



Research on abutment stress distribution of roof-cutting coalface: numerical simulation and field measurement

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Abstract During the processing of deep mining, revealing the distribution of abutment pressure is significant for controlling stability of the entry. In this study, the abutment pressure distribution of roof-cutting coalface was investigated by FLAC3D and self-developed flexible detection unit (FDU). In the numerical simulation, the double-yield model was built to analyze the goaf abutment pressure under the fracturing roofs to maintain entry (FRME). Compared with the non-fracturing side, the peak value of the advanced abutment pressure on the fracturing side

is reduced by 19.29% on average, the influence range (span) increases by 30.78% and the distance between the peak value and the working face increases by 66.7%. The goaf abutment pressure within 23m near the cutting side is significantly higher than other areas along the dip. The FDU was employed in the coalface to record the change of advanced abutment stress. And the field measured results are in well agreement with the numerical results.

Article highlights

- (1) The abutment pressure distribution of the whole panel was investigated.
- (2) The double-yield model was built to analyze the abutment pressure of the goaf.
- (3) The abutment pressure distribution of the goaf after adopting the FRME was obtained.

Keywords Abutment stress · Fracturing roofs to maintain entry (FRME) · Flexible detection unit (FDU) · Double-yield model

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1 Introduction

The mining of the working face will cause the redistribution of abutment pressure, which is affected by many factors, e.g., the strength of coal body, the

strength and thickness of roof strata, the buried depth of working face, the fracture pattern of main roof, geological conditions and even the mining sequence of working face (Wang et al. 2020, 2024). Numerous progresses have been achieved by many scholars on theoretical model and measured results of the abutment pressure. And the abutment pressure can be obtained quickly and easily by applying the theoretical model, which can well serve current production of deep resources (Wang et al. 2021). Based on key strata theory and Winkler's elastic foundation beam theory, the calculation method of the abutment pressure for coal body was derived which is affected by the key strata of overburden (Han et al. 2019). According to strain energy balance principle and a rheological constitutive model of caved materials, Rezaei et al. (2015) presented a time-dependent energy model to calculate the mining-induced stress over gates and pillars. Wang et al. (2017) proposed a new analytical model to calculate the distance of cover stress re-establishment within the goaf. However, we found the above models and methods cannot comprehensively summarize the factors that affect the redistribution of abutment pressure. And above mentioned is not enough to characterize and calculate the evolution of the abutment pressure and over-simplification will inevitably lead to inaccurate theoretical results. Moreover, the application of the design formulas is based on the assumption that the abutment pressure is not affected by the location of the roadway. Therefore, it is limited to use the formulas for calculating the abutment pressure and its results just only can be referred.

In order to comprehensively reflect the influence of special geological conditions, excavation sequence and excavation speed of the working face, numerical method has achieved much attentions and various factors can be introduced and considered to investigate the critical condition and factor that affecting the abutment pressure under complex environments.

FLAC^{3D} was adopted to choose a reasonable position among the large coal pillar for excavating the external tail entry of the next working face, and finally an innovative approach of "double U" roadway layout was developed (Yan et al. 2013). Bai et al. (2015) established the FLAC^{3D} numerical model to study the abutment pressure change of roadway's solid coal side and ahead of the coal face under the technology of gob-side entry driving. Cheng et al.

(2010) explored the relationship between the abutment pressure of the coal pillars with different widths and the abutment pressure of the coal mass in front of the working face through numerical simulation. In order to research the feasibility of coal body recovery at the boundary of mining area in Svea Nord Coal Mine, Shabanimashcool and Li (2013) established a 3D global model to explore the stress state of the mining area boundary after the mining of all working faces in the mining area was completed. Yao et al. (2015) analyzed the effects of the thickness and strength of immediate roof, the strength of coal body and buried depth of working face on lateral abutment pressure through numerical simulation. Shen (2014) evaluated the roadway abutment stress and deformation under different roof support designs through UDEC software and developed a new method of roadway support.

Nevertheless, numerical simulation just can obtain rough results, which cannot fully reflect the actual situation on the spot. In order to get more accurate results, it is necessary to cooperate with on-site monitoring. Shen et al. (2016) installed borehole stress meters in the coal pillar of the roadway to monitor its vertical stress with the advance of the working face under the condition of the hard roof of the thick coal seam. Shen et al. (2008) used vibrating wire stress meters at different depths of the roadway roof to monitor horizontal stress changes parallel to the roadway axis, perpendicular to the roadway axis, and 45°. Zhang et al. (2014) introduced vibrating wire borehole stress meters to research the advanced abutment pressure of the panel, it can be found that the vertical stress in front of the coalface was much larger than the horizontal stress. Wang et al. (2008) also investigated the variation laws of lateral abutment pressure with the mining and overburden movement of the working face through the monitoring of lateral abutment pressure of solid coal by borehole stress meters and the monitoring of roadway roof displacement by dynamic roof monitor apparatus. Chang (2011) explored the distribution laws of lateral abutment pressure and advanced abutment pressure in fully mechanized top-coal caving face through borehole stress meters. Singh et al. (2011) conducted on-site monitoring of the vertical pressure of the coal pillar in different working faces in India, and established an empirical relationship to estimate the magnitude and influence range of the vertical stress on

the coal pillar. Stas et al. (2011) designed a conical device for measuring the deformation at the bottom of boreholes, which can be used to observe the stress changes caused by underground mining activities for a long time. In addition to the conventional borehole stress meter, some innovate technologies and equipment (e.g., micro-seismic method Shen et al. 2008, electromagnetic radiation method Xu et al. 2012, passive seismic velocity tomography Hosseini et al. 2012, 2013; and improved borehole stress meter Ouyang et al. 2009) also have been applied to the field of abutment pressure monitoring. These new technologies and equipment have made outstanding contributions to abutment pressure monitoring.

From what have been discussed above, the research of abutment pressure is mainly concentrated in the traditional mining field, there are few studies on the abutment stress of the FRME. At the same time, it is easy to produce large error when measuring with rigid stress meters. In this study, abutment pressure was measured in the FRME with self-developed and more reliable FDU, and then the influence of cutting seam on the abutment pressure of solid coal is explored. Finally, the abutment pressure distribution of the whole stope will be analyzed by conducting the numerical researches.

2 Principle of measurement

2.1 Principle of the FDU

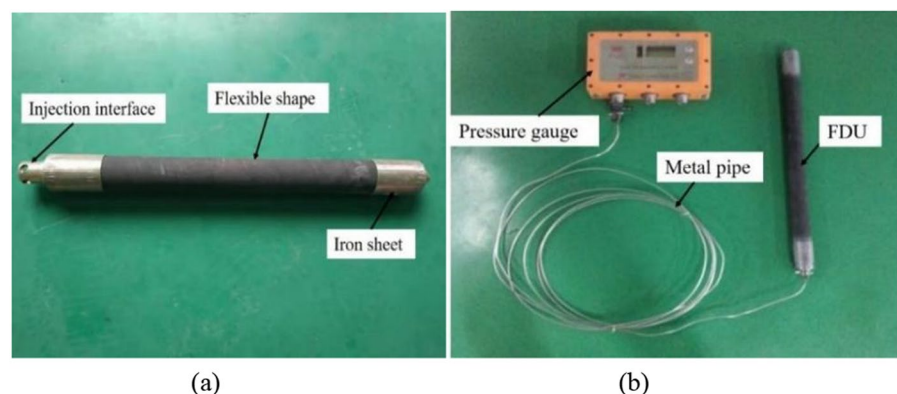
The flexible detection unit is consisted of iron sheet, injection interface, plug and flexible shape (thin steel wire and polymer material composition), as

shown in Fig. 1a. The measurement range of the FDU can reach 60 MPa and its accuracy is from 0.5 to 1.0% FS. The length of the FDU is 500mm and its diameter is 45 mm. The FDU is connected with the pressure gauge through a metal pipe to record the pressure of the FDU in time, as shown in Fig. 1b.

The change of abutment pressure is the radical cause of all kinds of strata behaviors. At present, the equipment used to monitor abutment pressure (i.e., borehole stress meter) is mainly made of rigid materials, and its disadvantages are as follows:

- (1) Equipment made of rigid materials cannot fit closely with the borehole wall and cannot change with the shape of the borehole, so the stress change of borehole surrounding rock cannot be monitored sensitively, and the monitored data is short of accuracy.
- (2) Monitoring equipment made of rigid materials can only monitor the stress change in fixed direction, but cannot monitor the change of horizontal stress and vertical stress simultaneously. In order to monitor the stress change in two directions of one borehole, it is necessary to install another pressure gauge in the borehole. However, if two pressure gauges are not in the same position, they will inevitably produce data deviation.
- (3) It is difficult to control the direction of the borehole stress meters during the installation of them. Hence, the existing rigid equipment can not accurately, timely and comprehensively reflect the stress change of the surrounding rock, causing us cannot accurately grasp the movement state of the borehole surroundings.

