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Study on Safety Mining Technology of Gob in Stopping Face by Replacing Pressure Equalization with Gob Pumping—A Case Study of Sitai Mine

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Abstract: Gas control in the upper corner of the natural coal mining face with high gas is always a difficult problem that troubles the safe production of the working face. Among them, a high gas-prone natural coal mining face with ground air leakage is more likely to cause gas and CO to exceed limits in the corner of the working surface and is difficult to control. The traditional treatment methods often have some problems; for example, it is easy to increase air leakage in the gob with the method of gas extraction in the gob, which is not conducive to the prevention and control of spontaneous combustion of coal in the gob. At present, the more effective method is the pressure-equalization method. However, the pressure-equalization measures need to establish a complex pressure-equalization system, and close cooperation between the systems is required; once the system power fails or equipment failure occurs, the pressure-equalization state changes randomly, and it is easy to cause gas over-limits and other faults. Therefore, this paper presents a new method to control gas in the gob of a coal seam by pumping the gob of the upper-adjacent layer, using the negative pressure of pumping, and balancing the negative pressure of the upper-adjacent layer and the gob of the coal seam to form a new pressure-equalization relationship. This method can prevent the toxic and harmful gases in the goaf of the upper-adjacent layer from escaping into the passageway of the gob of the local coal seam, reduce the air leakage in the goaf, and benefit the gas control and spontaneous coal combustion prevention in the goaf.

Keywords: coal; gas control; negative pressure extraction; pressure equalizing relation; numerical simulation

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1. Introduction

A total of 56% of coal mines in China have spontaneous ignition problems [1]. More than 70% of the state-owned key coal mines are high-gas mines [2], and high-gas mines, with a tendency of spontaneous coal ignition, account for about 50% [3]. Due to the double pressure of high gas emissions and easy, spontaneous combustion of coal during mining, the contradiction of safety production in coal mines is becoming more and more prominent. While improving the gas safety of the working face, the risk of natural ignition in the gob will increase; on the contrary, reducing the risk of natural ignition in the working face will easily lead to a gas over-limit, and there will be an imbalance between two types of safety hazards in the treatment of one or the other. At present, the loss of high-quality coal caused by natural-2 combustion has reached more than 4.2 billion tons, resulting in sluggish resource reserves of more than 200 million tons/year, and economic losses of up to billions of yuan. Safe production of coal mines is hindered by high-yield and high-efficiency fully mechanized mining, gas over-limits of fully mechanized caving faces, and spontaneous combustion of goaf coal.

In Inner Mongolia, Shaanxi, Qinghai, Shanxi, and other regions, there are shallow-buried deep coal seam group mines. Generally, the upper coal seam is mined first and then the lower coal seam. After mining the upper coal seam, the coal rock mass in the gob area collapses, the surface collapses, and the cracks in the gob area develop at the surface. When mining the lower coal seam, the gob of the lower coal seam is connected with the gob of the upper coal seam to form a super-large gob that connects with the surface. In the case of negative-pressure ventilation, when the pressure energy of the goaf is greater than the pressure energy of the working face, the gas in the goaf of the upper coal seam enters the working face through the return air side of the lower coal seam. As a result, gas in the upper corner of the lower coal seam exceeds the limit, which seriously affects the normal production of the mine and threatens the safety of underground workers [4,5].

The problem of gas leakage in the goaf of the upper coal seam occurs in shallow and deep coal-seam group mining, and the pressure-equalization technology has achieved a good control effect. The technique of pressure equalization is to rationally allocate the mine ventilation system so that the pressure difference between the air leakage sources can reach a certain degree of balance and reduce or eliminate air leakage. On the other hand, a normal pressure drop at both ends of the air leakage channel can be achieved by adjusting the air chamber, damper, etc., so as to reduce the oxygen supply by air leakage and prevent spontaneous coal combustion [6].

The technology of pressure-equalization regulation has been widely used in mines at home and abroad. A scholar summarized the method of pressure equalization to solve the problem of spontaneous coal combustion, which was successfully applied in coal mines. In the early 1960s, Polish scientists proposed unilateral and bilateral connecting pipe pressure to regulate gas chambers, etc., and added measures to equalize the pressure and prevent fires [7]. In the 1980s, a scholar [8] studied the influence of the parameters of the pressure-equalizing system, such as natural wind pressure, local fan working points, and fire wind pressure, on the pressure-equalizing system. In the 1990s, the KJ7 automatic monitoring and regulation system [9,10] for pressure equalization and fire prevention was jointly developed by several institutions. The development of equal pressure ventilation for fire prevention provides a theoretical basis for positive pressure ventilation to deal with the problems of return airflow and the emissions of toxic and harmful gases in the upper corner of the underground working face, as well as insufficient oxygen concentrations.

In recent years, a scholar [11] proposed that in order to prevent the massive accumulation of gas in the goaf from flooding into the working face in Sunjiawan Coal Mine, when the top coal is caved in layers under mining, pressure-equalization ventilation should be adopted to prevent the gas from flooding into the working face. Some scholars [12] implemented pressure equalization and fire prevention technology in the well when solving the problem of large-area CO leakages and achieved remarkable application effects. Some scholars [13] developed a complete automatic control pressure-balancing system based on pressure-balancing technology. A scholar [14] proposed that in order to prevent the emissions of toxic and harmful gases, such as CO, pressure-equalization ventilation measures should be adopted at the working face to make the pressure in the goaf lower than that at the working face, thus preventing harmful gases from entering the working face. A scholar [15] put forward that the main reason for the impact of support withdrawal due to the presence of a flammable caving face is that there are hidden dangers of spontaneous combustion in the goaf.

The ventilation method of the working face adopts positive pressure ventilation and other anti-fire measures, such as nitrogen injection into the goaf, to eliminate hidden dangers of combustion in the goaf and ensure the safe withdrawal of the working face. A scholar [16] established an automatically controlled positive pressure ventilation system based on research on the principle of working face pressure-equalization ventilation technology. The system was applied in the field to ensure that the pressure-equalization area was 50 Pa and that the pressure in the pressure-equalization area was controlled. Some scholars [17–19], while researching the actual situations of Xiaokang Mine, Yangquan Coal

Mine, and Liyang Coal Mine, proposed that during the demolition and withdrawal of the working face, the problem of fire prevention during the withdrawal was solved by adopting methods of pressure-equalization ventilation and nitrogen injection in the goaf. Some scholars [20,21] put forward the method of pressure-balancing ventilation combined with nitrogen injection in the goaf as well as plugging up and down the mine according to the abnormal CO emissions at the working face of Yunfei Mining and Longhua Coal mines, effectively preventing toxic and harmful gases from entering the working face. Based on the research by the above scholars, it was found that the local positive pressure ventilation technology of the coal mine's underground working face is mainly used to solve the abnormal emissions of toxic and harmful gases in the return airflow and upper corner of the working face, as well as the insufficient oxygen concentration. Measures such as nitrogen injection and grouting in the goaf during working face withdrawal are used to prevent fire in the goaf. Pressure-equalization technology can effectively prevent and control gas leakages in the goaf of upper coal seams and reduce the air leakage in goafs. But the equalizing technique has its own drawbacks. Pressure-equalizing technology relies too much on ventilation facilities, such as pressure-equalizing fans and wind windows, and once the relevant facilities fail, the pressure distribution of the working face will change abruptly. It causes the gas in the goaf of the upper coal seam to leak rapidly, which will easily produce unexpected consequences [22–24].

