Academic Journal of Science and Technology
ISSN: 2771-3032 | Vol. 14, No. 2, 2025

Research and Application of Efficient Undercover and Filling Technology Along the Top Tunneling Roadway

Zelin Zhu[1, *]

[1] Henan Polytechnic University, Jiaozuo, China

[*] Corresponding Author

Abstract: In order to solve the problems of large height of the roadway along the top of the Jiulishan 1613 working face, the difficulty of roof maintenance, the low resource recovery rate and the difficulty of replacing the working face, the bottom coal recovery work was carried out in the top excavation roadway in advance, and the top bag filling technology was used to maintain the stability of the surrounding rock of the roadway. Through theoretical analysis, the secondary support parameters and the required strength of the top backfill were determined. According to the mechanical properties of high-water materials at different temperatures, uniaxial and triaxial experiments were used to determine the optimal water-cement ratio. The support effect is verified by numerical simulation and industrial test. The results show that the final setting strength of the top filling body shall not be less than 1.68MPa, and the temperature has little influence on the mechanical properties of the high-water material; the maximum subsidence of the roof of the roadway after the top filling is 98mm and 170 mm, respectively, and the two gangs move closer to 130mm and 210mm, respectively, and the synergistic control system formed after filling has a significant effect, increasing the income of more than 70,000 tons of coal, and the direct economic benefit is 75 million yuan, and meeting the requirements of safe mining of the working face. It provides a reference for the control of surrounding rock in similar mine roadways.

Keywords: Ultra-high roadway, excavation along the top, ratio study, surrounding rock control.

1. Introduction

In recent years, with the increase of national energy demand, many mines have entered deeper mining, but a series of problems such as geothermal, high pressure, and surrounding rock instability need to be solved urgently [1-2]. In order to reduce the difficulty of mining and maintain the stability of the roof surrounding rock, a series of methods and technologies for controlling the surrounding rock of roadway have been developed, such as bolting rod anchor net cable, resin anchoring, and flexible spraying [3-6]. However, the support effect of this kind of support method on super-large section and "three soft" roadways is not satisfactory, and many roadways still have serious deformation; Moreover, since most of the coal mine roadways are mined along the bottom tunneling, under this arrangement, the coal pressing is serious along the upper and lower grooves and the working face supports[7]; For some mines with thick coal seams and poor coal quality, this method will cause the roadway to empty roof, and the roof is overwhelmed and collapses and crushes the support, which brings difficulties to roadway support and safe mining. Compared with bottom-to-bottom excavation, there will be no empty roof problem in the roadway along the top, and the gas overrun can also be avoided in some high-gas mines, and the roof maintenance is also easier. However, after excavation along the top, it is necessary to solve the problems of safe mining of thick coal seams and efficient recovery of triangular coal at the bottom.