Fig. 1 The picture of the FDU **a** real product picture of the FDU, **b** schematic diagram of connection between FDU and pressure gauge



Based on the study of above three issues, we invented the FDU. The FDU gives an approved setting load that is equal to the in-situ rock stress to the borehole surroundings, which actively applies pre-stress to the borehole. And the FDU changes from “passive” to “active” monitoring, completely eliminating the malpractice that there is no measurement data caused by the installation gap of rigid materials during the elastic deformation phase of coal mass, as shown in Fig. 2a. Meanwhile, because the FDU is fully coupled with the borehole surrounding rock, it can move in coordination with the borehole surrounding rock. Detection area of the FDU is the whole borehole wall rather than a single point, so the FDU can detect the whole process of surrounding rock from elastic deformation to plastic deformation, as shown in Fig. 2b. The FDU effectively solves the problem of data distortion caused by

single-point and one-way test of rigid materials in the plastic deformation phase of coal body.

To verify the precision of FDU in the laboratory experiment, pressure sensor is set into the surface of the FDU, and then emulsion at a specific pressure value is injected to the FDU by manual pump to make the FDU squeeze sensor, which will give the pressure readings of the FDU. The injection pressure is compared with the reading of the sensor to analyze the practicality of the FDU. The total testing process is shown in Fig. 3. In previous researches (Guo et al. 2021), similar experimental results show that the readings of the manual pump coincide with those of the sensor in Fig. 4, and the abutment pressure of coal by the FDU is successfully tested.

The emulsion is injected into FDU through a manual pump to make the pressure reach 5.25 MPa. After the FDU expanding, it will closely fit the surrounding rock of the borehole. As a result, when

Fig. 2 The deformation process of different materials for drilling hole **a** the deformation process of rigid material for drilling hole; **b** the deformation process of flexible material for drilling hole

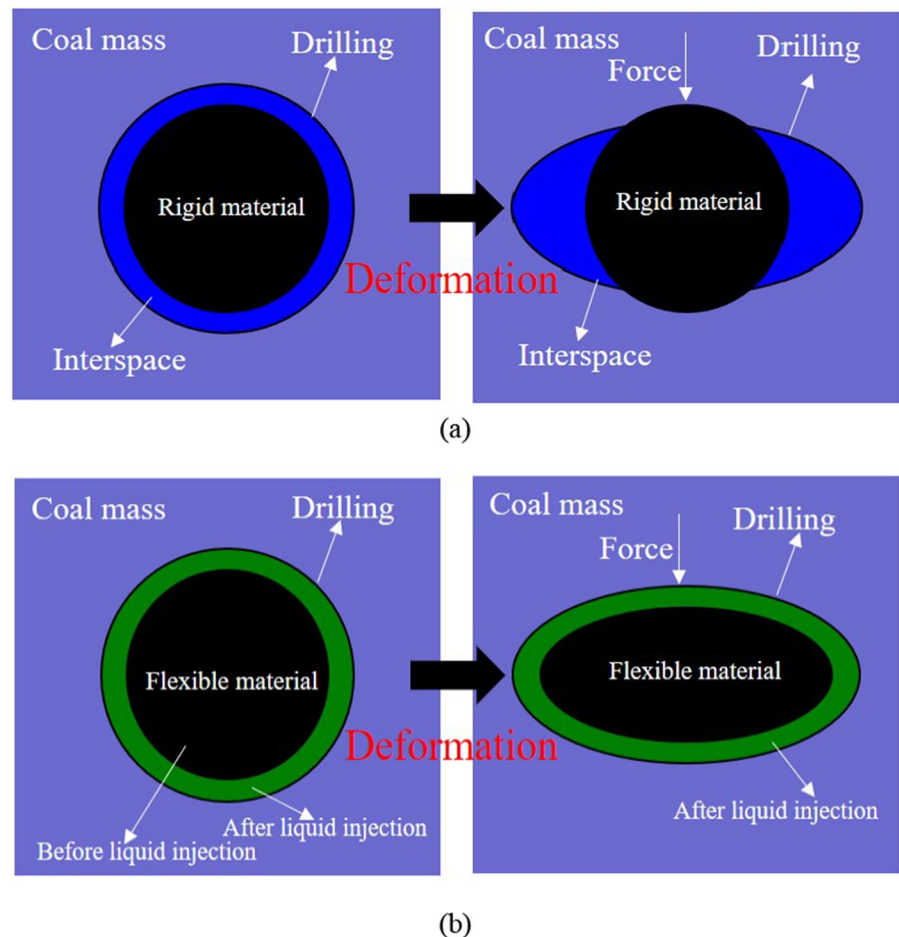


Fig. 3 FDU testing process **a** before testing, **b** after the test

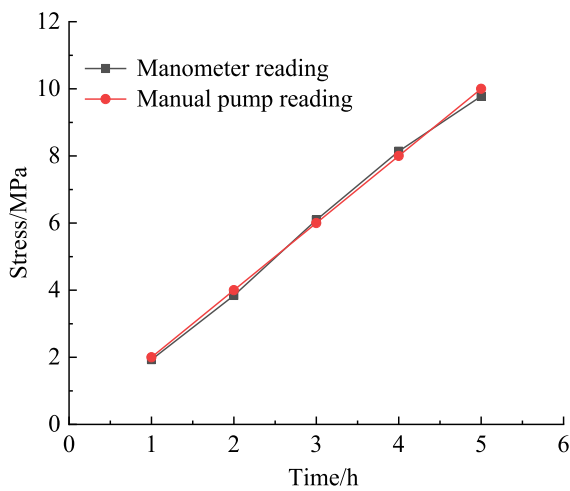
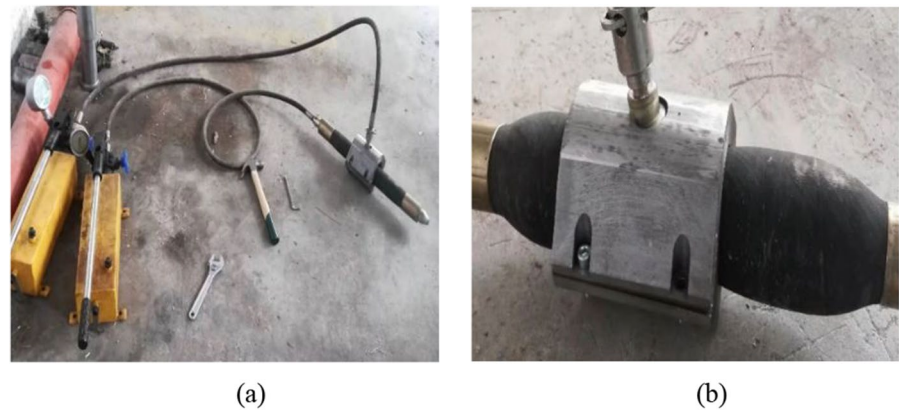


Fig. 4 Comparison between manometer reading and manual pump reading

the surrounding rock of the borehole is deformed due to the increase of abutment pressure, the coal body will squeeze the FDU, causing the fluid pressure inside the FDU to increase. At the same time, the pressure changes are sensed by sensors in the pressure gauge and shown in the gauge. The pressure value of the FDU will be transferred to the transmission substation through wireless communication, and then will be uploaded to the ground computer via optical cable. After receiving the data, the computer displays the pressure value of the coal body graphically through the data processing software, and then transmits the information to the monitoring and early warning center by satellite. The sketch map of monitoring system of the FDU is illustrated in Fig. 5.

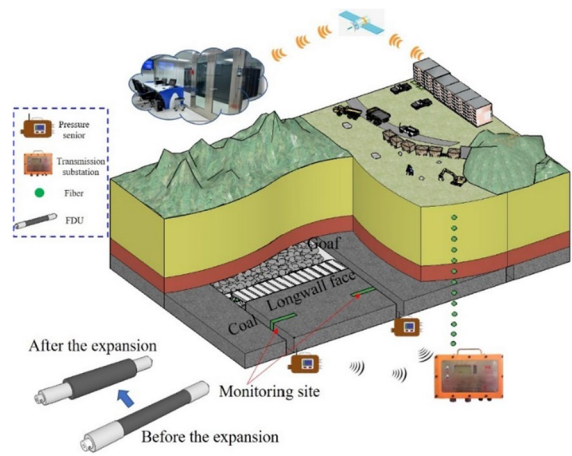


Fig. 5 Sketch map of monitoring system of the FDU

2.2 Working principle of the FRME

Before the mining of the coalface, the constant resistance large deformation (CRLD) anchor cables were constructed in the roadway to strengthen the roof, so as to prevent the disturbance of the roadway roof caused by the subsequent roof split blasting and the roof collapse of the goaf. Then the construction of the blasting hole is carried out on the goaf side of the roadway roof according to the designed height, angle and distance between the blasting holes. In the concrete operation, the cumulative energy tubes, emulsified blasting powders, detonators and stemming were put into the blasting holes sequentially to form a continuous slit line along the axis of the roadway. After the mining of the coalface, the goaf side of the roadway roof will automatically collapse along with the cutting seam under the action

of mine pressure and the gangue with different sizes and shapes in the process of roof collapse will be formed. Meantime, for preventing the gangue from rushing into the roadway in the course of falling and compaction, U-type steels, metal meshes and single hydraulic props were adopted to block the gangue, so that the compacted gangue can be used as gangue rib to continue to serve for the mining of the next panel (see Fig. 6).

