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Research Article

Mining Stress Distribution and Gas Drainage Application of Coal Seam Group under Fault Influence

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The existence of fault structures often seriously endangers the safe production of coal mines. In order to ensure the safe and efficient production of close coal seam group mines under the influence of faults, it is of wide engineering significance to study the gas occurrence and extraction technology between coal seam groups under the influence of faults. In this study, the mining and gas drainage of 13817 working face in Yueliangtian Mine were taken as the research background. Combined with theoretical research, laboratory analysis, and numerical simulation analysis, the pressure relief range of coal seam and the evolution characteristics of surrounding rock stress were studied, the coal and gas comining system under the influence of faults was determined, the gas drainage technology was optimized, and the effect was investigated and analyzed. The results show that the gas pressure reduction zone is formed in the range of 20 m and 6 m of the lower wall of F1 fault, and the pressure relief effect is good in the range of 40 m above the roof of the working face to 20 m below the bottom plate during the advancing process. The optimized gas drainage technology has a good effect on gas control in coal seam group mining under the influence of faults. The absolute gas emission of the working face is reduced to 28~36 m^3/min, the gas concentration in the upper corner of the mining face is about 0.6%, and the gas concentration of the return airflow is reduced to less than 0.5%. The gas extraction rate of No. 8 coal seam is 52.9%, and the gas content in the fault zone is reduced by 60%, realizing the safe and efficient coming of coal and gas. The research results have certain reference and application value for mine gas control with similar mining conditions.

1. Introduction

China is the largest coal-producing country in the world, with an output of 3.9 billion tons in 2020. At the same time, China is the largest consumer of coal, accounting for about 50.5% of world coal consumption in 2018 and is expected to remain so, at a proportion of 39%, until about 2040 [1, 2]. Due to China's energy resource condition of "lack of oil, lack of gas, and relatively rich coal," coal constitutes the base and important part of China's energy and mineral resources [3, 4]. And it is going to stay that way for quite some time. With the increase of mining depth and intensity, the geological and technical conditions of mines are becoming more and more complex [5]. Coal mine accidents often occur, such as fires, floods, roof disasters, rock bursts, and

gas accidents. These disasters have a serious and bad impact on coal mine safety production. Among them, gas explosion and coal and gas outburst accidents have high frequency and harm. According to statistics, coal and gas outburst accidents have occurred in major coal mining countries around the world [6], and China is one of the countries with frequent coal and gas outburst accidents in the world.

Guizhou Province is one of the regions with the richest coal resources in China, with an annual raw coal output of 131 million tons in 2021, accounting for 3.2% of China's total raw coal output. However, Guizhou Province is located on the Yunnan-Guizhou Plateau, and its territory is significantly affected by plateau tectonic and plate movements. Guizhou coalfields have strong deformation of coal measures, dense faults, and well-developed fold and folded structures [7, 8].
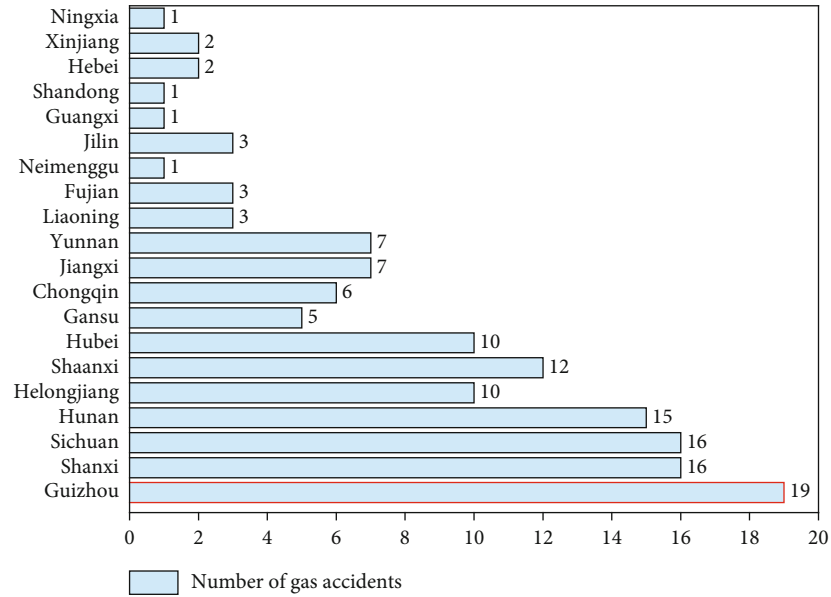
Figure 1: Regional distribution of coal mine gas accidents in China from 2015 to 2021.

At the same time, most of the coal mines in Guizhou are coal and gas outburst mines, which is one of the provinces with the most frequent gas disaster accidents. Figure 1 shows the regional distribution of coal mine gas accidents in China from 2015 to 2021. According to relevant data, there were 19 gas accidents in Guizhou Province from 2015 to 2021, and the form of coal mine safety production is very serious. In addition, faults, coal seam groups, and gas are the common characteristics of coal seams in Guizhou [9, 10], so it is very necessary to study the gas prevention and control of coal seam groups.

For a long time, due to the complex and changeable conditions of coal seam occurrence, the frequent occurrence of major gas dynamic disasters has long been the concern and attention of the mining industry. Gas is one of the main sources of mine disasters, which has become a key factor restricting the safe production of mines [11]. Gas control is the premise and foundation for safe and efficient mining in high-gas mining areas [12–15]. In the process of gas prevention, hydraulic fracturing technology is often used to increase the permeability of coal seam to improve the efficiency of gas extraction. Environmentally friendly anhydrous fracturing technology such as liquid nitrogen fracturing has also attracted more and more attention in coal seam permeability enhancement and coalbed methane development [16]. Many scholars have done a lot of research on improving the safety of coal mine production and reducing the risk of coal and gas outbursts. The protective layer mining and gas drainage are more economical and feasible methods and technical means to eliminate gas disasters in high gas mines [17, 18]. For mines without mining protection layers, gas drainage is usually used as a gas control measure to prevent coal and gas outbursts [19, 20]. Coal seam drilling gas drainage technology has been studied and applied in gas control [21–23]. But in soft coal seams where hydraulic fracturing has poor permeability enhancement effect, large-diameter borehole gas drainage is widely used, and good drainage effect has been achieved [24, 25]. However, in deep mining, drilling, and other engineering activities, the mechanical and seepage characteristics of coal and rock will be significantly changed under the influence of cyclic loading or even stratified cyclic loading [26]. In addition, the geological conditions and gas occurrence conditions of coal seams are significantly different in different mines, and the main control factors affecting gas occurrence are also different. Different gas extraction technology has great influence on the extraction effect. Especially under the condition of repeated mining of close coal seam group, the destruction of weathered rock, fracture development, gas distribution, and migration are more complex, and the gas control problems are also more complicated [27]. Liu and Cheng [28] adopted the analogy simulation method to study the dynamic evolution laws of overlying fractures induced by mining in the protective seam and the protective scope of pressure relief. Through numerical simulation software, such as FLAC^{3D}, Cheng et al. [29] investigated the stress level of the coal and rock mass around the protective layer during the excavation process and analyzed the characteristics of pressure relief protection. Chang and Tian [30] studied the distribution characteristics of overlying rock stress field, displacement field, and gas pressure and carried out an application analysis of the technical scheme of pressure relief gas drainage in multicoal seam mining areas. Under the influence of fault structure, the roof and floor failure characteristics, gas occurrence law, and migration characteristics of coal-rock mass between coal seam groups will be quite different [31, 32], and correspondingly different gas extraction technologies will have different effects. At present, many scholars have studied the gas extraction, utilization, and prevention of coal seam groups with short distance, low permeability, high gas content, and steep dip mostly with protective layer pressure relief mining as the main feature,
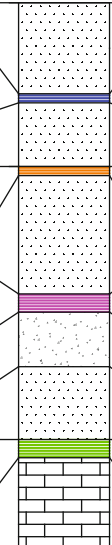
No.	Thickness (m)	Geological column	Petrographic description (m)
1	50.0		Grey siltstone
2	1.0		No.6 coal seam
3	7.0		Grey siltstone
4	1.0		No.7 coal seam
5	13.0		Grey siltstone
6	2.0		No.8 coal seam
7	6.0		Fine sandstone
8	8.0		Grey siltstone
9	2.0		No.10 coal seam
10	40.0		Siltstone

FIGURE 2: Coal seam comprehensive histogram.

while there are few studies on the gas occurrence and extraction technology of coal seam groups with fault structure.

This paper takes the gas drainage in 13817 working face of Yueliangtian Mine as the engineering background, analyzes the pressure relief range of coal seam and the evolution characteristics of surrounding rock stress, puts forward the coal seam group coal and gas comining technology system under the influence of fault, and optimizes the gas drainage technology reasonably and investigates the effect. This study will solve the gas control problem of coal seam group under the influence of fault in Yueliangtian Mine, which is of great significance to ensure the safe production of Yueliangtian Mine and study the gas control of coal seam group under similar geological conditions.

2. Overview of Geology and Gas in Mining Area

2.1. Mining Geological Conditions. The Yueliangtian Mine is located in Panjiang town Pan County, China, coal seam dip angle 8°~25°, average dip angle 10°, the use of inclined shaft development, to the long wall retreat type mining, comprehensive mechanized mining, the mine in 2004 as coal and gas outburst mine. The 13817 working face of the No. 8 coal seam is located in the south wing of F17 fault and the north wing of F15 fault in South Third mining area. The strike length is 284 m, the dip length is 170 m, and there is a normal fault with a drop of 6 m in the mining area. The original content of coal seam gas was 5.67 m³/t, and the original gas pressure was 0.52 MPa. The average thickness of coal seam is 2.2 m, and the average dip angle of coal seam is 7°. Mining face 13817 is the first mining face. The overlying, underlying, and underlying coal seams on this working face are not exploited, which belongs to mining in the pressure area. The direct roof of the No. 8 coal seam is composed of gray siltstone with horizontal bedding and oblique bedding locally. The thickness of the rock is 3 m; the direct bottom is composed of light gray mudstone with strong water absorption, and the thickness of the rock is 0.3~0.5 m.

Figure 2 shows a schematic diagram of the broad stratigraphic column of the 13817 working face at the Yueliangtian Mine. The mine is a coal and gas herniated mine with an absolute gas gush of 76.30 m³/min and a relative gas gush of 42.03 m³/t. The No. 8 coal seam has medium gas content, coal dust is explosive, and the propensity of coal seam to spontaneous combustion and fire belongs to class II. The location of this working face is between F17 and F15 faults, and a total of 2 large faults were found during the digging process, and the top plate of the coal seam near the fault is crushed and broken, and gas management needs to be strengthened during the back mining over the fault.

The 13817 working face is located in the north wing of the F15 fault. The geological structure is complex, and the F1 fault with a drop of 6 m is exposed in the mining process, and the F17 fault with a drop of 23 m is exposed in the tunneling process. Figures 3 and 4 show the section of the working face 13817 in relation to the location of the fault, and Table 1 shows the main parameters of the fault.

2.2. Overview of Mine Gas

2.2.1. Gas Source Analysis. The gas emission of coal mining face depends on the natural factors of coal seam, mining technical conditions, and many other factors. The source of gas emission includes the gas emission of coal wall, coal caving, and goaf. Figure 5 shows the composition of gas source in working face.