To sum up, traditional pressure-equalizing technology uses pressure-equalizing fan and air windows to increase the goaf's pressure in the lower coal seam and achieve the purpose of equalizing goaf pressure with that of the upper coal seam so as to avoid the goaf gas leakages in the upper coal seam. In order to avoid excessive reliance on the pressure-equalizing fan and wind windows and solve the problem of gas leakage in the goaf of the upper coal seam, this paper proposes the use of a gas extraction system to reduce the goaf pressure in the upper coal seam, which can also achieve the effect of pressure balance with the lower coal seam as well as avoid gas leakages in the upper coal seam and ensure normal mining of the lower coal seam. It also improves the reliability and disaster resistance of the system.

2. Research Method

2.1. Background of Coal Mine

Sitai mine is a high-gas, easily ignitable natural mine. At present, the mine is exploiting a Jurassic 14# coal seam, a 14# layer overlying an 11# layer and 12# layer gob, with a layer spacing of 5.2–11 m and an average layer spacing of 8.03 m, as well as a layer spacing of 16.3–21.3 m and an average layer spacing of 18.8 m. Due to the small spacing between layers 11# and 12#, each working face of layer 14# was affected by the goaf of layers 11# and 12#, as shown in Figure 1. After the early stage of mining, the goaf roof fell behind and formed a connecting relationship with the goafs of layer 11# and layer 12#'s overlying layer. Under the action of negative-pressure air flow, harmful gases were stored in the goaf and quickly leaked into the working face, resulting in the phenomenon of gas over-limit in the upper corner, with the highest gas concentration reaching 1.6%. Therefore, during the mining of layer 14#, pressure-equalizing system ventilation was used to control the gas of the working face. However, due to the poor stability of the pressure-equalizing system and many other safety hazards, exploring a set of reliable gas management technologies that can replace the pressure-equalizing system and solve the gas problem of the working face is urgent.

Taking 81113 as the working face and 8723 as the working face of layer 14# as examples, in order to solve the gas problem of layer 14#'s working face, this paper sets forward the comprehensive treatment measures of pre-pumping the goaf of layer 12#, overlaying the gas before mining, pumping while mining, and rationally deploying the air volume of the working face.

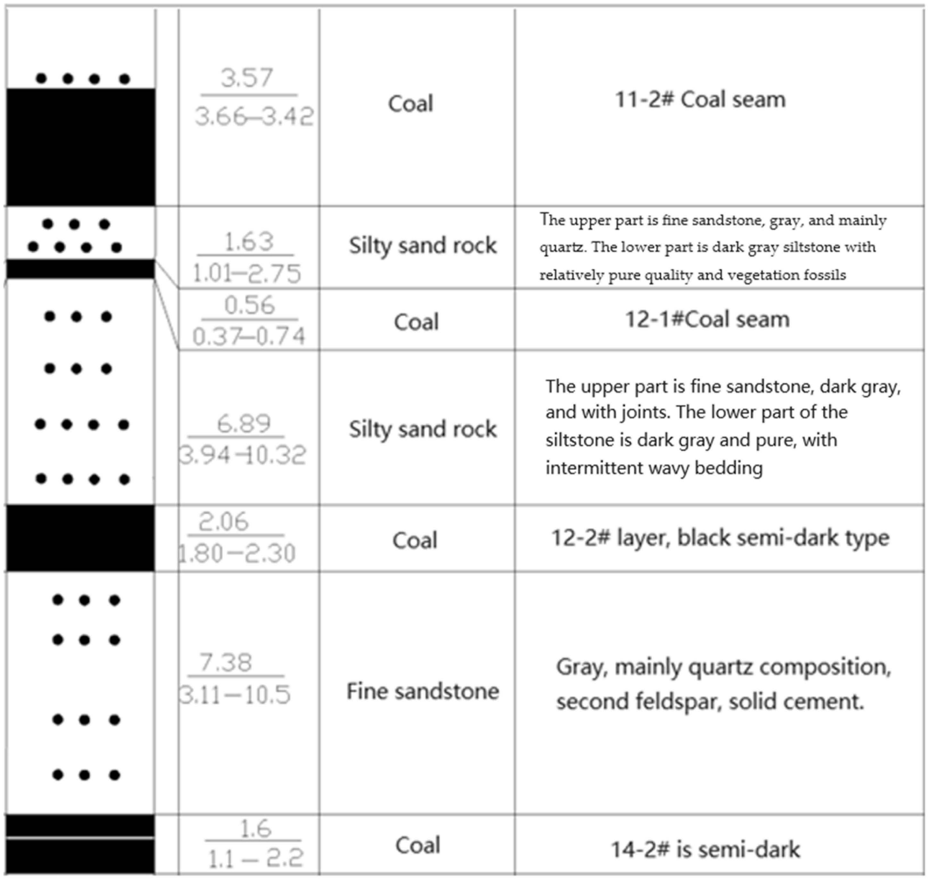


Figure 1. Coal seams 11#, 12#, and 14# relationship diagram.

2.2. *Overburden Failure Model of Shallow-Buried Coal Seam*

(1) The establishment of a numerical model

According to the material characteristics of a coal seam's overlying rock, the Mohr–Coulomb model was adopted by using UDEC7.0 numerical simulation software to study the failure state of overlying coal rock in a goaf during mining [25].

(2) Physical model establishment

The command flow was used to simulate coal-seam mining to realize the step-by-step mining process. The excavation sequence can be described by the following physical model according to the developing mining situation.

(3) UDEC can only establish a two-dimensional model of a coal seam, and the total size of the model (length × width) is 400 m × 346.85 m, as shown in Figure 2 below.

(1) The step-by-step excavation processes of the 11# coal seam, 12# coal seam, and 14# coal seam are as follows:

① 7.72 m × 200 (10) m;
② 0.4 m × 200 m (10) m;
③ 3.49 m × 200 (10) m;

(2) Setting of boundary conditions

The model's boundary conditions are set as follows: The model origin is set in the lower left corner of the coal seam; the lower surface of the model is a fixed boundary; the left and right sides of the boundary are fixed horizontally; the vertical direction is set as a free moving boundary; the initialization speed and displacement are zero; and the upper surface is set as a free moving boundary. In order to observe the change law of overlying rock subsidence and surface subsidence caused by mining, monitoring points were set at the positions of y = 24.8, 40.75 m. The monitoring points were arranged on the center line of the model along the strike (x = 0) of the working face to monitor the movement amount of overlying rock subsidence and surface subsidence.