In view of the problems of how to ensure the stability of the surrounding rock of the roadway in the working face and how to effectively manage the roof after the thick coal seam is excavated along the roof, Yu Bin [8] studied the instability mechanism of the surrounding rock of the thick coal seam roadway in view of the serious deformation of the roof of the thick coal seam roadway under the influence of the mining disturbance of the working face, and concluded that the gangue inclusion in the roof of the thick coal seam roadway and the joint cracks in the coal seam are the main reasons for the roof instability. In view of the serious deformation of the surrounding rock and the uneven stress distribution of the thick coal seam roadway, Zhang J [9] proposed a high preload bolt (cable) joint support scheme to support the thick coal seam roadway, and achieved good support results. In order to solve the problem of poor stability of the surrounding rock of the roadway in the thick coal seam, LiSC [10] established a mechanical model of the surrounding rock of the roadway in the thick coal seam with Zhaolou Coal Mine as the object, analyzed the deformation and stress evolution characteristics of the surrounding rock of the roadway under the anchor (cable) support system, and determined the instability mechanism of the surrounding rock of the roadway in the thick coal seam. Li Jixing [11] conducted research on the instability of the surrounding rock of the huge high roadway, and proposed to use the joint support method of high-strength anchor rod + steel belt + anchor cable to control the surrounding rock of the ultra-high roadway, so as to ensure that the ultra-high section of the roadway does not have accidents such as sheet gang and roof leakage, and ensure the safe mining of the working face. Chen Qiuyu [12] took the 3122 fully mechanized working face of Jingxin Coal Industry as the engineering background, and prepared an inorganic cementitious material with a certain foaming capacity to grouting and filling the empty roadway of the working face in view of the problems of insufficient support strength and large safety hazards of the working face crossing the empty roadway under the traditional support mode. The deformation of the surrounding rock is reduced by nearly 52% after roadway filling, and the empty roadway filling technology can effectively control the deformation of the surrounding rock of the roadway. He Xiangning et al. [13] simulated and analyzed the stability of the surrounding rock of the hollow roadway under three modes of bolt support, wood stacking support and high-water material support, and finally determined that high-water filling was the best support method. Combined with the properties of materials with different water-cement ratios and high water ratios, the filling scheme was formulated, and the ideal industrial test results were obtained. Through theoretical analysis and numerical simulation, Wang Cheng et al. [14] proposed the partition reinforcement technology of "three high" bolts (cables) and inorganic two-liquid grouting materials for surrounding rock grouting by using "three high" bolts (cables) and inorganic two-liquid grouting materials, which increased the integrity of the surrounding rock and significantly improved the stability of the surrounding rock, which provided a guarantee for the safe production and mining of large-section dynamic pressure mining roadway. Xiong Zuqiang et al. [15] studied the characteristics of the filling material with different proportions of filling excipients, and carried out the empty roadway filling test in a working face, and the results showed that increasing the content of the auxiliary materials could significantly improve the strength of the filling materials. Combined with the actual work on site, a reasonable filling system was designed and the self-flow transportation of the pipeline was realized. Sun Ruyi [16] proposed a bag filling support scheme for the top of the ultra-high roadway in the 1112 working face of Yong'an Hetuo Mine in view of the difficulty of supporting the ultra-high roadway at the 1112 working face, and determined that the required strength of the backfill body was 1.4MPa, and the reliability of the top bag filling scheme was verified by FLAC3D numerical simulation and field practice. Zhang Xingdong [17] established a numerical model of thick coal seam roadway in view of the large deformation of the surrounding rock and the difficulty of supporting the roadway in the thick coal seam, and studied the influence of the mining disturbance on the deformation of the surrounding rock of the roadway, and concluded that the height of the roadway is an important factor for the instability of the surrounding rock of the roadway. Hu Fangjian [18] aimed at the problems of obvious pressure and difficult support of the ultra-high roadway in Tengdong Coal Mine, and used the method of erecting a tripod and filling the wooden stack above to control the deformation of the surrounding rock of the roadway, and the actual field verified that the control effect of the surrounding rock of the roadway was good. Zhang Baolin [19] estimated the pressure of the top and bottom plate of the roadway and the two gangs in view of the difficulty of controlling the surrounding rock of the ultra-high roadway, used the vehicle-mounted safety platform to build the "well" wooden stack, and used the I-beam shed as the bearing body to effectively control the surrounding rock of the roadway. Cao Qijia et al. [20] analyzed the structure of the roof of the hollow roadway in the large buried working face, explained the failure mechanism of the surrounding rock of the hollow roadway in the working face, proposed the technology of filling the hollow roadway with high-water materials, and established the stability mechanical model of the backfill roof.

Many theories and technical means put forward by experts and scholars at home and abroad are aimed at controlling the deformation of surrounding rock in ultra-high roadways and ensuring the safe mining of working faces, and there are few studies on the efficient recovery of end triangular coal and undercover technology. Based on the above situation, the author takes the working face of Jiulishan 1613 as the engineering background, combined with theoretical analysis and field practice, and proposes a method of higher efficiency and stronger adaptability of the comprehensive excavator and the transfer machine to carry out the advanced undercover and remining triangular coal, anchor net cable + high-water material filling support management roof. Through numerical simulation analysis and field measurement, the feasibility of this new undercover process and the effect of the combined support method on the control of the surrounding rock of the ultra-high roadway are verified, which provides a reference for similar mines.

2. Engineering Background

2.1. Project Overview

The 1613 working face is designed to have a mineable strike length of 510m, an inclined length of 129m, a slope of 14.5°, and a recoverable reserve of 697,000 tons. The average thickness of the coal seam is 7.7m, the height of the inner and outer sections is uneven and the thickening trend from west to east, the coal quality of the main mining No.2-1 coal seam is soft, the Platts coefficient is 1.05, and the coal contains a large number of fractures with a high degree of development, compared with the top and bottom slate formations, the stability is poor, the strength is low, and it is easy to break the bed. The layout of the roadway and the lithological characteristics of the top and bottom slabs of the coal seam are shown in Figure 1 and Table 1, respectively.

The roadway design of the working face is excavated along the top, the excavation width is 4.6m, the height of the mining gang is 3.5m, the roof of the roadway adopts the anchor net cable + W steel belt with the reinforcement mesh support, the two gangs use the anchor net cable + the reinforcement ladder with the reinforcement mesh joint support, the anchor cable specification: φ17.8×4200m, the row spacing between the anchor cables is 800mm×900, mm, and a row of anchor cables is arranged between each row of supporting anchor cables to cooperate with the reinforcement ladder to reinforce the support.

