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ROOF WEAKENING OF HYDRAULIC FRACTURING FOR CONTROL OF HANGING ROOF IN THE FACE END OF HIGH GASSY COAL LONGWALL MINING: A CASE STUDY

OSŁABIENIE STROPU POPRZEZ SZCZELINOWANIE HYDRAULICZNE PRZY KIEROWANIU STROPEM W REJONIE PRZODKA W KOPALNI GAZOWEJ, PRZY WYDOBYCIU ŚCIANOWYM – STUDIUM PRZYPADKU

The occurence of hanging roof commonly arises in the face end of longwall coal mining under hard roof conditions. The sudden break and subsequent caving of a hanging roof could result in the extrusion of gas in the gob to the face, causing gas concentrations to rise sharply and to increase to over a safety-limited value. A series of linear fracturing-holes of 32 mm diameter were drilled into the roof of the entries with an anchor rig. According to the theory that the gob should be fully filled with the fragmentized falling roof rock, the drilling depth is determined as being 3~5 times the mining height if the broken expansion coefficient takes an empirical value. Considering the general extension range of cracks and the supporting form of the entryway, the spacing distance between two drilling holes is determined as being 1~2 times the crack's range of extension. Using a mounting pipe, a high pressure resistant sealing device of a small diameter-size was sent to the designated location for the high-pressure hydraulic fracturing of the roof rock. The hydraulic fracturing created the main hydro-fracturing crack and airfoil branch cracks in the interior of the roof-rock, transforming the roof structure and weakening the strength of the roof to form a weak plane which accelerated roof caving, and eventually induced the full caving in of the roof in time with the help of ground pressure. For holes deeper than 4 m, retreating hydraulic fracturing could ensure the uniformity of crack extension. Tested and applied at several mines in Shengdong Mining District, the highest ruptured water pressure was found to be 55 MPa, and the hanging roof at the face end was reduced in length from 12 m to less than 1~2 m. This technology has eliminated the risk of the extrusion of gas which has accumulated in the gob.

Keywords: high gassy; longwall mining face; hard roof; hanging roof; hydraulic fracturing

Stropy wiszące spotykane są powszechnie w rejonie przodka przy wybieraniu ścianowym, w warunkach dużych obciążeń stropu. Nagłe zarwanie stropu i powstały zawał mogą doprowadzić do wypływu gazu ze zrobów do strefy przodka, wskutek czego stężenia gazu gwałtownie wzrosną, przekraczając

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bezpieczne wartości. W stropie wywiercono serię liniowych otworów szczelinowych o średnicy 32 mm. Zgodnie z teorią, że zroby winny zostać całkowicie wypełnione rozdrobnionymi fragmentami skał ze stropu, głębokość wiercenia oblicza się jako 3-5 krotność wysokości wybierania, a wartość współczynnika rozszerzania szczelin otrzymuje się empirycznie. Uwzględniając zakres rozszerzalności szczelin i obecność obudowy chodnika, obliczono rozstęp pomiędzy kolejnymi otworami wierconymi jako 1-2 krotność zakresu rozprzestrzeniania się szczeliny. Przy pomocy rury, do wyznaczonego punktu przesłano szczeliwo odporne na wysokie ciśnienia, dla potrzeb szczelinowania hydraulicznego skał stropowych. Szczelinowanie hydrauliczne spowodowało powstanie szczeliny w skale stropowej, zmieniając tym samym strukturę stropu i powodując jego osłabienie poprzez powstanie płaszczyzny o zmniejszonej wytrzymałości, co przyspieszy zapadanie stropu, a w końcu wywoła pełny zawał, czemu sprzyjać będą także naciski powierzchniowe. W przypadku otworów o głębokości powyżej 4 m cofanie szczelinowania hydraulicznego zapewni równomierność rozprzestrzeniania się szczeliny. Metoda ta została przetestowana i zastosowana w kilku kopalniach w zagłębia Shengdond, najwyższe zastosowane ciśnienia wody wyniosły 55 MPa, zaś długość wiszącego stropu w rejonie przodka zmniejszono z 12 do poniżej 1-2 m. Zastosowanie tej technologii pozwala na wyeliminowanie ryzyka wypływu gazu nagromadzonego w wyrobisku.

Słowa kluczowe: kopalnia gazowa, przodek ściany, strop, strop wiszący, szczelinowanie hydrauliczne

1. Introduction

In China, there are many coal longwall mining faces with hard roofs. Although most of the roof in the central part of face could fall in time, roof hanging problems at the end of the face are also common because the support of the coal pillar is beside the entry. The shape of the hanging roof is mostly evident in the form of a long narrow board along the strike of the entry. The area taken by the hanging area is generally larger than dozens of square meters. Hanging roof in the face end is seen through the periodic process of hanging – caving – hanging again. From the perspective of ground pressure and strata control, this won't cause shocks to the coal mining face. However, if hanging roof suddenly caves, toxic and harmful gases in the gob may be squeezed out and pushed into the face space, leading to a safety risks, especially for high gassy coal mines. An excess of gas in the face within a very short time could cause gas concentrations to exceed the allowable limit while mines are in production. At the same time, if the hanging roof suddenly caves in, the crash among caved rocks may produce sparks, providing another potential safety risk of inducing the gathered gas in the gob to explode.