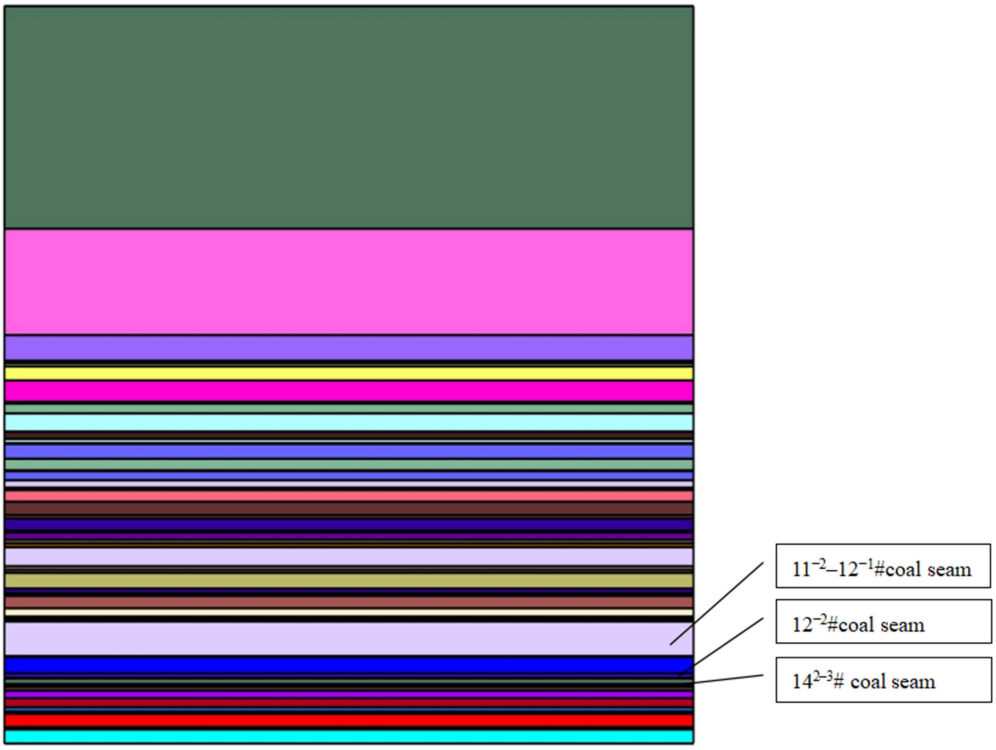


Figure 2. Physical model of coal-seam excavation.

2.3. The 81113 Surface Crack Leakage Test

2.3.1. Principle of Continuous Constant Release of Tracer Gas to Detect Mine Air Leakage

A continuous quantitative release method was adopted; that is, a certain amount of SF6 gas was continuously released at the main inlet of the air leakage path, and then gas samples were taken at the exits of several pre-estimated air-leakage channels. By analyzing whether the gas samples contained SF6 or concentrations of SF6, the air leakage channel and the amount of air leakage were specifically determined.

Air Leakage Detection Method

In a roadway with continuous negative-pressure air leakage, the release device was placed at R on the air inlet side of the air leakage point, and the SF_6 tracer gas was continuously and quantitatively released, with the release amount being q (m^3/min). The sampling point Si was placed below the release point, and the concentration C_i of the SF_6 tracer gas was measured by sampling. The sampling point S_{i+1} was placed on the downwind side of the air leakage point and the concentration of SF_6 tracer gas C_{i+1} was measured by sampling. The release and reception positions of the tracer gases are shown in Figure 3. According to q, C_i, and C_{i+1}, we can calculate the air leakage of the shaft and roadway.

The air leakage in the shaft and roadway can be calculated according to Equation (1).

$$\Delta Q_i = \frac{q(C_{i+1} - C_i)}{C_{i+1}C_i} \quad (1)$$

In the equation

ΔQ_i—air leakage between the R_i point and R_{i+1} point in the detected well lane, m^3/min;

q—SF_6 gas release, m^3/min;
C_i, C_{i+1}—the SF_6 gas concentrations at measuring points *i* and *i* + 1, respectively.

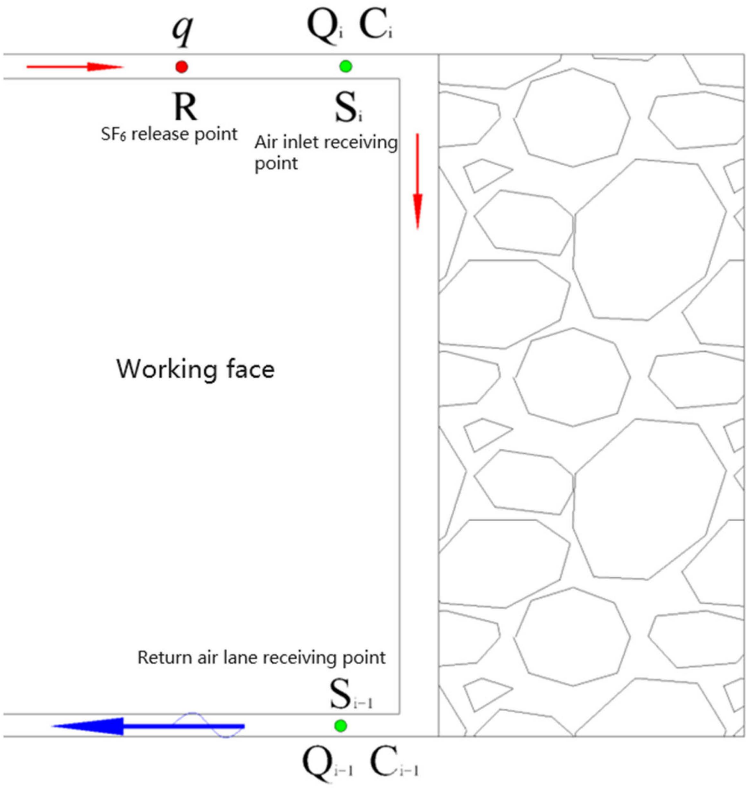


Figure 3. Schematic diagram of the release and reception positions of the tracer gas.

2.3.2. Determine the SF_6 Tracer Gas Release Point

The negative-pressure ventilation is at the 81113 working face, and the SF_6 tracer gas release point should be at the stable air flow at the air inlet. Figure 4 shows the schematic diagram of the air leakage measurement of the 81113 working face.