According to the data analysis of 13817 working face in Yueliangtian Mine, No. 8 coal seam is 13 m away from the upper adjacent No. 7 coal seam, 21 m away from the upper adjacent No. 6 coal seam, and 28 m away from the upper
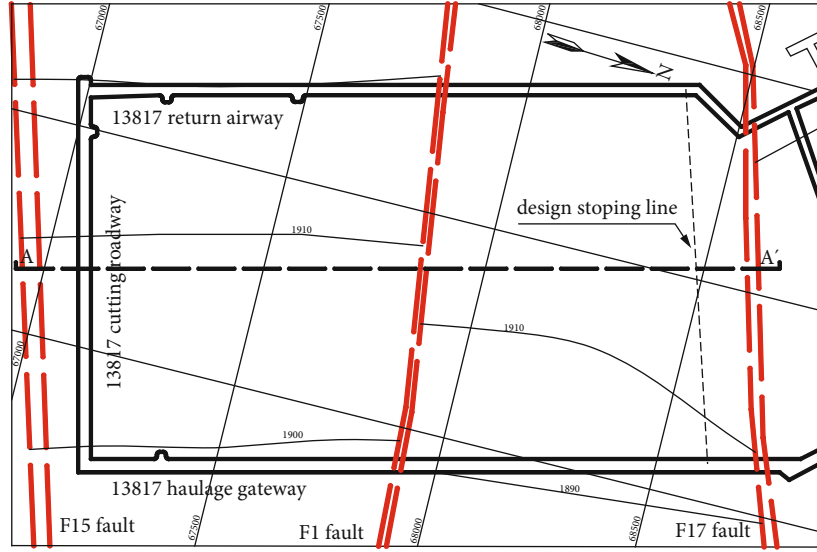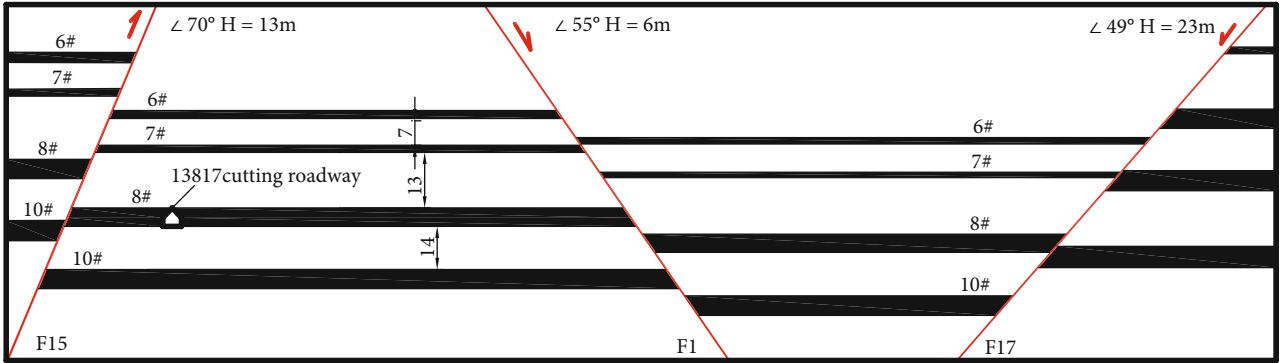
FIGURE 3: 13817 mining face mining plan.

FIGURE 4: Schematic section of the working face along A-A$'$ strike.

TABLE 1: Main parameters of the fault.

Faults	Trend (°)	Dip	Angle of dip (°)	Fault properties	Fall (m)
F15	80	NNE	70	Reverse fault	13
F1	83	NWW	55	Normal fault	6
F17	77	SE	49	Normal fault	23

adjacent No. 5 coal seam. According to the upper three-zone theory and the lower two-zone theory [33], the upper adjacent No. 5 coal seam, No. 6 coal seam, and No. 7 coal seam are all in the fracture zone after the mining of the No. 8 coal seam. The upper adjacent coal seam is affected by mining, and the analytical gas will all enter the goaf of the No. 8 coal seam. No. 8 coal seam is 14 m from the lower adjacent No. 10 coal seam and 35 m from the lower adjacent No. 12 coal seam. The lower adjacent No. 10 and No. 12 coal seams are both within the influence range of the No. 8 coal seam mining, and the analytic gas in the coal seam will all enter the goaf of the No. 8. Therefore, the gas in 13817 working face of the No. 8 coal seam mainly comes from the pressure relief

gas of the coal seam, the upper adjacent No. 5, No. 6, and No. 7 coal seam and the lower adjacent No. 10 and No. 12.

2.2.2. Analysis of Gas Emission. According to "Mine Gas Emission Prediction Method," when thin and medium-thick coal seams are mined without layers, the gas emission of this coal seam is shown in the following formula:

$$q_1 = k_1 \cdot k_2 \cdot k_3 \cdot \frac{m}{M} (W_0 - W_c), \tag{1}$$

where q_1 is relative gas emission of the coal seam, m^3/t; k_1 is the gas emission coefficient of surrounding rock (value is
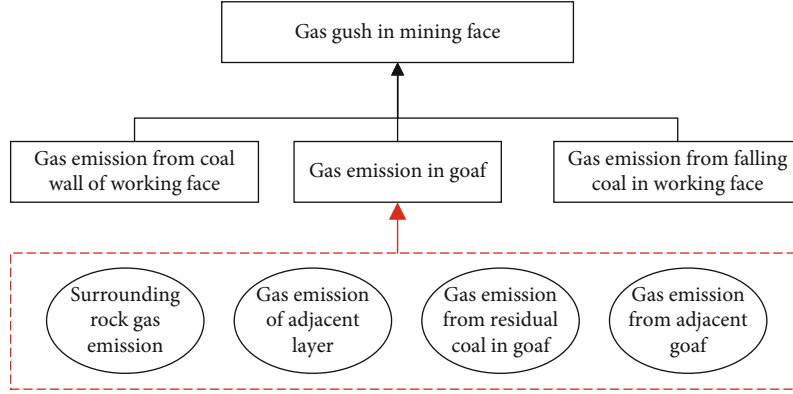
FIGURE 5: Composition of gas source in working face.

TABLE 2: Relevant parameters for calculating gas emission in adjacent layers.

Coal seam	W_{0i}	W_{Ci}	m_i	M	d	η_i
5#	5.48	0.82	1.8	1.8	28	70%
6#	4.76	0.71	1.3	1.3	21	70%
7#	4.83	0.72	1.4	1.4	13	70%
10#	10.18	1.53	2.2	2.2	14	70%
12#	12.37	1.86	2.8	2.8	35	60%

1.2); k_2 is the residual coal gas emission coefficient (value 1.05); k_3 is the influence coefficient of roadway predrainage gas (value 0.858); m is the coal seam thickness, m; M is the coal seam mining thickness, m; W_0 is the original gas content of the coal seam; m^3/t; W_C is the gas content of residual coal seam, m^3/t.

According to formula (1), the relative gas emission of the coal seam can be obtained by substituting relevant parameters into calculation as follows:

$$q_1 = 1.2 \times 1.05 \times 0.858 \times (2.2 \div 2.2) \times (5.67 - 1.42) = 4.59 \, (m^3/t). \tag{2}$$

The relative gas emission from adjacent coal seams is shown in the following formula:

$$q_2 = \sum_{i-1}^{n} (W_{0i} - W_{Ci}) \cdot \frac{m_i}{M} \cdot \eta_i, \tag{3}$$

where q_2 is the relative gas emission of adjacent coal seam; m^3/t; m_i is the thickness of the adjacent seam, m; M is the mining height, m; W_{0i} is the gas content in adjacent seam, m^3/t; W_{Ci} is the residual gas content in adjacent layers, m^3/t; η_i is the gas emission rate of adjacent layer.

The relevant parameters of each coal seam adjacent to 13817 working face are shown in Table 2.

According to formula (3) of the data shown in Table 2, it can be concluded that the relative gas emission in the adjacent layer of 18717 working face is as follows:

$$q_2 = q_{5\#} + q_{6\#} + q_{7\#} + q_{10\#} + q_{12\#} = 21.34 \, (m^3/t). \tag{4}$$

The gas emission of the No. 8 coal seam is 4.59 m³/t, and the relative gas emission of 13817 working face is 25.93 m³/t. It can be seen that in the gas source of the No. 8 coal seam 13817 working face, the coal seam gas emission accounted for 17.7%, the adjacent layer gas emission accounted for 82.3%, the upper adjacent layer accounted for 34.6%, and the lower adjacent layer accounted for 47.7%. Therefore, in the mining process of 13817 working face in Yueliangtian Mine, the gas drainage of adjacent layers will become the focus of gas control.

Similarly, the absolute gas emission of the working face is an important index to evaluate the gas grade of the mine. Based on the calculation results of the relative gas emission of the working face, the absolute gas emission of the working face can be calculated. Equation (5) shows the calculation method of absolute gas emission in working face.

$$Q_q = \frac{q_3 \times A}{24 \times 60}, \tag{5}$$

where Q_q is the relative gas emission quantity, m³/t; q_3 is the absolute gas emission quantity, m³/t; A is the design daily output, t.

Daily output of 13817 working face is 3074 tons, and the absolute gas emission is 55.35 m³/min. The absolute gas emission of 13817 working face is far greater than 5 m³/min, and the risk of gas outburst is high. It is necessary to eliminate outburst in working face.

2.2.3. Influence of Faults on Gas Occurrence. In the tunneling process of 13817 return airway and haulage gateway, the occurrence and gas of the No. 8 coal seam in front of the
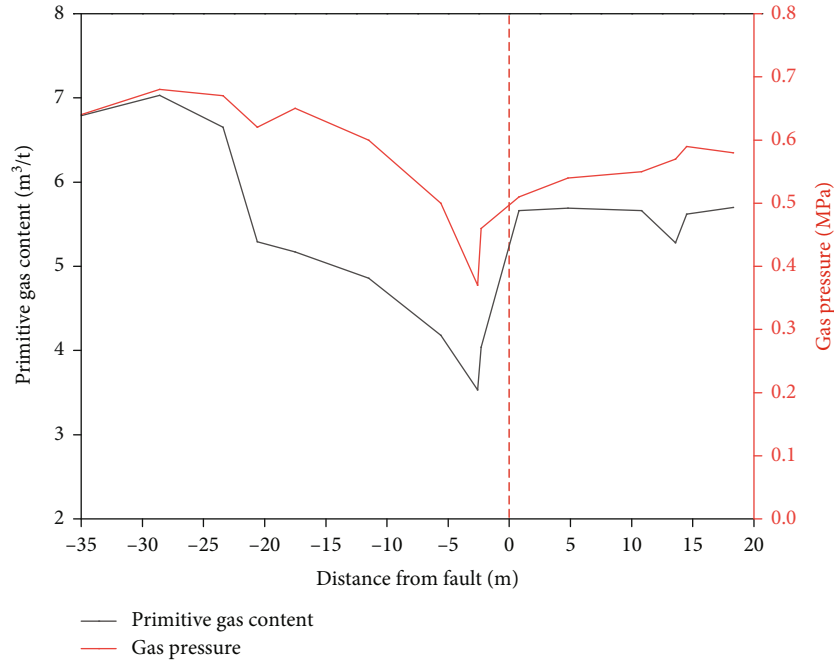
FIGURE 6: Relationship diagram of original gas content and gas pressure at different distances from measuring point to fault in heading face of 13817 return airway.
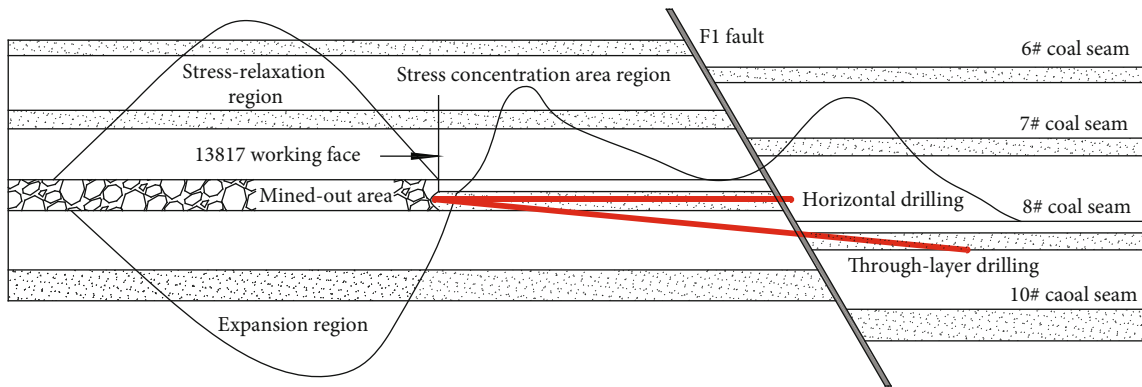

FIGURE 7: The schematic diagram of 13817 working face advancing towards fault along strike.

tunneling area were detected by exploratory drilling before the tunneling working face construction. After coring (coal sample), the original gas content and original gas pressure were determined. The F1 normal fault layer was exposed at 145 m in 13817 return airway. The original gas content and original gas pressure of coal samples taken from the forward drilling holes at different distances from the fault were measured during the excavation of the return airway. Figure 6 shows the original gas content and pressure diagram obtained from the measured data at different distances from the fault.

