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COMMENTS/DISCUSSIONS AND REPLIES

Research on Gas Outburst Coal Seam Mining Technology Based on the Theory of Gob-Side Entry Retaining

Yongkui Shi · Xin Wang · Jian Hao · Jiahui Lin · Weijia Cheng · Jun Chen · Xiao Li

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Abstract Under the condition of traditional mining technology of gas outburst coal seam, the continuous mining of working face is tense, the waste of resources is serious, the amount of roadway excavation is large, and the stress of surrounding rock is large, which results in frequent occurrence of gas outburst, large deformation of surrounding rock, and serious damage to the benefit of coal mine. In this study, a technical method system of gob-side entry retaining, layer-by-layer drainage and pumping roadway without digging rock is put forward. The technology of roof-cutting pressure relief and gob-side entry retaining is adopted to relieve the tension of continuously mining in working face, and to ensure the stability of surrounding rock while improving the recovery rate of resources, so as to provide sufficient time and space for gas drainage along layers in the next section. Gas control roadway instead of rock bottom roadway is used to optimize the layout of mining area and greatly reduce the pressure of roadway working end. Finally, by analyzing the effect of retaining roadway and drainage effect, it is concluded that the average displacement of roof and floor of roadway is 304 mm, the average displacement of two sides is 254 mm, the maximum resistance of support is 44.1 MPa, and the effect of retaining roadway is ideal; the effect of gas drainage is 1.88 times of the original scheme, and 14,989,000 m^3 has been drained, and the outburst prevention effect is ideal.

Keywords Gob-side entry retaining · Flat layer Gas drainage · Not digging rock bottom pumping gas roadway · Safe and efficient

Y. Shi · X. Wang (✉) · J. Hao · J. Lin · W. Cheng · J. Chen · X. Li
College of Mining and Safety Engineering, Shandong University of Science and Technology, Qingdao 266590, China
e-mail: woxingwangao@163.com

J. Hao
State Key Laboratory of Mining Disaster Prevention and Control, Shandong University of Science and Technology, Qingdao 266590, China

J. Hao
Shandong Taian Hua Ning Group, Taian 271400, China

1 Introduction

There are many restrictive factors for safe and efficient mining under gas outburst coal seam. The reason lies in the large amount of roadway excavationworks under traditional mining mode, the difficulty of stress control of roadway surrounding rock, and the tense continuation of mining face, which can notprovide enough time and working space for gas control in mine (Zhang and Li 2019). The situation of gas concentration exceeding the limit can not beimproved, resulting in rock burst and gas explosion during mining face, so that disasters such as explosion, gas emission, coal and gas outburst occur fromtime to time (Zhang et al. 2017; Chen et al. 2013; Li et al. 2018), threatening the life safety of underground workers, serious loss of mine benefits, thesurvival and development of coal mining enterprises are limited.

In order to avoid rocky roadway excavation in high gas area, reduce resource waste, and alleviate the tension between working face mining, many scholars have done a lot of research in ensuring the effect of coal mine gas control and Control surrounding rock deformation. Zhang et al. (2018) used technology for CO_2 cracking and increasing permeability to effectively prevent gas overrun during roadway excavation and improve tunneling speed. Szlązak et al. (2019) studied the gas control situation of coal mines in Poland, discussed the ventilation system under longwall mining, and formulated a detailed solution plan. Liu and Liu (2019) achieved the pressure relief and increasing permeability of coal seams through hydraulic stamping and pressure-relieving technology, and greatly improved the gas drainage efficiency of coal seams. Konicek et al. (2019) analyzed the stress changes during the advancement of the longwall face and the possible induced seismic activity, and applied the decompression blasting technique to release the stress to ensure safe mining. Luo (2014) implemented the roadway retaining technology along the analysis of surrounding rock structure and roadway deformation characteristics, improved the ventilation mode of the roadway, and can prevent gas transfinite problem successfully of the upper Corner. Zhang et al. (2016) ensured the stability of the roadway by carrying out specific process design on the roadway support resistance and support scheme. Guo et al. (2018) analyzed the compression characteristics of the loose gangue mass under different conditions, and analyzed the stability characteristics of the surrounding rock deformation of the gob-side entry retaining to improve the coal recovery rate. Shi (2014) aimed at the disadvantages of large displacement of the top and bottom of the gob-side entry retaining of wood pile, and used high water material to fill technology of the gob-side entry retaining, and the surrounding rock deformation control effect was obvious. The application of other traditional lanes to fill the gob-side entry retaining is ideal it can also achieve “Y” type ventilation, solve the problem of harmful gas overflow in the goaf, and reduce the gas concentration in the working face (Wang and Xufei 2013; Liu 2016; Liu et al. 2019a, b); Luan et al. (2017), Li (2015), Zhang (2017), etc. combined the way of filling along the goaf and retaining the roadway with the way of gas extraction technology, and implemented them in Qinglong Coal Mine, Songhe coal mine and Xinyuan coal mine respectively, Basically, the problem of gas exceeding the limit has been eliminated, and the advancing speed of the working face has been greatly improved. These studies all start with practical engineering problems and obtain rich research results, However, due to the complexity of the research object in this paper, the coal pillar retaining lane, the gob side entry driving and the retaining roadway along goaf by filling body beside roadway can not be successful. Therefore, the research on the combination of gas control technology and the roadway retaining technology remains to be further studied.

This study provides a technical method system for safe and efficient mining of gas outburst coal seam, including gob-side entry retaining, seam-side drainage and no rock bottom drainage, so as to introduce roof cutting and pressure relief technology to change the traditional coal mining mode of gas outburst coal seam. By using this technology, the deformation of surrounding rock is controlled and sufficient gas control time is guaranteed, and the gas control roadway is innovated to replace the rock bottom drainage roadway, breaking the tradition of pre-drainage of coal seam gas in high gas mine excavation roadway. The technical method system has been successfully implemented in Linhua Coal Mine, Guizhou Province, or it may provide a reference for safe and efficient mining of gas outburst coal seams abroad.

2 Technical System for Gob-Side Entry Retaining, Flat Layer Gas Drainage, and not Digging Rock Bottom Pumping Gas Roadway

2.1 System Introduction

The technical method system for gob-side entry retaining, flat layer gas drainage, and not digging rock bottom pumping gas roadway of the rock is proposed on the basis of the existing construction of Lin Hua Coal Mine, which means that the technology of gob-side entry retaining is used to retain the return airway roadway of the previous working face, providing construction space for the flat layer gas drainage, the flat layer gas drainage is used for gas extraction from working face, and the gas control roadway is used to replace rock bottom drainage roadway. The main process: analyze the geological conditions of coal seams in coal mines, determine the way of gob-side entry retaining the suitable for coal mines, and design the process plan; Analyze the occurrence conditions of coal seam gas, reasonably design the gas drainage process plan of the flat layer gas drainage, and extract the gas in the next working face while excavating roadway of gob-side entry retaining; Reasonably arrange the next working face, dig the gas control roadway at the appropriate position on the working face, and carry out the gas drainage operation of the remaining coal body on the working face. Figure 1 is a system flow chart.