2.2. Engineering features

The initial mining height of the two lanes of the 1613 working face is 3m when the top is excavated, and the bottom coal is close to 4m, resulting in a waste of nearly 70,000 tons of resources, as shown in Figure 2 (b); In order to recover resources and reduce waste, a fully integrated excavator can be used to excavate the bottom to recover the triangular coal to improve the recovery rate, but if the undercover bottom coal is mined, an ultra-high roadway of about 7-8m will be formed, as shown in Figure 2(c), the original support system is difficult to continue to maintain the stability of the roof surrounding rock; Moreover, due to the increase of the roadway section during the excavation of the working face, the mine pressure activity will be violent, the displacement of the surrounding rock will increase significantly, and the risk of fracture and instability will increase, which will seriously threaten the safety production of the working face.

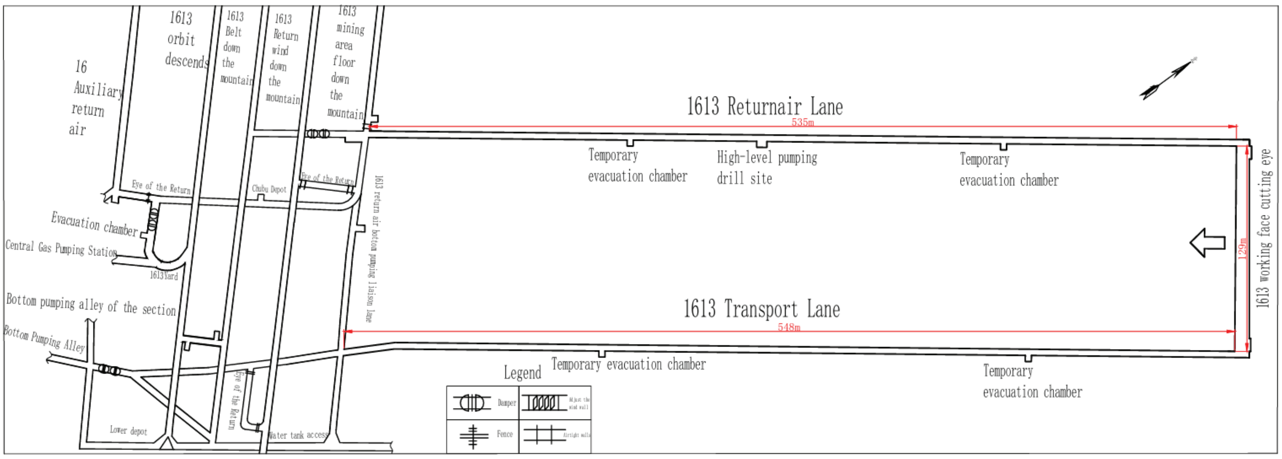


Figure 1. Working face roadway layout drawing

Table 1. The condition of the top and bottom plates of the working face

Comprehensive histogram	Bottom layer thickness/m		The name of the rock	The name of the top and bottom plate	Lithological characteristics
	Tired/m	Layer thickness /m			
	440.9	18.39	Medium-grained sandstone	Old top	Gray-dark gray, mainly quartz, with a small amount of muscovite flakes and carbonaceous films.
	459.29	3.01	Siltstone	Direct top	Grayish-black, silty-grained, layered, silty composition, containing fossilized plant leaves.
	462.3	0.4	Carbonaceous mudstone	False top	Carbonaceous mudstone (locally missing)
	470	7.7	II#1 coal	Seam	It is mainly end-shaped and block-shaped, and the upper part is about 1.0m of end-like soft coal with dull luster, and the lower part is mostly semi-bright medium-hard coal, and the coal seam structure is simple.
	471.03	1.03	Mudstone	Direct bottom	Black, dense, argillaceous composition, rich in plant fossils
	471.65	0.62	L_9#Limestone	Old bottom	Dark grey, lumpy, with calcite veins.