Many studies have been conducted on the hanging and fracturing mechanisms of the working faces of hard roofs in mining (Niu, 1988; Zhu et al., 1991; Jin & Xu, 1994; Altounyan & Taljaard, 2001; Banerjee et al., 2003). Here, the primary control method used is blasting. Due to the incidence of gas in the gob, traditional blasting methods seem too complex to construct and manage, and are found to be unsuitable. Thus, it is necessary to find a new control technology which is safe, effective and easy to carry out. In a situation where a hard roof is hanging in a coal mine face and where high levels of gas are present, a control technique of hydraulic fracturing with a small diameter-size (ϕ32 mm) roof-borehole (drilled by bolter) ahead of the working face in the entry, is proposed. This technique has been found to control end-roof overhang safely and effectively, and has the advantages of less quantities, operational simplicity, and low cost and without disrupting normal production. As such, this technique is found to be superior, especially in high gassy coal mine.

2. Analysis of hanging roofs at the face end of longwall mining

Shendong Mining District is China's largest underground coal mining base. A large number of underground longwall mining faces have problems with hanging roofs at face end. The following section characterizes a typical mine to illustrate the problem.

2.1. The condition of a longwall mining face

No. 2 mine of Cuncaota belongs to the Shenhua Group which is a mining and energy company in China. The fully mechanized face 22115 is located in the western part of the 2-2 coal seam and is a working face of this mine. The ground level of the face is +1275 ~ +1328 m, and the coal seam floor level is +1015.5 ~ +1040.3 m. The geologic column of the coal seam is shown in Figure 1. It's average thickness is found to be 2.49 m. It can be seen that the roof seam is integrated, and that the joints and fissures are not developed (Fig. 2, Table 1). The face's strike-length is 2650 m and its lean-length is 300 m. The mining method used was mechanized overall height mining, and all caving methods are used to deal with the gob.