2.2 Technical System Engineering Implementation

At present, two rock bottom pumping gas roadway have been excavated in the coal mine. The system is proposed to adapt to the reality of Lin Hua Coal Mine. The purpose is to avoid re-exploitation of the lower section rock roadway, improve resource recovery rate and ease the tension of mining. Figure 2 shows the mining area the layout. 20917 working face is the first working face.

Analyze the geological conditions of No. 9 coal seam in Lin Hua Coal Mine, determine the way of gob-side entry retaining that is suitable for coal mines, and design the process plan. Lin Hua Coal Mine implemented the technology of the gob-side entry retaining in air return roadway of 20917 working face (red area in Fig. 3). Figure 3 shows the position of the gob-side entry retaining.

Analyze the gas storage conditions of 9# coal seam in Lin Hua Coal Mine, arrange 20915 working face

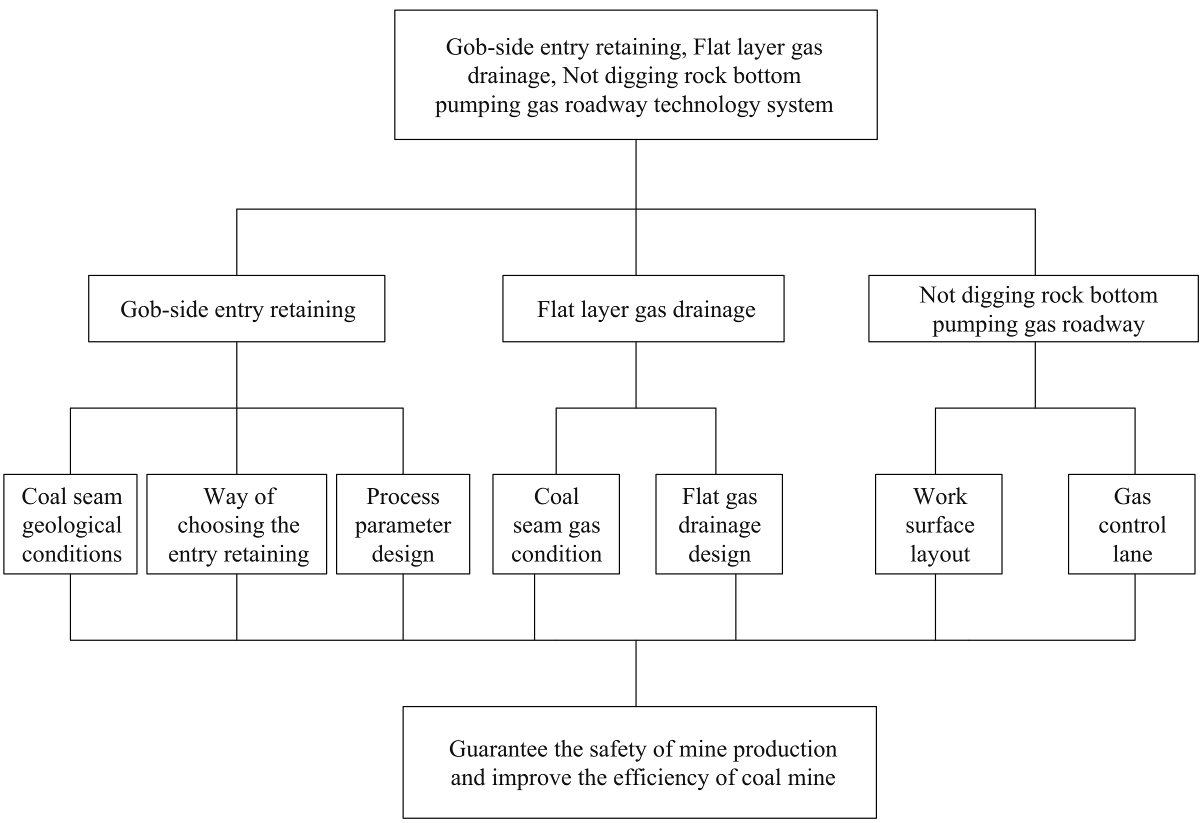


Fig. 1 System flow chart map

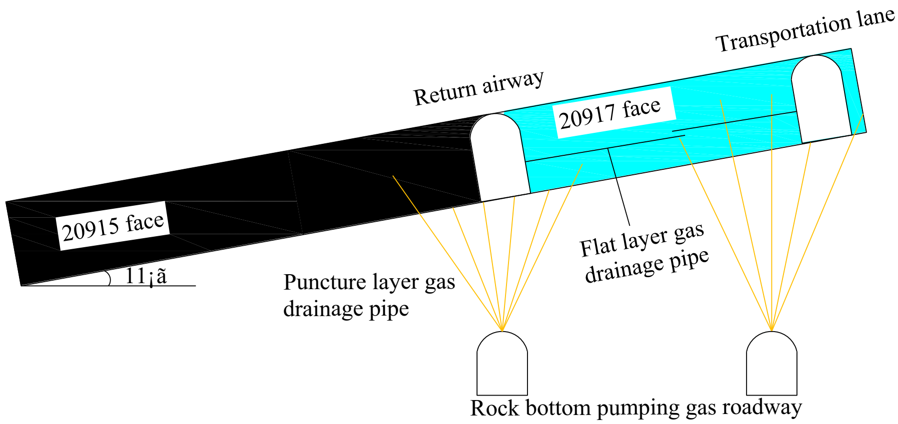


Fig. 2 The mining area layout map

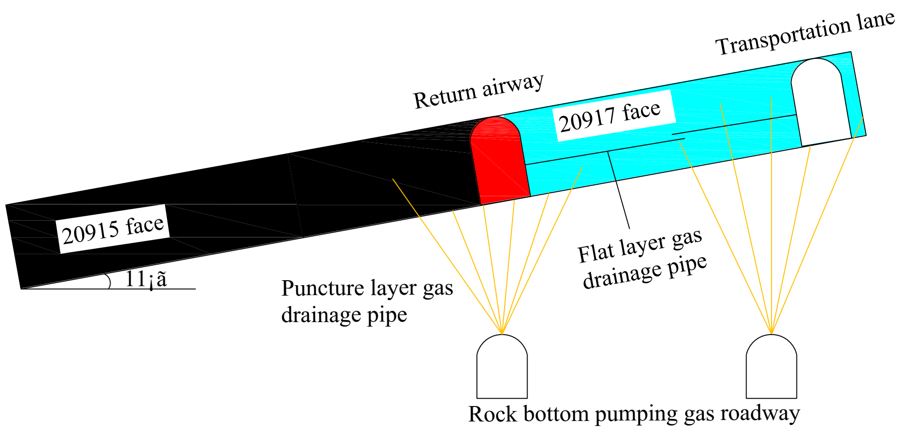


Fig. 3 Position map of the gob-side entry retaining

reasonably, and design the gas drainage plan for 20915 working face. While tunneling along the roadway of gob-side entry retaining, gas drainage along seam is adopted to extract gas from 20915 working face. When the 60 days drainage requirements are met, dig the gas control roadway in the 20915 working face. Figure 4 is a layout diagram of the gas control roadway.