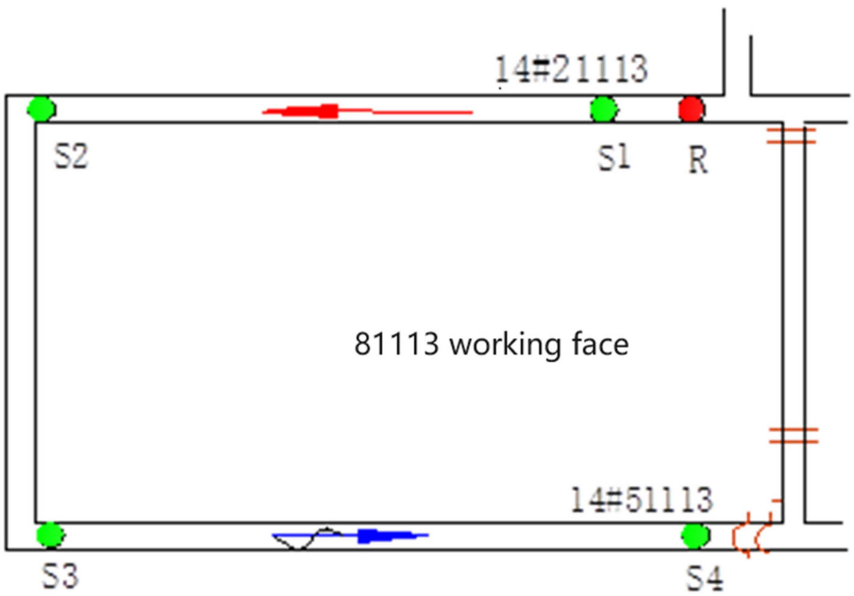


Figure 4. The 81113 working face air leakage measurement diagram.

2.3.3. Tracer Gas Detection and Gas Sample Collection

According to Equation (1), the influence coefficient ranges from 4 to 5. The minimum concentration of SF_6 tracer gas is expected to be 2×10^{-6}. The air distribution volume of the 81113 working face is 1240 m^3/min. The estimated SF_6 release volume ranges from 9.9 to 12.4 L/min. The actual value is 10 L/min.

The safety requirements of the SF_6 release device are connected, and the personnel at each sampling point carry the SF_6 measuring device for the coal mine in place and are ready to begin the measurement work. The release point R begins to release SF_6 tracer gas continuously and steadily at a release rate of 10 L/min. The measurement results are shown in Table 1.

Table 1. Measurement results.

Measuring Point	Determination of Concentration (ppm)	Remark
S_1	10.1	/
S_2	10.07	/
S_3	9.64	/
S_4	0.15	/

2.3.4. Calculation of Air Leakage

The calculation results of the air leakage volume and air leakage rate are as follows:

(1) $\Delta Q = q(C_2 - C_1)/C_2 \times C_1 = 10 \times 10^{-3} \times (9.15 - 10.1) \times 10^{-6}/(9.15 \times 10^{-6} \times 10.1 \times 10^{-6}) = -102.8$ m^3/min

The negative-pressure air leakage is 102.8 m^3/min.

(2) $\alpha = (C_2 - C_1)/C_2 \times 100\% = (9.15 - 10.1) \times 10^{-6}/9.15 \times 10^{-6} \times 100\% = -10.38\%$

The negative-pressure air leakage rate is 10.38%.

2.4. Test Method of Pressure Equalization Effect

The dynamic resistance distribution of the 81113 working face was measured in order to study the resistance distribution along the inlet and return roadway of the working face when the working face was advancing. A total of 7 measuring points were arranged for the measurement of the working face, as shown in Figure 5.

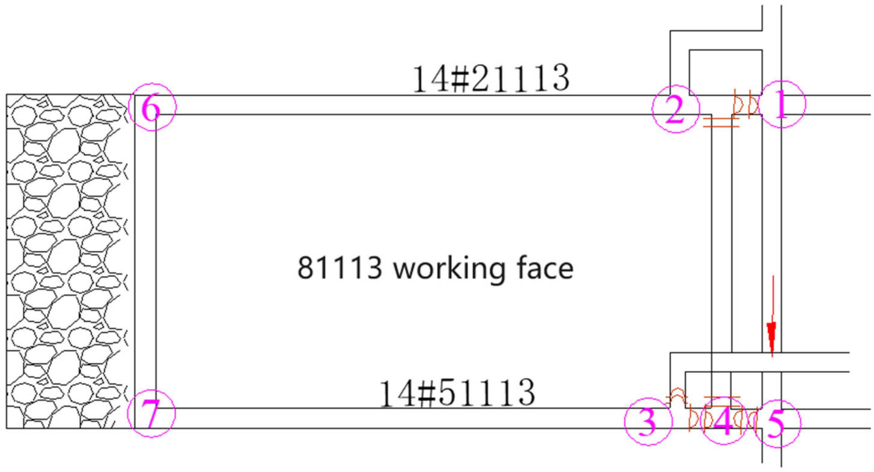


Figure 5. The 81113 schematic diagram of measuring points for dynamic resistance measurement of the working face.

