum M$ is the mining height of coal seam, and the height of No. 3 coal seam is 4.5 m; According to the calculation, the height of caving zone of No. 3 coal seam is 9.0~13.4 m, and the height of crack zone is 36.1~47.3 m.

(2) Determination of key strata.

The key stratum has the characteristics of thick and hard, which can be determined by judging the stiffness and strength of the strata. The discrimination of rock stratum stiffness is based on the load of the overlying strata acting on the hard rock layer of the first layer, which is calculated by Eq. (8). The discrimination of rock stratum strength is based on the broken distance of rock stratum, which is calculated by Eq. (9)¹.

$$q_n = \frac{E_1 h_1^3 (\gamma_1 h_1 + \gamma_2 h_2 + \dots + \gamma_n h_n)}{E_1 h_1^3 + E_2 h_2^3 + \dots + E_n h_n^3} \tag{8}$$

$$L_n = h_n \sqrt{\frac{2R_{Tn}}{r_n h_n}} \tag{9}$$

The conditions for judging the stiffness and strength of key stratum are as follows Eq. (10):

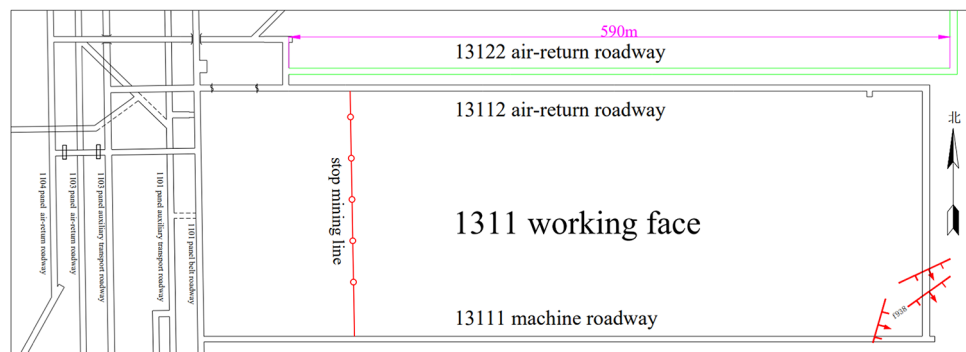


Fig. 8. Position diagram of the working face.

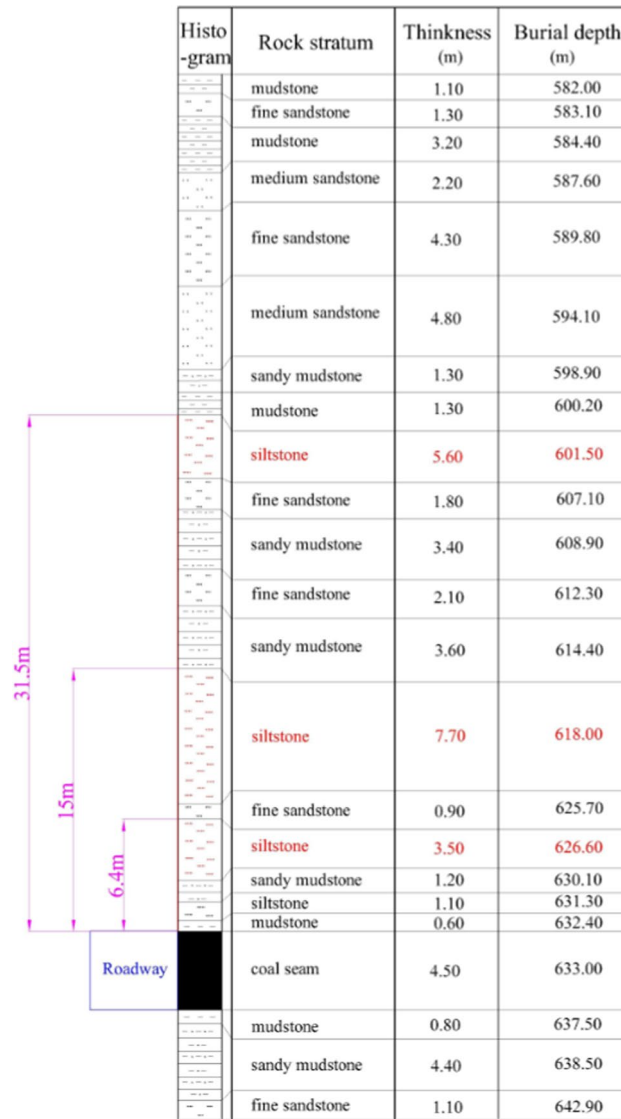


Fig. 9. Rock stratum histogram of working face.

$$\begin{cases} (q_{n+1})_1 < (q_n)_1 \\ l_{n+1} > l_n \end{cases} \quad (10)$$

By substituting Eqs. (8), (9) into equations (10), the identification conditions for the stiffness and strength of the key stratum after simplification are as follows:

$$\begin{cases} \frac{\sum_{i=1}^{n+1} \gamma_i h_i}{\sum_{i=1}^{n+1} E_i h_i^3} < \frac{\sum_{i=1}^n \gamma_i h_i}{\sum_{i=1}^n E_i h_i^3} \\ h_{n+1} \sqrt{\frac{2R_{T_{n+1}}}{\gamma_{n+1} h_{n+1}}} > h_n \sqrt{\frac{2R_{T_n}}{\gamma_n h_n}} \end{cases} \quad (11)$$

where, $(q_n)_1$ is the self-weight of the first rock stratum and the load of the overlying n-layer on it, kPa; l_n is the breaking distance of the nth layer (key stratum), m; E_i is the elastic modulus of the ith layer, GPa; γ_i , γ_n is the volumetric weight of the ith and nth stratum, N/m³; h_i , h_n is the thickness of the ith and nth stratum, m; R_{T_n} is the tensile strength of the nth stratum, MPa.

| Rock stratum | Density/kg·m ⁻³ | Bulk/GPa | Shear/GPa | Tensile/MPa | Cohesion/MPa | Friction/(°) |
|------------------|----------------------------|----------|-----------|-------------|--------------|--------------|
| Medium sandstone | 2550 | 7.33 | 6.25 | 4.76 | 9.36 | 35 |
| Siltstone | 2630 | 10.31 | 8.24 | 6.22 | 11.07 | 34 |
| 3# coal seam | 1459 | 2.72 | 1.52 | 0.56 | 2.25 | 24 |
| Mudstone | 2655 | 5.23 | 4.15 | 3.52 | 6.82 | 26 |
| Sand mudstone | 2620 | 5.06 | 4.12 | 3.45 | 7.03 | 28 |
| Fine sandstone | 2585 | 9.26 | 7.18 | 5.64 | 7.85 | 36 |

Table 1. Physical and mechanical parameters of rock and coal strata.

| Serial number | Rock stratum | Thickness/m | Load of rock /kN | Breaking distance /m | Notes |
|---------------|----------------|-------------|------------------|----------------------|---------------|
| 8 | Siltstone | 5.6 | 0.41 | 42.02 | Key stratum 3 |
| 7 | Fine sandstone | 1.8 | 0.43 | / | |
| 6 | Sand mudstone | 3.4 | 0.40 | / | |
| 5 | Finesandstone | 2.1 | 0.33 | / | |
| 4 | Sand mudstone | 3.6 | 0.28 | / | |
| 3 | Siltstone | 7.7 | 0.03 | 49.09 | Key stratum 2 |
| 2 | Fine sandstone | 0.9 | 0.11 | / | |
| 1 | Siltstone | 3.5 | 0.09 | 29.90 | Key stratum 1 |

Table 2. Roof rock load and lateral roof cantilever length.