The analysis of the above diagram shows that the gas content and gas pressure at different distances from the fault are quite different, and the gas content and gas pressure curve have roughly the same trend. When the distance from the fault is 20~0 m, the gas content curve is relatively stable, and the gas content is basically maintained at an average of 5.70 m^3/t. After crossing the fault ($S = 0$ m), the gas content

decreased sharply, and the minimum value appeared at about 3 m ($S = -3$ m) after crossing the fault, which was only about 3.5 m^3/t. After that, the gas content increased gradually and stabilized at about 5.0 m^3/t when crossing the fault 10 m~20 m. After crossing the fault 20 m ($S = -20$ m), the gas content increased sharply, from an average of 5.57 m^3/t to 7.03 m^3/t. From the fault 20~0 m range, the gas pressure changes gently and slightly decreased, and the average is about 0.55 MPa; after crossing the fault, the gas pressure decreased sharply, and a minimum value of only 0.37 MPa appeared at about 3 m after crossing the fault. After that, the gas pressure increased again and stabilized at about 0.65 MPa. It can be seen from the whole curve that the gas pressure reduction zone is 6 m ($S = -6$ m) after crossing the fault ($S = 0$ m).

Figure 7 shows the schematic diagram of 13817 working face advancing along the strike to the fault. When the working face is close to the fault zone from the footwall, due to
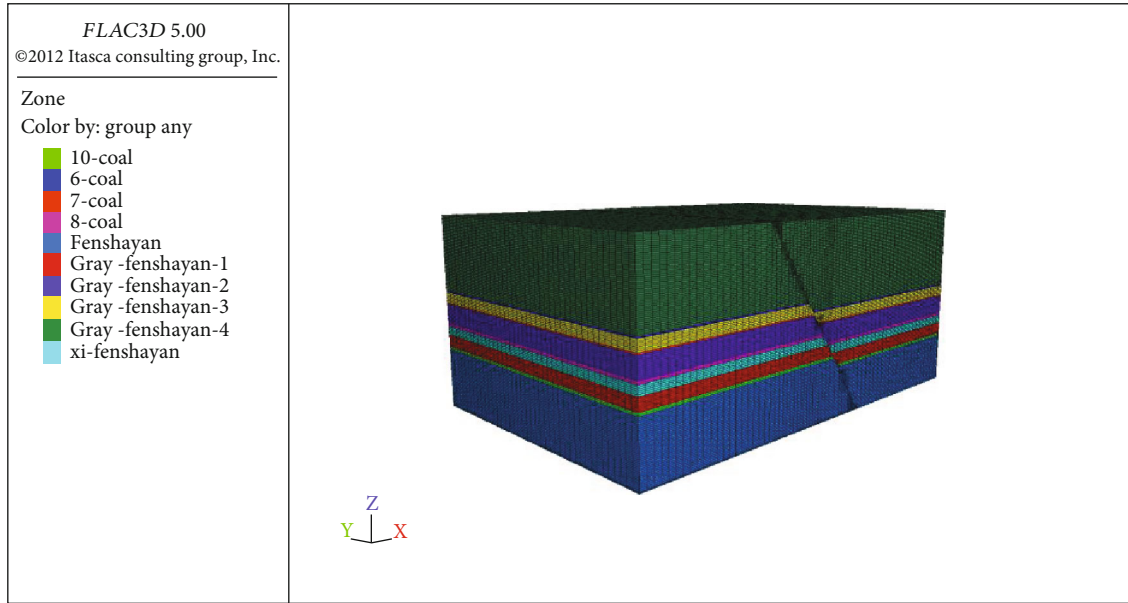
FIGURE 8: Stress distribution model and meshing of surrounding rock in coal seam mining.

the influence of mining, the stress concentration will be formed in front of the work, superimposed with the fault stress concentration area, and the fracture is closed, which hinders the gas migration in the coal seam. Before the working face exposes the fault, due to the release of stress in the mining area on the lower side of the fault, the gas pressure is reduced, and the relative equilibrium state of coal seam gas is destroyed. The gas content and pressure in the upper and lower walls of the fault are greatly different. When the gas pressure in the upper wall of the fault exceeds the coal strength, it is easy to produce coal and gas to be sprayed into the newly exposed mining space, resulting in a sharp increase in gas content. Therefore, when the coal mining working face is close to a certain distance from the fault, it is not only necessary to arrange the borehole in the coal seam to predrainage coal seam gas, but also to arrange the borehole through the seam to the other side of the fault to extract high pressure gas. The predrainage of the borehole in advance makes the analytical gas accumulate to the predrainage borehole area, the gas pressure decreases continuously, the adsorbed gas is quickly resolved into free gas, the gas content decreases continuously, the strength and stability of the coal body will also be improved, and the occurrence of coal and gas outburst will be avoided.

3. Analysis of Numerical Simulation Results

3.1. Numerical Simulation of Coal Seam Mining

3.1.1. Geometric Model and Mesh Generation Diagram. By considering the actual conditions of field working face and basic assumptions, the stress distribution law of surrounding rock in the 13817 working face of the No. 8 coal seam is simulated. The model is reasonably simplified, down to No. 10 coal seam floor 40 m, up to No. 6 coal seam roof 50 m, which are near horizontal coal seam. The mining height of the

simulated working face is 2.2 m, and the working face length is 170 m. The excavation area is in the central position of the model. The open-off cut is located at the position of 50 m on the left side of the model, advances to the right along the coal seam strike, and simulates the stress change during the 200 m advancement of the working face. The model and mesh generation diagram are shown in Figure 8.

3.1.2. Selection of Yield Criterion and Mechanical Parameters of Coal Strata. According to the characteristics of coal and rock and geological conditions, Mohr-Coulomb plastic model is selected for calculation [34]. The yield criterion of Mohr-Coulomb model is shown in the following formula:

$$f_s = \sigma_1 - \sigma_3 \frac{1 + \sin\varphi}{1 - \sin\varphi} + 2C\sqrt{\frac{1 + \sin\varphi}{1 - \sin\varphi}}, \tag{6}$$

where σ_1 is the maximum principal stress; σ_3 is the minimum principal stress; C is the binding power; φ is the angle of friction.

Through the core drilling of 13817 working face, the mechanical parameters of rock samples were tested, and the physical and mechanical parameters of the studied coal and rock strata were obtained when FLAC^{3D} was used for numerical simulation. Figure 9 shows the relevant experimental sites of physical and mechanical experiments, and the mechanical parameters of each coal and rock strata are shown in Table 3.

3.1.3. Model Boundary Condition. The rolling boundary conditions are adopted for the left and right boundaries, the front and rear boundaries, and the lower boundaries of the numerical model, and the vertical stress is applied on the upper boundary, which is horizontally free. The vertical stress of the upper boundary is determined according to the
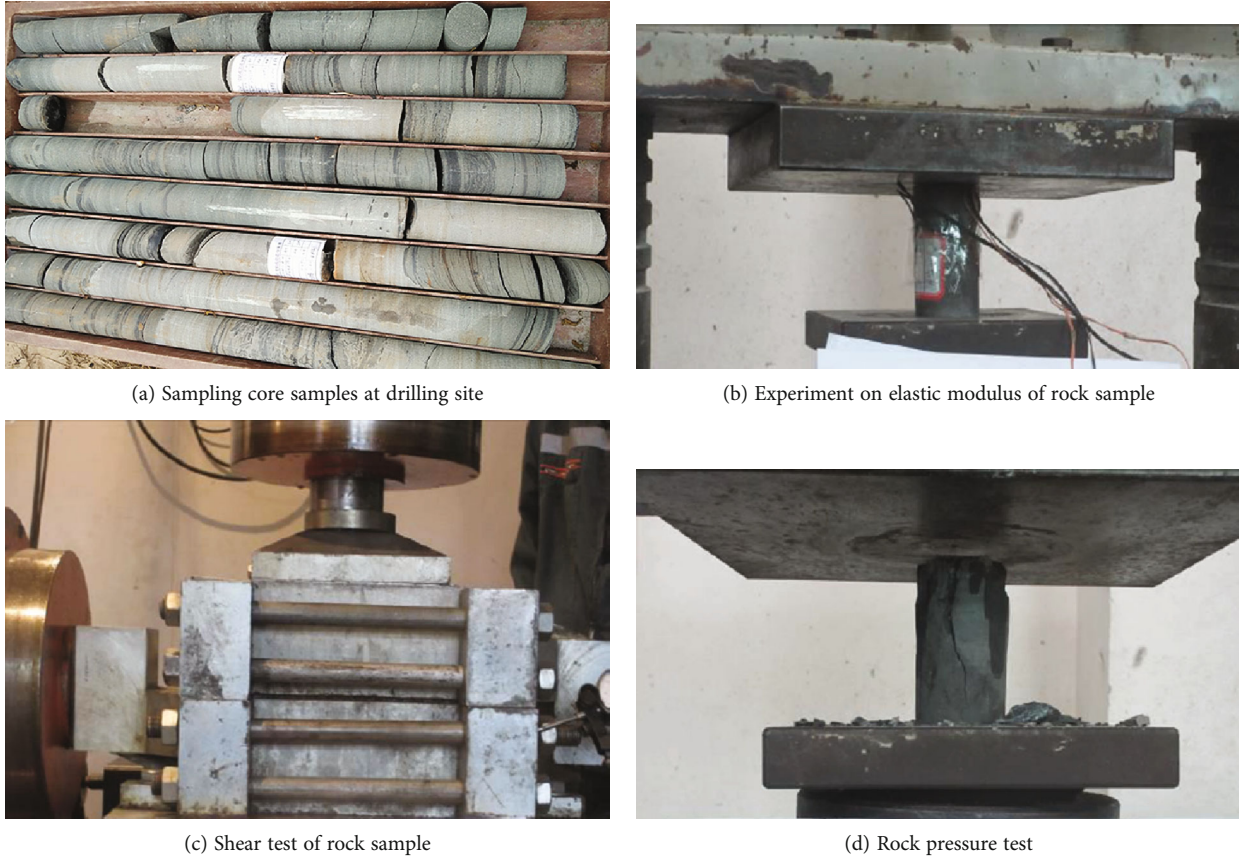
(a) Sampling core samples at drilling site

(b) Experiment on elastic modulus of rock sample

(c) Shear test of rock sample

(d) Rock pressure test

FIGURE 9: Field rock samples and physical and mechanical experiments.

TABLE 3: Mechanical parameters of coal strata.

Lithologic characters	Volumetric weight ρ\kg/m^3	Elastic modulus E\GPa	Poisson ratio μ	Force of cohesion c\MPa	Angle of internal friction φ\°	Tensile strength R_m\MPa
Grey siltstone	2,500	17.8	0.26	1.8	41.5	1.2
No. 6 coal seam	1,400	3.4	0.21	0.5	30	0.21
Grey siltstone	2,500	17.8	0.26	1.8	41.5	1.2
No. 7 coal seam	1,400	3.4	0.21	0.5	30	0.21
Grey siltstone	2,500	17.8	0.26	1.8	41.5	1.2
No. 8 coal seam	1,400	3.4	0.21	0.5	30	0.21
Fine sandstone	2,500	10.2	0.22	2.5	42	1.5
Grey siltstone	2,500	17.8	0.26	1.8	41.5	1.2
No. 10 coal seam	1,400	3.4	0.25	0.5	30	0.21
Siltstone	2,500	28.5	0.28	2.8	41.5	1.4

following: the weight of the rock above is the original rock stress, the distance from the upper boundary to the surface is 686 m, and the vertical stress is 17.15 MPa. The left and right horizontal stress is considered by hydrostatic pressure, and the horizontal stress before and after is 0.25 of the vertical stress.

3.2. Simulation Result Analysis

*3.2.1. Stress Evolution Characteristics of Surrounding Rock during Coal Seam Mining.* In the process of advancing 200 m along the strike of 13817 working face in No. 8 coal seam, the stress distribution of surrounding rock was studied when the working face was advancing 60 m, 90 m, 120 m, and 160 m, respectively. Figure 10 shows the vertical stress distribution nephogram along the strike and inclination when the working face advances at different distances.