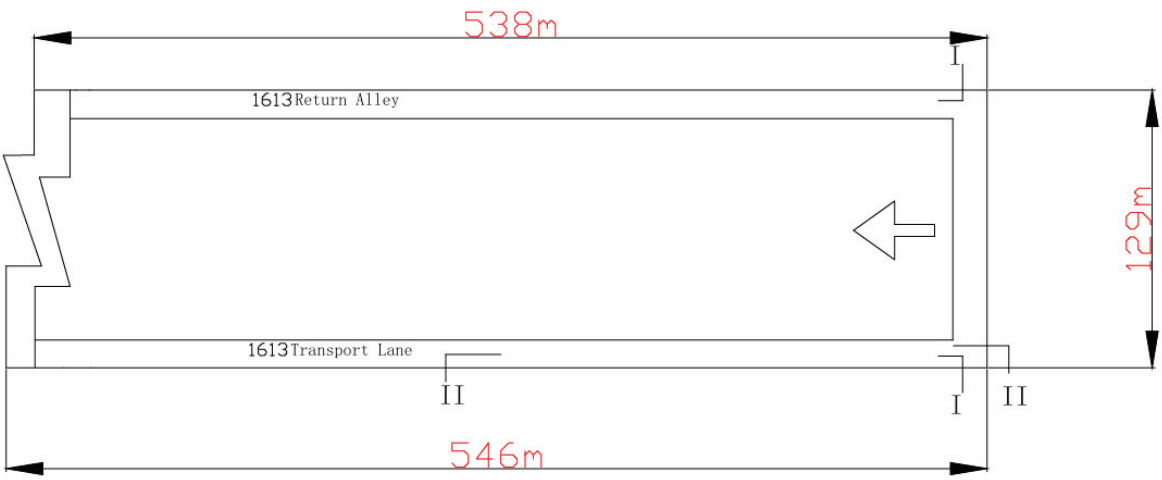


Figure 2(a). Layout of the working face

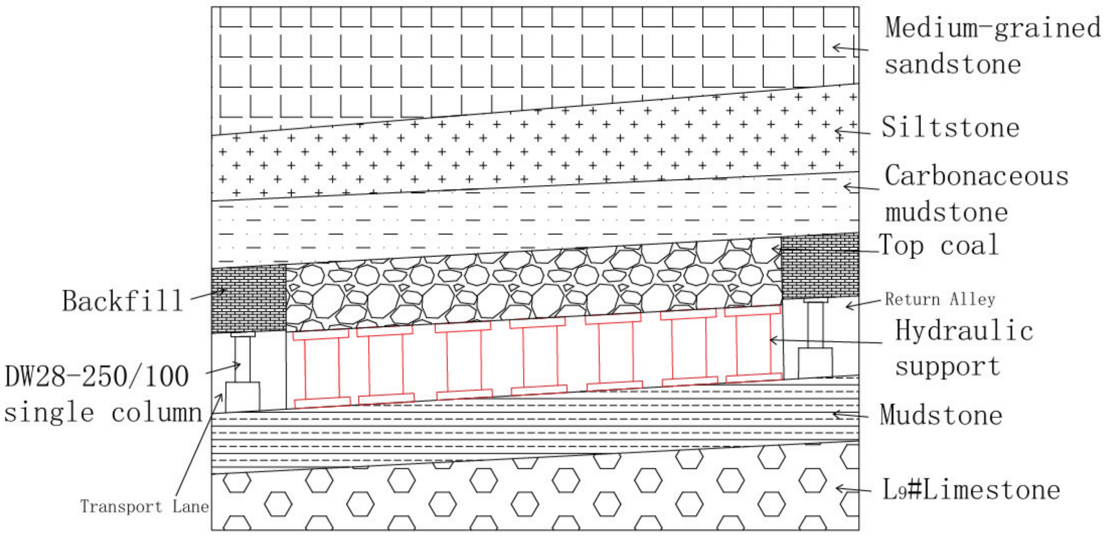


Figure 2(b). I-I Sectional view

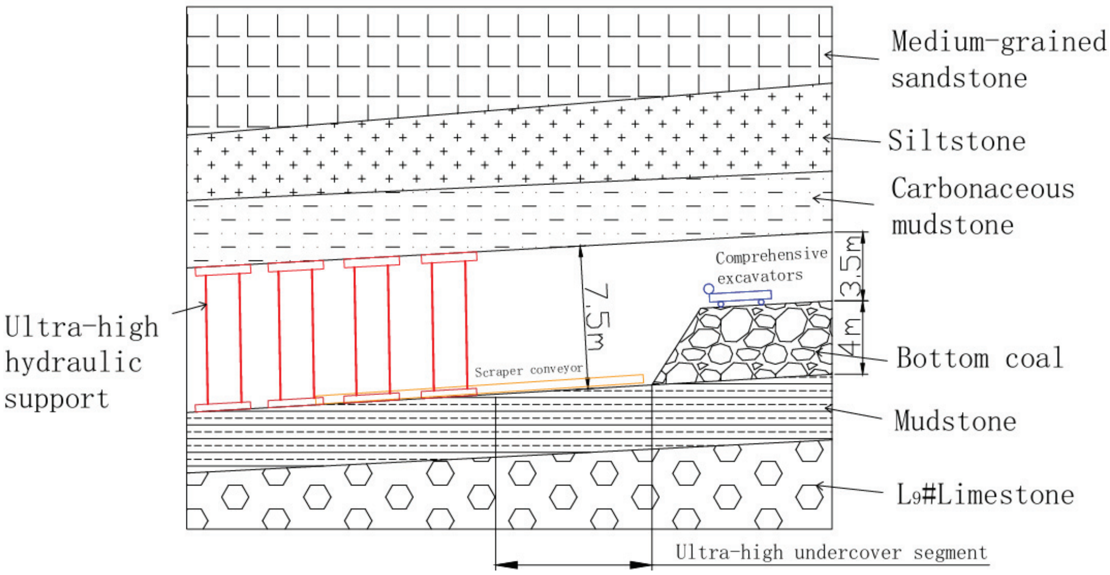


Figure 2(c). II-II Sectional view

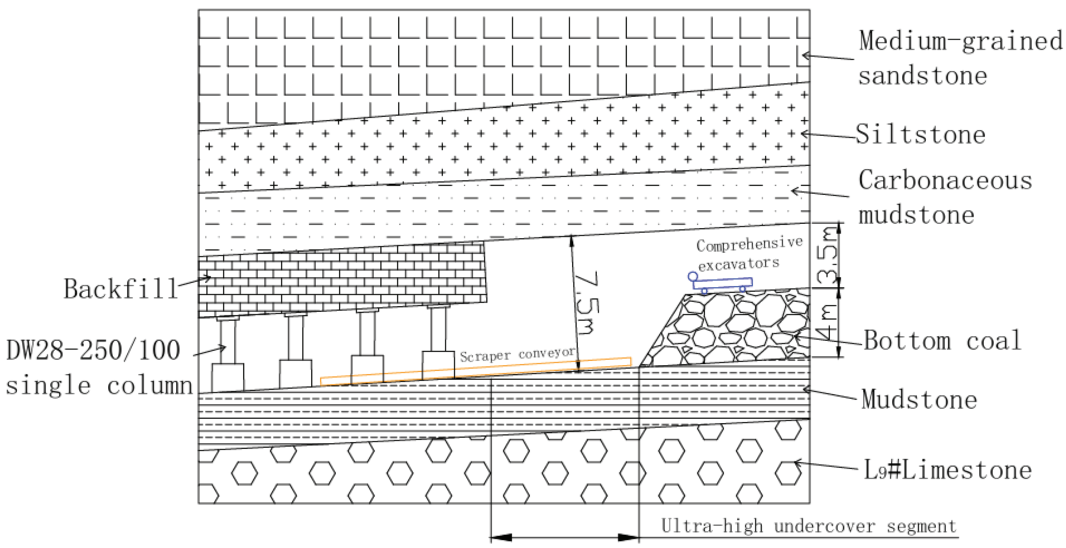


Figure 2(d). II-II Sectional view (improved)

3. Maintenance of Over-high Roadway Roofs

The management and support of ultra-high roadway roof has always been a common problem in coal mining, and the commonly used support measures and advantages and disadvantages in China at this stage are shown in Table 2. Combined with the actual situation of the site and after a comprehensive comparison of various schemes, it was finally determined that all filling and supporting materials should be used to form a joint support system with anchor net cable + backfill as the main body to jointly maintain the stability of the surrounding rock of the roadway. This is shown in Figure 2(d).

Table 2. Comprehensive comparison table of ultra-high roadway roof support scheme

Scheme	Peculiarity
Wooden stack support	The bearing capacity of the wooden stack is low, the compression deformation is serious, and the stability is poor; The wooden stack acts directly on the support, and the safety performance is poor.
Bolt anchor cable reinforcement support	The surrounding rock of the roadway is seriously damaged, and it is difficult to reinforce the support; The empty roof above the support has a large range, the support is difficult to advance, and the stope mine pressure is violent, and the roof is likely to collapse in a large area.
All fill support	After cementation, the backfill can improve the stress state of the surrounding rock, reduce the intensity of the stope pressure, and the control effect of the surrounding rock is significant. However, the performance requirements for the backfill are different in different environments.

3.1. Secondary support after undercover

Due to the increase of the exposed area of the two gangs after the undercover, the increase in the degree of fragmentation of the surrounding rock, the poor maintenance effect of the original roadway support measures on the roadway after the undercover, and the risk of the roof of the piece gang is aggravated, so it is necessary to carry out the secondary reinforcement support of the gang in time. The site is reinforced.