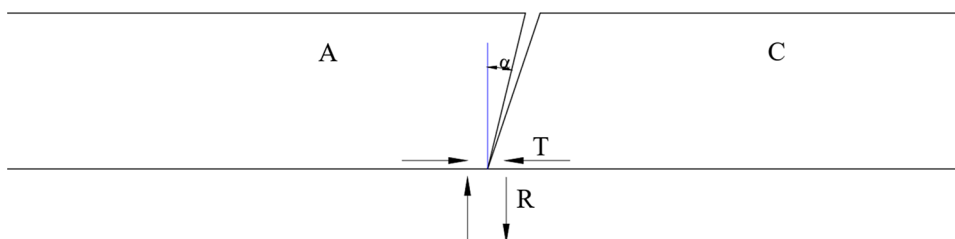


Fig. 10. Occlusal force analysis of key blocks A and C.

The physical and mechanical parameters of No. 3 coal seam and its roof and floor are shown in Table 1. According to the rock parameters in Table 1, the load and breaking distance of each stratum of coal seam roof are calculated by using formulas (9) and (10). The results are shown in Table 2.

From the calculation results, it is known that there are three key strata in the caving zone and the crack zone, which are the 1st, 3rd, 8th rock stratum shown in Table 2, namely siltstone at 6.4 m, 15 m and 31.5 m above the coal seam shown in Fig. 9. Therefore, the target strata of the deep hole are the upper part of the siltstone at 15 m and the siltstone at 31.5 m above the coal seam. The target strata of the shallow hole are the siltstone at 6.4 m and the lower part of siltstone at 15 m above the coal seam. In order to facilitate construction, the length of the deep hole is designed to be 31 m and the length of the shallow hole is 10 m, so as to ensure that the key strata can be cut off completely.

Calculation of the angle of roof cutting

According to the previous content, after the roof cutting, the key strata are broken along the roof cutting top line, so the key block B does not exist, and the key block A and the key block C bite directly, and the occlusal surface is the broken section formed by the fracture of the roof along the roof cutting line after blasting. The angle between the fracture surface and the horizontal plane is the dip angle of the roof cutting hole, and the angle α formed by the fracture surface and vertical plane is the complementary angle to each other. When cutting the roof, it will affect the effect of roof cutting that the dip angle of the drill hole is too large or too small: if the inclination angle of the hole is too small, the weak surface with large inclination angle will be formed, which will cause the roof to collapse ahead of time, which is not conducive to the stability of the mining roadway, and there is still a certain length of “cantilever beam” above the coal seam; if the inclination of the hole is too large, the friction between the rocks on both sides of the broken section increases, which is not conducive to the collapse of the key block C. Therefore, only when the key block C slips along the broken section after mining, the best roof cutting effect can be achieved. The hinged block formed by key block A and key block C after occlusion and force³⁸ is shown in Fig. 10., and formula (12) is the instability condition of the key block C¹.

$$T \sin(\varphi - \alpha) = R \cos(\varphi - \alpha) \quad (12)$$

$$\text{That is, } a = f - \arctan(R/T) \quad (13)$$

where, T is the horizontal thrust of the rock mass, according to the condition of block equilibrium, simplification to $T = q_c L / 2(h_G - S_c)$, kN; and q_c is the load of key block C, kN; h_G is the thickness of key stratum, m. S_c is the subsidence of the key block C after touching the gangue, m. R is the shear force of rock mass in the case of instability, which can be simplified to $(R = q_c L_f)$ by the condition of block equilibrium, kN; φ is the angle of friction between rocks, ($^\circ$).

The mining height of 1311 working face is 4.5 m, the thickness of immediate roof is 2.9 m, the thickness of basic roof is 12.1 m, the breaking distance of initial pressure is 25.4 m, the subsidence of contact gangue is 2.7 m, and the friction angle between rock blocks is $\varphi = 38^\circ \sim 45^\circ$. Calculated with these parameters, $\alpha = 1.5^\circ \sim 8.5^\circ$, corresponding to the Angle of cutting the top hole is $81.5^\circ \sim 88.5^\circ$, in order to facilitate construction, the drilling Angle is 85° .

Calculation of roof cutting hole spacing

Under the influence of blasting energy and explosive gas, centered on the roof cutting hole, the rock around the drilling hole forms three failure zones in the radial direction of the drilling hole, namely, the crushing zone, the crack zone and the elastic vibration zone; the radius of the crushing zone and the crack zone directly affects the spacing of the roof cutting hole. When the cylindrical uncoupled charge is adopted, the radius of the crack zone R_p ^{39,40} can be calculated by the following formula.

$$R_p = \left(\frac{\sqrt{2}\sigma_{td} b}{2\sigma_{td}} \right)^{\frac{1}{\beta}} \cdot \left(\frac{\rho_0 D_v^2 n K_2^{-2\lambda} l_e b}{8\sqrt{2}\sigma_{cd}} \right)^{\frac{1}{\delta}} \cdot r \quad (14)$$

where, σ_{td} , σ_{cd} are the dynamic tensile strength and the dynamic compressive strength of rock between the two regions, MPa; b is the lateral stress coefficient, $b = \nu / (1 - \nu)$, and ν is Poisson's ratio; β is the attenuation coefficient of shock wave; ρ_0 is explosive density, kg/m^3 ; D_v is detonation velocity of the explosive, m/s; n is the explosion pressure increase factor, generally taken as 10; K_2 is the uncoupled charging coefficient of the hole, generally taken as 1.25; λ is the expansion adiabatic coefficient of the blasting product, generally taken as 3; l_e is the axial coefficient of drilling charge, due to the use of continuous charge, no air column, taken as 1; δ is the load propagation attenuation index, $\delta = 2 - \nu / (1 - \nu)$; r is the radius of the top cutting hole, m.

The grade III coal mine permissible emulsion explosives is used during the roof cutting, and the density is 1100 kg/m^3 , the detonation velocity of the explosive is 3600 m/s, and the diameter of the hole is 75 mm. According to the test results of rock physical properties, the radius of the fracture circle is about 1.1 m.