The working face is advanced by 60 m (at this time, it is 40 m away from the fault), the supporting pressure of the overlying and underlying coal seams continues to decrease, and the pressure relief area is basically symmetrical with
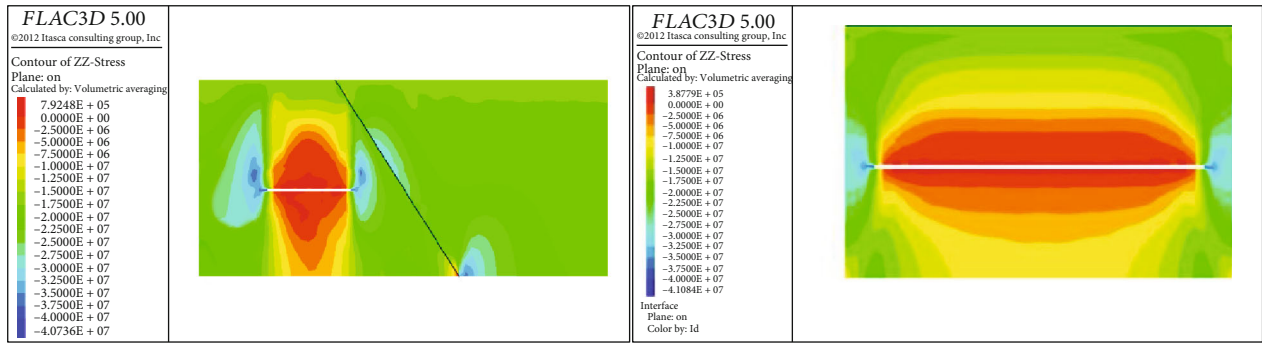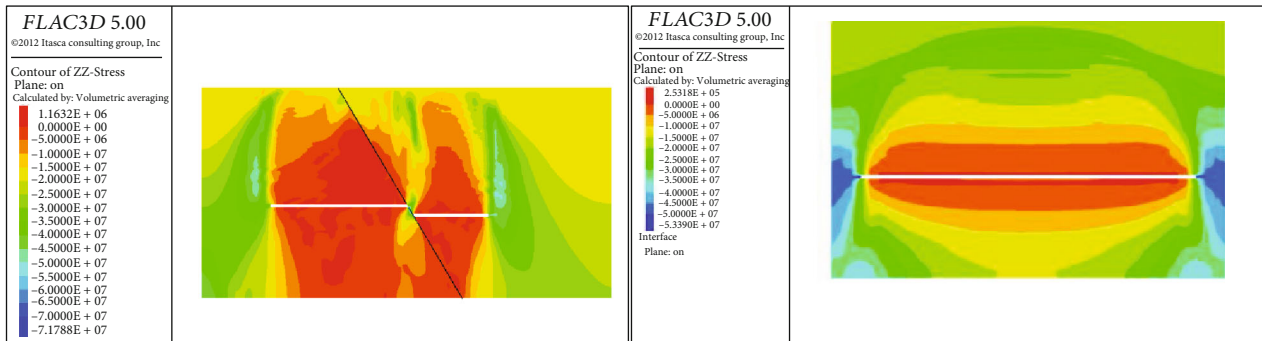
(a) Vertical stress distribution nephogram along strike and dip when working face advancing 60 m
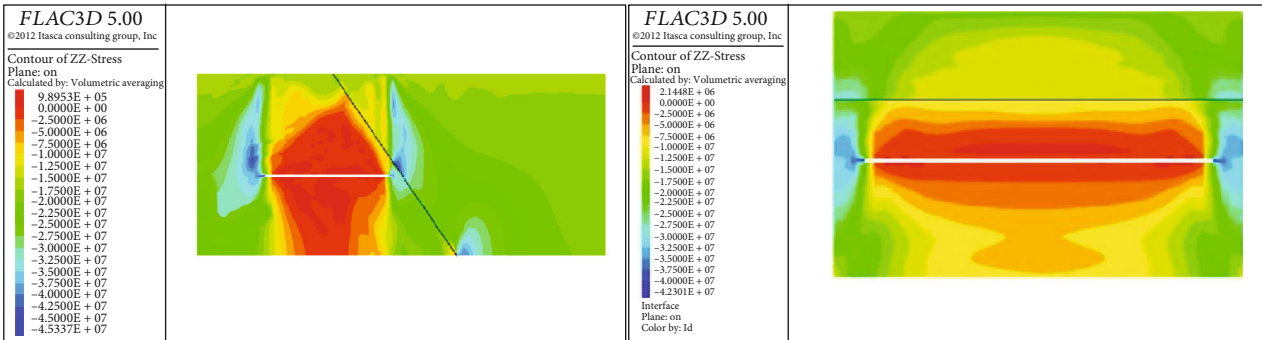
(b) Vertical stress distribution nephogram along strike and dip when working face advancing 90 m
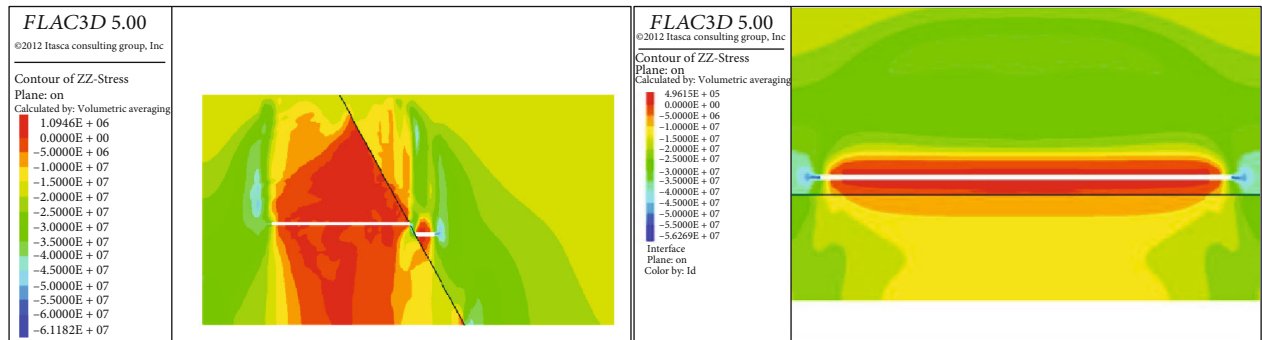
(c) Vertical stress distribution nephogram along strike and dip when working face advancing 120 m

(d) Vertical stress distribution nephogram along strike and dip when working face advancing 160 m

FIGURE 10: The distribution nephogram of vertical stress along tendency and strike under different advancing distances.

the middle of the goaf as the center, and the overlying and underlying coal seams are divided into two parts. It is a stress concentration area and a stress reduction area. The upper and lower parts of the goaf correspond to the stress reduction area, and a compressive stress concentration area

is formed near the opening and in front of the working face. At this time, the working face is still far from the fault. The stress concentration area in front of the working face and the stress concentration area in the middle of the fault have not been connected. There are at least 20 m original stress zones
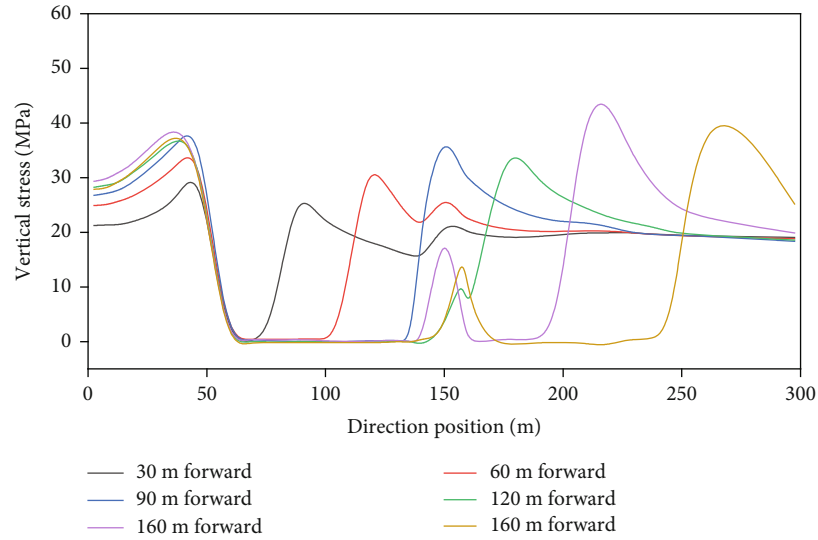
FIGURE 11: The vertical stress distribution law of the No. 8 coal seam roof when the working face advances at different distances.

between them, and the gas in the hanging wall of the fault is still in a closed state. With the continuous expansion, it can be preliminarily seen that the fault plane blocks the development trend of the pressure relief area in the upper part of the goaf to the fault hanging wall (Figure 10(a)). When the working face advances 90 m (10 m from the fault), the upper and lower coal rock strata pressure relief effect is very significant. The No. 6 and 7 coal seams located in the upper part of the goaf and the lower No. 10 coal seam are fully relieved. The stress concentration area in front of the working face and the fault stress concentration area are superimposed, resulting in a large stress concentration (Figure 10(b)). When the working face continues to approach the fault and cross the fault (20 m across the fault), the entire fault is relieved to varying degrees, and the mining space on its hanging wall forms a small range of pressure relief. The pressure relief area and the pressure relief area caused by the mining of the fault footwall are not connected (Figure 10(c)). The pressure relief areas of the upper and lower walls are connected to each other, and the entire fault is in the pressure relief area formed by mining. The fault is relieved to a large extent and further undergoes large displacement, and deformation and is destroyed. The surrounding fractures are fully developed and connected to each other, which are adjacent layers. The migration of pressure relief gas provided a channel, the sealing effect of the entire fault on the hanging wall gas disappeared, and a large amount of pressure relief gas gushed out (Figure 10(d)).

3.2.2. Pressure Relief Law of Overlying and Underlying Coal and Rock Masses in No. 8 Coal Seam. In order to clearly see the change law of the vertical stress of the roof when the working face advances at different distances, when the advancing distance continues to increase, the stress monitoring and recording points are arranged along the advancing direction of the working face 50 m in front of the No. 8 coal seam roof cutting front to the right boundary 50 m. The

monitoring point data are organized and recorded in Figure 11.

When the working face continues to approach the fault, it has a great influence on the stress of the fault, and the stress of the fault zone increases greatly. The influence range of this stress concentration is from the fault plane to 38 m in front of the fault, and the maximum stress reaches 39.10 MPa, the pressure relief range continues to increase, and there is also a large stress concentration in front of the working face. At this time, there is gas with high content and high pressure in the hanging wall of the fault. If the stress exceeds the strength of the coal and rock mass, the coal and gas outburst accident will occur. Therefore, when the working face advances by 60 m (40 m from the fault), the gas within 20 m of the hanging wall of the fault is extracted.

After the No. 8 coal seam is advanced for 200 m, monitor the stress distribution of the coal and rock mass within 70 m above the roof of the No. 8 coal seam (a monitoring line is arranged every 10 m) and record the monitoring data, as shown in Figure 10, to determine the No. 8 coal seam. The size of the pressure relief range of the overlying coal strata after mining provides a basis for rationally arranging gas drainage in the fractured zone in the upper adjacent strata.

After 200 m of the No. 8 coal seam is pushed forward, the stress distribution of coal rock mass in the range of 70 m above the roof and 55 m at the bottom of the No. 8 coal seam was monitored (a monitoring line is set every 10 m), and the monitoring data were recorded. The monitoring results are shown in Figures 12 and 13. In this way, the pressure relief range of the overlying and underlying coal seams can be determined after the mining of the No. 8 coal seam, which provides a basis for the rational arrangement of gas drainage in the fault zone of the upper adjacent layer and the bottom heave fault zone of the lower adjacent layer.