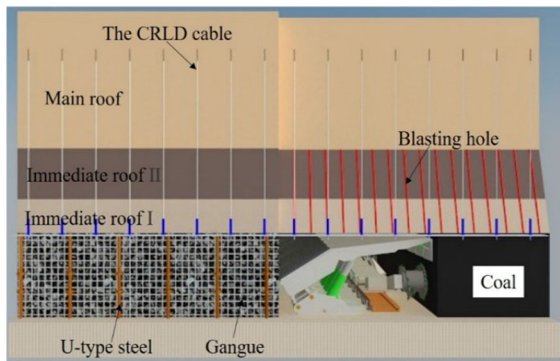


Fig. 6 Side view of the FRME

3 Research on abutment pressure by field measurement

3.1 Engineering geological conditions

The Lvtang coal mine is located in Guizhou Province, China, whose geographical location is shown in Fig. 7a. The average buried depth of S204 working face, the strike length and the dip length are 210 m, 310 m and 115 m, respectively. The retained roadway is the tail entry of S204 working face, which will serve for the mining of S205 working face that displayed in Fig. 7b.

The size of the entry is 4.2 m × 3.0 m in width and height. The original supporting measure adopts bolt and common anchor cable. The specification of the bolts is $\phi 20 \times 2200$ mm and the spacing is 800 × 800 mm. The specification of the common anchor cables is $\phi 15.24 \times 6000$ mm and the spacing is 1600 × 1600 mm. Two rows of the CRLD cables were arranged in the entry, perpendicular to the roof and along the entry lengthwise. The specification of the CRLD cables is $\phi 21.8 \times 10,000$ mm and the spacing is 1600 × 1600 mm. The length of the cutting line is 8000 mm, with a deviation of 15° toward goaf. The average thickness of coal is 3.0 m. The immediately floor is sandy mudstone and coal, with a thickness of 3.2 m. The basic floor is muddy sandstone, with a thickness of 4.5 m. The immediately roof of the coal seam is sandy mudstone and coal, with a thickness

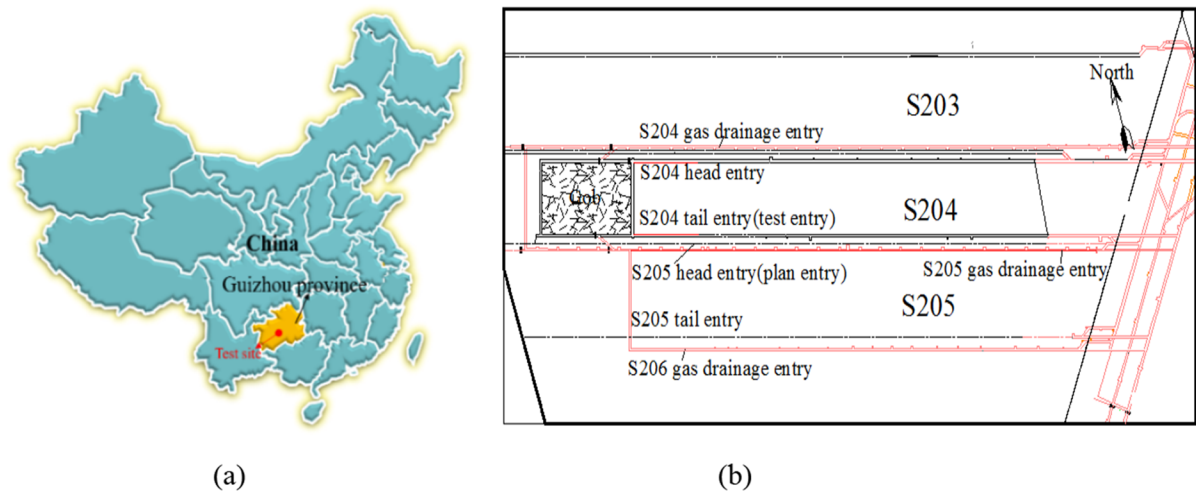


Fig. 7 The location and layout of Lvtang coal mine, **a** Lvtang coal mine location; **b** plan view of S204 working face layout

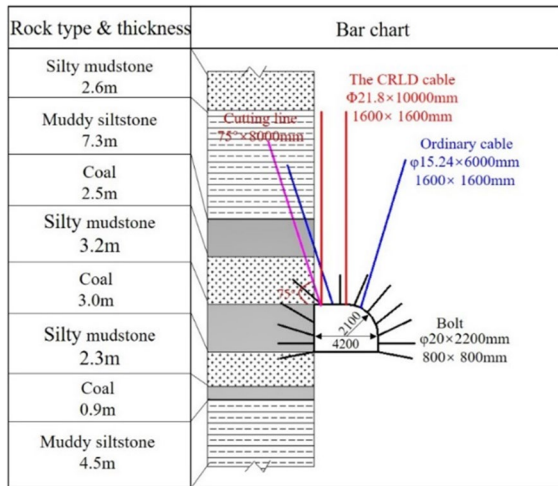


Fig. 8 The geological drilling column of working face

of 5.7 m. The basic roof is muddy sandstone, with a thickness of 7.3 m. The geological drilling column of working face is shown in Fig. 8.

The original S204 tail entry is straight wall semi-circular arch. During the production of panel mining, the panel side of the entry roof would form triangle areas and then the suspended roof will be caused. Hydraulic props will provide a force to the suspended roof, that cannot contact the entry roof, so that the deformation of entry roof is serious. Therefore, the panel side of straight wall semicircle arch entry will be transformed from rectangle to eliminate triangle areas before mining and hydraulic props can timely and effectively support the roof to reduce the deformation of the entry.

3.2 Field measurement and data analysis of the FDU

In order to clarify the difference of advanced abutment pressure on the fracturing and non-fracturing sides of coalface, two groups of measuring stations were arranged on both sides of the coal body of the working face, and each group of measuring stations was installed at 100 m in front of the coalface. Three sets of the FDU at different depths were arranged in each group, and the depths of FDU are set as 3, 6 and 9 m. The installation diagram of the FDU is shown in Fig. 9.

As shown in Fig. 10, the variation curve of the advanced abutment pressure for coal mass on the

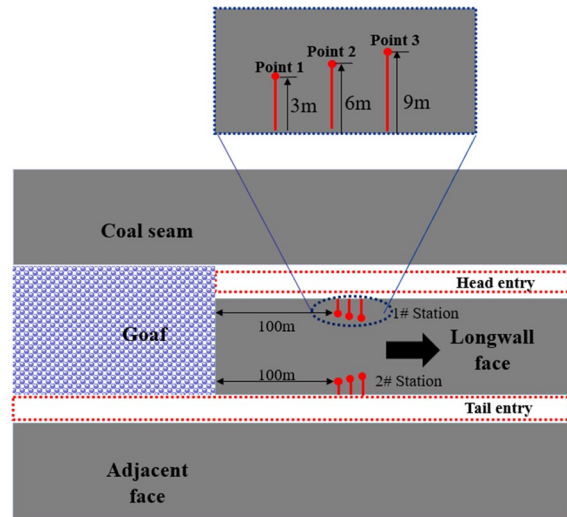


Fig. 9 Installation diagram of the FDU

slotted side and non-slotted side of panel with the mining of longwall face.