The principle of bolt support is to combine several thin rock layers through the anchoring force of the anchor rod, so that it is anchored as a whole with the thick and stable rock layer, increase the friction between the rock layers, and the bolt itself also provides a certain shear resistance to prevent the relative movement between the rock layers, so as to form a composite beam similar to the anchor reinforcement to the roof to reinforce the roof. The length of the anchor cable is generally selected by the bolt suspension theory:

$$L=kH+L_1+L_2. \tag{1}$$

Where: L-Anchor cable length,m;

H-The height of the falling arch is consistent with the height of the balance arch, and the 1613 working face takes 3.417,m

k-The safety factor is generally 2;

L_1-The depth at which the anchor cable is anchored into the stable rock formation, m;

L_2-The exposed length of the anchor cable is 0.2m;

The row spacing a between the bolts is calculated, (usually the row spacing between the bolts is equal):

$$a=\sqrt{\frac{Q}{kH\gamma}} \tag{2}$$

Where: a-Row spacing between bolts,m;

Q-Anchoring force, kn/root;

H-Falling arch height, m;

γ-Bulk density of anchor rock formation, kn/m³;

k-The safety factor is generally 2.

Based on the geological conditions of the 1613 working face combined with the above calculations, the length of the anchor cable is 6.2m. The minimum anchoring force of the anchor rod selected for the secondary support after undercover shall not be less than 24MPa, and the distance between the rows is set to 900mm×800mm, and the deviation is -50mm~50mm. The specification of the first row of anchor cables below the junction surface of the undercover bottom plate is Φ17.8×6200mm, and the specifications of the anchor cables extending down are Φ17.8×4200mm. For the ultra-wide location between individual rows, the anchor cable of the repair point is supported. At the same time of laying the anchor cable, the gang lays 2600mm×1100mm metal welded mesh, and the lap between the mesh pieces shall not be less than 100mm. The secondary support parameters of the two lanes after the undercover are shown in Figure 3 and 4.