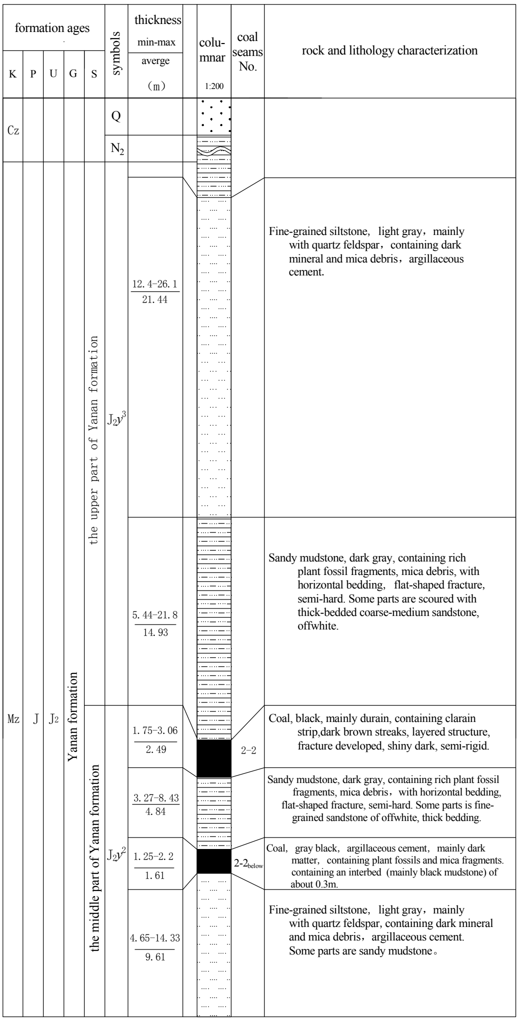


Fig. 1. Representative geologic column

Fig. 2. Integrity of coal seam roof

TABLE 1

Physical and mechanical properties of roof

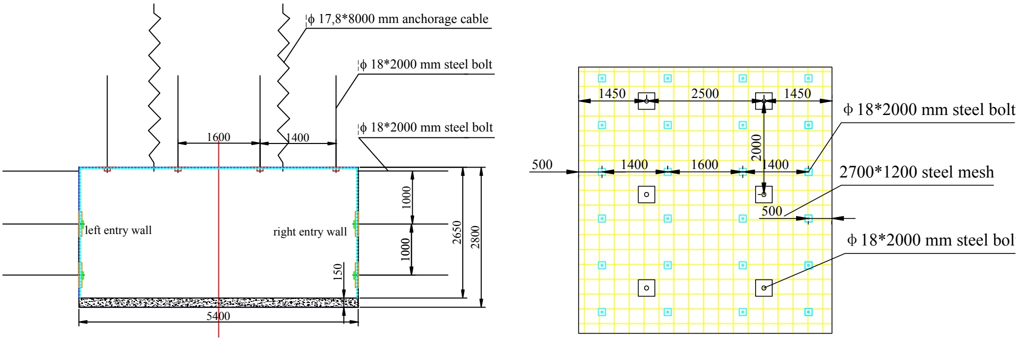
Strata position	Lithology	Compressive strength /MPa	Strength of extension /MPa
roof	fine sandstone	28.3~61.2	0.79~2.88
roof	sandy mudstone	19.2~31.2	1.02~3.31
2-2 coal seam	coal	29.34	1.25

In order to ensure the surrounding rock stability of the entry, and meet the need of production, the width of coal pillar between faces is 20 m. The section shows a rectangular shape of 15.12 m^2, and the entry is 5.4 m wide and 2.8 m high. The entry is supported by the anchor and anchor bolt as seen below:

The support parameters of the left and right coal walls are as follows: steel bolt of $\phi18\times2000$ mm, wooden pallets of $500\times200\times50$ mm, $10^{\#}$ iron wire mesh of 1.2×10 m. Three rows of bolts are fixed on the mesh with an inter-row spacing of 800×1000 mm.

Roof support parameters: as shown in Figure 3, the support form involves anchor + bolt + steel mesh. Many bolts sized $\phi18\times2000$ mm support the roof, and each 4 make up a row. The bolt row spacing is 1000 mm. The leftmost bolts are 500 mm from the left coal wall and the rightmost bolts are 1900 mm from the right coal wall. The left and right bolts are placed in a symmetrical arrangement. As with the bolts, the anchorage cables of size $\phi17.8\times8000$ mm are arranged as each row made of 2 anchorage cables. The anchorage cable row spacing is 2000 mm. The leftmost anchorage cables are 500 mm from the left coal wall, and the rightmost ones are placed in a symmetrical arrangement.

The roof in face end is supported by ZY10200/14/28 shield hydraulic support. The entry is reinforcing supported ahead of the coal wall of working face. During the normal mining, the distance of reinforcing support for the entry is 20 m. The entry is additional supported by two row individual hydraulic props, the spacing of individual hydraulic props is 1 m, and the distance of individual hydraulic props to the wall of the entry is 1 m.



(a) Cross sectional view (b) Plan view

Fig. 3. Support form of entry

2.2. End-roof overhang and its impact on safety production

Some portions of the roof at the point of machine entry are made of fine-grained sandstone. The fine-grained sandstone is difficult to break and cave. Besides, the supporting strength is big. As such, the roof undergoes a cyclical process of hanging roof – caving – hanging roof in gob with face advancing. Most of the time, the end-roof overhang develops into a long and narrow plate shape along the strike direction of the entryway (Fig. 4). The maximum length of the overhang is 20 m long, and the maximum hanging roof area can be larger than 120 m^2.

Fig. 4. Hanging roof in the end of longwall mining face

When a large area of the roof breaks and caves, it could result in the extrusion of gas that has accumulated in the gob, leading to a sharp rise in gas concentrations that go over a safety-limited value. On November 20, 2012, the hanging roof (about 3 m wide) which was 8 m behind the hydraulic support in the upper corner of the return entry formed a 24 m^2 space of hanging roof. Before the hanging roof broke and caved, the gas concentrations beneath the roof, 2.5 m behind the cut roof line, were 0.42%. Furthermore, the gas sensors measured the gas concentrations beneath the roof at the upper corner of the return entry as being 0.34%. After these measurements were taken, the hanging end-roof broke and caved, resulting in a gas ultra-limit alarming accident. On that day, at 22:07:01, the gas concentrations reached a maximum of 2.14% (as monitored by the gas sensor at the upper corner of the return entry). The gas ultra-limit alarm continued for 1 minute and 36 seconds. The alarm automatically cut off the non-MKD-I Power, production stopped, and all the workers were evacuated from the working face. At 22:40, about 33 minutes after the alarm had initially gone off, as monitored by plot reconnaissance, the gas concentrations at the upper corner of the return entry, in the central part of the working face were 0.36%, 0.20% and 0.10% respectively. This situation put the workers at great risk, and seriously affected production levels.

3. Control principle of hydraulic fracturing for end-roof overhang

3.1. Stability analysis for end-roof overhang

The form of the hanging roof at the gob end can be considered to have two fixed sides, and two handing sides (Fig. 5). As impacted by the hard roof and high-strength entry support, the shape of the hanging roof at the gob end varies, but generally approximates a rectangular or triangular shape. To facilitate calculation, the shape is simplified as a rectangle with length *a* and width *b*.