Numerical simulation and analysis

Numerical model

In order to study and compare the effect of roof cutting and pressure relief by shaped charge blasting with combined and grouped of deep and shallow holes and with single hole in different depths, according to the engineering geological conditions of 1311 and 1312 working faces, a numerical calculation model is established by using numerical simulation software FLAC^{3D} 7.0. The size of the model is 300 m (length) \times 300 m (width) \times 62 m (height) m, as shown in Fig. 11. As the stress boundary, the top of the model is applied a vertical stress of 14.26 MPa (overlying strata load) according to the corresponding buried depth, and the other is the displacement boundary. Mohr-Coulomb model is used in rock mass. The rock physical and mechanical parameters of coal strata are shown in Table 1.

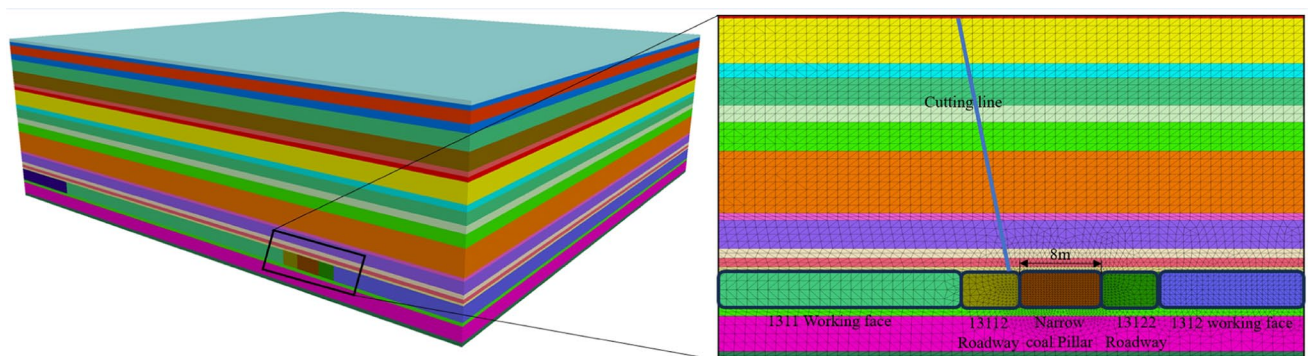


Fig. 11. Numerical model generated using numerical simulation software FLAC^{3D} 7.0.

| Scheme | A | B | C | D | E |
|------------------|----|-----------------------|-----------------------|------------------------|-----------------------|
| Width/m | 8 | | | | |
| Angle/° | 85 | | | | |
| Cutting length/m | 0 | 6.4 m (key stratum I) | 15 m (key stratum II) | 30 m (key stratum III) | 31 m (all key strata) |

Table 3. Scheme of numerical simulation.

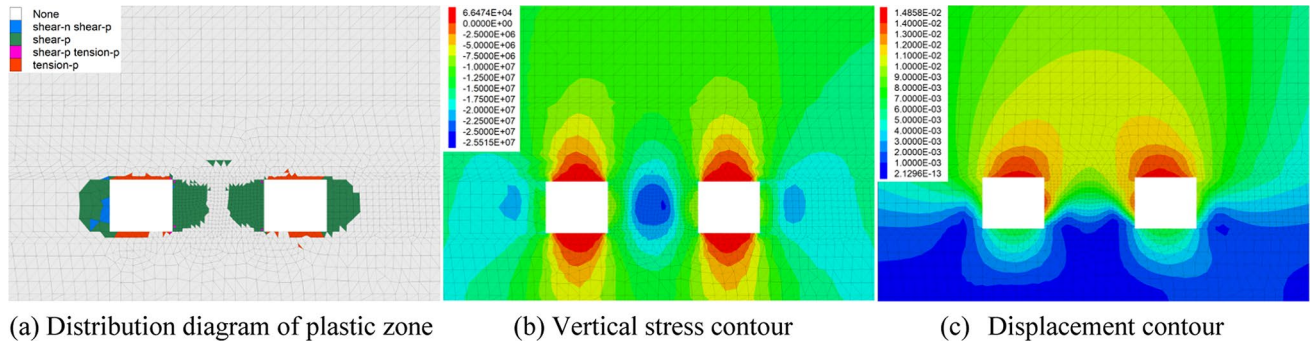


Fig. 12. Numerical simulation diagram of double roadway excavation.

Simulation scheme

In order to be closer to the reality, the numerical simulation is divided into the following processes: excavating 13,112 and 13,122 roadways with small coal pillars, cutting the roof on the side of the coal pillar in roadway 13,112, mining 1311 working face and monitoring data. In the numerical simulation, firstly, the stability of roadway during entry driving along gob is simulated and analyzed, and then the effects of shaped charge blasting with combined and grouped of deep and shallow holes and with single hole in different depths on small coal pillar and surrounding rock are simulated and analyzed. When simulating the roof cutting, the width of the coal pillar and the angle of roof cutting remain unchanged, and the roof cutting height is set to no roof cutting, 6.4 m (only cutting the key stratum 1), 15 m (only cutting the key stratum 2), 31 m (cutting the key stratum 3 and part of the key stratum 2) and the combination of deep and shallow holes (cutting the key stratum 1 ~ 3). Details are shown in Table 3.

Numerical simulation analysis

Simulation analysis of double roadways excavation

According to the specification size and support parameters of 13,112 and 13,122 roadways, double roadway excavation is simulated, and the simulation results are shown in Fig. 12. It can be seen from Fig. 12a–c that when the 8 m small coal pillar is set up for double roadways excavation, the range of the plastic zone of the roof and floor is approximate 0.6 m, and the range of the plastic zone in the both sides of the coal seam is about 2.5 m, and there is an elastic zone of about 3 m in the coal pillar. There are high stress areas in the middle of the coal pillar and both sides of the roadway. The high stress area of the coal pillar appears in the range of 3 m to 4.9 m, the maximum stress value is 25.3 MPa; the high stress area of the sides appears in the range of 3 m, and the maximum stress value is about 22 MPa. During the period of double roadway excavation, the maximum vertical displacement of roof is 14 mm and the maximum displacement of coal pillar is 13 mm.

Simulation and analysis of different roof cutting methods

The simulation results of no roof cutting and different roof cutting heights for different key strata are shown in Fig. 13. As shown in Fig. 13, when the roof is not cut, the plastic zone of small coal pillar and surrounding rock of entry retained is more developed under the influence of mining stress. Compared with the unroof-cutting, the development range of the plastic zone of the small coal pillar and the entry retained is reduced when the roof is cut by a single depth blasting hole for a single key stratum; however, because only the single key stratum is cut off, and the mining stress conduction path is not completely cut off, the change is not great. However, after roof cutting and pressure relief by shaped charge blasting with combined and grouped of deep and shallow holes, the key stratum in the caving zone and crack zone are cut off, and there is no stress transfer channel, and the stress cannot be transferred to the surrounding rock of the small coal pillar and the roadway along the gob. Therefore, the plastic zone has the smallest development range in the surrounding rock of the small coal pillar and the entry along the gob, and there is still a large range of elastic zone, that is, the stability of the small coal pillar and the remaining entry is the best under this method of roof cutting.