Dig the gas control roadway in the 20915 working face, at the same time, carry out the flat layer gas drainage operation to the remaining coal body of the

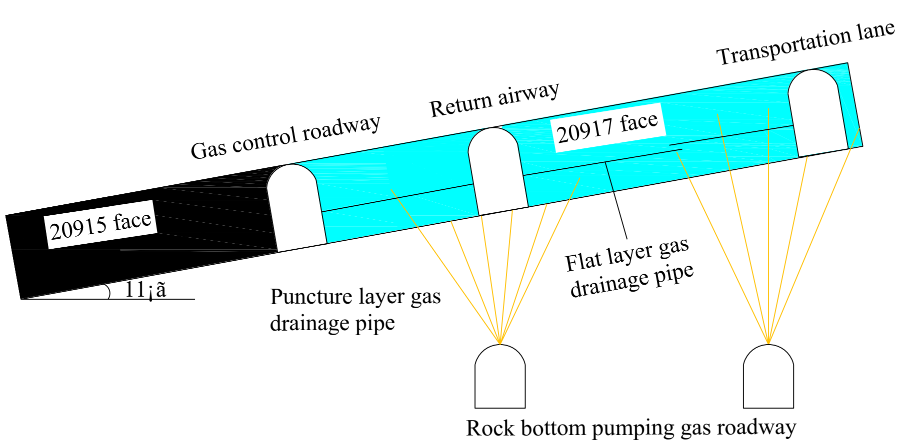


Fig. 4 Gas control roadway map

20915 working face, and began to dig into the return airway of the 20915 working face when the 60 days drainage requirements are met. Figure 5 is a layout diagram return airway of the 20915.

The air return roadway of 20917 work surface is finished for renovation, it was used as the 20915 working face transportation lane to prepare for the 20915 mining work. Each process shall be cross-constructed according to the specific deployment of the coal mine, and all working face mining work shall be completed in turn. Figure 6 is a plan view of the mining area.

The method system proposed by the research institute differs from the traditional mining method in that the technology of gob-side entry retaining is used to retain the return airway roadway of the previous working face, the flat layer gas drainage is used for gas extraction from working face, and the gas control roadway is used to replace rock bottom drainage roadway. The system can provide sufficient drainage time for working surface gas and working space, which can ensure safe production of coal mines and improve economic benefits.

3 Project Overview

3.1 Coal Seam Geological Conditions

Lin Hua Mine is located in the southwest of Jinsha County, Guizhou Province, 10 km away from the center of the county. The administrative division is under the jurisdiction of Xinhua Township and Xi Luo Township. The 20917 working face of coal mine is located in the second mining area of 9# coal seam. The average thickness of the working face is 3.5 m; the strike length is 1200 m, the trend length is 190 m; the roadway width is 4.2 m and the height is 3.5 m; the 9# coal seam is of gas outburst. The average inclination angle is 11°; the roof is silty mudstone and fine sandstone with a thickness of 14.34 m; the bottom plate is argillaceous silty, silty mudstone or fine sandstone with a thickness of 31.65 m. Figure 7 shows the location of the mine and the plane of the 20917 face mining project.

3.2 Coal Seam Gas Occurrence Conditions

The mineable and locally mineable seams in Lin Hua Coal Mine are 5 seams, numbered 4, 5, 9, 13 and 15 seams, all of which are gas outburst seams. M9 is the main coal seam, and the maximum content of gas in this coal seam is 23.42 m^3/t. It is dangerous and difficult to extract. Table 1 is gas test table of 9 # coal seam.

4 Engineering Test

4.1 Roadway Retention and Process Parameter Design

Through the analysis of the geological conditions of the 9# coal seam in Lin Hua Coal Mine of Guizhou Province, the air return roadway of 20917 working face was retained by the technology of roof cutting and pressure release, and process parameters were designed. The no-coal column roof cutting and