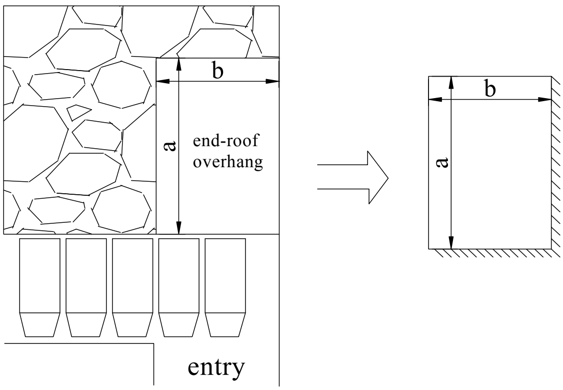


Fig. 5. Schematic of end-roof overhang

Generally, *b* equals the entry width. When $a < b$, the hanging roof area will not have much impact on production; when $a > b$, the roof needs to be controlled and weakened artificially. Reducing the width of coal pillar can make the coal pillar to be damaged, and is conducive to the breaking of roof in face end. Due to the need of surrounding rock control for the entry, the hanging roof can not be controlled by reducing the width of coal pillar.

Solving the issues related to the state of roof stress is a complex process. However, in mining engineering, according to plate theory, in order to solve the problem on a macroscopic level, a simplified Marcus arithmetic is adopted. The plate's fracturing step (i.e. limit overhang distance) can be calculated as follows (Qian & Shi, 2003):

$$a = \frac{2h}{1-\mu^2}\sqrt{\frac{\sigma_s}{3q}\frac{1+\lambda^4}{1+\mu\lambda^2}}$$

Where μ is the poisson ratio of rock strata; q is the self-weight of the rock strata and the load above it; h is strata thickness; σ_s is rock tensile strength; λ is the geometry coefficient of the hanging roof, $\lambda = a/b$.

Suppose that the poisson ratio of strata μ and the geometry coefficient λ are known, given that parameter k:

$$k = \frac{2}{1-\mu^2}\sqrt{\frac{1+\lambda^4}{1+\mu\lambda^2}}$$

We can get:

$$a = kh\sqrt{\frac{\sigma_s}{3q}}$$

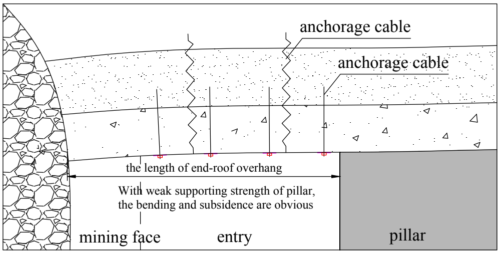
Hydraulic fracturing can produce cracks in rocks, destroy rock integrity and reduce rock layer thickness h; Wetting power and transforming the rock structure can reduce tensile strength σ_s (Huang et al., 2011). Thus, according to this formula, hydraulic fracturing can minimize length a of the hanging roof.

3.2. Control principle of hydraulic fracturing for end-roof overhang

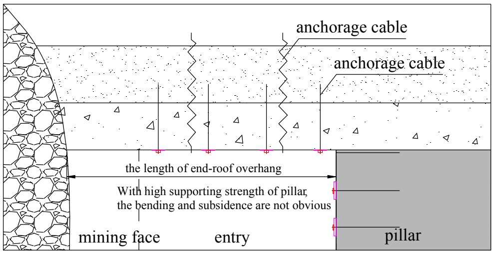
The water pressure parameters and hydraulic crack extended morphology of hydraulic fracturing for coal-rock mass is related to the stress field, mechanical properties of the rock and rock mass structure. The structure of coal-rock mass is transformed by the main hydraulic crack and the airfoil branch crack, the coal rock's permeability is improved; water penetrates inside the coal-rock mass through the crack resulting in an increase in moisture content and a weakening of the strength of the coal-rock. Through the extension of the main hydraulic crack and airfoil branch crack, as well as the effects of water wetting, the technical goals of structure transformation, strength-weakening and improvement in permeability etc. are achieved. Different engineering projects would thus place different levels of focus and emphasis with respect to the three above effects (Huang et al., 2011).

The reasons that the hanging roof has an overall stability at the face end are as follows:

1) The roof is hard, joints and cracks undeveloped, and it has a thick sub-layer. 2) The entryway's supporting strength is high. With rock bolt + anchor cable + steel mesh, the roof rocks squeeze each other and form a whole, increasing the integrity of the roof. 3) The high supporting strength of entry-wall strengthens the coal rock stiffness at the entry-side, leading to minimal roof bending and subsidence (Fig. 6). Low levels of bending and curvature lead to a peak value of abutment pressure on the coal pillar near the entry wall, i.e., the hanging-roof's double clamped points of cantilever beam are close to the entry-wall.