Figure 14 shows the contour of the vertical stress without roof cutting and with different cutting heights, and Fig. 15 shows the stress curve in the center of the coal pillar, 13,122 roadway and solid coal in the range of 20 m from the left side of the roadway. It can be seen from Figs. 14 and 15 that stress concentration occurs in the middle of the coal pillar and the right side of the roadway long the gob side during entry retained along the

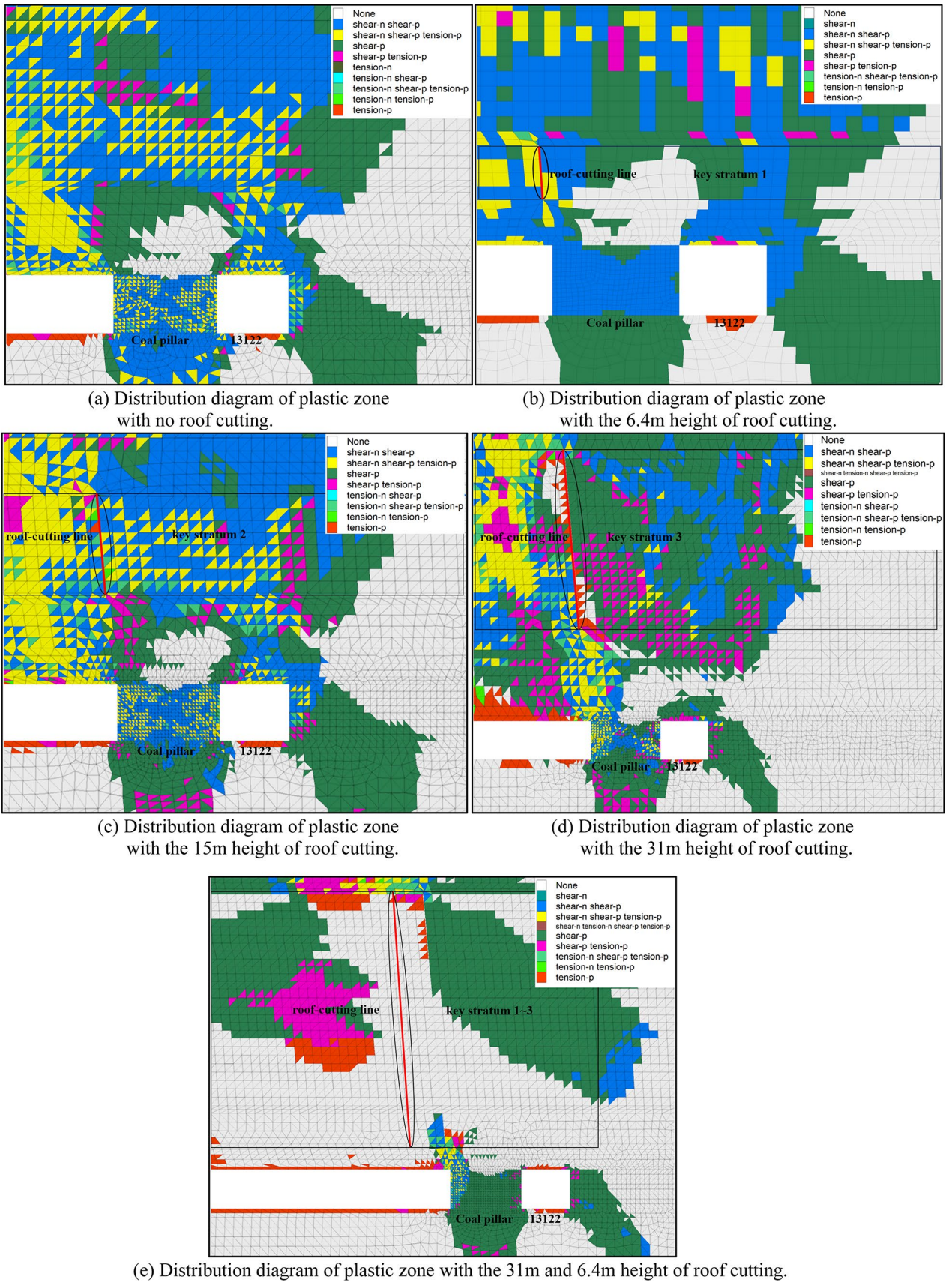


Fig. 13. Distribution diagram of plastic zone under different roof cutting methods.