It can be seen from Fig. 8 that after pre-splitting the entry roof, compared with the non-slotted side of longwall face, there are obvious characteristics that found for the advanced abutment pressure of the coal body on the slotted side of longwall face, which are expressed as follows:

- (1) The trend of the advanced abutment pressure curve is more stable. There is only a small amount of up-and-down vibration of the advanced abutment pressure curve on the slotted side of working face. Meanwhile, the vibration amplitude of the advanced abutment pressure of coal mass at each gauging point is significantly smaller than that on the non-slotted side of longwall face (see Fig. 8a). Significant and variable frequency vibration occurs on the non-slotted side of longwall face in the advanced abutment pressure curve (see Fig. 8b). The appearance of vibration may be related to the fracture of the overlying strata which powerfully indicates that the movement state of overlying rock has changed after pre-splitting the entry roof, and then the impact of dynamic loads on the coal and roadways is reduced (Jiang et al. 2017; Sinha and Walton 2019).
- (2) The advanced abutment pressure is reduced. The abutment pressure of the coal body at each

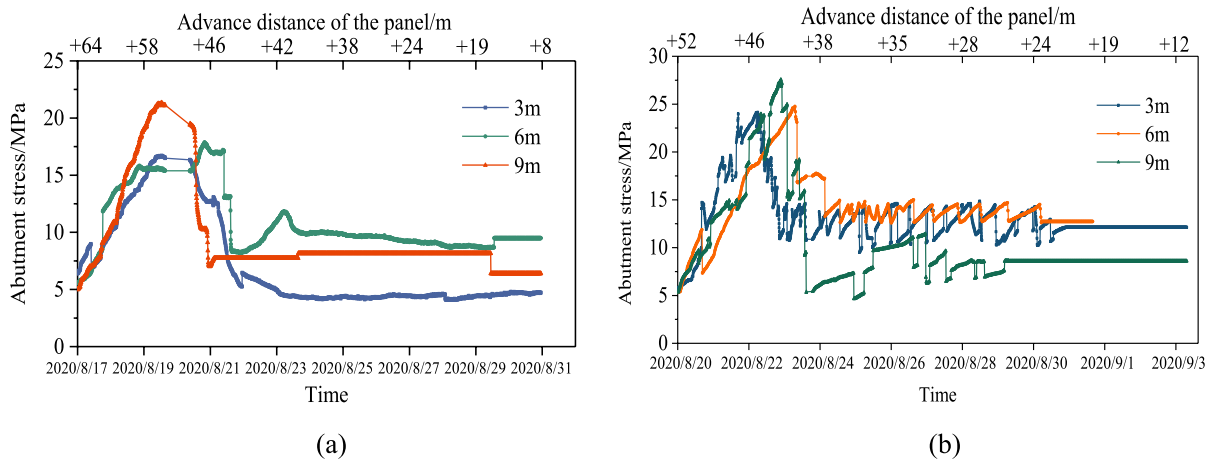


Fig. 10 The variation curve of the advanced abutment pressure of coal mass with the mining of longwall face. **a** Advanced abutment pressure curve of coal mass on the slotted side; **b** advanced abutment pressure curve of coal mass on the non-slotted side

measurement point on the slotted side of the panel is significantly lower than that of the non-slotted side of the panel. The advanced abutment pressure of the coal at 9 m on the slotted side is approximate 22.7% lower than that on the non-slotted side, and the advanced abutment pressure at 6 m and 3 m on the slotted side is 27.7% and 30.8% lower than that on the non-slotted side respectively. This consequence shows that the pressure relief effect of the slotted side is very obvious, which is conducive to maintaining the stability of the roadway on the slotted side. Due to the limitations of the current technical conditions, it is impossible to measure and study the abutment pressure in deeper coal body. It is certain that the influence range of the cutting fissure is limited and the state of the abutment pressure in the whole panel cannot be altered completely. Because the height of the cutting fissure is limited, it can only change the movement state of the roof within the range of the slotted, and cannot influence the higher overburden. It is meaningful to determine the length of “fracturing-affected area”, hence, following studies in this area should be strengthened.

(3) The influence range of the advanced abutment pressure goes up. The measured data shows that the rising of abutment pressure at each measuring point on the slotted side begins to be measured at 64 m ahead of the longwall face, while the change of abutment pressure at each measur-

ing point on the non-slotted side is measured at 52 m in front of the working face, which can be accounted for after pre-splitting and cutting the roof, the movement state of the overlying strata on the panel roof has changed, while the movement state of overburden at the unslotted side of longwall face remains unchanged.

(4) The abutment pressure changes significantly at 3 m. The abutment pressure at this place first rises rapidly and then decreases sharply and finally stabilizes at 5 MPa (the initial pressure of the equipment). The increase of abutment pressure is affected by coal mining. When the abutment pressure reaches to the yield limit of the coal mass, the coal mass will be broken and the abutment pressure will decrease rapidly. At this moment, the load-bearing capacity of the coal will decline, and there are slight expansions on the goaf side of the roadway, which has none influence on the normal use of the entry. The coal body on the non-slotted side of the working face is affected by abutment pressure of a certain amplitude, which cannot reach to the yield limit of the coal mass. Hence, the gateway will remain normal. Through the investigation of the actual conditions on the spot, it is found that there is obvious water gushing phenomenon on the slotted side of panel in the examination area, while the non-slotted side of panel is normal. The strength of coal mass on the slotted side of panel will decline after it encounters water, hence, the

coal mass at 3 m is damaged under the influence of advanced abutment pressure.

4 Study on abutment pressure by numerical simulation

4.1 Introduction of global model

The calculation model is spatially divided into three directions, X, Y and Z, where X, Y and Z are inclination direction of the panel, mining direction of the panel and the gravity direction respectively, and the upward direction is positive. Since S204 working face belongs to nearly flat coal seam, it can be simplified as flat stratum for simulation. Previous researches also indicate this simplified method was used to perform numerical calculation (Gao et al. 2019; Li et al. 2018). The dimensions of the whole model in X, Y and Z directions are 244 m, 160 m and 50 m respectively, and X and Y constitute the

horizontal plane. The trellis of three-dimensional calculating model is shown in Fig. 11. The model adopts the Mohr-Coulomb criterion and the load applied on the upper surface is 5.25 MPa to simulate the weight of the overlying rock. The buried depth of the working face is shallow and there is no structural zone, the coal body is almost not affected by horizontal stress, so the displacement constraint is adopted around and at the bottom of the model. The fully constrained displacement boundary conditions on the bottom surface and surrounding areas were adopted to restrict the movement of model. The mechanical parameters of each rock mass are shown in Table 1. Among them, the parameters of coal body and roof rock are obtained by GSI rock mass classification method on the basis of rock mechanics parameters obtained by laboratory uniaxial compression test and field borehole peep to estimation GSI value. The parameters of floor sandy mudstone are obtained by GSI rock mass classification method based on the measured mechanical parameters of coal mine exploration.

According to the specific implementation steps of the FRME, the entire process of simulation calculation is divided into 4 steps. Step 1: the numerical calculation model is established and the calculation is carried out until the balance is reached. Step 2: the roadways are excavated, and then anchor cables and bolts are constructed to support entry surroundings. The calculation is carried out until the balance reaching to simulate the entry excavation. Step 3: the fracturing line along the positive direction of the Y axis is excavated to simulate the roof cutting. Step 4: by gradually excavating the working face along the positive direction of the Y axis, the required simulation results of the longwall face stress are extracted, and then screened, sorted and analyzed.

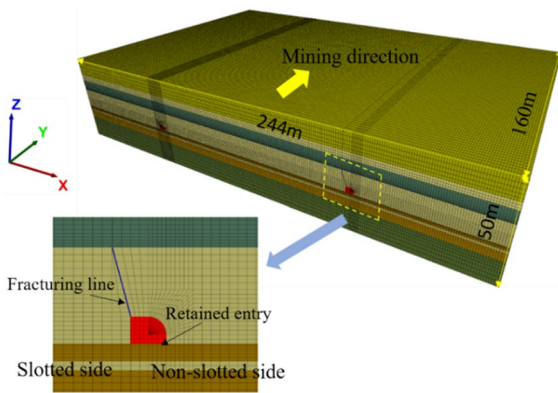


Fig. 11 The trellis of three-dimensional calculating model

Table 1 Mechanical parameters of rock mass

Lithology	Density (kg/m ³)	Tension (MPa)	Cohesion (MPa)	Friction angle (°)	Bulk modulus (GPa)	Shear modulus (GPa)
Coal	1550	2.35	1.45	25.2	1.31	1.44
Muddy siltstone	2240	1.43	3.07	32.0	6.47	4.33
Silty mudstone	2450	3.52	2.18	35.1	4.16	7.4
Splitting face	2046	2.43	2.23	30.7	3.98	4.39

4.2 Presentation of gob model

Numerous achievements related to the stress change of the surrounding rock through numerical simulation after the coal seam is mined have been obtained, which greatly push the development of coal industry (Jiang et al. 2016; Li et al. 2015; Wu et al. 2015). However, in the process of numerical simulation, a great many researchers ignored the load-bearing capacity of gangue body in the goaf and believed that the weight of the overlying strata is completely borne by the coal body in front of the working face and coal pillars on both sides of the working face, which will lead to a higher degree of stress concentration in the surrounding rock. As a result, there is a certain deviation between the current researches and the reality (Zhang et al. 2018). Due to the special position of the goaf, it is difficult to comprehensively measure its internal pressure. Hence, the numerical simulation has become an effective method to research the stress changes in the goaf (Wang et al. 2018). In this paper, the double yield model is used to simulate the goaf materials.