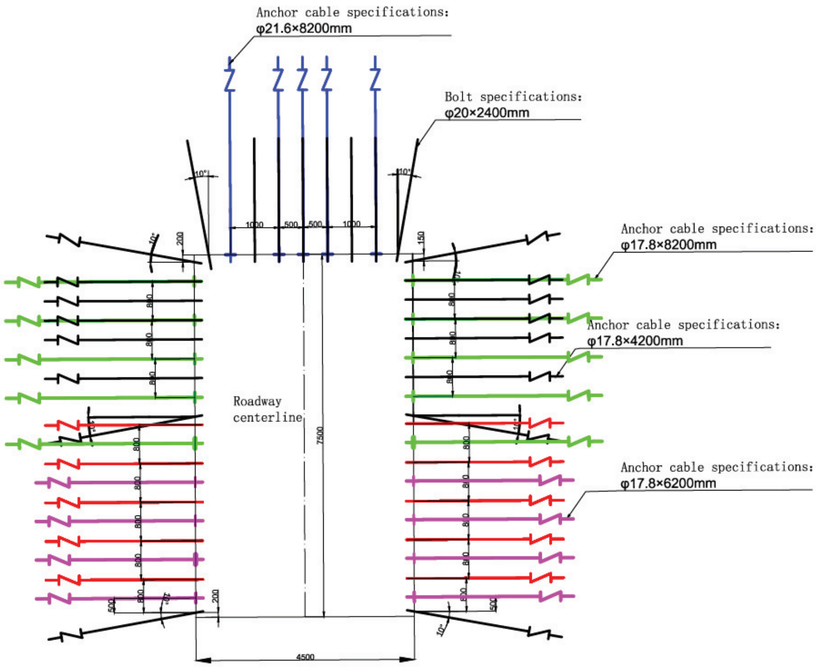


Figure 3. Parameter diagram of secondary support of transportation lane

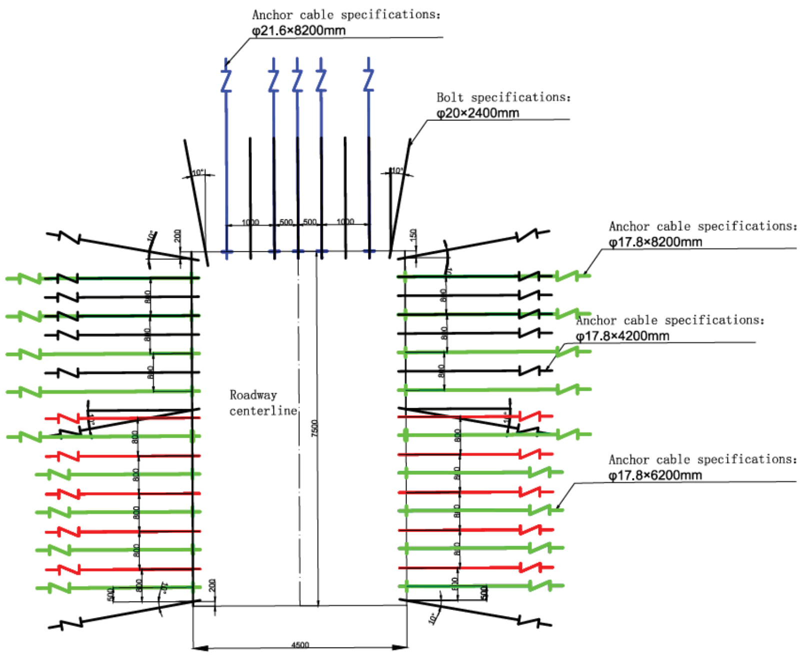


Figure 4. Parameter diagram of secondary support of return air lane

3.2. Top bag filling

Based on the geological conditions of the site combined with the filling cost and other factors, the inorganic two-liquid high-water material is selected for the filling work, which is composed of material A (sulfoaluminate cement) and material B (lime + gypsum), which has the advantages of complete flame retardant, no peculiar smell, non-toxic and non-corrosive, small material consumption, and long-distance pumping. By controlling the content of additives (TC1 early strength agent, TC2 quick-setting agent, TC3 retarder) and the ratio of main material, the solidification time and cementation strength can be freely controlled. The reaction equation is shown in (3, 4, 5).

$$C_4A_3\bar{S} + 2CSH_2 + 34H \rightarrow AFt + 2AH_3(gel). \tag{3}$$

$$C_2S + 2H \rightarrow CSH(I) + CH. \tag{4}$$

$$AH_3(gel) + 3CH + 3CSH_2 + 20H \rightarrow AFt. \tag{5}$$

3.2.1. Analysis of the reasonable strength of the backfill.

As an important index of filling performance, the reasonable strength of the backfill determines the final filling effect. Using the empirical estimation method, the reasonable strength required for the backfill is analyzed with the hydraulic support of the working face:

$$p_t = hrk \tag{6}$$

Where: p_t- Reasonable strength of advance support, Mpa;

h- Support height, m

γ- The bulk density of the roof rock is 25kN/m³;

k- Generally, it is 4~8, which should be reasonably selected according to the specific situation. The cycle pressure of 1613 fully mechanized mining face is more obvious, which is calculated according to the maximum value of 8.

After calculation, it can be seen that p_t=0.639MPa. After the completion of the filling, a supporting structure will be formed with the backfill body, anchor anchor network cable, and overlying rock layer as the main body. In order to simplify the calculation, the strength of the backfill was preliminarily calculated according to the full filling method, and the stress of the backfill was simplified into the load of the overlying rock layer and the horizontal extrusion force generated by the crushing and slipping of the surrounding rock of the roadway.

Thomas et al. used the limit equilibrium analysis method in rock and soil mechanics to propose a method for determining the strength of the backfill through the three-dimensional wedge stability analysis of the backfill, and the calculation formula is shown in (7, 8, 9).

$$p_c = p_1 + p_2 \tag{7}$$

$$p_1 = 9.8\gamma h \tag{8}$$

$$p_2 = \frac{h^2\gamma(\cot\theta - \sin\theta)\cos\theta\cos\left(45° - \frac{\pi}{2}\right)}{2h_1} \tag{9}$$

Where p_1 is the uniform load of the overlying rock layer on the backfill, and p_2 is the horizontal pressure of the loose coal body in the unstable triangle due to the slip of its own weight on the backfill. h_1 is the filling height, and the 1613 working face is 3.5m.

After calculation, it can be seen that p_c=1.68MPa. The greater the strength of the backfill, the smaller the damage degree of the coal, roof and the plastic zone of the two gangs, and the more stable the roadway. However, the strength is too high, and the backfill body can not fall with the mining, resulting in the increase of the suspended area of the roof, forming an unstable beam-type structure, which is easy to cause the roof to collapse; In addition, for high-water materials, increased strength means a lower water-cement ratio and a surge in consumption costs. Therefore, the strength of the design backfill is not less than 1.68MPa, the undercover depth of the roadway of the 1613 working face is 3-4m, and the final height is 7.5-8m, and the double-liquid filling material with a water-cement ratio of 3:1, a strength of 2.1MPa and a grout mixture can reach 80% of the final strength in 7 days and more than 95% in 14 days to meet the actual needs according to the site [21].