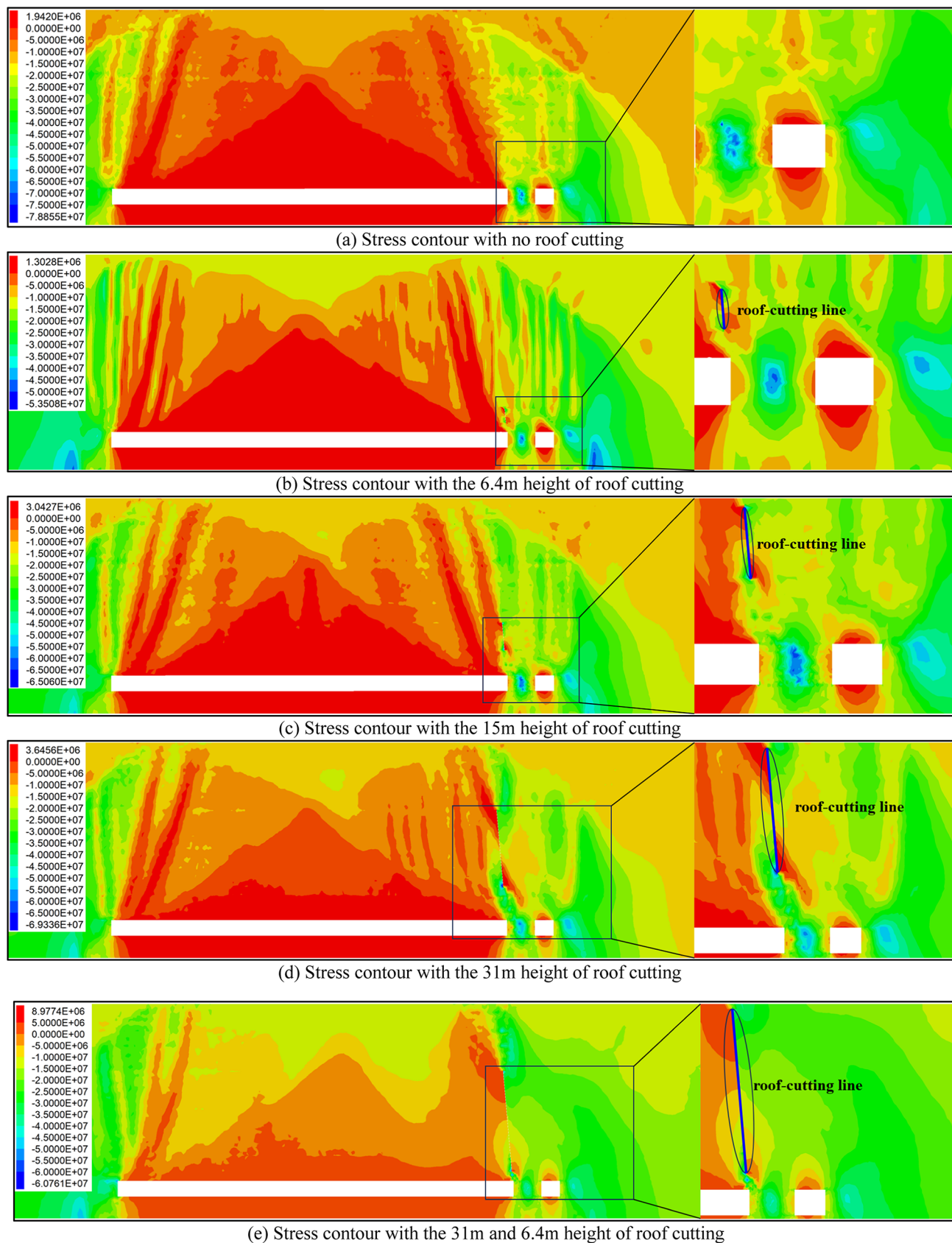


Fig. 14. Stress contour under different roof cutting methods.

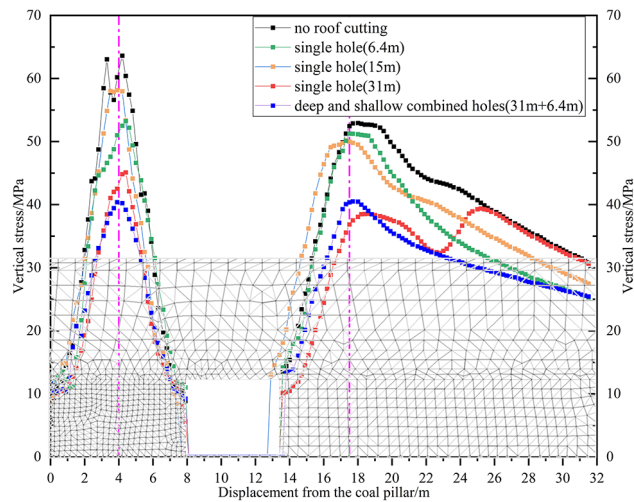


Fig. 15. Vertical stress curve under different roof cutting methods.

gob side with small coal pillar. When the roof is not cut off, the maximum stress value in the middle of the coal pillar is 63.6 MPa. When roof cutting and pressure relief by a single depth blasting hole to a single key stratum, the stress in the middle of the coal pillar gradually decreases with the increase of the blasting hole depth (the value of the stress decreasing from 58.2 MPa to 45.1 MPa); however, when the height of roof cutting is 15 m, the internal stress of the coal pillar is greater than that when the height of roof cutting is 6.4 m and 31 m; by analysis, we find that when the cutting top height is 15 m, only the second key stratum (7.3 ~ 15 m) is cut off, and the low and high key strata are not cut off, which does not really play a role in relieving pressure. In contrast, the stress in the middle of the coal pillar decreases to 40.4 MPa after shaped charge blasting with combined and grouped of deep and shallow holes. From the comparative analysis of different cutting depths and methods, it can be seen that the pressure relief effect is most obvious when the shaped charge blasting with combined and grouped of deep and shallow holes is used.

Engineering application analysis

According to the above theoretical analysis, calculation and numerical simulation, the application of entry retained along gob side with small coal pillar was carried out in the working face 1311 and 1312: a small coal pillar with width 8 m is set up between roadway 13,112 and 13,122, and the coordinated control scheme of “roof cutting and pressure relief by shaped charge blasting with combined and grouped of deep and shallow holes” + “anchor-cable combined support” + “coal pillar strengthened” is adopted to improve the stress environment of small coal pillar and the roadway to the greatest extent and improve the stability of the surrounding rock.

Roof cutting scheme of shaped charge blasting with combined and grouped of deep and shallow holes scheme

Key parameters of deep and shallow hole construction

Through the above calculation and numerical simulation, the key parameters of “shaped charge blasting with combined and grouped of deep and shallow holes” are determined, as follows:

1. Position of roof cutting hole: 0.3 m away from coal pillar side.
2. Length of roof cutting hole: the deep hole is 31 m, and the length of shallow hole is 10 m.

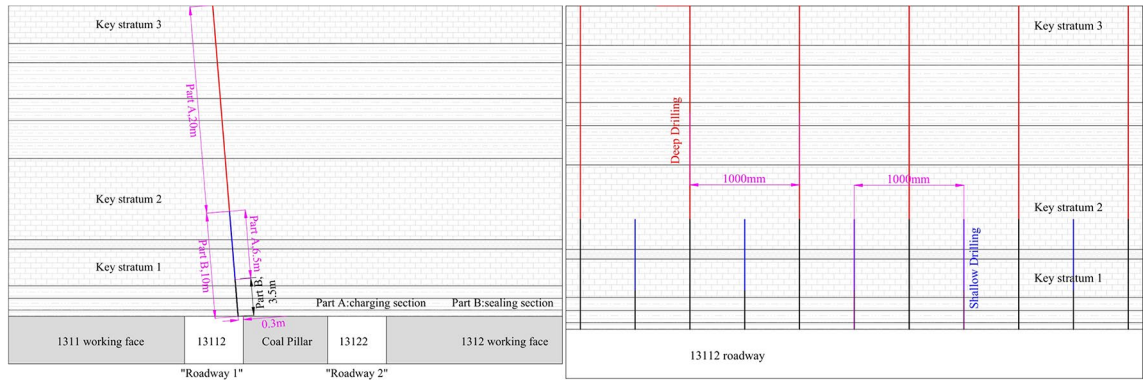
According to the calculation results of the key strata, the distance from the upper key layer to the roof of the coal seam is 31.5 m. When the dip angle of the top drilling hole is 85°, the drilling depth is 31.6 m; however, during the on-site construction, the length of each drill pipe is 1 m, if the drilling depth is 31.5–31.6 m, it increases the difficulty of construction and is not conducive to the accurate construction of drilling, therefore, the drilling depth is designed to be 31 m and does not affect the effect of roof cutting.