3. Results and Discussion

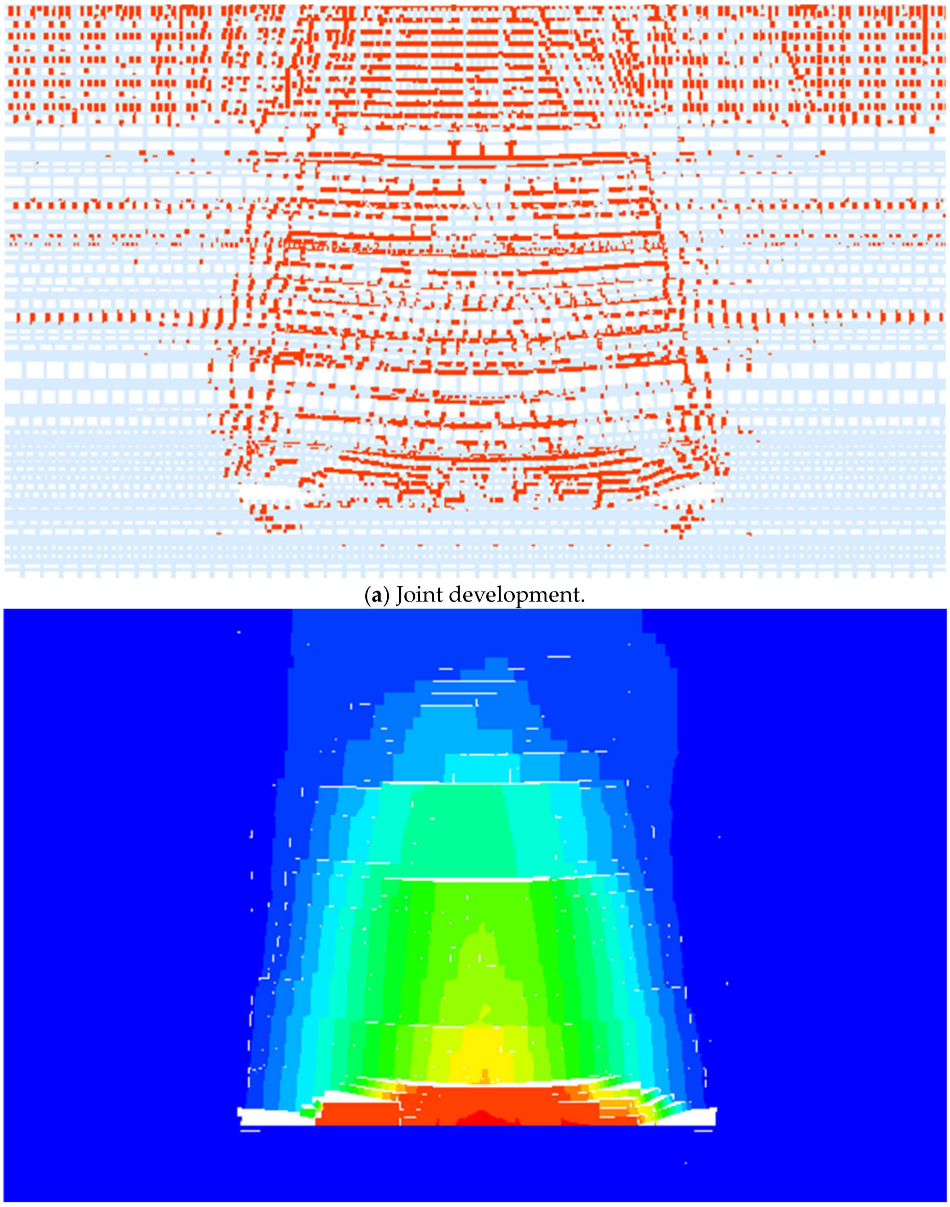
3.1. Research on Overburden Failure Law in Mining of Shallow-Buried Coal Seam Group

The distribution of the plastic zone in the numerical simulation results reflects the failure of rock mass at the top and bottom of the coal seam after mining. The thickness of the 14^{2-3}# coal seam in Sitaizhu Coal Mine is 3.49 m, which belongs to medium-thick coal-seam mining. The higher the mining height, the more intense overburden activities, and the more serious mining deformation and failure. There are 11^{-2}–12^{-1}# coal seams above the 14^{2-3}# coal seam in Sitai Coal Mine, with a total of 7.72 m; the 12^{-2}# coal seam is 0.4 m, and the interval between the two coal layers is 2.22 m. The mining sequence of this coal mine is to mine 1^{1-2}#, 12^{-1}#, and 14^{2-3}# coal seams from top to bottom. Therefore, when mining the 14^{2-3}# coal seam, the mining of the 11^{-2}–12^{-1} coal seam has ended, the roof of the coal seam has collapsed, and the rock stress will be re-distributed; so it will have a greater impact on the mining of the 14^{2-3}# coal seam.

The fracture development and fracture zone of the rock layer when the No. 11^{-2}–12^{-1} coal seam was mined are shown in Figure 6a,b. With the gradual mining of the working face, different degrees of mining failure occur in the overlying coal strata and the underlying coal strata. With the continuous progress of the working face, the mining failure depth of the overlying coal strata gradually increases, and the distribution characteristics of mining failure of the overlying coal strata present an “inverted trapezoid” distribution, respectively. The middle of the overlying strata collapsed to the bottom of the coal seam, and cantilever support beams were formed at both ends. When the overlying strata were extracted at 200 m from the working face, the failure height of the roof’s overlying rock in the goaf reached 121.82 m, and the failure depth of the coal rock on the bottom floor reached 6.38 m. As the working face continues to advance, limited by the mining height of the mining coal seam, the mining failure height of the overlying coal strata in the goaf is basically maintained at 120–125 m, and the mining failure depth of the underlying coal strata is basically maintained at 7.43 m left to right. The joint development and fracture zone distribution after the mining of the No. 11 and 12 coal seams are shown in Figure 6.

Coal seam 14^{2-3}# is 8.23 m away from the No. 11 coal seam and 5.61 m away from the 11^{-2}–12^{-1}# coal seam. The spacing between coal seams is small, and the overlying coal rock is greatly disturbed when mining the 14^{2-3}# coal seam. The mining process was simulated by UDEC, and the results of the overlying strata are shown below, where Figure 7’s joint development and fracture zone distribution of coal seam 14^{2-3}# after mining is displayed.

Since the No. 11 and No. 12 coal seams have been mined during the mining of the 14^{2-3}# coal seam, the rock layer has been affected by mining and has produced more cracks, up to 35 m. With the gradual mining of the working face of the 14^{2-3}# coal seam, the cracks of the overlying coal strata are also evolving, while the mining damage of the underlying coal strata occurs to different degrees. With the continuous progress of the working face, the mining failure depth of the overlying and underlying coal strata in the goaf gradually increases, and the distribution characteristics of mining failure of the overlying coal strata are “trapezoidal,” respectively. When the 14^{2-3}# coal seam is mined, the failure height of the roof’s overlying rock in the goaf reaches 141 m, and the failure depth of the bottom coal rock reaches 21 m. With the continuous advancement of the −15 working face, the mining failure height of the overlying coal strata in the goaf is basically maintained at 135–145 m, and the mining failure depth of the underlying coal strata is basically maintained at about 22 m due to the restriction of the mining height of the coal seam.

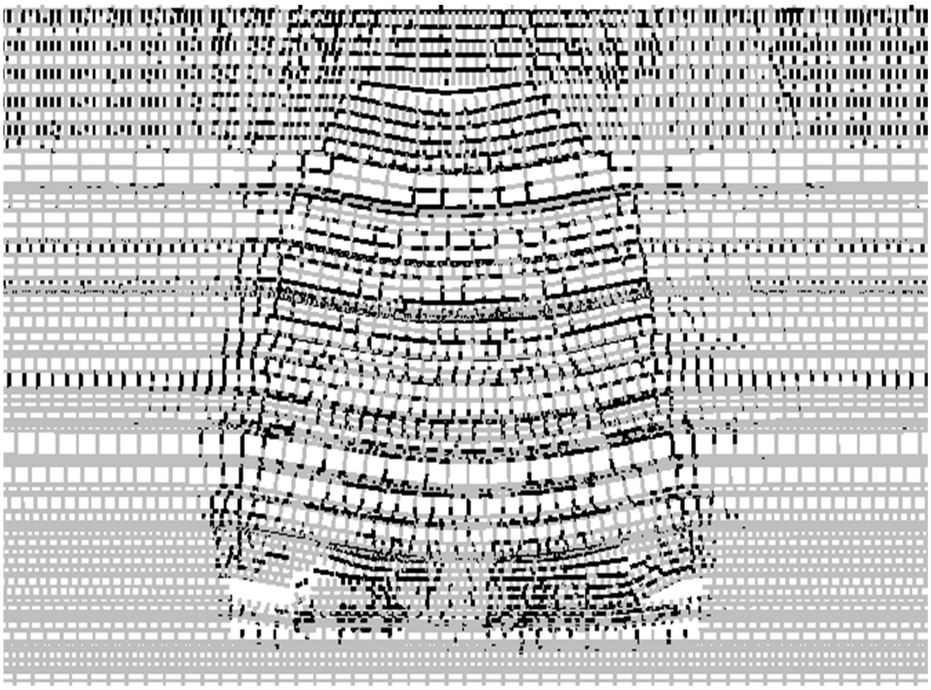


(**a**) Joint development.