The farther it is from the roof of the No. 8 coal seam, the smaller the reduction of the vertical stress of the coal seam. From the roof of the No. 8 coal seam to 40 m above, the
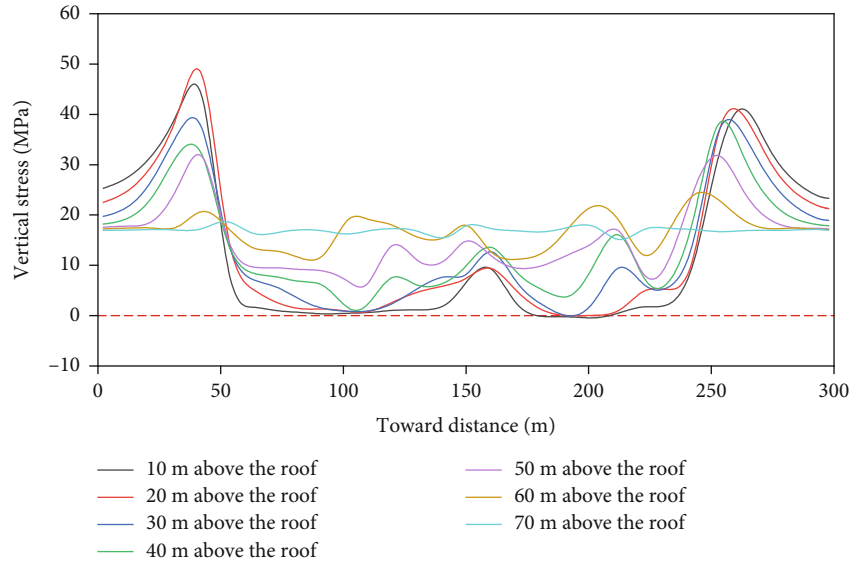
FIGURE 12: Distribution law of vertical stress in overlying roof strata of the No. 8 coal seam.
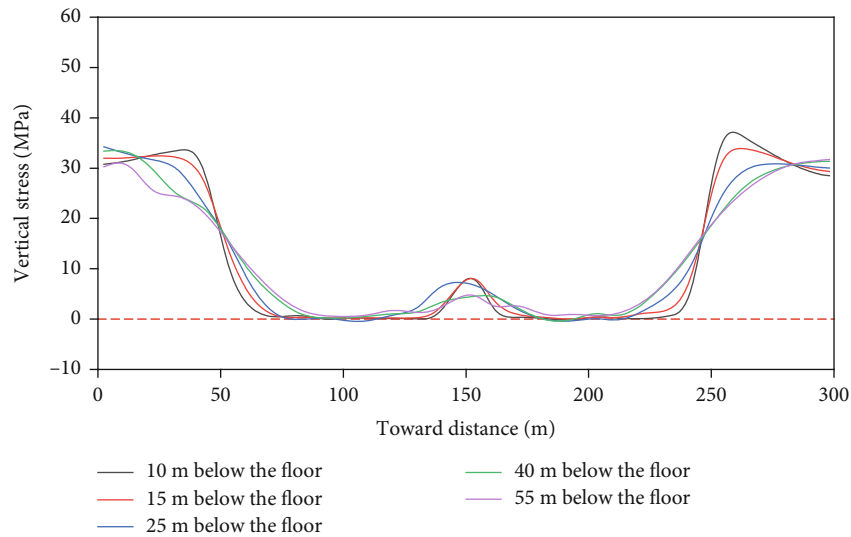

FIGURE 13: Distribution law of vertical stress in the underlying floor rock of the No. 8 coal seam.

vertical stress is low, and it is below 10 MPa as a whole, which has a good pressure relief effect. The overlying No. 6 and No. 7 coal seams are located in this pressure relief area. The pressure relief effect is poor at 50 m to 70 m above the roof, and the vertical stress at 70 m is basically maintained at about 18 MPa of the original in situ stress. Stress concentration occurs in the range of 10~20 m in front of the working face, and the farther away from this range, the smaller the influence on its stress. At the same time, the variation of the stress near the fault decreases with the increase of the distance from the roof, and the same rule is also observed behind the incision hole. It can also be seen from the figure that in the middle position along the advancing direction, the pressure fluctuates and rebounds. This is because with the increase of the advancing distance, the top plate collapses and the bottom plate stress increases.

Similarly, the farther the vertical stress of the underlying coal seam is from the No. 8 coal seam floor, the smaller the reduction of the coal seam stress. In the range from the lower part of the bottom plate to 25 m, the vertical stress is about 0 MPa, which has a very good pressure relief effect. In the range from 25 m to 45 m at the bottom of the bottom plate, the vertical stress is about 2 MPa and also has a good pressure relief effect. Near the fault, the reduction in stress is relatively small. The underlying No. 10 coal seam is located in the pressure relief area after of the No. 8 coal seam after mining.

When the No. 8 coal seam is being mined, the pressure relief gas from the overlying No. 6 and No. 7 coal seams and the underlying No. 10 coal seam will continuously move to the fissure zone and accumulate in the fissure zone, affecting the normal mining of the 13817 working face. The coal
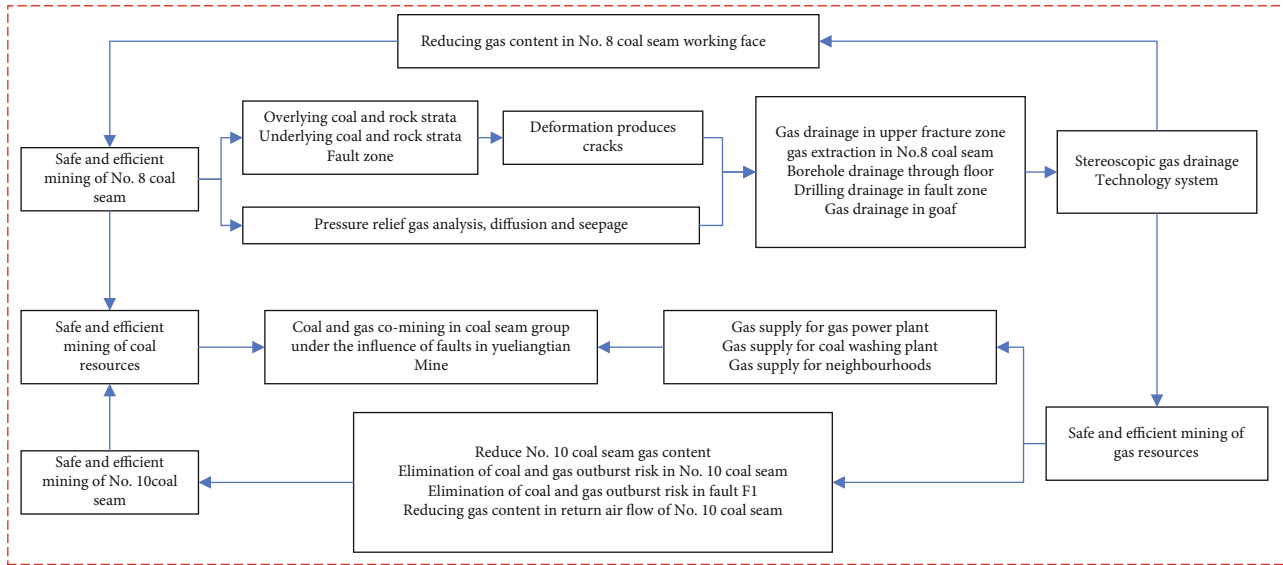
FIGURE 14: Co-mining technology system of coal and gas in coal seam group mining under the influence of fault in Yueliangtian Mine.

and rock mass within 40 m above the top of the working face and within 25 m of the bottom of the bottom plate have better pressure relief effects. Reasonable drainage of the gas accumulated in this range can ensure the safe production of the working face.

4. Design of Gas Drainage Scheme for Coal Seam Group under the Influence of Fault

4.1. Yueliangtian Mine Comining Technology System of Coal and Gas. Coal and gas coming technology means that coal and gas existing in coal seams are also mined as mine resources [35]. When mining coal seams, two complete systems of coal mining and gas extraction are formed. The movement and deformation of the surrounding rock can effectively extract the gas due to the pressure relief effect of the gas, so as to realize the safe and efficient mining of the working face. According to the technical characteristics of coal and gas comining and the occurrence characteristics of coal seams in Yueliangtian Mine, taking the No. 8 coal seam 13817 working face as an example, the technical system of coal and gas comining in the coal seam group under the influence of faults in Yueliangtian Mine is constructed. The technical system is shown in Figure 14.

The coming of coal and gas in Yueliangtian Mine can be divided into three stages according to the relationship between extraction and mining time: (1) predrainage of gas before mining, (2) coal and gas comining, and (3) gas extraction after mining.

4.2. The No. 8 Coal Seam Gas Predrainage. The predrainage of coal seam gas is to reduce the gas content and gas pressure of coal seam through a certain number of boreholes after a period of drainage, so that the elastic potential of coal seam is reduced, the stress of coal body is reduced, and the strength of coal body is correspondingly improved, and the permeability of coal seam is enhanced, so as to promote

the discharge of coal seam gas [36]. According to the above analysis, after passing through the F1 fault, the original gas content and pressure of the No. 8 coal seam have increased greatly, and the 13817 cutting roadway is close to the F15 fault, and there is a great risk of outburst in the annex of the cutting roadway. Therefore, in the No. 8 coal seam, it is not only necessary to take the predrainage of the coal seam, but also to take the predrainage of the cross-layer drilling.

During the excavation of 13817 haulage gateway and return airway, the coal seam drilling was adopted between F17 and F1 fault to extract the coal seam gas in the area. The coal seam drilling was used to control the contour of the left and right sides of the roadway, respectively, 15 m, and 9 boreholes were constructed in each group. The coal seam drilling retained 20 m advance distance; After the heading face passes through F1 fault, not only the boreholes in the coal seam are arranged between F1 and 13817 cutting roadways to extract the coal seam gas in the heading area, but also the construction cross-layer boreholes in the roof gas extraction roadway are arranged to preextract the coal seam gas in the heading area.

The horizontal distance between the roof gas extraction roadway and the return airway is 20 m, and the heading is carried out along the No. 6 coal seam (21 m from the roof of the No. 8 coal seam). A total of 417 m of the roof gas extraction roadway is constructed. A group of cross-layer boreholes are constructed every 10 m of the gas extraction roadway, and seven cross-layer boreholes are constructed in each group. The No. 8 coal seam predrainage gas drilling layout schematic diagram shows in Figures 15–18.

At the same time, according to the occurrence of the No. 8 coal seam in this area, considering the inclination length of coal seam working face, the change of coal seam thickness, and other factors, the bedding drilling is arranged in the haulage gateway and the return airway to extract the gas of the coal seam. The extraction of the No. 8 coal seam can
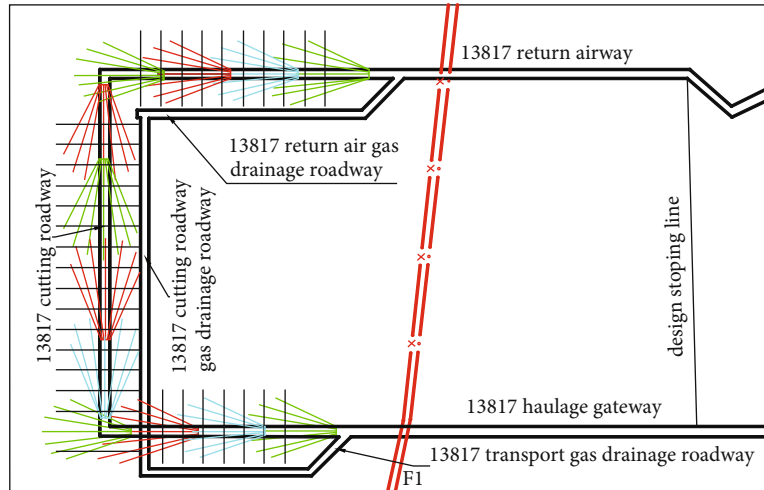
FIGURE 15: Layout plan of drainage borehole in 13817 heading face.
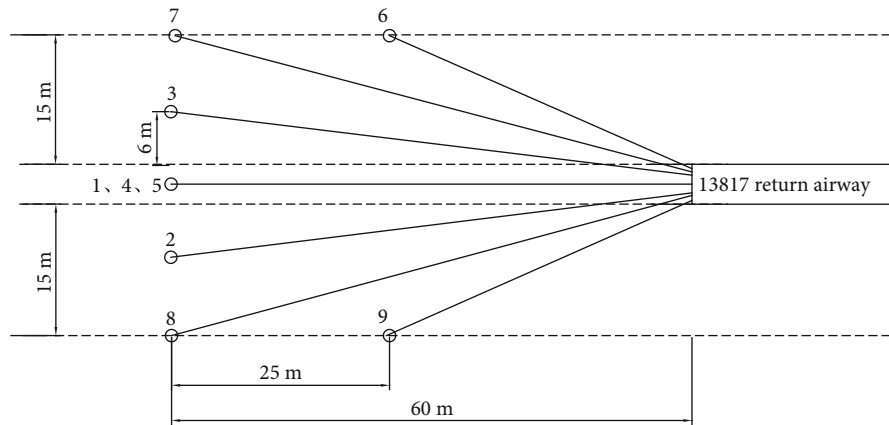

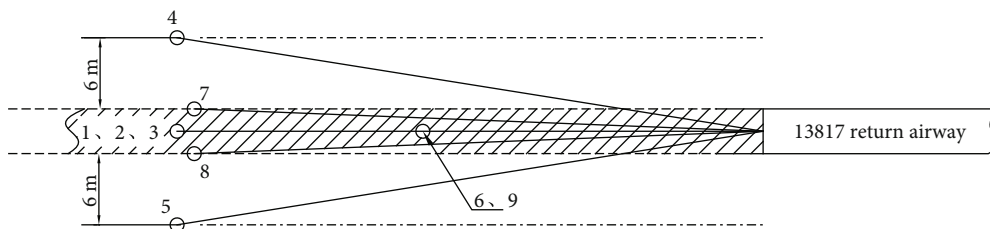FIGURE 16: 13817 drainage borehole layout plan of return airway.