3. Angle of roof cutting hole: the inclination angle of deep and shallow hole is 85°.
4. Spacing of roof cutting hole: according to the previous calculation, the radius of the crack zone of the blasting hole is 1.1 m. In order to ensure the effect of pressure relief, the spacing between the deep and shallow holes is set to 1 m, and the deep and shallow holes are staggered.

The combined arrangement of deep and shallow holes is shown in Fig. 16.

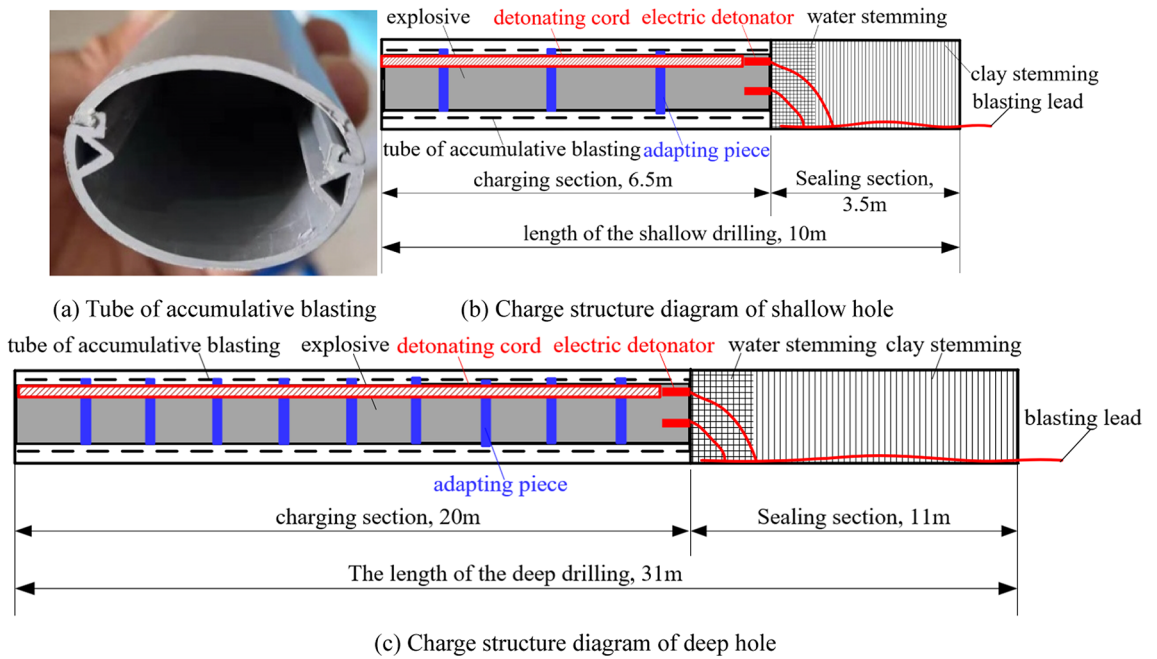
Construction scheme of combined grouping blasting with deep and shallow holes

The deep and shallow blasting holes are all arranged on the side of the small coal pillar of roadway 13,112, away from the 300 mm of the small coal pillar. The construction of the hole is ahead of the working face, and the leading distance is not less than 50 m. While cutting the roof, grouping blasting is adopted according to depth of the



(a) Dip profile of roof-cutting drilling (b) Strike profile of roof-cutting drilling

Fig. 16. Blasting roof-cutting diagram of combination with deep and shallow drillings.



(a) Tube of accumulative blasting (b) Charge structure diagram of shallow hole (c) Charge structure diagram of deep hole

Fig. 17. Diagram of tube of accumulative blasting and charge structure.

hole, that is, the blasting holes are divided into two groups : one group of deep holes and one group of shallow holes ; the shallow hole is blasted firstly, meanwhile, the deep hole is used as the guide hole of the shallow hole. After the completion of shallow-hole blasting, charging and blasting will be carried out on the deep hole. The specific construction scheme is as follows:

- (1) Construction equipment of the hole: DCA-45 type advance roof cutting drilling rig, matching with $\Phi 75$ mm drilling bit.
- (2) Diameter of the hole: 75 mm.
- (3) Sealing length of roof cutting hole: in accordance with the requirements of the Coal Mine Safety Regulations, for medium and deep holes, the sealing length should not be less than 1/3 of the hole depth; so the sealing length of deep hole is 10 m and that of shallow hole is 3.5 m.
- (4) Blasting method: shaped charge blasting.
- (5) Explosive: grade III coal mine permissible emulsion explosives, specification 35 mm \times 300 mm/roll, each roll weight 333 g; continuous charging.
- (6) Charging and sealing structure: adopting “O” shaped charge tube, which is made of flame retardant and antistatic material, diameter 48 mm, length of each 2 m–1 m, as shown in Fig. 17a.

In accordance with the requirements of the Coal Mine Safety Regulations, for medium and deep holes, the sealing length should not be less than 1/3 of the hole depth; so the sealing length of deep hole is 11 m and