The parameters of the double-yield model are composed of cap stress and material properties. To determine the cap stress, the stress-strain relationship of the material needs to be obtained through the following formulas (Shabanimashcool and Li 2012; Yavuz 2004), which can be expressed as follows:

$$\sigma = \frac{E_0 \varepsilon}{1 - \varepsilon / \varepsilon_m} \quad (1)$$

$$E_0 = \frac{10.39 \sigma_c^{1.042}}{b^{7.7}} \quad (2)$$

$$\varepsilon_m = \frac{b - 1}{b} \quad (3)$$

$$b = \frac{H_c + h}{H_c} \quad (4)$$

where σ is the uniaxial stress applied to the goaf material (MPa), ε is the occurring strain under the applied stress, E_0 is the initial tangent modulus (MPa), and ε_m is the maximum possible strain of the bulked rock material. σ_c is the compressive strength of the gangue within the goaf, b is the initial bulking

factor. H_c is the height of the caved zone (m). h is the mining height (m).

According to the engineering geological conditions of Lvtang coal mine, the mining height is 3 m, the height of caving zone can be determined to be 8 m that can be obtained by field measurement, and $\sigma_c = 30$ MPa is determined which can be obtain by laboratory test. Accordingly, b , E_0 and ε_m are calculated to be 1.37, 31.84 and 0.27 MPa, respectively.

Finally, the stress-strain relationship of the goaf material that meets the actual conditions of Lvtang coal is obtained:

$$\sigma = \frac{31.84 \varepsilon}{1 - \varepsilon / 0.27} \quad (5)$$

The cap stress of the double yield model is illustrated in Table 2.

A sub-model of 1m in length, width and height is established. Displacement constraints are imposed on the periphery and bottom of the model and its top is subjected to simulation loading at a speed of 10^{-5} m/s. The comparative results between numerical model and Salamon's model are shown in Fig. 12. The final parameters of the double yield model are obtained by repeated iterative calculation in Table 3.

4.3 Monitoring line arrangement

For more comprehensively studying the abutment pressure of coalface after using the technology of the FRME, the model when the coalface advances to 80m is selected as analyzed example and 19 monitoring lines were set to quantitatively describe the abutment pressure of stope face. Among them, the 1–5# monitoring lines with a spacing of 5m were located

Table 2 Cap pressures for the double-yield model

Volumetric strain (%)	Stress (MPa)	Volumetric strain (%)	Stress (MPa)
0	0	0.14	9.258
0.02	0.688	0.16	12.504
0.04	1.495	0.18	17.194
0.06	2.456	0.20	24.562
0.08	3.620	0.22	37.826
0.10	5.057	0.24	68.774
0.12	6.877	0.26	223.517

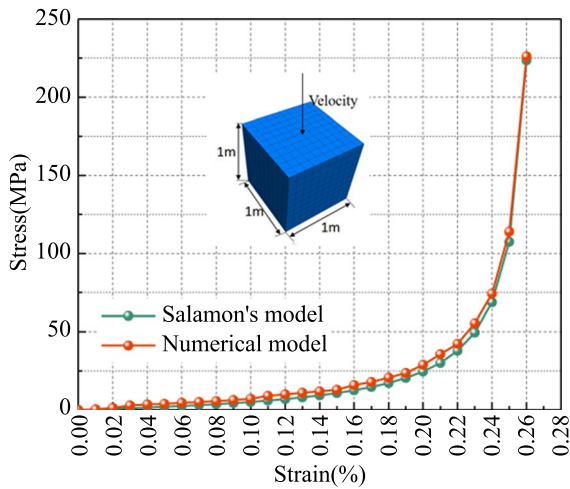


Fig. 12 Gob stress–strain curves of numerical and Salamon’s models

Table 3 Material parameters of the gob model

Bulk modulus (GPa)	Shear modulus (GPa)	Density (kg/m ³)	Friction (°)	Dilation (°)
9.56	7.25	1000	22	6

in front of coalface, which are used to explore the lateral abutment stress at different distances ahead of coalface. The 6~10# monitoring lines were located at the back of coalface. Since it takes a certain amount of time for the gangue to break down and compact, the spacing of the 6~10# monitoring lines were set to 10m, which is used to analyze the lateral abutment stress at different distances behind the coalface. The 11–13# monitoring lines were used to evaluate strike advanced abutment pressure on the non-slotted side of coalface, and the interval is 5m. The 14–16# monitoring lines were used to research strike advanced abutment pressure on the slotted side of coalface, and the interval is 5 m. The 17–19# monitoring lines were used to investigate the strike abutment pressure in the goaf. The 17# and 19# monitoring lines were 10m away from the edge of the gateway respectively, and the 18# monitoring line was located in the middle of the goaf. The location diagram of monitoring lines is illustrated in Fig. 13. Omni-directional stress monitoring of the stope face is also to explore the difference of abutment pressure distribution about the

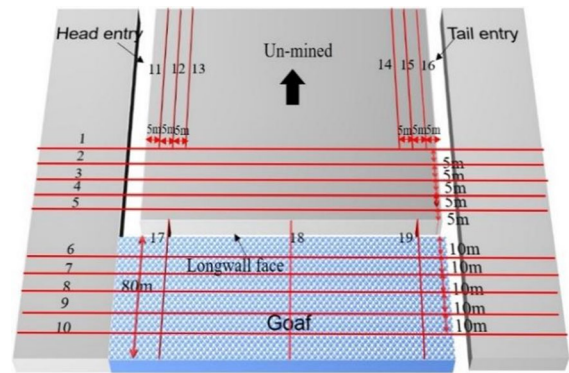


Fig. 13 Location diagram of monitoring lines

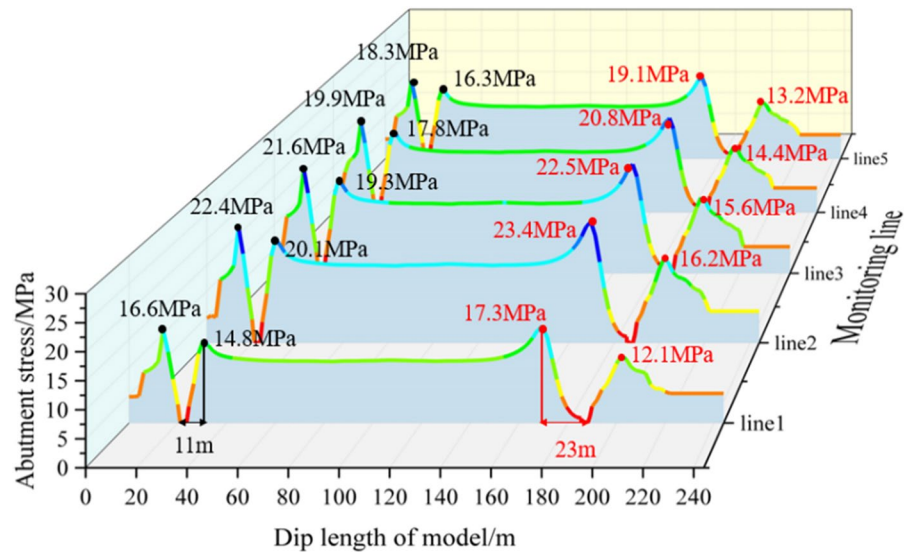
slotted and non-slotted sides of longwall face, and to further guide the field application of the FRME.

5 Research on inclined abutment pressure in front of coalface

The inclined abutment stress curves at different distances in front of coalface when the stope face advances to 80m are shown in Fig. 14. From this diagram, we discovered that the trend of each lateral abutment pressure curve at different distances ahead of coalface is same. There is an obvious stress concentration area in the coal body in the front of the working face, and the advanced abutment pressure presents an asymmetric saddle shape. The peak value of advanced abutment pressure on the slotted side of longwall face is higher than that on the non-slotted side of longwall face, which may be related to the collapse sequence of the roof. The roof of slotted side bends downward greatly under the action of advanced abutment pressure, while the roof far away from the slit is less affected by the cutting fissure and has a little downward bending. The difference of bending moment makes the roof far away from the slit accumulate energy, so that the abutment pressure in the depths of the coal body is high.

The distance between the roadway and the peak position of the advanced abutment pressure on the slotted and non-slotted sides of coalface is different. Numerical simulation results prove that the peak position of the advanced abutment pressure on the non-slotted side of coalface is 11 m away from the head entry, while the peak position of the advanced

Fig. 14 Inclined abutment stress curve at different distances in front of coalface



abutment pressure on the slotted side of coalface is 23 m away from the tail entry, e.g., the range of ‘fracturing-affected area’ is 23 m, which explains the above phenomenon that the peak value of the advanced abutment pressure on the slotted side of coalface increases while the surrounding rock conditions of the roadway is improved. This phenomenon indicates that the influence range of the slit is limited and the abutment pressure just can be reduced in a certain range. Hence, the state of the abutment pressure in the whole working face cannot be altered, that is, the slit can only change the movement of overlying rock in a limited range. A clear understanding of above problem is valuable to master and apply the technology of the FRME.