(a) Weak entry-wall support



(b) High strength entry-wall support

Fig. 6. Schematic of the cantilever beam for end-roof overhang

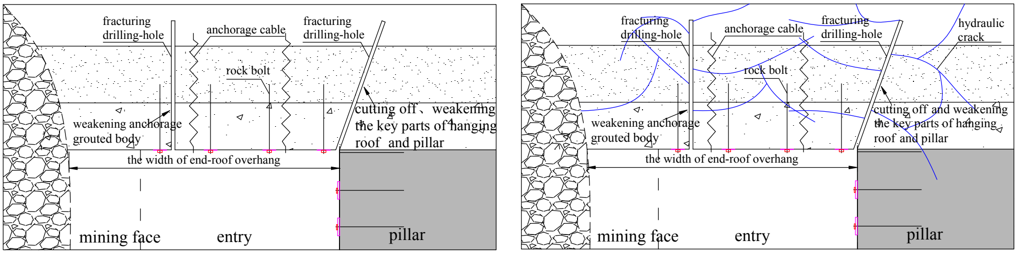
According to the principle of rock weakening by hydraulic fracturing, for the anchorage grouted body right above entry, even if the hydraulic fracturing generating the hydraulic crack weakens the overall performance relative to what it had previously been, the squeezing action of the anchorage system makes the broken rock form a whole structure and the structure becomes stable. As such, the key is to fracture and weaken the roof above the entry-side of the coal. The control principle for end-roof overhang is as follows: 1. Cut off the roof from the entry's upper corner, or weaken the roof strength above the coal pillar; 2. Weaken the overall strength of the roof support; 3. Bring the level of pillar stiffness down.

3.3. Control methods of hydraulic fracturing for end-roof overhang

Taking the overall stability characteristics of the roof and its system of support in the longwall mining face into consideration, according to the crack propagation laws of hydraulic fracturing (Alekseenko et al., 1997; Takatoshi, 2008; Cheng et al., 2012; Huang et al., 2012; Huang et al., 2013; Huang et al., 2013), the control method is determined as "cutting the end-roof mainly along the entry side + weakening the overall mechanical properties of the anchorage grouted body" (Fig. 7). A drilling-hole arrangement for hydraulic fracturing is also designed. With the anchor cable machine and a small diameter-size (ф32 mm) drilling-hole, a set of equipment for roof control by hydraulic fracturing, especially installation pipe with a small diameter-size and a high-pressure as well as a sealing device are developed.

Therefore, in order to reduce the roof strength for falling and caving in time, it is determined to be effective to drill a drilling-hole of a small diameter size along the two edges of the entry roof. Following this, hydraulic fracturing is carried out to weaken the strength of the anchorage grouted body above the entry, the roof above the entry-side, and the coal mass of the pillar.

In order to ensure for the homogeneity of crack propagation when fracturing a drilling-hole deeper than 4 m, a backward sectionalized fracturing technique is used. For example, we begin by sealing in the hole end and then conduct the fracturing. Following this, we draw back the sealing device, reseal the hole and repeat the fracturing again. Finally we draw back the sealing device, and if necessary, repeat the sealing and fracturing again until the propagation is homogeneous. For different structures and lithologies of roof rock, especially those which have a rock strata that is thick and difficult to control, a varying technique employing an alternate arrangement of



(a) Arrangement of drilling-holes

(b) Structure transformation

Fig. 7. Method of hydraulic fracturing for end-roof overhang control

deep and shallow drilling-holes provides a good alternative. Due to the small diameter-size of the equipment, this hydraulic fracturing method, which employs the anchor cable machine, can also be used to control the roof overhang in the middle part of mining face in the gob through the creation of fracturing drilling-holes among the supports.

4. Test Program

4.1. Determination of the fracturing rock strata

The immediate roof of 2-2 coal seams is composed of 5.44~21.8 m thick sandy mudstone. It has a semi-hard horizontal bedding with some parts comprising of thick-bedded coarse, medium sandstone. As seen in Table 2, drilling exploration at the mining face has revealed that the immediate roof rock of face 22115 (mining 2-2 coal seam) is comprised of 7 m ~ 22 m sandy mudstone or fine-grained sandstone.