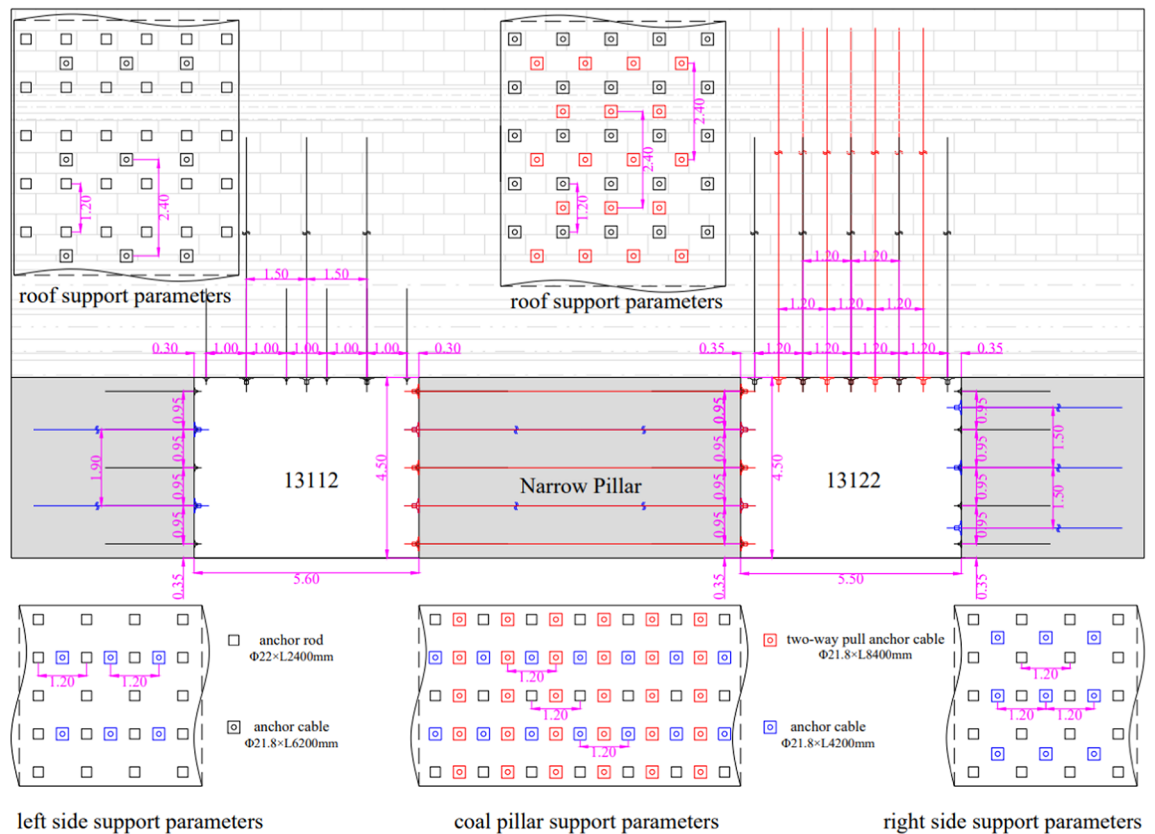


Fig. 18. Support parameter diagram of 13,112 and 13,122 roadway.

charging length is 20 m, that of shallow hole is 3.5 m and 6.5 m. Deep and shallow holes are sealed with clay and blister mud. The charge and sealing structure are shown in Fig. 17.

- (7) Detonation mode: double detonation with electric detonator and detonating cord.

When detonating in this way, two electric detonators are arranged in parallel with the outermost shaped charge tube, one detonating explosive and one detonating the cord. After the detonating cord is installed, the detonator is bound with the detonating cord, and the exposed end of the detonating cord is inserted into the lowest roll of explosives. The detonator blasting lead leaves 200 mm, and the rest is cut; the length of blasting lead needs to exceed 2 m outside the hole.

- (8) Initiation sequence and quantity: first blasting shallow holes, each group of 10; then blasting deep holes, each group of 10.

Control scheme of small coal pillar and surrounding rock of roadway

13,112 roadway and 13,122 roadway are excavated at the same time, and the small coal pillar and 13,122 roadway will experience mining disturbance of 1311 and 1312 working face respectively, so 13,112 roadway, 13,122 roadway and the small coal pillar adopt the surrounding rock control scheme of “anchor-cable combined support” + “coal pillar strengthened”. Especially, the support of small coal pillar and 13,122 roadway need to be strengthened during construction, the specific support methods as follows:

13,112 roadway support scheme: the 13,112 roadway is a rectangular section with a section size of 5.6×4.5 m (width \times height). The left-handed non-longitudinal steel bolt is used as the foundation support; the bolt specification is $\Phi 22 \times 2400$ mm, and the pretightening moment is 400 N·m; the row spacing of the roof is 1.0×1.2 m, and the left side is 0.95×1.2 m, and the right side (coal pillar) is 1.9×1.2 m. Using the anchor cable as the reinforcement support, the roof reinforcement anchor cable specification is $\Phi 21.8 \times 6200$ mm, and the pretightening force is 300kN; the row spacing is 1.5×2.4 m. The reinforcement anchor cable specification of the two sides is $\Phi 21.8 \times 4200$ mm, and the pretightening force is 180kN; the anchor cable in the left side is arranged between two rows of bolts, while the anchor cable and the bolts in the right side are arranged in the same row, and the row spacing is 1.9×1.2 m.

13,122 roadway support scheme: the 13,122 roadway is a rectangular section with a section size of 5.5×4.5 m (width \times height). The anchor cable is adopted as the foundation support in the roof, and the anchor cable specification is $\Phi 21.8 \times 6200$ mm, with the pretightening force 300kN, and the row spacing is 1.2×1.2 m. The

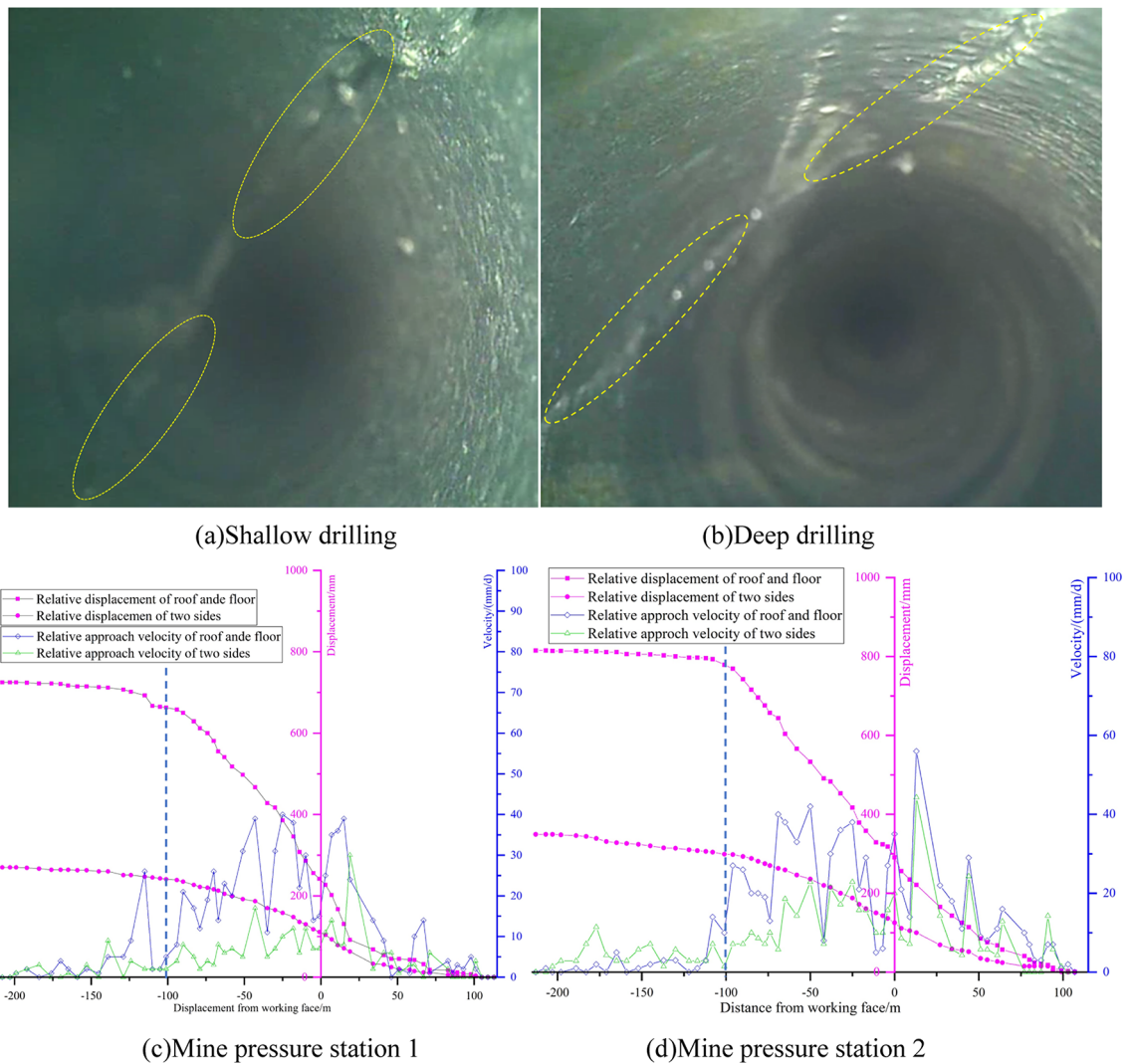


Fig. 19. Effect diagram of accumulative blasting and surrounding rock control.