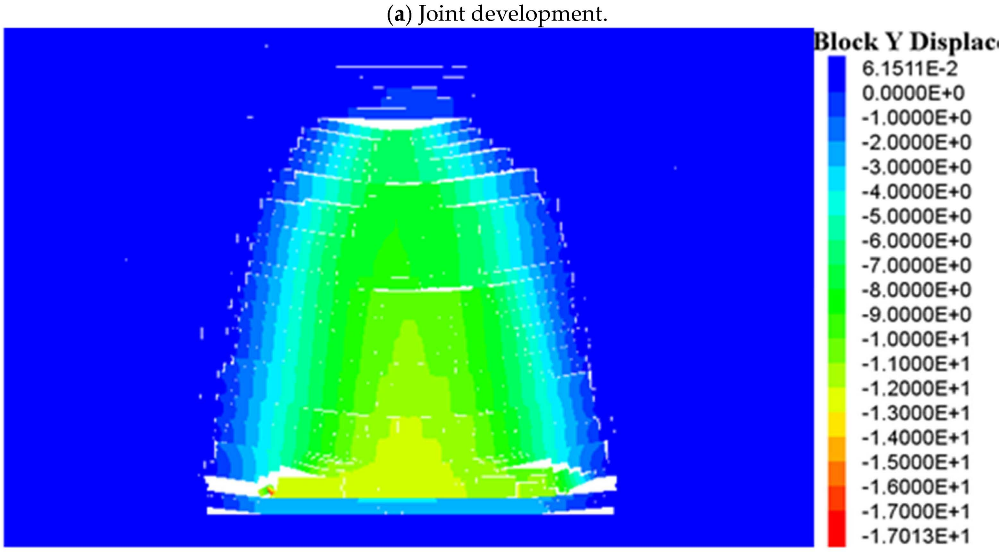
(**b**) Distribution of fracture zone.

Figure 6. Joint development and fracture zone distribution of No. 11 and No. 12 coal seams after mining.

The average distance between the 14^{2-3}# coal seam and the surface is 322.05 m. After the mining of the 14^{2-3}# coal seam, the cracks have developed at the surface under the joint action of the goaf of the No. 11 and No. 12 coal seams, so the surface air leakage has a significant impact on the goaf of the 14# coal seam. According to the observation of gas change data in the upper corner during the mining period of panels 410 and 412, after the roof caving of the ancient tang during the initial mining period, the goaf on this surface formed a connection relationship with the goaf on the overlying layer and the adjacent surface. Combined with the influence of atmospheric pressure and the mine's negative pressure on the goaf, the harmful gas in the overlying and adjacent goaf was abnormally released, resulting in insufficient oxygen in the upper corner area. The oxygen concentration is between 12% and 16%, which is prone to low oxygen injury accidents.



(**a**) Joint development.



(**b**) Distribution of fracture zone.

Figure 7. The 14^{2-3} coal seam joint development and fracture zone distribution after mining.

3.2. The 81113 Implementation Method and Effect Analysis of Common Pressure Equalization on Working Face

The general pressure-equalizing method of the 81113 working face is to install a local ventilator with an air door in the inlet lane of the working face, set up a regulating air window in the return air lane of the working face, and increase the pressure in the inlet lane and the return air lane of the working face to reduce the pressure difference between the two points. The adjustment facilities are installed, as shown in Figure 8.

When the combined pressure regulation of the fan and the air window is adopted, the pressure of the fan is consumed by the resistance of the regulating air window when the air volume is unchanged. The system has little influence on the ventilation system of the whole mine, and the wind pressure of the ventilation network outside the pressure-regulating system will not be affected.

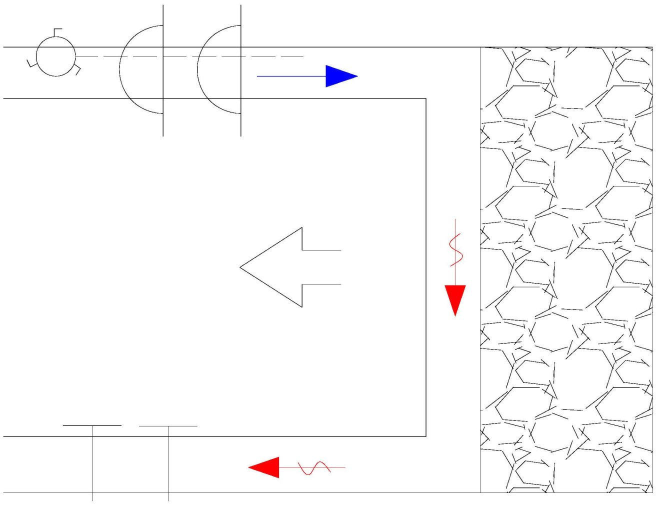


Figure 8. Diagram of pressure regulation between fan and window.

In the production process of the 81113 working face, the pressure change of each measuring point is shown in Figure 9, and the pressure difference between each measuring point and the wellhead is shown in Figure 10.