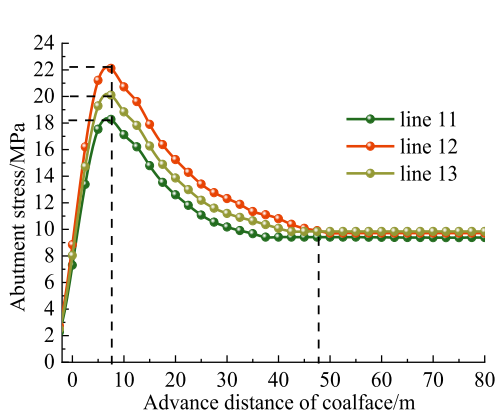
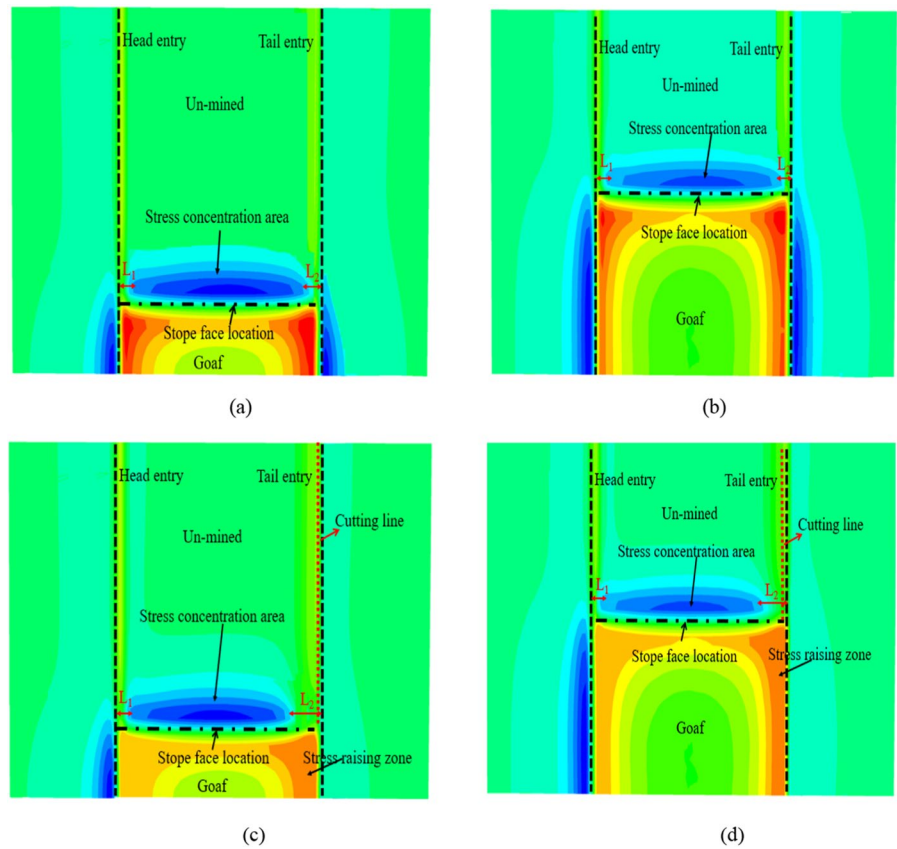
Meanwhile, although the variation tendency of the advanced abutment pressure on the solid coal side of two roadways is the same, the peak value is different. The peak value of the advanced abutment pressure of the solid coal in the non-slotted entry is higher than that in the slotted entry, which can be interpreted the cutting fissure can cut off the relationship between the goaf roof and the roadway roof, and then the effect of impact load in the goaf on the solid coal of the gateway is alleviated or even eliminated. Hence, the abutment pressure of solid coal on the slotted entry is reduced.

It is illustrated in Fig. 13, with the mining of the working face, the stress concentration area is formed in front of the coalface. However, the stress

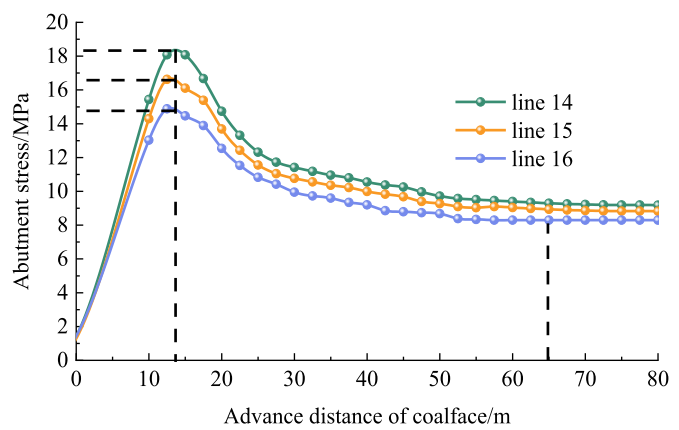
concentration area presents the characteristic of asymmetric distribution (see Fig. 15c, d), which is obviously different from the symmetrical distribution of traditional coal pillar mining (see Fig. 15a, b).

It is worth noting that the stress concentration area of advanced abutment pressure acts on the internal coal mass of coalface, and there is still a certain distance from the edge of the entry. In addition, we also discover that the peak value of advanced abutment pressure is different when the coalface advances to different distances. The peak value of the advanced abutment pressure is 20.2 MPa with the panel advancing to 40 m, and the peak value of the advanced abutment pressure is 18.1 MPa with the panel advancing to 80 m, which is reduced by about 10.5%. This is because after the coal seam is mined, the roof will failure and cave in. It will take a certain time for the gangue to completely fill the goaf. Hence, the contact between the overlying strata and the gangue also tends to lag behind the coalface at a certain distance. During this period, the coal mass of the coalface withstands more weight of the overlying rock. When the coalface continues mining, compacted gangue will support overburden rock, at this time, the weight of the overlying rock borne by the coal of coalface will be reduced, so the advanced abutment pressure will be reduced.

Fig. 15 The vertical stress distribution diagram of the coalface floor under different stopping distances. **a** Stress distribution diagram of conventional mining with the face advancing to 40 m; **b** stress distribution diagram of conventional mining with the face advancing to 80 m; **c** stress distribution diagram of the FRME with the face advancing to 40 m; **d** stress distribution diagram of the FRME with the face advancing to 80 m



(a)



(b)

Fig. 16 Strike advance abutment pressure curve of coalface. **a** strike advanced abutment pressure distribution curve on the non-slotted side; **b** strike advanced abutment pressure distribution curve on the slotted side

6 Research on strike advanced abutment pressure

The strike advanced abutment pressure curves are drawn by extracting the data of the 11–16# monitoring lines, as shown in Fig. 16. Figure 16 shows that the trend of advanced abutment pressure on the slotted side and non-slotted side of coalface is the same which shows the change laws of increasing first and then decreasing. The main effects of the slit on the advanced abutment pressure are as follows:

- (1) The peak value of advanced abutment pressure is reduced: In Fig. 16a, the peak values of advanced abutment pressure in the strike of the coalface at lines 11–13 are 18.3 MPa, 22.1 MPa and 20.1 MPa respectively, while those at lines 16–14 in Fig. 16b are 14.5 MPa, 16.1 MPa and 18.1 MPa, respectively. The peak values at the same distance away from tail entry and head entry are reduced by 20.77%, 27.15% and 9.95%, respectively, with an average decrease of 19.29%, which shows that pressure relief effect of the slit is remarkable and the surrounding rock condition of the entry is obviously optimized.
- (2) The influence range (span) of advanced abutment pressure is enlarged: The influence range (span) of the advanced abutment pressure on the non-slotted side is about 49 m, and it on the slotted side is about 65 m that increases by 30.78%. However, the advanced abutment pressure on the slotted side of coalface decreases and becomes smooth and steady, which is conducive to the stability and maintenance of the entry.
- (3) The peak position of the advanced abutment pressure is far away from the coalface: The peak value of the advanced abutment pressure on the non-slotted side is 7.5 m away from the coalface, and it on the slotted side is 12.5 m away from the coalface, with an increase of 66.7%. The peak value is far away from the working face and drops, so that the pressure of the coal mass on the slotted side of working face is smaller, which is conducive to the stability of the coal body, at the same time, the safety factor of workers is higher.