FIGURE 17: 13817 return airway drainage borehole layout section.

be divided into two stages: The first stage is predrainage. The gas in No. 8 coal seam is preextracted by the borehole along the seam in advance. The second stage is that when the working face is normally advancing, the gas in the pressure relief zone in front of the working face can be effectively extracted by pumping while mining, and the drilling is gradually scrapped. Due to the mining action, the pressure of coal seam has been released, which greatly increases the permeability of coal seam and has good drainage effect.

The layout of predrainage boreholes in the coal seam of the haulage gateway and the return airway is as follows: the haulage gateway and the return airway are 15 m away from the cut position to the design stop line, and the roadway lengths are 301 m and 297 m, respectively. The upward and downward boreholes in the coal seam are constructed perpendicular to the haulage gateway and the return airway along the strike direction. The boreholes are all located in the mining coal seam, and their dip angles are consistent with the coal seam dip angles. The borehole diameter is $\Phi75$ mm, the borehole spacing is 3 m, and the borehole depth is 85 m. There are 100 prepumping boreholes in the upstream and 100 in the downstream of the construction.
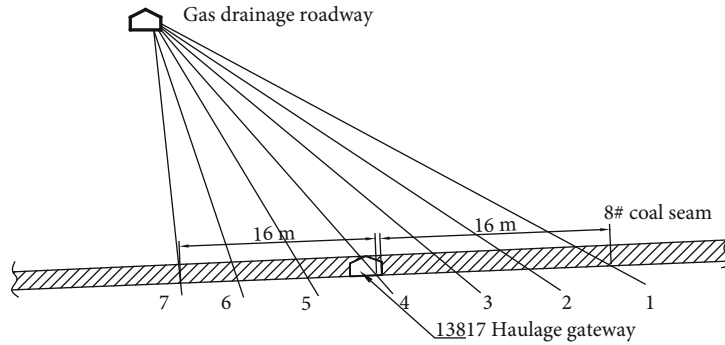
FIGURE 18: 13817 transport roadway heading face drainage borehole layout section.
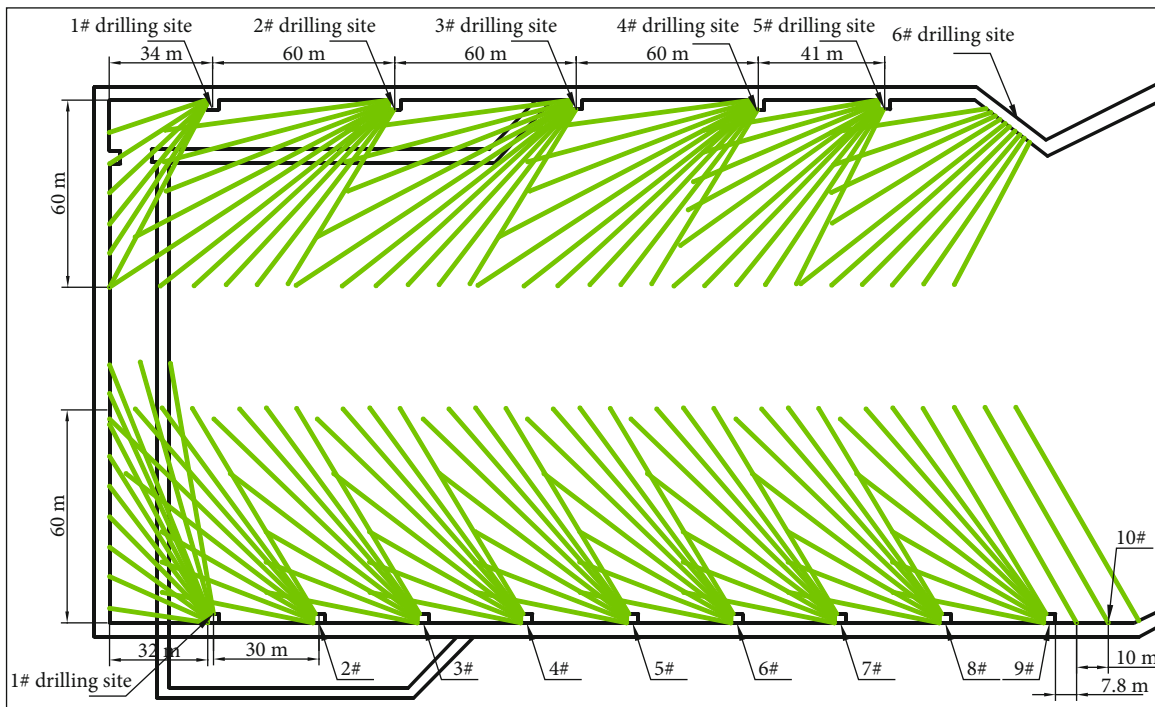
FIGURE 19: 13817 floor drilling layout plan of mining face.

4.3. The No. 8 Coal Seam Floor Crossing Borehole Gas Drainage.

Before mining at the mining face, a layer penetrating borehole is constructed in the haulage gateway and return airway of the No. 8 coal seam to intercept the pressure relief gas of adjacent No. 10 coal seam. The drilling was carried out 5 m below the floor of the No. 8 coal seam, which was located in the floor heave fracture zone of the No. 8 coal seam.

(1) *Layout of Drilling Hole through Haulage Gateway.* 10 drilling yards were arranged in the haulage gateway, and 1# drilling yard was constructed 32 m away from the cutting lane. 11 floor predrainage holes were constructed in the 1# drilling yard, and then, one drilling yard was arranged every 30 m. Eight floor predrainage holes were constructed in each drilling yard, and 3 floor predrainage holes were constructed

from the 10# drilling yard. The hole diameter was 75 mm, and the hole spacing was 10 m

(2) *Layout of Cross-Layer Drilling Holes for Return Airway.* Six drill yards were arranged in the return airway. 1# drill yard was constructed at 34 m outside the cutting lane, 6 floor drill holes were constructed in 1# drill yard, and 10 floor drill holes were constructed in 2#~6# drill yard, respectively. A total of 56 floor drill holes were constructed in return airway, with hole diameter of φ75 mm and hole spacing of 10 m.

The drill hole layout is shown in Figures 19 and 20.

4.4. Gas Drainage in High Drainage Roadway in Upper Fracture Zone.

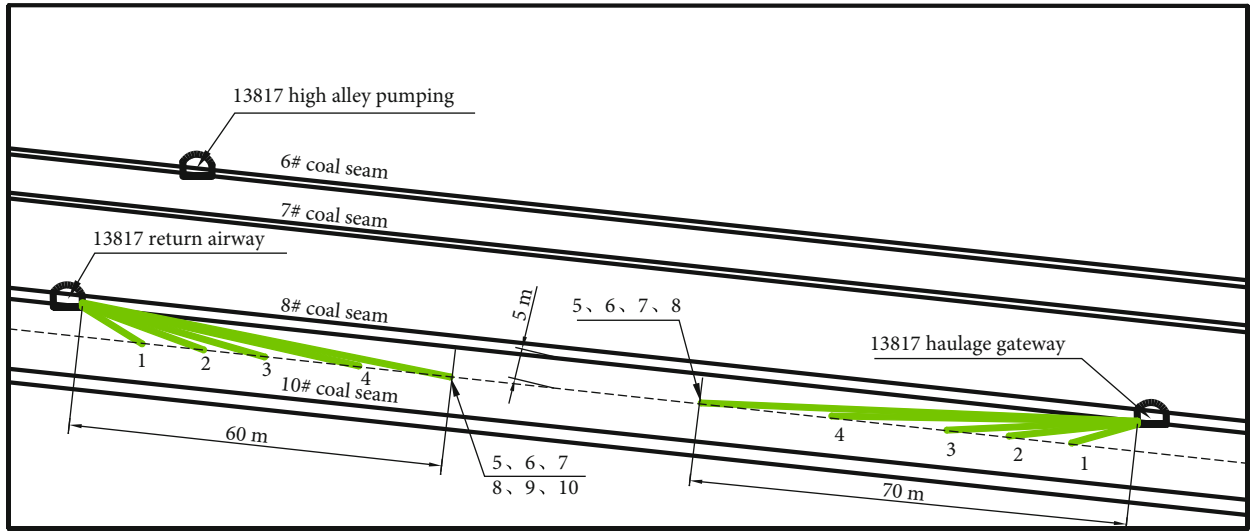By fault under the influence of close distance

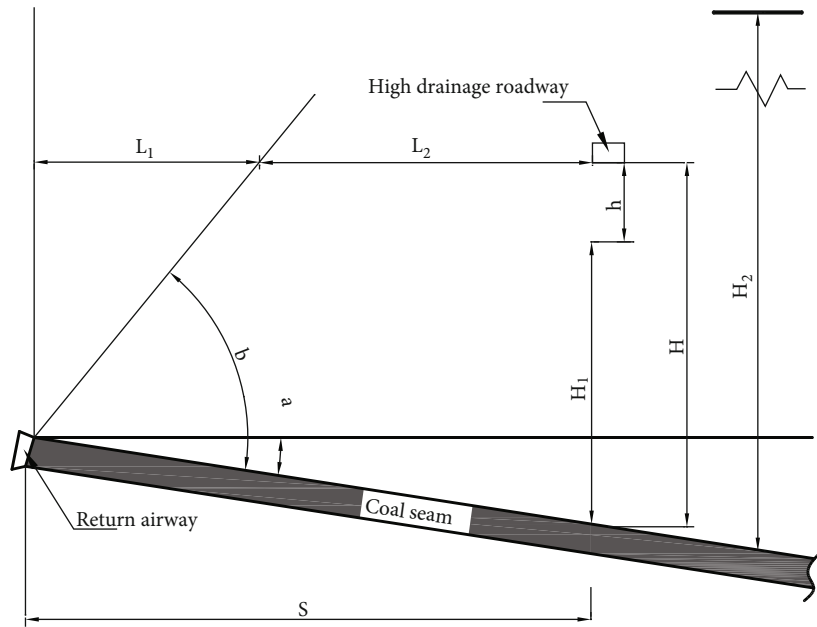Figure 20: Borehole profile of floor crossing layer of 13817 mining face.

Figure 21: Reasonable level layout of high drainage roadway.

coal seam group mining stress distribution and crack evolution law of the theoretical analysis and numerical simulation, it shows that after the No. 8 coal seam mining, due to the effect of mining, the No. 8 coal seam working face roof and floor and the fault zone will appear a lot of wear layer and delamination fracture. After fault activation, the gas near the fault will also migrate along the fracture to the mining space. A large number of mining-induced fissures provide passage and space for the migration and storage of pressure relief gas in No. 8 coal seam mining face. Goaf gas is one of the main sources of gas emission in working face. The determination of the best location of gas drainage in goaf is an important factor affecting the effect of drainage and determines whether the gas emission in working face

can be effectively reduced [37]. In order to maximize the gas extraction in the fissure zone enrichment area and goaf area above the roof of the No. 8 coal seam and avoid affecting the safety production of working face, the high drainage roadway is arranged to extract the gas in the goaf and fissure zone enrichment area of 13817 first working face.