TABLE 2

Lithology of immediate roof of 2-2 coal seam as revealed by a variety of drillings

Drilling-hole No.	**Thickness of immediate roof (m)**	**Lithology describe**
B16	6.81	siltstone: gray, giant-thick bedded, containing a small amount of pyrite nodules and incomplete fossil of plant leaf, semi-hard.
B54	22.85	sandy mudstone: light gray or gray, medium-thick bedded, with horizontal texture, wavy bedding, local lenticular bedding, angular and irregular-shaped fracture, containing incomplete fossil of plant leaf.
BK-1 drilling-hole	20.66	fine-grained sandstone: off-white, mainly with quartz feldspar, containing dark mineral and mica debris, argillaceous cement, semi-hard.
BK-2 drilling-hole	14.05	sandy mudstone: dark gray, contain rich plant fossil fragments, mica debris, with horizontal bedding, flat-shaped fracture, argillaceous cement, sandwiched thin siltstone, semi-hard.

Among the four drilling-holes, the BK-1 drilling-hole locates beside the air return entry, about 500 m from the open-off cut. It is the nearest drill hole to the hydraulic fracturing test area, so strata lithology is mostly understood through the BK-1 drilling-hole. If information from the other three drilling holes are taken as a reference, the strata positions of the hydraulic fracturing can be determined.

With the mechanized mining method that is utilized, the mining height is 2.49 m. For safe production, the roof should cave timely. According to the principle that the gob must be filled with broken fallen rock from the roof, we arrive at a theoretical formula for the caving zone height *h*:

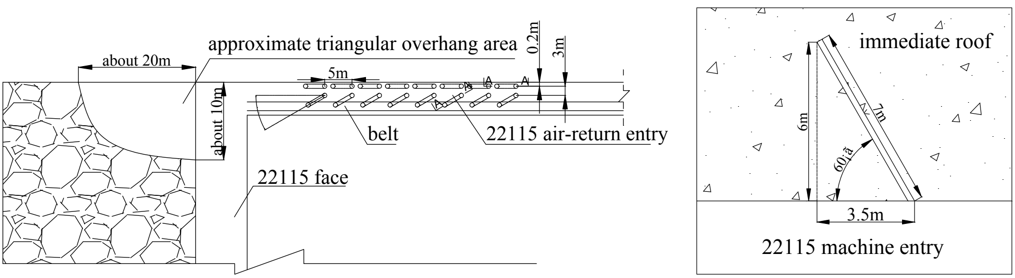
$$h = \frac{M}{K_z - 1}$$

Where *M* is the height of the coal seam, and K_Z is the average broken expansion coefficient of the immediate roof and the main roof.

Given $K_Z = 1.5$, then $h = 4.98$ m; otherwise, if $K_Z = 1.3$, then $h = 8.3$ m. Additionally, taking a combination of drillings from the geologic column, the hydraulic fracturing hole should be 7 m deep.

4.2. Arrangement of drilling-holes

The drilling construction plan for the construction of hydraulic fracturing holes is shown in Figure 8. Constructed using an anchor cable machine with a ф32 mm drilling bit, two drilling-holes were drilled every 5 m at the machine entry. All the drilling-holes were 7 m deep and built using an anchor cable machine with a drill-rig of ф32 mm diameter. The drilling-holes were made close to the pillar side which was inclined at 60° up towards the direction of the gob, 0.2 m from the pillar. The other rows of drilling-holes were 3.2 m from the pillar, inclined horizontally at 30° towards the entry wall, and inclined at 60° up in the direction of the gob. Seen from the direction that is perpendicular to the entry-wall, the sectional views of the two rows of drilling-holes are similar (Fig. 8b). The vertical height of the drilling-hole was 6 m. Taking crack extension into consideration, the control height is more than 8 m. In order to decrease the effect of the anchor cable holes on hydraulic fracturing, drilling-holes should be constructed as far as possible from the anchored cable holes.



(a) Plan view of drilling-holes (b) Sectional view of drilling-hole

Fig. 8. Layout of drilling-holes for hydraulic fracturing

4.3. Hydraulic fracturing equipment and technology

The hydraulic fracturing area lies at the machine entry of Face 22115. Drilling-holes can be drilled out of the advance support area ahead of time, while hydraulic fracturing must be carried out in advance reinforcing support area. The roof was cut into rock blocks by hydraulic fractures. The rock blocks of roof still squeeze bite each other because of the support systems of anchor cable + anchor bolt + steel strip and so on. The roof of the entry still has strong stability. It has been proved by pre-trial practice of hydraulic fracturing for the entry roof. Moreover, the advance reinforcing support can effective support the entry roof. A lot of practice has proved that the technology is safe.

The fracturing pump and other equipment are placed within the first crossheading, ahead of the face. The fracturing pump has a 63 MPa rated pressure and a 120 L/min rated flow. The hydraulic fracturing equipment and layout are shown in Figure 9. Using pine (pressure and flow resistant), the sealing device is sent to the designated position in the fracturing hole (Fig. 10). It is worth noting that the sealing length should be over 2 m to ensure a high level of pressure and safety. Once these procedures have been implemented, the fracturing pump should be turned on, initiating hydraulic fracturing. If the drilling-holes do not deform adequately, backward sectionalized fracturing techniques can be adopted to improve the fracturing effect.