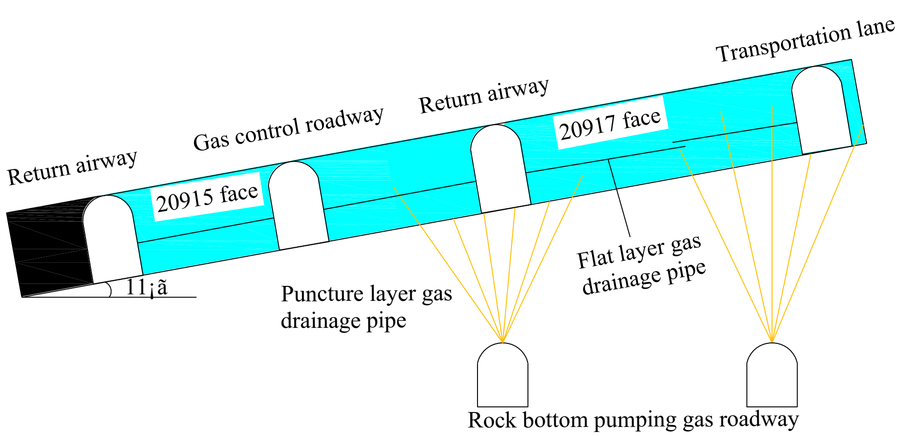


Fig. 5 Return airway of the 20915

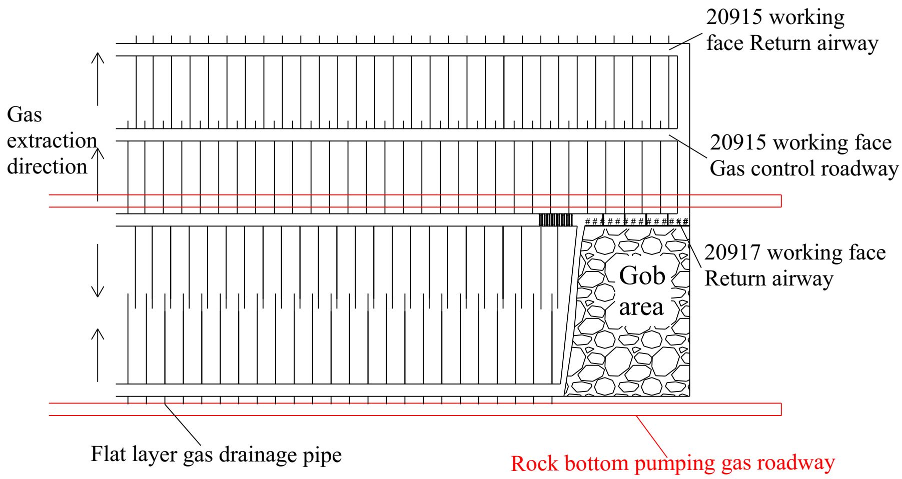


Fig. 6 Plane layout of the mining area

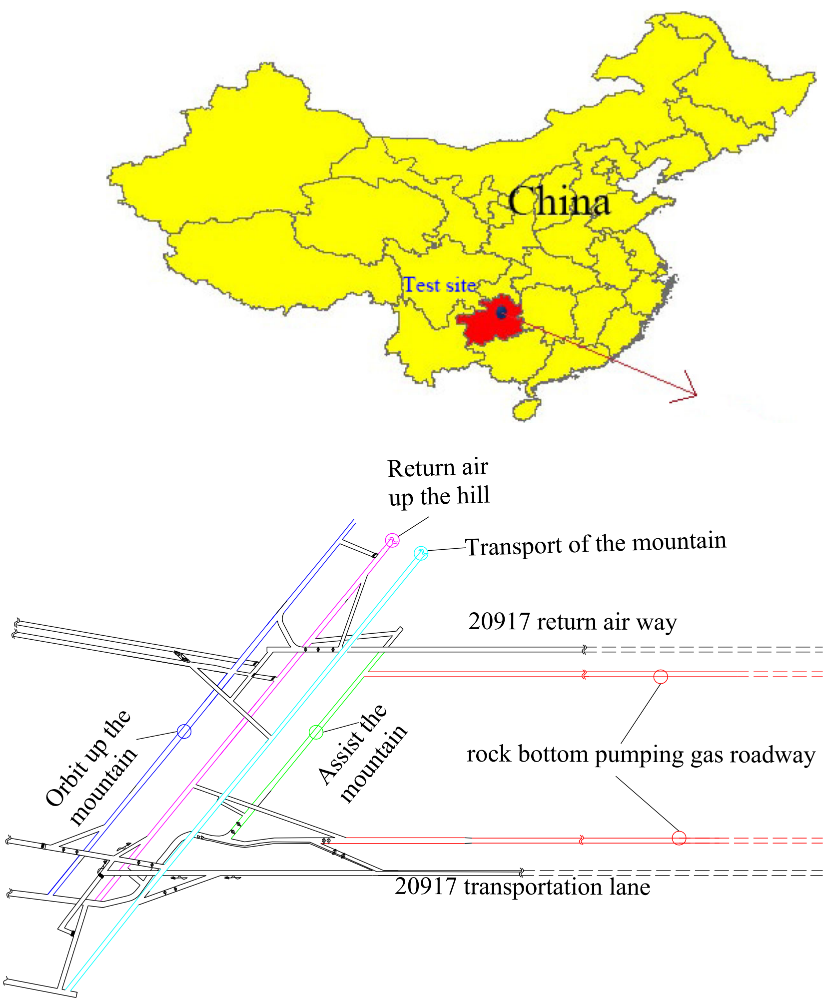


Fig. 7 Lin Hua mine location and 20917 floor plan

Table 1 The gas test table for 9# coal seam

Coal seam	Elevation (m)	Buried depth(m)	Gas absolute pressure (MPa)	Raw coal gas content (m^3/t)	Gas release initial velocity Δp (mmHg)	Gas permeability coefficient (MPa^2 d)	Drilling gas flow attenuation coefficient (d^{-1})
9#	+ 810	395	1.5		30	0.05–7.17	0.064–0.165
	+ 787	393	2.15	23.38			
	+ 788	412	0.34	20.82	32		
	+ 833	472	0.96				
	+ 815	511	0.75	23.42	29		
	+ 810	505	0.65				
	+ 850	385		22.65	32		
	+ 810	550	0.87		28		
	+ 811	543			33		

pressure release technology refers to Pre-splitting blasting at a distance ahead of the roof in the air return roadway gob side of the working face, cutting off the mechanical connection between the roof of the roadway, and the overburden of the goaf, the overlying strata in the goaf collapsed under the action of ground pressure, forming a lane gang (Ma et al. 2018a). This technology realizes the transition from long arm beam to short arm beam, enhances structural stability (Liu et al. 2019a, b), and fundamentally changes the stress distribution law of surrounding rock in the roadway along the roadway under the influence of mining (Wang et al. 2018; Ma et al. 2018a, b; Yajun et al. 2018). Figure 8 is the schematic diagram of roof cutting and pressure releasing.

Y-type ventilation is adopted for roof cutting and gob-side retaining. Compared with the traditional “U” ventilation mode, it can effectively solve the problem of gas accumulation in the upper corner and reduce the risk of explosion in the working face. The “Y” ventilation schematic diagram is as follows (Fig. 9).

Support design: advanced support adopts I-beam steel + single hydraulic prop + no longitudinal reinforcement threaded steel anchor + anchor cable + metal mesh + W steel belt, 0–60 m support from mining face; two coal Gang the protection adopts the bolt + metal mesh support method; Stabilization section with masonry wall and U36 steel leg support, the lag section adopts the wood pile to strengthen the support method, and lapped along the U36 steel leg well, and the top is tightly connected, as shown in Fig. 10.

Calculate the roof cutting depth of Lin Hua Coal Mine by selecting the theoretical formula of rock mass expansion:

$$H_{Kerf} = (H_{coal\ thickness} - \Delta H_1 - \Delta H_2)/(K - 1) \quad (1)$$

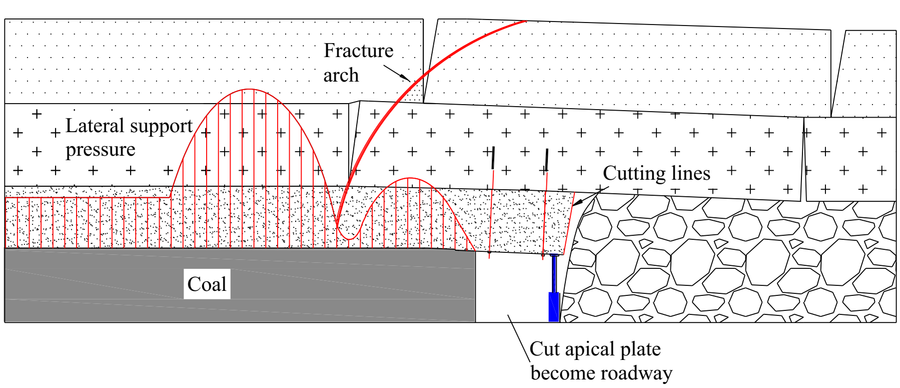


Fig. 8 Roof cutting and pressure release schematic

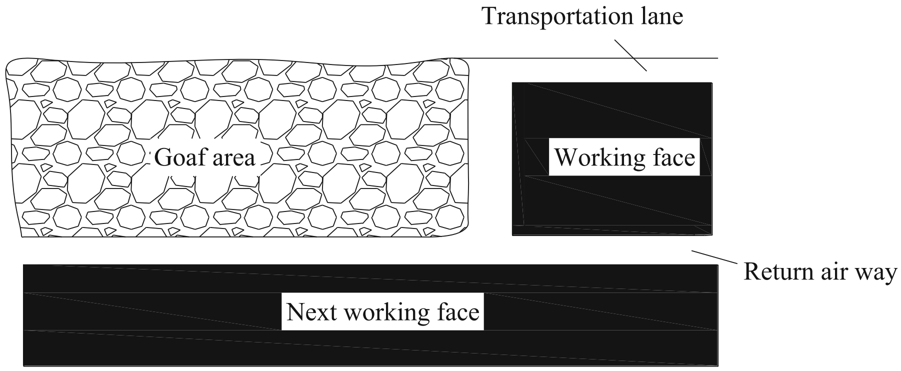


Fig. 9 The "Y" ventilation schematic diagram

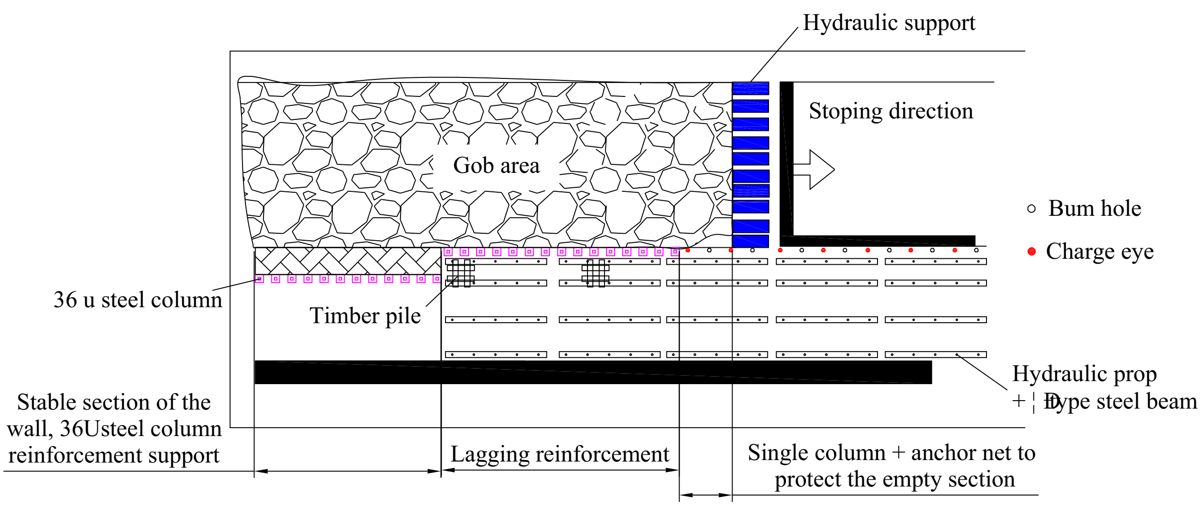


Fig. 10 Return airway support diagram

In formula: $\Delta H1$ is the amount of roof subsidence, m; $\Delta H2$ is the bottom drum amount, m; K is the expansion coefficient of 1.35; the coal thickness is 3.5 m.

When the working face is not mined, the bottom drum and the roof subsidence are 0 and the designed slit hole depth is 10 m and the slit angle is 15°. The drilling construction position is at the angle between the top plate and the working surface of the 20917 return airway, the hole diameter is φ42 mm, the drilling distance is 500 mm, using spacer hole charge. The roof cutting construction should be synchronized with the roadway ahead support. Before the roof cutting, the top plate is reinforced with a constant resistance large deformation anchor cable (Sun et al. 2014; Ma et al. 2018a, b, c). The drilling arrangement is shown in Fig. 11.

The three-stage emulsion explosives coils allowed by the coal mine are used, and the explosives are injected into the energy-concentrating tubes by pneumatic injection guns. Each of the energy-concentrating tubes is 2 m, and six medicine rolls are strictly prescribed. Drill hole sealing 2 m, charging 8 m, 3 detonators are connected in parallel, one at 2 m from the orifice, one at 4 m, one at 6 m. a reinforced cartridge at the bottom of the hole for positive detonation. Each time five roof cutting blasting holes are detonated, and the blasting holes are detonated in series (Hu et al. 2019, Yang et al. 2019). The structure of the charges is shown in Fig. 12.

The double-layer diamond mesh + double-section U36 steel leg block is used to prevent the gangue from goaf flowing into the roadway. The diamond mesh and the top metal mesh overlap the length of 500 mm, and