The selection of reasonable horizon of high drainage roadway is shown in Figure 21.

It can be known from the figure above that the vertical height of the high drainage roadway from the roof of the mined coal seam is as follows:
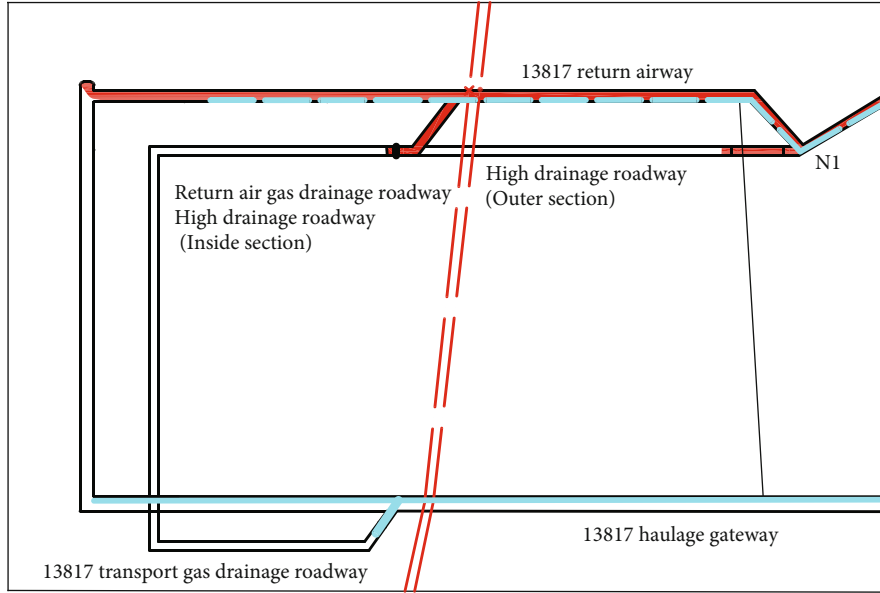
$$H = H_1 + h, \tag{7}$$

FIGURE 22: 13817 high drainage roadway layout plane diagram.

where H is the vertical height of high drainage roadway distance from mining coal seam roof, m; H_1 is the caving zone height, m; h is the height of high drainage roadway from the top of caving zone, m.

In order to achieve the best gas drainage effect, the height between the high drainage roadway and the top of the caving zone and the highest position of the high drainage roadway in the fracture zone can be determined by the following equation:

$$\begin{cases} h \geq (1.0 - 1.2)M, \\ h \leq H_2, \end{cases} \quad (8)$$

where M is the mining height, m; H_2 is the height of the fracture zone, m.

Equation (9) shows the determination method of horizontal distance between high suction lane and return airway.

$$S = L_1 + L_2, \quad (9)$$

where S is the horizontal distance between high drainage roadway and return airway, m; L_1 is the horizontal distance between unrelieved pressure zone and return airway, m; L_2 is the horizontal distance between high drainage roadway and unrelieved pressure area, m.

The horizontal distance L_1 between the unrelieved pressure zone and the return airway can be calculated by the following equation.

$$L_1 = H \cdot \cot(a + b) = \frac{H}{(a + b)}, \quad (10)$$

where a is the dip angle of coal seam, °; b is the pressure relief angle of roof strata of coal seam, °.

To ensure that the high drainage roadway is completely arranged in the fracture zone of overlying strata, the horizontal distance S between the high suction roadway and the return airway should meet the requirements of $S \leq (X/3)$, where X represents the mining length of working face (m).

After the No. 8 coal seam mining, the maximum height of caving zone on working face is 7.1 m, and the maximum height of fracture zone is 36.5 m. According to the above formula, the location range of high drainage roadway can be calculated: In the vertical direction, it is 9.3~43.6 m away from the roof of the No. 8 coal seam, and 11.5~30.5 m away from the return air passage in the horizontal direction. Combined with the above numerical simulation results, the pressure relief effect of coal and rock mass is good in the range below 40 m on the upper roof of the working face, cracks develop, and gas concentration is high in this range. High drainage roadway construction in the middle of the fracture zone can achieve better gas drainage effect. Meanwhile, considering that the position of the No. 6 coal seam is 21 m above No. 8 coal seam, it is easier to construct high drainage roadway along No. 6 coal seam than rock roadway. Therefore, the high drainage roadway of mining face 13817 is arranged 21 m above the roof of the No. 8 coal seam, and the horizontal distance between it and the return airway is 20 m.

Figure 22 shows the layout of high drainage roadway. The outer section of high suction roadway is opened from the measurement point N1 of 13817 return airway. After tunneling forward to expose No. 6 coal seam according to 166° azimuth angle and+17° inclination angle, the roof of the No. 6 coal seam is constructed to a horizontal distance of 30 m outside the F1 fault with a drop of 6 m. The horizontal distance between high suction roadway and return airway is 20 m, and the vertical distance between high suction

TABLE 4: 13817 coal mining face through the fault drilling parameters table of this coal seam extraction.

Hole number	Position\°	Angle of dip\°	Hole depth\m	Aperture\mm	Hole spacing\m	Quantity	Length\m
1~15	346	1	30	75	5	15	450
16~33	346	1	40	75	5	18	720

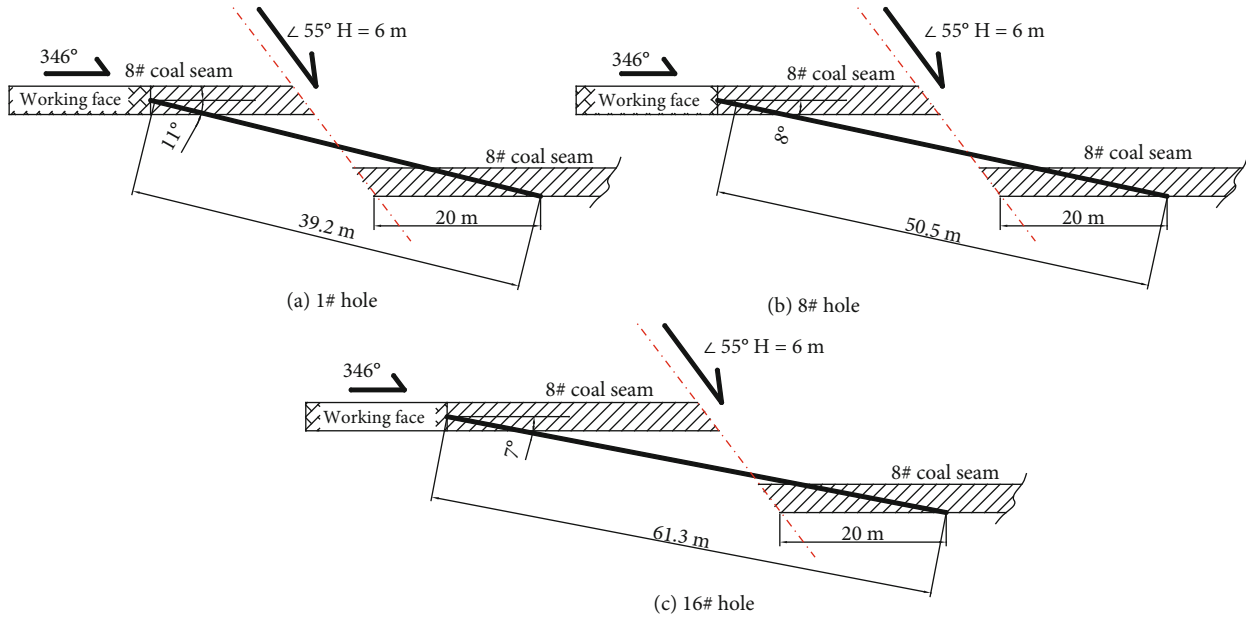(a) 1# hole

(b) 8# hole

(c) 16# hole

FIGURE 23: Drilling through layer layout profile.

roadway and return airway is 21 m. The 13817 return air gas drainage roadway was used as the inner section of the high drainage roadway, which was combined with the outer section of the high drainage roadway to extract the gas in the fissure zone and goaf above. Two extraction pipelines with diameter of 350 mm were installed in the slope changing position of the high suction roadway, and the high suction roadway was closed for extraction.

4.5. Drilling Gas Drainage. Before the 13817 mining face crosses the fault, the coal seam drilling and layer penetrating drilling are carried out to extract gas from the coal seam and surrounding rock within the fault range. The specific arrangement is as follows.

(1) The coal seam extraction drilling hole

When the 13817 coal mining face is mined to 30~40 m away from the fault, a local coal seam drill hole is constructed every 5 m along the coal wall of the working face. 1#~15# drill hole is 30 m deep, 16 #~33 # drill hole is 40 m deep, and a total of 33 drill holes are constructed. The drilling angle can be properly adjusted to ensure that the drilling construction in the coal body, the completion of construction in time to seal the hole network pumping, and the negative pressure of the hole to adjust to more than 13 KPa. Drilling parameters are shown in Table 4.

(2) Drilling hole for through-layer extraction

The drilling angle can be adjusted appropriately to ensure that the drilling is carried out in the coal body. According to the numerical simulation analysis, when the working face is 40 m away from the fault, the influence range of stress concentration in the fault zone is 20 m from the fault plane to the front of the fault. The maximum stress reached 27.04 MPa, so the drilling hole for translayer construction should cover 20 m range of the No. 8 coal seam on the other wall of the fault. In order to extract gas as much as possible, the cross-sectional view of the drilling arrangement is shown in Figure 23. At the end of construction, the hole should be sealed and pumped in a timely manner, and the negative pressure of the hole should be adjusted to more than 13 KPa. The drilling azimuth is 346°, the dip angle is -7°~-11°, the hole depth is 39.2~61.3 m, and the aperture is 75 mm.

4.6. Gas Drainage with Buried Pipe in Upper Corner. After the No. 8 coal seam was mined, the pressure relief gas from the adjacent coal seam will flow into the gob along the cracks. Affected by air leakage, the gas in the gob will gather on the return airway of the upper corner, and the gas concentration is high. Buried pipe in this area for gas extraction has good effect on the control of gas accumulation in the upper corner. The drainage main pipe is arranged in the
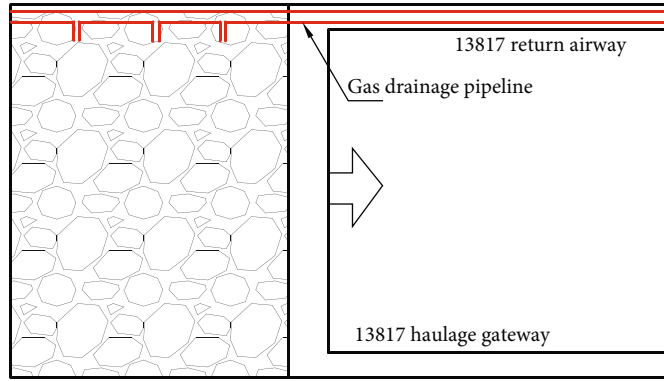
FIGURE 24: Schematic diagram of gas in goaf with buried pipe at upper corner.
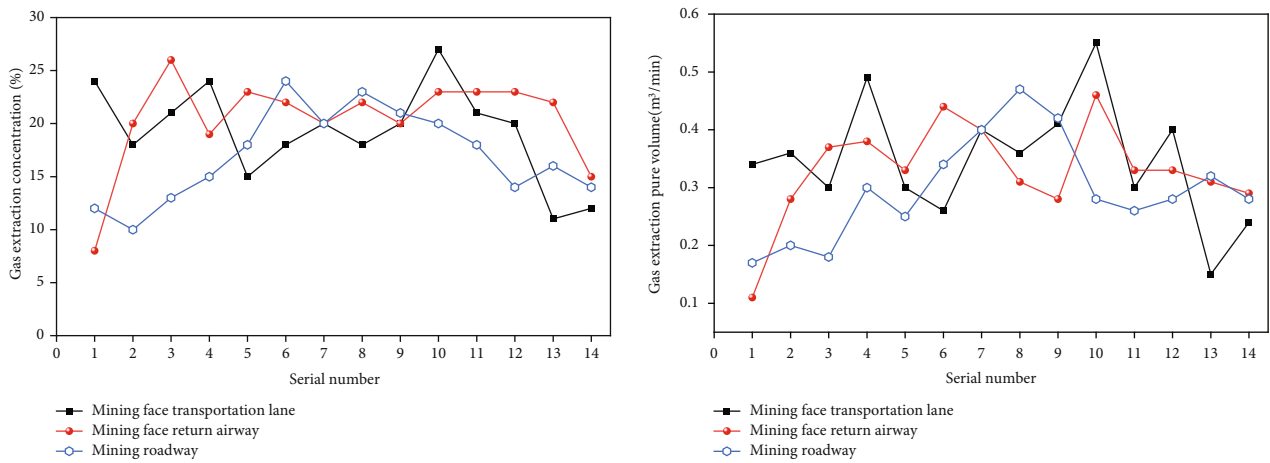

FIGURE 25: The concentration and scalar quantity of gas drainage in this coal seam.

return airway, and a tee is installed every 30 m on each branch pipe, and a gate valve is added to the tee, shown in Figure 24.