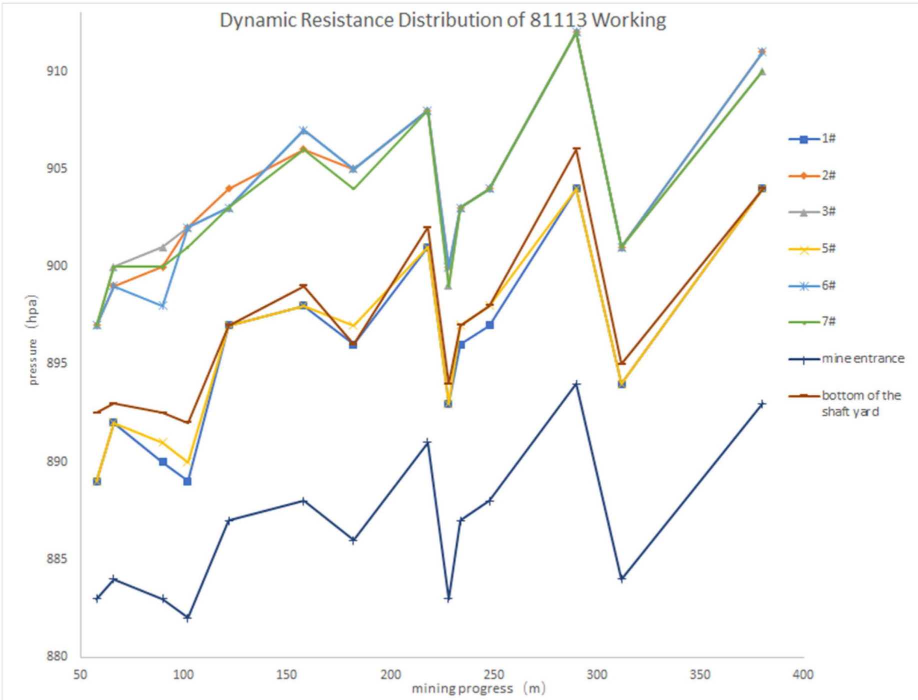


Figure 9. The 81113 schematic diagram of dynamic resistance change of working face with propulsion.

As can be seen in Figure 9, the pressure change at one point of the working face varies with the influence of surface atmospheric pressure. In order to accurately study the dynamic distribution law of the resistance of the inlet and return roadway of the working face, the influence of surface atmospheric pressure should be excluded to directly reflect the distribution law.

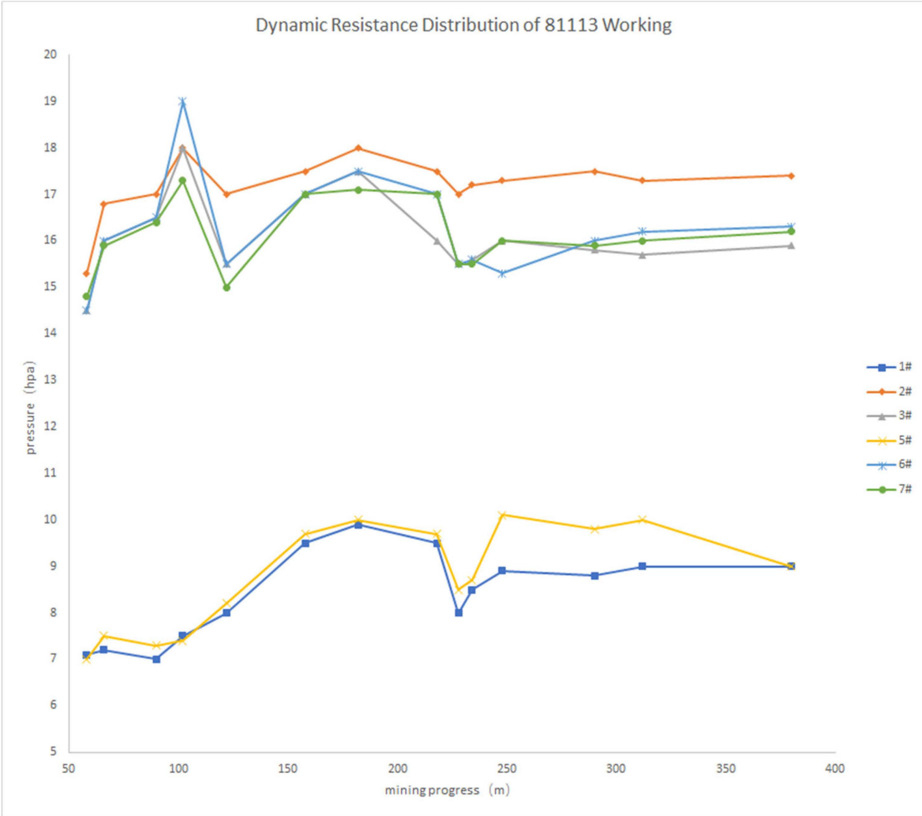


Figure 10. Schematic diagram of the difference in pressure values between each measuring point and wellhead.

As can be seen in Figure 10, during the mining process, when periodic pressure is applied, the cracks between the goaf and the overlying goaf increase, and harmful gases from the overlying coal-seam rush into the goaf of the coal seam. Only by increasing the wind pressure of the equalizing fan—the pressure along the inlet and return air roadway—can the harmful gases from the overlying goaf flow to the working face of the coal seam be effectively prevented. After periodic pressure, we advanced with the working face; after the collapse of part of the gob, the crack channel between the overlying gob and the coal seam was less than that without the collapse. It is necessary to appropriately reduce the air pressure of the equalizing fan; that is, reduce the pressure along the inlet and return air roadway, which can effectively inhibit the harmful gas flow to the coal seam working face in the overlying gob. Therefore, the preliminary analysis shows the variations in dynamic resistance distribution in the inlet and return roadway of the pressure-equalizing working face, which are affected by the variations in atmospheric pressure and the periodic pressure of the working face.

Although the pressure-equalizing system can effectively inhibit the intrusion of toxic and harmful gases into the working face, the stability of the system is restricted by the pressure-equalizing fan. Once the pressure-equalizing fan fails, the pressure of the working face drops, and toxic and harmful gases will immediately enter the working face, posing a serious threat to working face staff, so it is necessary to adopt a new pressure-equalizing method.

3.3. Implementation Method and Effect Analysis of Pressure Equalization in Fissure Pumping of 8723 Working Face

In order to improve the reliability of the pressure-equalization system, the fracture pumping pressure-equalization method is adopted in the 8723 working face. By reducing the pressure of the fracture zone, the pressure balance between the fracture zone and the goaf is achieved so as to prevent the toxic and harmful flow of the overlying goaf to the goaf and then prevent the gas from entering the working face.

3.3.1. Implementation of Fracture Zone Extraction

(1) Supporting gas pumping station

Using the newly built underground 307-panel gas pumping pump station, which is located between the return lane of track 2725 on the 12th # layer, the pumping system was officially placed into operation on 1 December 2017. The type of pumping pump is a ZYB-200/250 water ring vacuum pump with a rated flow rate of 200 m^3/min and motor power of 250 KW. A total of two gas drainage pumps are installed in the room; one is running, and one is on standby.

(2) Gas drainage method

From the return air lane of the 307-panel area to the 12# layer 8723 goaf covered by the 14# layer 8723 working surface, three groups of holes (four holes in each group) are constructed to carry out the gas drainage of the goaf covered by the 14# layer's 8723 working surface, so as to achieve "full drainage" and prevent gas leakage of the overlying goaf.

It can be seen in Figure 11 that gas drainage is carried out by 12 boreholes in three groups on the 8723 working face to prevent gas leakage in the overlying gob.

Figure 11. The 307-panel pumping system diagram and drilling layout diagram.