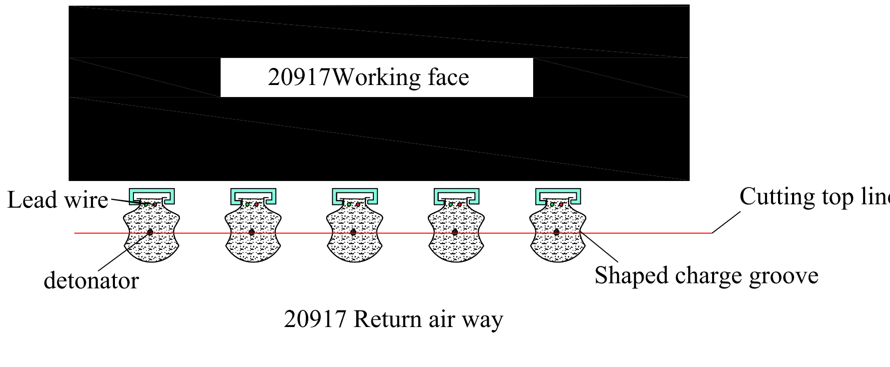


Fig. 11 Schematic diagram of the roof cutting hole arrangement

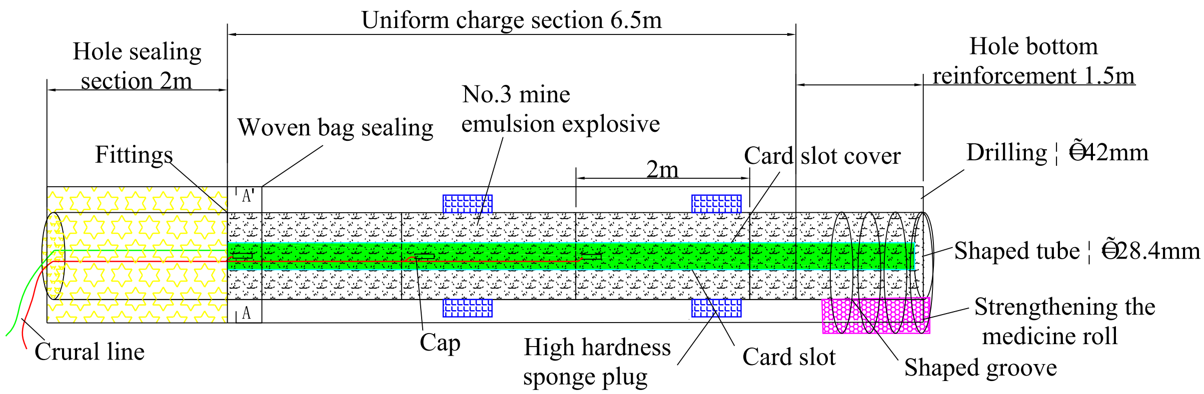


Fig. 12 Charge internal structure diagram

the 14# wire double-strand joint is used every 100 mm, repeat the connection twice. The horizontal direction mesh is overlapped with the mesh by 10 cm; the vertical direction mesh is used for lap joints, the lap length is 10 cm, and the mesh bottom leave behind is 60 cm and bend 90°. The steel legs are 0.6 m apart, and the bottom of the column and the column cap are welded to the column legs with iron plates. The bottom of the steel leg is padded of wood, and the steel leg groove side is opposite to the goaf. After the steel leg is set, an anchoring agent is placed in the anchor hole reserved by the column cap by the roofbolter, and the anchor rod is used for limiting. Figure 13 is blocking device for coal gangue.

In order to prevent gas overflow in the goaf, seal the roadway. The high negative pressure gas pipe and the low negative pressure gas pipe are used to extract the gas in the goaf, and the gas concentration in the goaf is reduced. According to the mining face of the working face, sandbag walls are installed at intervals of 5 m in the roadway to seal the roadway to prevent gas spillage in the goaf. Figure 14 System support diagram in the roadway.

4.2 Flat Layer Gas Drainage Scheme Design

Next working face gas drainage design: In the 20917 return air alley, the construction applied to be the coal seam tends. The φ 94 mm drill hole is used to pre-pump the 20915 working face gas, and the "two plugs and one injection" sealing method is adopted (Li et al. 2019). The drilling hole diameter is 94 mm, and the negative pressure is 21 kPa, drilling design depth of 90 m, effective extraction radius of 1.5 m, drilling distance of 3 m; During the actual construction of the drilling, the drilling depth can be adjusted according to the actual production situation and the time of the mining standard. The spacing of the drilling holes in the 20915 drill hole can be shortened according to the extraction time. Table 2 shows the drilling parameters of the 20915 face.

Gas drainage design in goaf: 20917 return airway lays a low negative pressure gas drainage pipe, not less