left-handed non-longitudinal steel bolt is used as the foundation support of two sides, the bolt specification is $\Phi 22 \times 2400$ mm, with the pretightening moment 400 N·m, and the row spacing of the left side (coal pillar) is 1.9×1.2 m, and the row spacing of the right side is 0.95×1.2 m. The roof reinforcement anchor cable specification is $\Phi 21.8 \times 8200$ mm, and the pretightening force is 300kN as well, the arrangement in the form of “4-3-4”, the row spacing 1.2×2.4 m. The reinforcement anchor cable specification of the both sides is $\Phi 21.8 \times 4200$ mm, and the pretightening force is 180kN; In the left sides, the anchor cable and the anchor rod are arranged in the same row, and the row spacing is 1.9×1.2 m, while the anchor cable in the right is arranged between two rows of bolts with a spacing of 1.5×1.2 m.

Reinforcement scheme of small coal pillar: in order to improve the bearing capacity of small coal pillar, bidirectional grouting cables are applied in the small coal pillar arranged between the original supports. The specification of anchor cable is $\Phi 21.8 \times 8400$ mm, the pretightening force is 300kN, and the row distance is 0.95×1.2 m. The support parameters are shown in Fig. 18.

Analysis of the effect of roof cutting and pressure relief and surrounding rock control

In order to inspect the roof cutting effect of shaped charge blasting with combined and grouped of deep and shallow holes, while the shallow hole blasting, the deep hole is used as the effect check hole, and the crack development in the deep hole surrounding rock is observed by the borehole peep instrument, then the roof cutting effect of the shallow hole is analyzed according to the peep results. Similarly, during deep hole blasting, the shallow hole is deepened for peeping analysis. As shown in Fig. 19a,b, the cracks caused by blasting can be clearly observed on both sides of the borehole, whatever deep or shallow hole, indicating that the blasting method can completely cut the surrounding rock between the holes under the current hole spacing and charging parameters.

Figure 19c,d shows the mine pressure monitoring results of two monitoring stations in 13,122 roadway during the mining period of 1311 working face. According to the displacement monitoring data of the monitoring

station, it can be seen that during the mining period of the 1311 working face, the roadway begins to deform when the distance between the working face and the monitoring station is not more than 100 m, but the deformation is small. When the distance between working face and monitoring station is about 27 m, the roadway begins to deform obviously, at this stage, the roadway has been affected by the advanced abutment pressure of the working face. When the mining position of the working face passes through the monitoring station and does not exceed 100 m, the roadway deforms violently, which is due to the overlying rock movement in the gob after mining, mainly caused by the breaking and subsidence of the key strata layer by layer in the caving zone and crack zone. When the mining position of the working face exceeds 100 m of the monitoring station, the 13,122 roadway begins to stabilize. According to the relative displacement rate of the roadway surface, it can also be found that when the 1311 working face passes through the 13,122 monitoring station, the roadway surface displacement rate increases obviously in the range from 30 m behind the monitoring station to 100 m in front of the monitoring station, showing an intermittent jump trend, which is affected by the periodic pressure of the working face.

According to the monitoring data of the station, during the mining period of the 1311 working face, the relative displacement of the roof and floor of the 13,122 roadway is about 800 mm, and the relative displacement of the two sides is about 350 mm, but according to the field observation, the relative displacement of the roof and floor is mainly the floor heave; in addition, the deformation of the coal pillar side is obvious, but there is no support failure and surrounding rock instability in the roadway, and the surrounding rock control effect is good.

Conclusion

By establishing the mechanical model of entry along gob side, the stress source of entry retained along gob side with small coal pillar is analyzed, the principle of roof cutting and pressure relief is analyzed, and the roof cutting pressure relief technology of shaped charge blasting with combined and grouped of deep and shallow holes is put forward, in the process the industrial test is carried out. the main conclusions are as follows:

1. By analysis, the stress source of the small coal pillar and roadway surrounding rock is the “triangular combination” composed of the “cantilever beam” structure in the caving zone and the “voussoir beam” structure in the crack zone. The main stress acting on the small coal pillar and roadway surrounding rock is the self-weight and rotary extrusion pressure of the “triangular combination”.
2. By cutting off the multi-layer key strata in the caving zone and the crack zone (close to the small coal pillar), the formation of the “triangular combination” is avoided, the hard roof breaking distance is reduced, and the pressure of the small coal pillar and roadway surrounding rock system can be effectively depression.
3. The technology of shaped charge blasting with combined and grouped of deep and shallow holes is put forward. In view of the immediate pressure characteristics of the caving zone, giving priority to blasting shallow holes, meanwhile deep holes play the role of guiding holes to strengthen the effect of shallow hole cutting; then blast deep holes, to cut off the key strata in the crack zone and alleviate the delayed loading. Through the staggered arrangement of deep and shallow holes, the multi-layer key strata in the caving zone and crack zone can be effectively cut off;
4. The industrial test shows that after adopting the cooperative control scheme of “roof cutting and pressure relief by shaped charge blasting with combined and grouped of deep and shallow holes” + “anchor-cable combined support” + “coal pillar strengthened”, the pressure of small coal pillar and roadway surrounding rock system is reduced. It improves the stability and bearing capacity of surrounding rock structure, effectively controls the deformation of surrounding rock of small coal pillar retaining roadway along gob, and provides a reference for the control and maintenance of surrounding rock of this kind of roadway.

Data availability

Some or all data, models, or codes generated or used during the study are available from the corresponding author by request.

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Author contributions

Author Contribution Lei Li: Supervision, Writing-review and editing. Yuzhong Yang: Data curation, methodology, writing-original draft. Liyun Wu: Conceptualization, Project administration, Supervision. Weiyu Zhang: Project administration, resources. Wanli Yang: Project administration, resources. Yaowei Zhai: Conceptualization, project administration.

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The authors declare no competing interests.

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Additional information

Correspondence and requests for materials should be addressed to Y.Y.

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