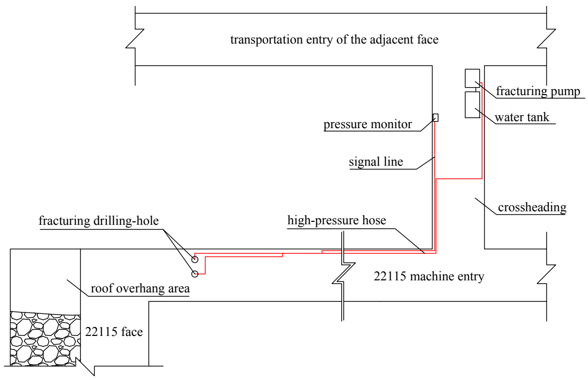


Fig. 9. Hydraulic fracturing equipment and its layout

Fig. 10. Hydraulic fracturing in underground coal mine

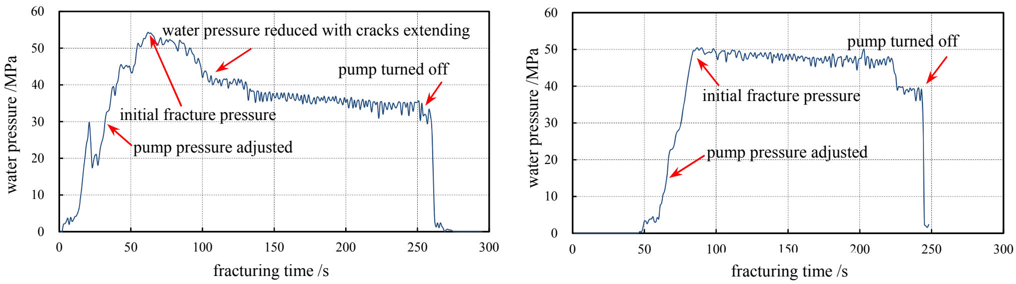
5. Testing process and effect analysis

5.1. Water pressure curve of hydraulic fracturing

A total of 18 drilling-holes were fractured. According to the data obtained from the underground hydraulic fracturing monitor, the initial fracture pressure was approximately 50 MPa. Where the roof is complete, the fracture pressure reached approximately 55 MPa. With the hydraulic fracturing cracks extending to the roof surface, water pressure was released, allowing the water pressure to decline gradually. Where roof joints is developed, the initial fracture pressure was found to be approximately 30 MPa. The general flow was recorded as being approximately 86.13 L/min~111.47 L/min. The representative drilling-hole fracturing is shown in Figure 11.

For drilling-hole 1, the pump pressure rose gradually as the fracturing began. The initial fracture pressure of the roof was 54.3 MPa. With cracks extending, water pressure declined gradually, eventually stabilizing at around 35 MPa. After stopping the pump, the water pressure was rapidly reduced to 0. The monitored flow was 91.34 L/min.

For drilling-hole 2, the initial fracture pressure of the roof was 50.3 MPa. As the cracks extended, the water pressure slowly declined. After turning the pump off, the water pressure rapidly reduced to 0. The flow was monitored at 89.14 L/min.



(a) Drilling-hole 1 (b) Drilling-hole 2

Fig. 11. Representative hydro-pressure curve for drilling-hole hydraulic fracturing

5.2. Morphology of hydraulic fracturing crack in the end-roof

The conditions surrounding the drilling-hole during hydraulic fracturing are shown in Figure 12. It is evident from the figure that a relatively large amount of water leakage has occurred around the anchor and anchored cables around the fracturing drill holes. The farthest water leakage anchor is 5.6 m distance from the drilling-hole. It indicates that cracks have extended to the roof surface or some cracks have link through the original fissures on the surface. In any case, cracks have extended to this surface position, so the crack propagation range is seen to be not less than 5 m from the fracturing drilling hole.

When hydraulic fracturing was completed, the pump was stopped and the roof was observed as having cracks extended to the surface of the roof (Fig. 13). As a result of in-situ stress, some cracks were noted as having different degrees of closure. Some cracks were so small that they