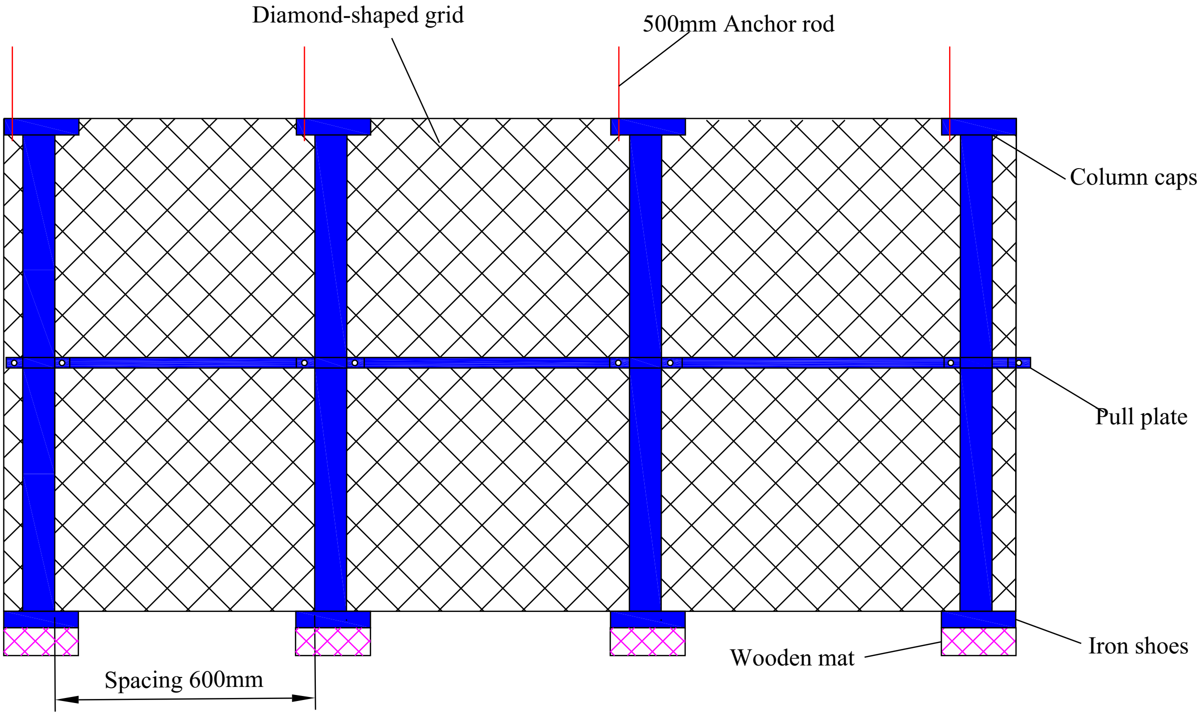


Fig. 13 Blocking device for coal gangue

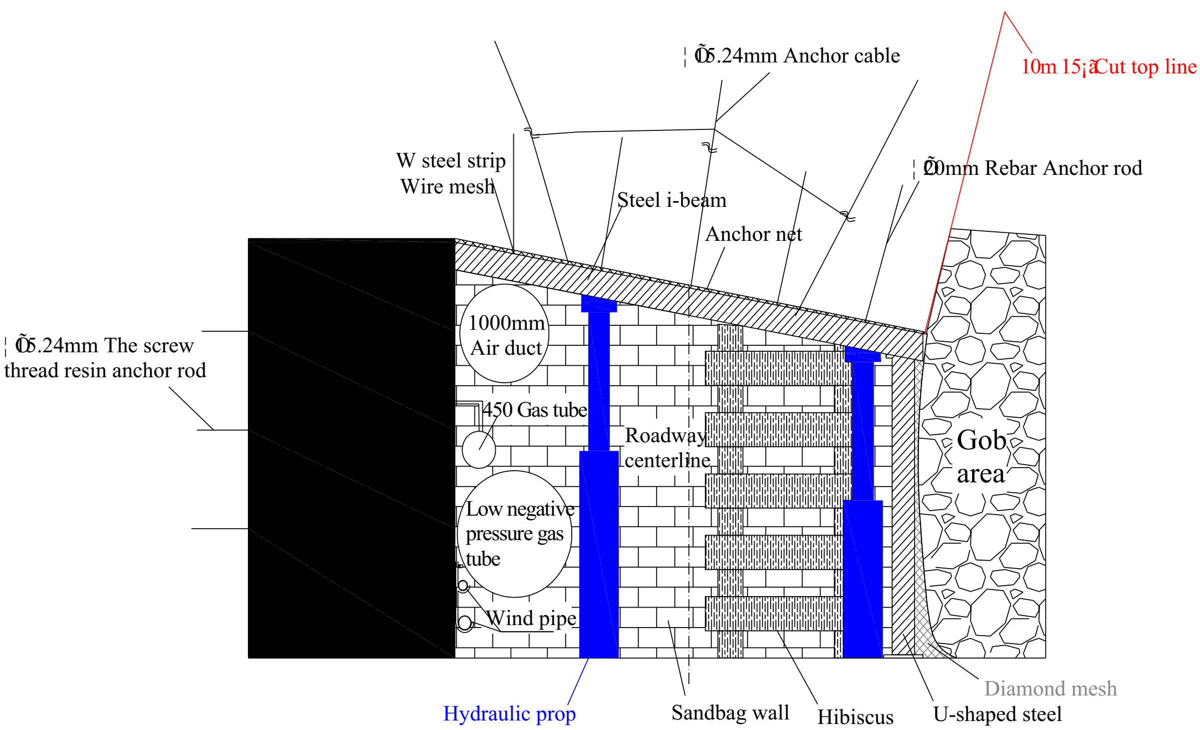


Fig. 14 System support diagram in the roadway

Table 2 20915 working face drilling parameter table

Construction location	Coal seam dip (°)	Drill hole dip (°)	Height from floor (m)	Drill hole depth (m)
20917 return airway	− 11	− 11	1	90

than DN300 mm once. The front end of the drainage pipe is processed into a sieve hole. Before roof caving, the gas pipe is placed on the wall constructed by the goaf and as close as possible to the top of the roadway. The gas pipeline is installed with a three links every 36 m. The three links is connected with the elbow to connect the DN300 pumping tube into the goaf. The upper and lower gas pipelines are connected to each other at a distance of 36 m, forming both upper and lower pumping.

5 Engineering Effect Analysis

5.1 Analysis of Roadway Effect

In order to grasp the situation of the working face in detail, the pressure monitoring of the hydraulic support is detected by the fully mechanized support pressure computer monitoring system. This study analyzes the pressure of the 200 m top plate in front of the work, and draws the resistance change diagram of the bracket according to the test data. Figure 15 shows.

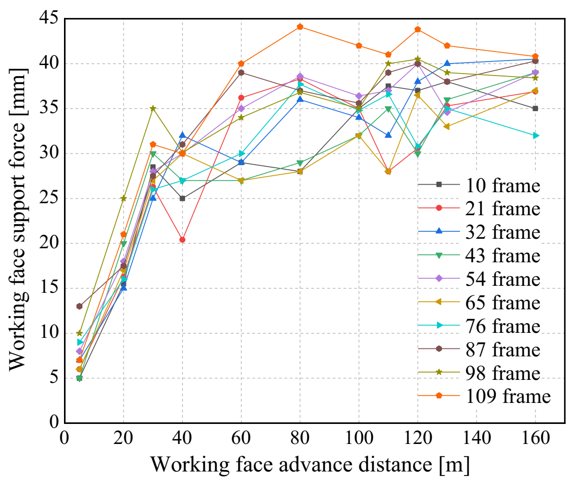


Fig. 15 Bracket resistance change diagram

According to the above table analysis, the maximum support resistance of the working face is 44.1 MPa. The specific performance characteristics are: When the working surface is advanced by 25 m, the hydraulic support pressure of the lower part, the middle part and the upper part of the working face begins to increase gradually, and the First mine pressure is coming, the pressure in the lower part and the upper part of the working surface is relatively slow; When the working surface is advanced by 31 m, the pressure of the working face support pressure increases again, and the first ground pressure is appear; When the working face advances 42 m, the support resistance increases, and the second ground pressure is appear at 46 m; When the working surface is advanced by 63 m, in addition to the slow pressure of the support on the working Upper part, the other sub-segments have already been ground pressed for the third time. During the fourth to seventh ground pressure of the working face, except for the individual areas, the force of the support during the working face is reduced compared with the previous three times.

5.2 Roadway Deformation Analysis

In order to understand the overall roadway effect of the working face, measure the height and width of the roadway, as shown in Fig. 16.

In the above figure, the distance between the collection points is 10 sheds, and the scatter plot is drawn. Statistics show that the average height is 3226 mm and the average width is 4016 mm after the roadway is stable. Compared with the front of the roadway, the average moving amount of the top and bottom of the roadway is 304 mm and the maximum is 403 mm; the average moving amount of the two coal gang is 254 mm and the maximum is 320 mm. Figure 17 is a rendering of the project site.