5. Analysis of Optimization Effect of Gas Drainage in Coal Seam Group under the Influence of Fault

The investigation of the optimized gas drainage effect of the 13817 working face will help to clarify the actual application situation on the site and to supplement and improve the gas drainage technology in the future, so as to improve the overall gas drainage effect and ensure the safe production of the working face.

5.1. Analysis of the Effect of Gas Drainage in Coal Seam. In order to investigate the gas drainage effect of the coal seam, we analyzed the gas drainage of the coal seam within 70 m outside the roadway of 13817 working face and measured and counted the gas in the transportation roadway, return airway and roadway of 14 groups of working face every 7 days. For sampling concentration and scalar, statistics is shown in Figure 25.

From the extraction situation, the gas extraction concentration of haulage gateway and return airway is about 20%,

the pure amount is about 0.35%, the gas extraction concentration of cutting roadway is about 17%, and the pure amount is about 0.3%. The cumulative drilling drainage amount of the coal seam in the haulage gateway is 55,460.38 m³, the cumulative drilling drainage amount of the coal seam in the return airway is 46,776.87 m³, the cumulative drilling drainage amount of the coal seam in the cutting roadway is 5,966.61 m³, and the cumulative gas drainage amount is 108,203.86 m³. According to the calculation formula of residual gas content after gas predrainage [38], it can be concluded that the gas extraction rate of coal seam reaches 52.9%, and the extraction concentration is about 20%, which reduces the gas emission of 13817 mining face and largely ensures the safe production of the working face.

5.2. Effect Analysis of Mining Face Extraction System. In the high-drainage roadway, two 14-hour gas extraction pipelines are used for extraction. The gas concentration is 26%~30%, and the mixed flow rate of extraction is 111.32~121.63 m³·min⁻¹. During the mining process, the local gas concentration in the upper corner is between 1.3% and 1.8%. The gas source of the mining face is analyzed. The No. 6 and No. 7 coal seams overlying the 13817 mining face are all thin coal seams, and the gas content is small and
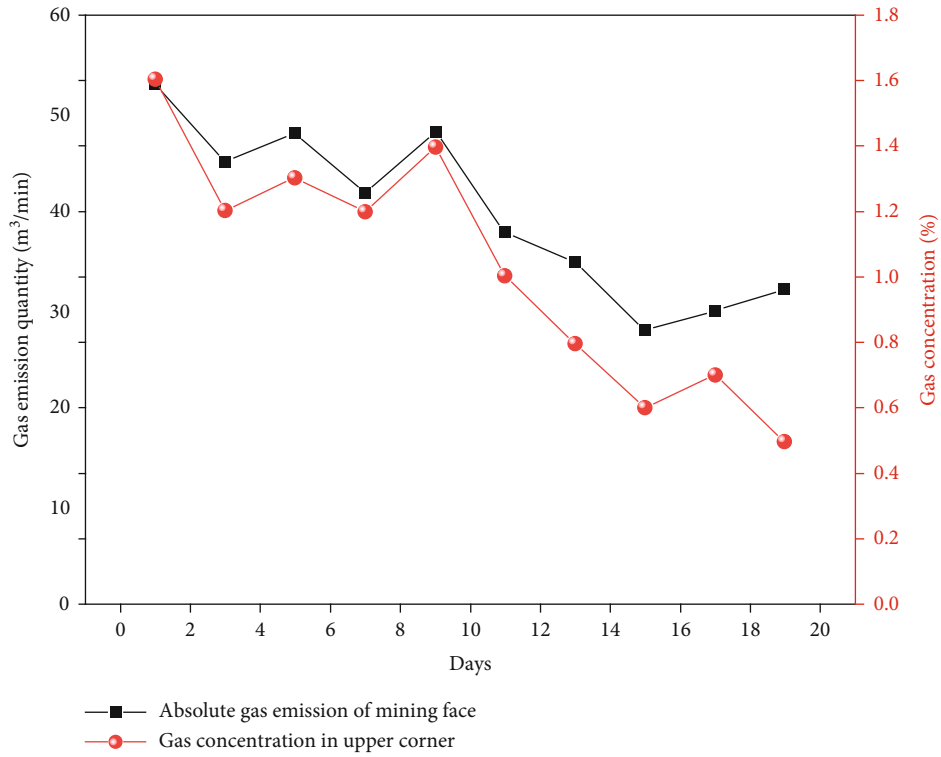
FIGURE 26: Statistical chart of absolute gas emission and gas concentration in the upper corner of the mining face.
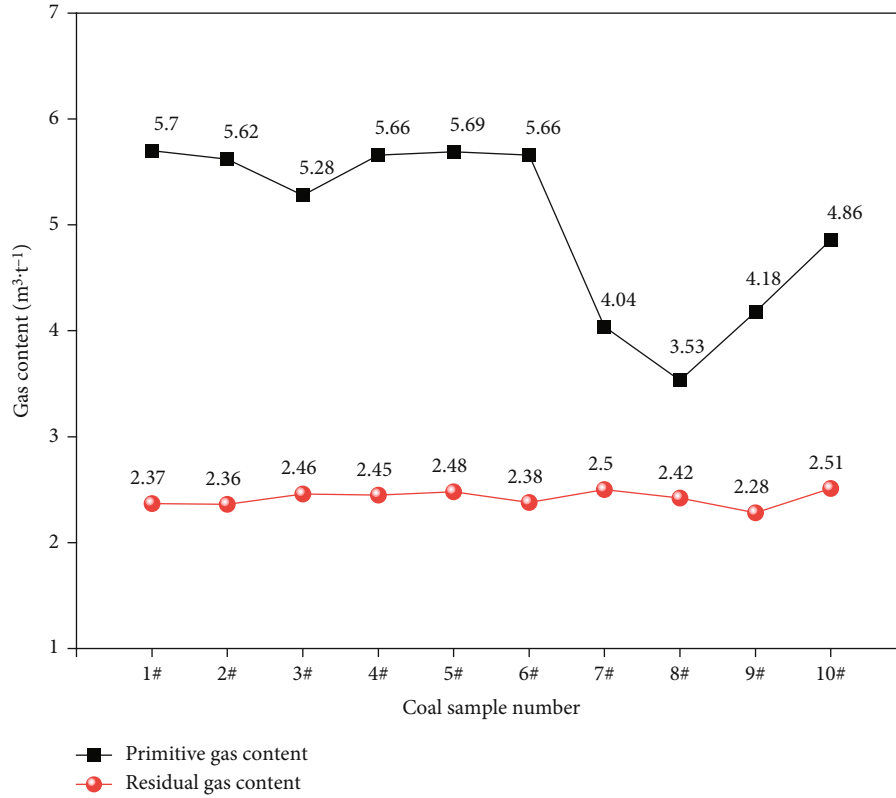

FIGURE 27: Determination results of gas content after coal seam extraction in fault zone.

high. Part of the gas extracted from the roadway is invalid for gas control. Therefore, the extraction system is adjusted, and another extraction pipeline with a high extraction roadway is used to extract the corners of the mining surface. The extraction situation after statistical adjustment is shown in Figure 26. The absolute gas emission from the mining face has decreased to a certain extent, from $42\,m^3\cdot min^{-1}$ to $53\,m^3\cdot min^{-1}$ to $28\,m^3\cdot min^{-1}$ to $36\ m^3\cdot min^{-1}$, the decrease rate is about 33%, and the local gas in the corners of the mining face has also been effectively controlled and reduced to less than 1%.

5.3. Analysis of Gas Drainage Effect in Working Face. In the process of mining, due to the short predrainage time of borehole in the coal seam of the mining face and the thick coal seam, the gas concentration of the gas in the back flow increases by 0.2%~0.3%, and the gas concentration in the back flow reaches 0.65%~0.78%. In order to reduce the amount of gas in the coal seam of the mining face, the discharge boreholes are evenly arranged along the direction of mining face. The construction depth of the drainage hole is 8 m to 10 m, and the spacing of the holes is 1.5 m. The drainage hole after construction is completed shall be drained in a timely manner, and the drainage shall be stopped at the mining surface to allow natural gas discharge. The lead distance of the drainage hole shall be not less than 5 m, and the drainage hole shall be continued after the recovery is in place. After the optimization of borehole layout, the analytical gas of coal falling on the mining face is greatly reduced, and the gas concentration of return flow is reduced to less than 0.5%.

5.4. Analysis of Gas Drainage Effect in Fault Zone. After drilling the coal seam gas in the fault zone, the residual gas content of the No. 8 coal seam within 20 m near the fault was measured. The comparison between the measurement results and the original gas content of the coal seam is shown in Figure 27. It can be seen that the fault zone drilling after the gas was extracted, and the gas content decreased by 60% compared with the original gas content, which largely eliminated the prominent danger of the fault zone and ensured the safe passage of the working face through the fault.

6. Conclusions

This paper takes the coal mining and gas drainage of 13817 working face in Yueliangtian Mine as the research background, analyzes the gas source of working face and the influence of fault on gas, studies the pressure relief range of fault zone through laboratory research and FLAC^{3D} numerical simulation analysis method, and optimizes the design and effect investigation of coal seam group coal and gas comining system. Through research and analysis, the following conclusions are drawn:

(1) The original gas content and pressure were measured by sampling the coal near fault F1, and the influence of fault on the gas occurrence of No. 8 coal seam was analyzed. It was concluded that the gas content and pressure in the upper wall of fault were high, and the upper wall of fault had a closed effect on gas, and the lower wall of fault had an open dissipation effect on gas. The gas content and pressure reduction zones were formed within 20 m and 6 m of the lower wall of fault F1

(2) According to the mechanical parameters obtained from the test, the vertical stress changes of adjacent coal seams and the deformation and failure characteristics of surrounding rock during mining near F1 fault in No. 8 coal seam are studied by numerical simulation experiment. On this basis, the pressure relief range and gas migration law of coal seam in the fault zone were analyzed, and the pressure relief effect was good within the range of 40 m at the top of the working face and 25 m at the bottom of the floor

(3) The coal seam group coal and gas comining system under the influence of faults in Yueliangtian Mine was constructed, and the gas comprehensive extraction was carried out for each coal seam and fault zone. The layout and parameters of boreholes and roadways were designed and optimized. The investigation results show that the absolute gas emission of the working face is reduced to 28~36 m^3/min, and the gas concentration in the upper corner of the working face is reduced to about 0.6%. The analytical gas of falling coal in coal mining face is greatly reduced, and the gas concentration of return air flow is reduced to less than 0.5%. The gas extraction rate of No. 8 coal seam is 52.9%, and the gas content in the fault zone is reduced by 60%, which achieves the expected extraction effect

In summary, through theoretical research, laboratory analysis, and numerical simulation analysis, the coal and gas comining system under the influence of faults in Yueliangtian Mine was determined, and the gas extraction technology was reasonably optimized and designed, and good results were achieved. However, it is worth noting that this work is based on the moon mine, and the geological conditions of different mines are different. It should be further studied to improve the efficiency and versatility of gas drainage technology under complex geological conditions.

Data Availability

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

Conflicts of Interest

The authors declare that they have no conflicts of interest.

Acknowledgments

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