7 Research on inclined abutment pressure in the goaf

Figure 17a and b are cloud diagrams of three-dimensional stress distribution when the working face is mined toward 40 m and 80 m under the mode of conventional mining. It can be seen that stress concentration showing symmetrical distribution appears in front of working face and the solid coal on both sides of the roadway during conventional mining. Figure 17c and d show the cloud maps of three-dimensional stress distribution when the working face is mined toward 40m and 80m by using the FRME. Those cloud images display an obvious feature of asymmetric distribution, which is different from that of Fig. 17a and b. These differences are mainly manifested in the following aspects:

- (1) The stress concentration area in front of the working face has an evident arch on the slotted side, which is not connected with the stress concentration zone on the solid coal of the roadway.
- (2) The abutment pressure in a certain range of the gob on the slotted side is significantly higher than that in other areas of the goaf.
- (3) The abutment pressure of the solid coal on the slotted side is obviously lower than that on the unslotted side.

In order to analyze the influence of the slit on the abutment pressure in the goaf in more detail, the abutment pressure in the goaf on the fracturing and non-fracturing sides is cored. And the abutment pressure at 10 m, 20 m, 30 m, 40 m and 50 m behind the working face (lines 6–10) when the working face is mined to 80 m is selected as the research object. The curves are shown in Fig. 18.

In Fig. 18, it can be seen that with an increase in the lagging coalface distance, the abutment pressure in the goaf increases gradually. The abutment pressure at 10 m behind the coalface is the lowest and the abutment pressure is the highest at 50 m behind the coalface. If this trend is correct, we can predict that as long as the distance of the lagging coalface is far enough, the abutment pressure in the goaf will return to the original rock stress, even greater than the original rock stress. This is closely related to the overburden movement. When the distance of the lagging coalface is close, the gob roof will not completely

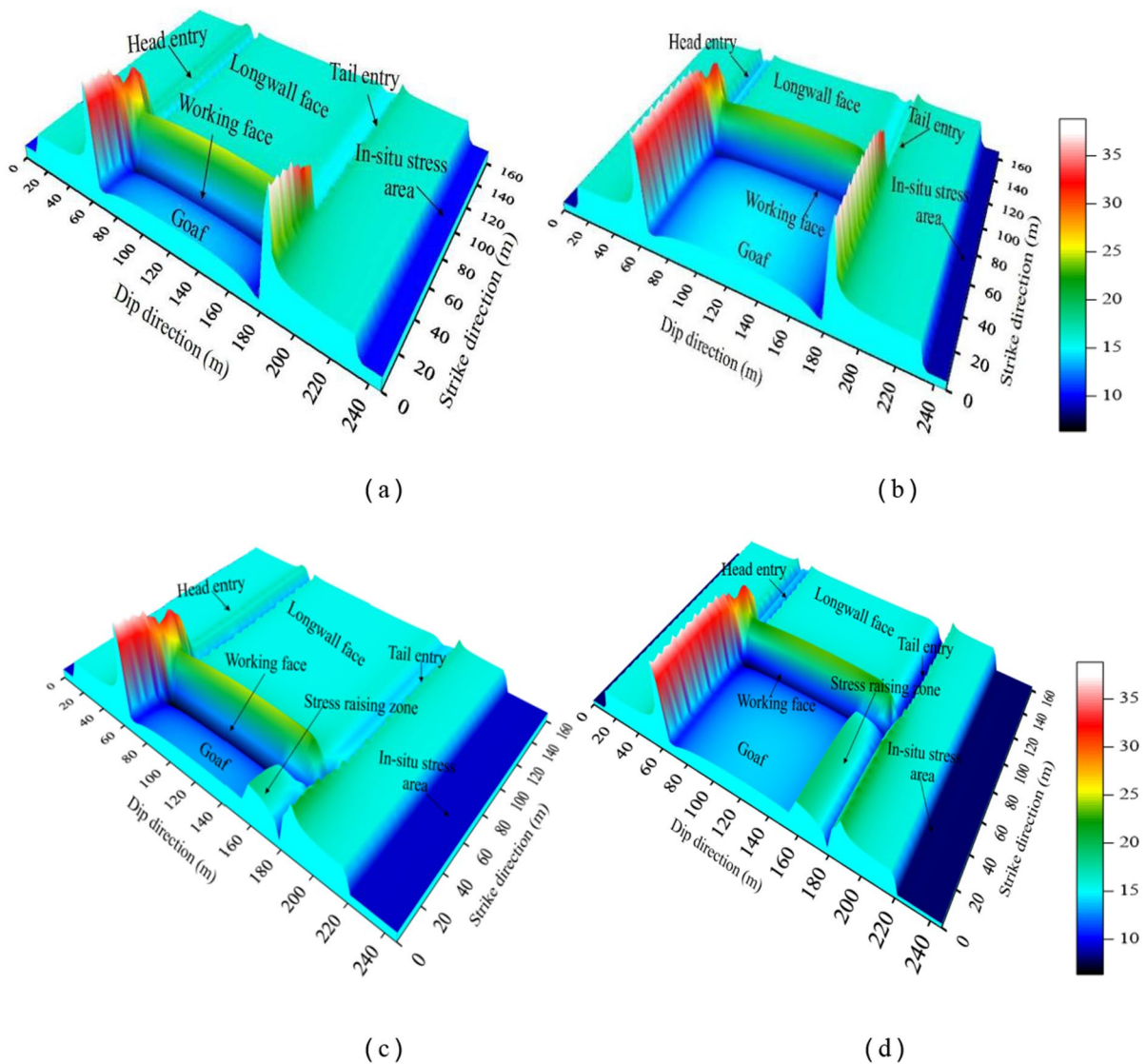


Fig. 17 Three-dimensional stress distribution cloud maps of conventional mining and the FRMR. **a** Three-dimensional stress distribution cloud map of conventional mining with the face advancing to 40 m; **b** three-dimensional stress distribution cloud map of conventional mining with the face advanced

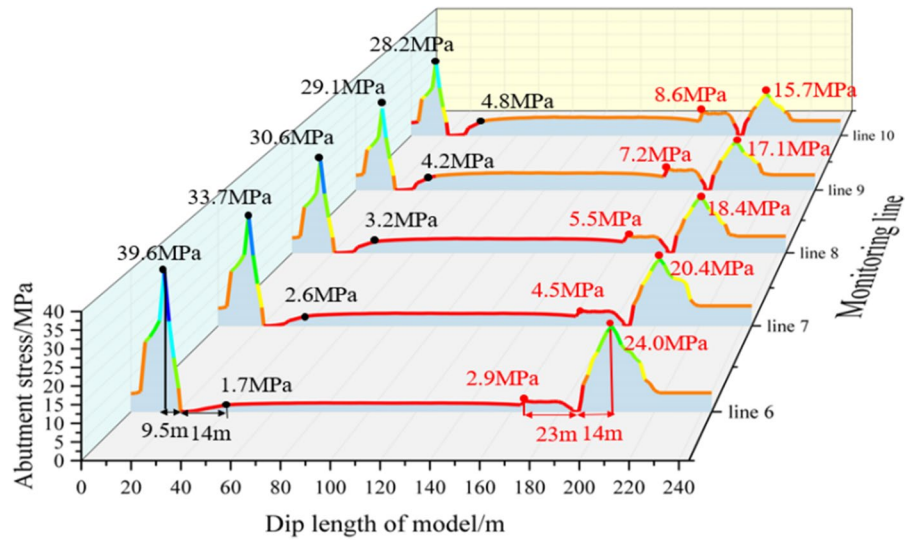
ing to 80 m; **c** three-dimensional stress distribution cloud map of the FRME with the face advancing to 40 m; **d** three-dimensional stress distribution cloud map of the FRME with the face advancing to 80 m

collapse, hence, the gangue will not come into contact with the gob roof. At this time, the abutment pressure in the goaf is the weight of the gangue so that the abutment pressure is lower. As the working face continues to advance, the gob roof will collapse more and more fully until the gangue comes into contact with the gob roof. Then the gangue will bear the weight of the overlying strata when overburden is

bent, subsided, unstable, and broken, so the abutment pressure in the goaf increases clearly.

The inclined abutment pressure of the goaf on the slotted side within the range of 23 m is obviously higher than that in other areas, indicating that the influence range of the slit is about 23 m. The goaf roof is more likely to collapse under the influence of slit, which makes the caving gangue contact with

Fig. 18 Inclined abutment stress curve at different distances of lagging coalface under the condition of the FRME



the gob roof quickly and support roof. Hence, the inclined abutment pressure of the goaf on the slotted side is greater. In addition, regardless of the slotted side or the non-slotted side, the side abutment pressure in the goaf gradually increases from 0 at the edge of the entry.

The abutment pressure at the solid coal of the slotted side is obviously lower than that on the non-slotted side. An interesting phenomenon is found that the abutment pressure on the side of solid coal decreases with the increasing of abutment pressure in goaf. It is easy to understand that after coal seam is mined, the weight of the overlying strata is borne by the coal body of the coalface, the gangue in the goaf and the solid coal on both sides of the coalface. When the distance of the lagging coalface is far enough, due to the limited load bearing capacity of the coal at the coalface, at this time, the overburden weight is mainly borne by the gangue in the goaf and the solid coal on both sides of the coalface. When the abutment pressure in the goaf gradually recovers, the lateral abutment pressure in the solid coal on both sides of the coalface will be reduced.