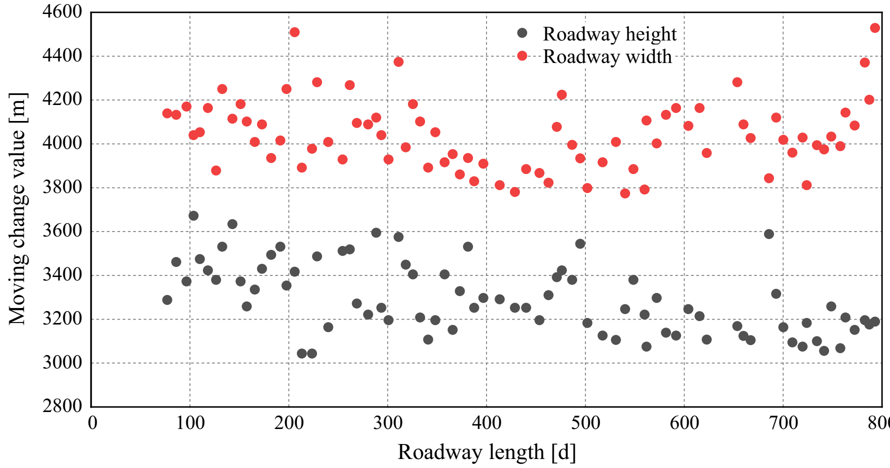
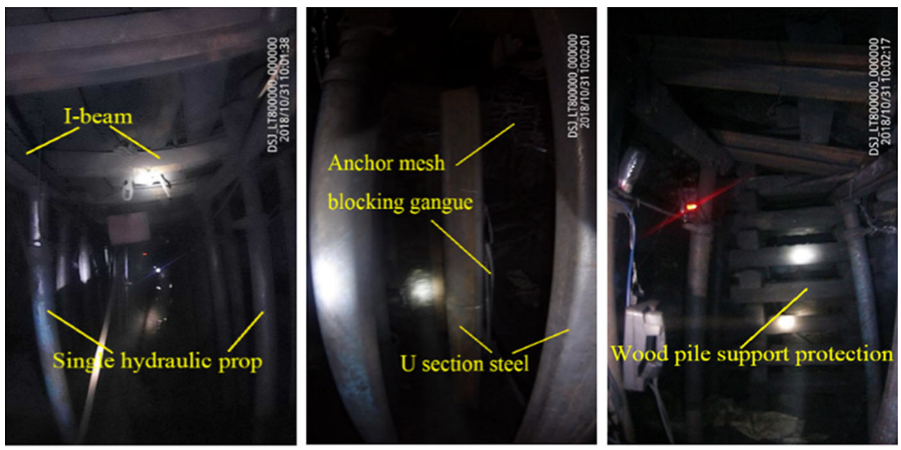


Fig. 16 Overall roadway renderings



(a) Roadway support map **(b)** Blocking gangue map **(c)** Wood pile support diagram

Fig. 17 Project renderings

5.3 Security Effect Analysis

Combined with on-site measurement, the SF6 tracer gas method and gas content method (Yang et al. 2013) are used to analyze the influence radius and effective radius of gas drainage in flat layer gas drainage (Cao et al. 2009; Liu et al. 2011). Under the condition of extracting borehole diameter of 94 mm and pumping negative pressure of 21 kPa, the relationship between the influence radius r(m) and the extraction time t(d) function of the No. 9 coal seam of Lin Hua Mine is:

$$r = 1.3879t^{0.4977} \tag{2}$$

In the range and beyond the extraction limit time, the effective radius (y) and the extraction time (t) of the gas drainage in the flat layer gas drainage follow the following functional relationship:

$$y = \begin{cases} 1.035\ln(t) - 2.7248, & 0 < t < 510d \\ 3.7, & t > 510d \end{cases} \tag{3}$$

A comparison chart of the influence radius and effective radius function of the flat layer gas drainage is obtained, as shown in Fig. 18.

It can be seen from the figure above that the influence radius curve of the extraction is always above the effective radius curve, indicating that the gas content test result is valid and the obtained effective radius formula is correct. The working face length is 820 m. If two excavation teams are working at the same time, the tunneling of the gas control roadway can be completed at 164 days after the

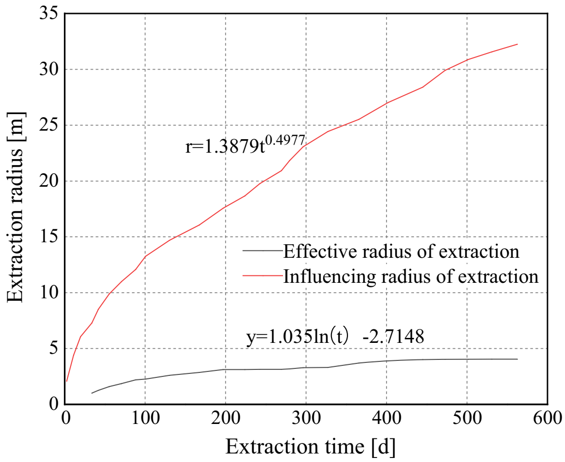


Fig. 18 Map comparison of the influence radius and effective radius function curve of flat layer gas drainage

excavation of the 20917 double lanes; When the gas extraction time of the remaining coal body of 20915 working face reaches 60 days, the 20915 return airway is excavated, and 205 days can be excavated. All roadways on the 20915 working face were completed within 205 days, The mining time of 20917 working face is expected as 315 days, and the roadway renovation 102.5 days, total 417.5 days. Therefore, at least 212.5 days of drainage time can be guaranteed for the working surface. Substituting 212.5 days into Eq. (3):

$$y = 1.035\ln(212.5d) - 2.7248 = 2.82\ \mathrm{m}$$

The effective radius of extraction is y = 2.82 m. The effective radius of the flat layer gas drainage in the original scheme is designed to be 1.5 m. Through the application of this system, the gas drainage effect of the 20915 working face of Lin Hua Coal Mine is 1.88 times of the original scheme, and the effect is ideal.

At present, the gas drainage volume of the working surface of 20915 has reached 14.98 million m^3, which is an increase of 2.0054 million m^3 compared with the mine defense standard of 12.963594 million m^3. 14.949 million m^3 is calculated according to the gas utilization rate of 24% (Gao et al. 2019). The gas utilization capacity is 3.5879 million m^3, and the gas power generation capacity can reach 1.198 million KW h. According to the use of gas power generation 1 kWh, the complete cost is about 0.33 yuan/kWh, the purchase of pure gas is 0.2 yuan/m^3, the power generation is calculated at 0.5 yuan/kWh: From January 2018 to the end of September 2019, the cumulative power generation was about 1.198 million kWh (all calculated according to electricity consumption), the total cost of power generation was 0.1 yuan/kWh (compulsory custodian fee), and the purchase of on-grid electricity price was 0.65 yuan/kWh:

Saving electricity costs: 119.8 × (0.65–0.1) = 659,900 yuan;
Subsidies available: 119.8 × 0.3 = 359,400 yuan;
The above total: 1.0183 million yuan.

6 Conclusion

1. This paper proposes a technical method system for gob-side entry retaining, flat layer gas drainage, and not digging rock bottom pumping gas roadway. It has been successfully implemented in Lin Hua Coal Mine and realized the duality of “integration for mining, pumping gas and retention roadway” and “coal and gas joint mining” idea.
2. Using the technology of roof cutting and pressure relief, the design of roadway support, cutting parameters and gangue sealing scheme is carried out, which effectively reduces the deformation of roadway surrounding rock, provides sufficient time and space for gas drainage in the next working face, greatly alleviates the problem of mining continuity tension in the working face, and steadily improves the resource recovery rate and mine benefit.
3. In this system, the way of gas control roadway replacing rock bottom drainage roadway optimizes the layout of mining area and liberates the pressure of roadway heading end. With the technology of gas drainage along seam as the center, the effect of gas drainage can reach 1.88 times of the original plan, and the gas drainage volume has exceeded 14,989,000 m^3, which meets the requirement of outburst prevention.

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