3.3.2. Implementation Effect of Fracture Zone Pumping

(1) The gas in the gob covered by layer 12# 8723 was reduced from 10.5% to 4.5% through the gas extraction of the gob covered by layer 12# 8723 for 5 months before mining. Both the gas sampling and laboratory results of the gob and the on-site manual observation data show that the gas concentration in the gob showed an obvious downward trend. The absolute gas emission in the 307-panel area is 7.39 m^3/min, the gas extraction concentration in 307 is 6.55%, the gas extraction pump quantity is 107.84 m^3/min, the negative pumping pressure is 11 kpa, and the extraction rate is 52%.

(2) During the mining of the 8723 working face of layer 14#, the goaf of layer 12# 8723 continues to be pumped to ensure that the working face of layer 14# 8723 can still produce normally without taking pressure-boosting measures, and the gas concentration in the upper corner is stable at about 0.3% during this period. Compared with the gas concentration of 0.7% in the upper corner of layer 14# 8721 of the previous working face, the gas concentration has decreased significantly. The change in gas concentration in the corner of the working surface of layer 12# 8723 goaf and layer 14# 8723 are shown in Figures 12 and 13.

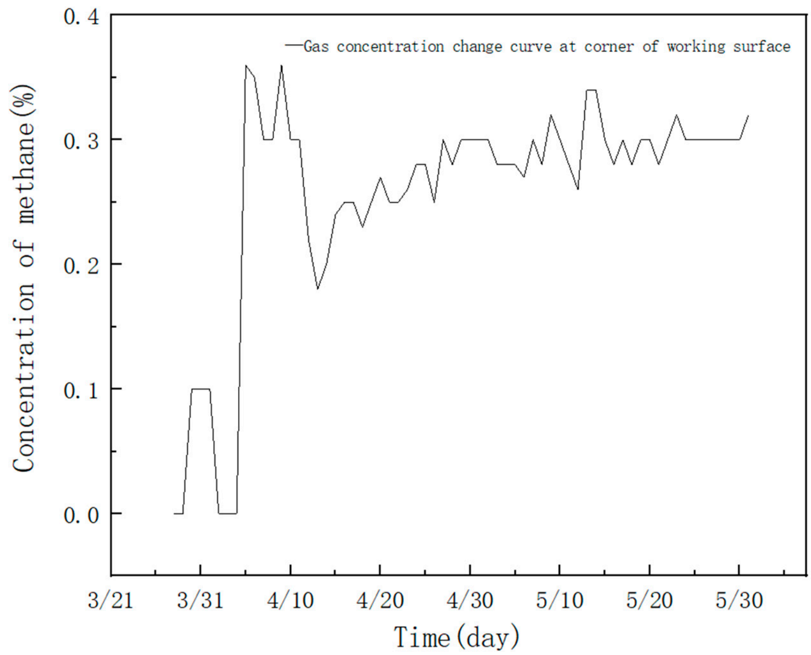


Figure 12. It can be seen in Figure 12 that the gas content in the upper corner of the working face is low after mining. After the first collapse, 3/4 days later, the gas content increases to about 0.3%.

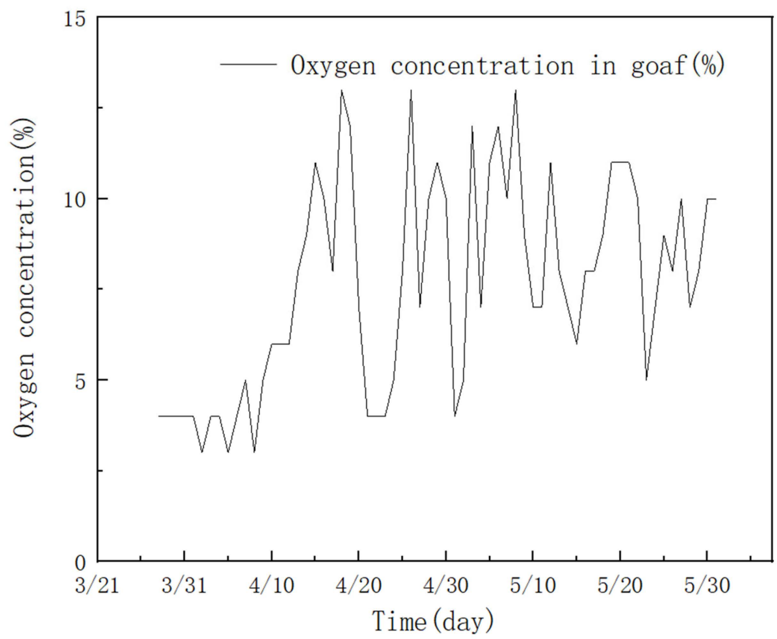


Figure 13. It can be seen in Figure 13 that the oxygen concentration in the goaf varies greatly, which is related to the pump opening time.

(3) The analysis of the overall change trend of gas is as follows: ① The gas concentration in the goaf shows an overall decline trend because the gas concentration in the goaf decreases after a long time of extraction; ② the oxygen concentration in the goaf increases as a whole because the fresh air flow from the air inlet trough of the working face 8723 of layer 14# enters goaf 12# through the working face and maintains a low balance after stopping; ③ and the carbon monoxide concentration has no significant change, which has been maintained below 24 ppm.

4. Conclusions

In this study, we discussed a new coal mine safety mining technology that is, by using upper-adjacent layer goaf gas extraction instead of the traditional pressure-equalization method to solve the problem of high-gas mine safety mining. Through the case study of the Sitai mine, this study verified the effectiveness of extraction technology in reducing gas over-limits and preventing spontaneous coal combustion. The following is a summary of the main research results.

The technology proposed in this paper is based on gas extraction from the overlying gob and adjusts the gas pressure to balance the pressure difference between the upper and lower coal seams by establishing a negative-pressure environment. This method effectively reduces the concentration of gas in the working face, reduces the risk of gas exceeding the limit, and helps to control the possibility of spontaneous combustion of coal seams.

After adopting the extraction technology proposed in this paper, the ventilation effect of the mine was improved, and the safety environment of the working face was significantly enhanced. The practical application results show that gas extraction can effectively control gas concentrations within a safe range and reduce coal mine accidents caused by gas accumulation.

Although gas extraction technology has obvious safety production advantages, there are also some technical and management challenges in the implementation process. For example, maintenance of extraction equipment, optimization of borehole layout, and accuracy of monitoring systems are all issues that need to be focused on. Therefore, it is suggested that mine management should strengthen the regular inspection and maintenance of extraction equipment and optimize the design of extraction systems to ensure efficient operation of the system.

Future studies can further explore the adaptability and optimal configuration of extraction technology in different mine environments. In addition, combining modern sensing technology and data analysis, the development of more intelligent gas monitoring and control systems will help achieve more accurate gas management and higher safety outcomes.

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