8 Research on strike abutment pressure in the goaf

The strike abutment pressure curves at different positions for the goaf were drawn according to the data extracted by the monitoring line 17# (the dip length

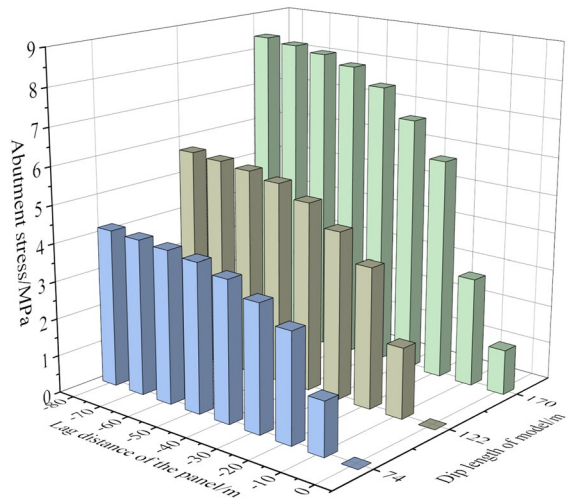


Fig. 19 Bar graph of strike abutment pressure of different positions in the goaf

of model is 74 m), monitoring line 18# (the dip length of model is 122 m), monitoring line 19# (the dip length of model is 170 m), as shown in Fig. 19, it shows that in the range of about 10m behind the working face, the abutment pressure in the goaf is relatively small. The maximum abutment pressure in this range at monitoring line 17 and monitoring line 18 is about 2 MPa, and the maximum abutment pressure at monitoring line 19 is 2.9 MPa, which is obviously higher than that at monitoring line 17 and monitoring line 18. This is because the slit makes the

direct roof fully collapse and the weight of gangue borne by the goaf floor is larger. Hence, the abutment pressure of monitoring line 19 is higher than that of monitoring lines 17 and 18.

It is worth noting that although the gangue contacted with the basic roof in this range, there is a large interspace among the gangue blocks, which does not bear the load of overlying strata. The evident stress recovery phenomenon begins to appear after the position of goaf is lagging behind the working face at 10 m and the abutment stress of the goaf increases rapidly. In this interval, the goaf roof moves from bottom to top, and gradually appears the phenomenon of instability, breaking, bending and sinking. After the goaf roof contacts with the gangue, it will start to squeeze the gangue, so the weight of the overburden will be transferred toward the goaf floor.

Generally speaking, the abutment pressure on the slotted side of the gob is obviously higher than that on the middle and unslotted side of the goaf. While the abutment pressure in the middle of the gob is greater than that on the unslotted side of the gob, it may be related to the movement of the overlying rock. According to the simple algorithm of Marcus's thin plate theory, the fracture of main roof first occurs in the center of the long side, that is the middle of the working face, and then gradually extends to both sides.

9 Discussion

Figure 20 reveals that the trend of lateral abutment pressure under the two mining methods is still the same, however, the pressure values will be different. As far as the advanced abutment pressure is concerned, the trend in the strike direction of the working face is also the same, but it is different in the dip direction of the working face, compared with the non-cutting side, the inclined abutment pressure peak on the cutting side is farther from the working face and entry. The advanced abutment pressure of the conventional mining is symmetrically distributed in the inclined direction of the working face. However, the advanced abutment pressure of the FRME is asymmetrically distributed in the inclined direction of the working face, which means the advanced abutment pressure on the slotted side of the coalface is larger

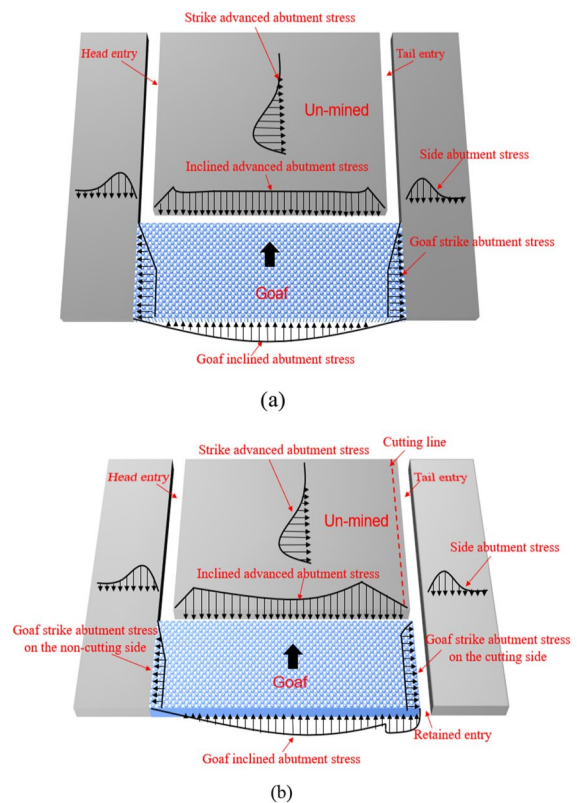


Fig. 20 Stope abutment pressure distribution diagram of different mining methods. **a** Abutment pressure distribution of conventional mining stope; **b** abutment pressure distribution of the FRME

than that of the non-slotted side of the coalface under the technique of the FRME.

Compared with conventional mining and the FRME, the difference of abutment pressure in the goaf is the most obvious. In Fig. 19a, the abutment pressure in the goaf is regarded as a linear variation, which is only related to the buried depth. However, Sinha and Walton (2019) proved that the relationship between stress distribution in the goaf and buried depth is not unique. Abutment pressure in the goaf is related to geological conditions, mining height, mining speed and roof lithology and so on, and its distribution is complex and changeable, not a simple linear change. The abutment pressure in the goaf is modified in Fig. 19b, which shows that after the entry roof is split, the rising speed of the abutment pressure on the slotted side of the gob is faster than that on the non-slotted side of the gob, indicating that the slit can augment the bulk coefficient of the roof, and the

gangue in the goaf can quickly contact with the basic roof and then bear the weight of the overlying rock.

10 Conclusions

In order to achieve efficient mining of coal resources, a variety of innovative technologies emerge as the times require, in which the FRME as a representative of efficient, safe and green mining is more and more widely used. For the sake of better guidance of the on-site production practice, the numerical simulation and field measurement are used to study the abutment pressure of the whole working face. The main conclusions are expressed as follows:

- (1) Under the technology of the FRME, with the mining of the working face, a stress concentration area is formed in front of the working face, but this area shows the characteristic of asymmetric distribution, which is obviously different from the symmetrical distribution of traditional coal pillar mining. The specific performance is as follows: the distance between the peak value of advanced abutment pressure and the edge of roadway is more distant than that of traditional coal pillar mining.
- (2) The influence range of the slit is limited, which can not affect the abutment pressure distribution of the whole working face. This scope is named as ‘fracturing-affected area’. Under the geological conditions of Lvtang mine, the length of the “fracturing-affected area” is 23 m. The length may be different under different geological conditions.
- (3) Cored with the non-slotted side, the peak value of the strike advanced abutment pressure of the slotted side in the same position drops by 19.29% on average, and the influence range of the strike advanced abutment pressure is increased by 30.78%, and the peak distance from the working face is increased by 66.7%. This conclusion shows that the slit can obviously improve the stress environment of the surrounding rock of the roadway, which has practical significance for maintaining the stability of the roadway and the safety of workers.
- (4) The abutment pressure distribution in the goaf of the FRME has the greatest change compared

with coal pillar mining. The abutment pressure in the goaf on the slotted side is higher than that in other areas of the goaf, which is related to the change of the overburden movement. The bulk coefficient of the goaf roof is affected by the slit increases, which makes the caving gangue contact with the roof rapidly and support it. Bearing the weight of the overlying rock earlier makes the abutment pressure of the goaf on the slotted side great.

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Author contribution YG: Supervision, conceptualization. XL: Investigation, visualization. WL: Writing—original draft, conceptualization. FD: Visualization. JM: Data curation. RQ: Supervision. NH: Data curation.

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Data availability The data used to support the findings of this study are available from the corresponding author upon request.

Declarations

Ethical approval This research adheres to the highest ethical standards, ensuring originality of the work, data integrity, proper citations, and avoidance of untrue statements or personal attacks.

Consent for publication The authors confirm that all necessary consents have been obtained from individuals and institutions involved in this research. Participants’ informed consent has been secured for the use of their data, images, or any other identifiable information. This ensures compliance with ethical standards and permits the publication of this research in academic journals or other publications.

Competing interest The authors declare that they have no known competing financial interests or personal relationships that could have appeared to influence the work reported in this paper.

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