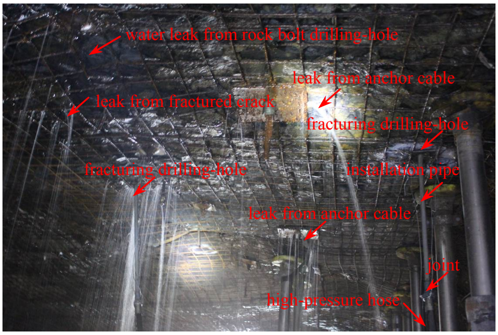
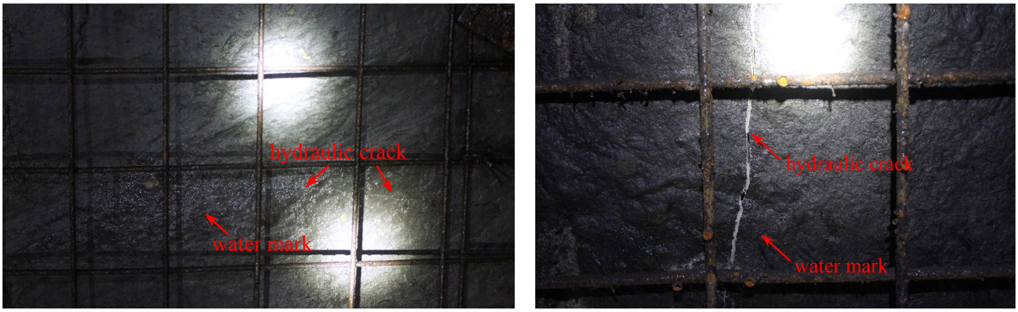


Fig. 12. Conditions surrounding the drilling-hole during hydraulic fracturing



(a) Fracture morphology

(b) Partially enlarged detail

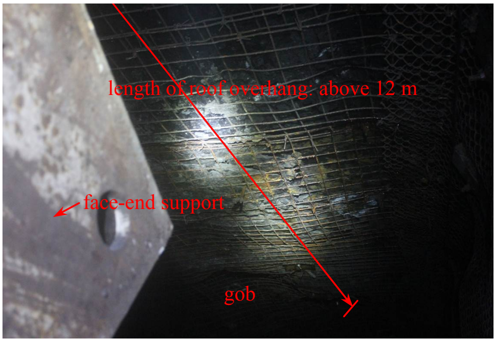
Fig. 13. Hydraulic fracture morphology on roof surface

were not immediately visible to the naked eye. However, it was noted that the cracks formed a weak plane, and the original roof structure had been transformed.

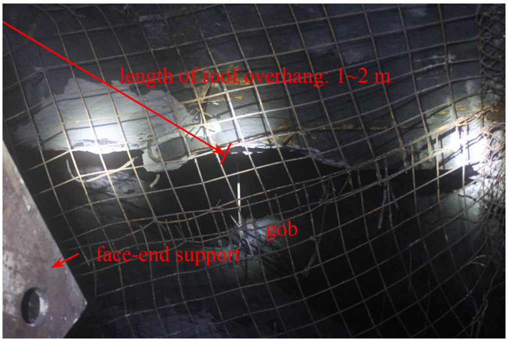
5.3. Control effect

After weakening the end-roof through hydraulic fracturing, the length of the hanging roof was cut off sharply from its original state of over 12 m to 1~2 m (Fig. 14), and the roof caved in time as the hydraulic supports were moved forward. As such, the potential gas risks caused by the suddenly falling of the hanging roof were eliminated.

As this application of hydraulic fracturing for the control of the hanging roof at the face end has proven successful, the technique has been widely applied in Buertai Mine, Cuncaota Mine, No. 2 Mine of Cuncaota, Liuta Mine, Wulanmulun Mine, Baode Mine of Shendong Mining District. As well, related techniques and equipment have also been applied in other mining districts including Datong Mining District and Yulin Mining District.



(a) Before hydraulic fracturing



(b) Effect after hydraulic fracturing

Fig. 14. Control effect

6. Conclusion

(1) A design for the control of the hanging roof at the face end through hydraulic fracturing is put forward. This design primarily employs the control method of cutting the roof along the coal wall and weakening the overall mechanical properties of the end-roof which is supported through the use of bolts and anchor cables.
(2) A set of hydraulic fracturing equipment for roof control were developed using anchor cables of a small diameter-size (ϕ32 mm). The fracturing water pressure reached above 60 MPa. Easy to master and operate, this technique can be applied industrially, and particularly in the field.
(3) For hard roofs, the crack extension range of every single fracturing borehole was not found to be less than 5 m. As a result, the controlled roof range was not less than 5 m.
(4) Tests and applications in many of the coal mines of Shendong Mining District indicate that the hanging roof length at the face end could be reduced from over 12 m to 1~2 m. This process would thus, eliminate the risk of a sharp rise in gas concentrations which could result from gas extrusion from the gob in the case that a large area hanging roof suddenly falls and caving at the face end.
(5) The small diameter-size of the borehole (based on the anchor drilling machine) hydraulic fracturing method can also be constructed between two supports in order to control the hanging roof in the middle part of mining face.
(6) In order to provide a standard for borehole arrangement and fracturing techniques for different roof conditions, further research is needed with respect to the three-dimensional extension rule of hydraulic cracks at the face end.

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