3.2.2. Performance test of the backfill.

Due to the presence of geothermal energy, the temperature of the roadway in the 1613 working face is higher than that of other mines buried in the same depth, and the temperature of the two lanes can reach up to 42°C during the advancement of the working face. In order to explore the influence of high temperature environment on the performance of the backfill, the experimental improvement was carried out on the basis of the original experimental ratio to obtain a more stable effect.

Referring to the geothermal temperature of the two lanes of the working face, the SX2-4-10 box-type resistance furnace was used in the laboratory to design four temperature gradients of 20 °C, 30 °C, 40 °C and 50 °C to simulate the curing environment of the sample, the water-cement ratio of the inorganic two-liquid high-water material was set to 3:1, the test water was the laboratory tap water, and the test water temperature was 20 °C. Uniaxial compression and triaxial compression experiments were used to simulate the downhole stress of the backfill, and the test blocks at each curing age were analyzed and tested. The experimental equipment and experimental results are shown in Figures 5, 6, 7 and 8.

It can be seen from Fig. 7 and 8 that in the uniaxial and triaxial compressive tests, when the curing temperature is 30°, the strength of the two-liquid material with a water-cement ratio of 3:1 reaches the highest after condensation into a cemented body. The peak value of uniaxial strength is slightly greater than that of triaxial strength, and with the increase of temperature, the strength curve shows a trend of first increasing and then weakening, and the triaxial strength is smaller than that of uniaxial, which is due to the increase of residual strength caused by the re-compacting of the broken test block in the triaxial test. The experimental results show that the temperature has little effect on the performance of the backfill, and the strength of the cemented body changes little with the increase of temperature, which can still meet the 1.68MPa required in the above calculation.

Figure 5. SX2-4-10 box-type resistance furnace

Figure 6. RMT-301 rock mechanics testing machine

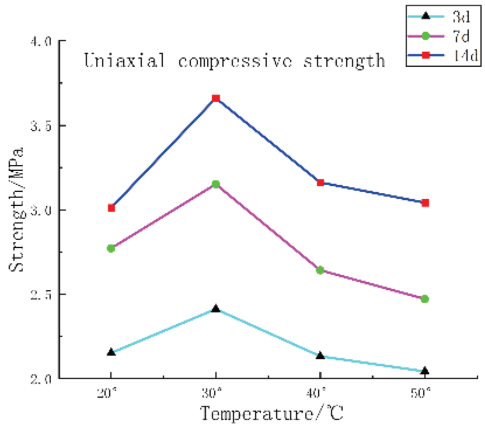


Figure 7. Uniaxial strength as a function of temperature

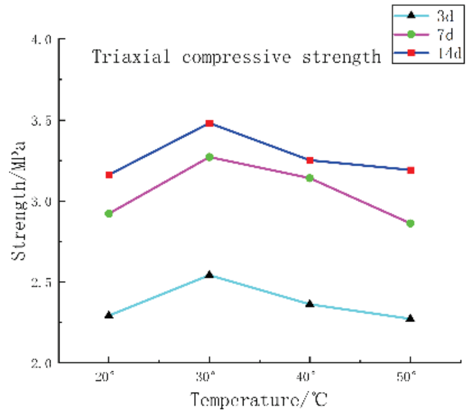


Figure 8. Triaxial strength as a function of temperature

3.2.3. Fill timing selection

The right filling time plays a crucial role in the filling effect; The earlier the filling, the higher the inhibition of the backfill on the deformation of the surrounding rock, the smaller the strength loss of the surrounding rock, the earlier the roadway support, and the more obvious the support effect. If the filling time is too late, the micro-cracks in the surrounding rock will further develop, the degree of fragmentation will increase, and the backfill will have a poor effect on the roadway support, thus losing the filling significance.

Therefore, the filling time should be selected when the surrounding rock has good bearing capacity and anti-deformation ability. The relationship between the deformation velocity and time of each stage of the surrounding rock is shown in Fig. 9, the deformation velocity of the surrounding rock is aggravated in the O-A section, and the elastic strength of the surrounding rock is greater at this time, and the filling at this time has higher requirements for the strength and anti-deformation performance of the backfill. The A-B stage is the slowdown section of the deformation rate of the surrounding rock, and the surrounding rock is in a plastic state with a certain bearing capacity. The surrounding rock of the BC section is basically in the crushing stage under the action of severe confining pressure, and the residual strength is low, so it is difficult to rely on the backfill to continue to maintain the roof.

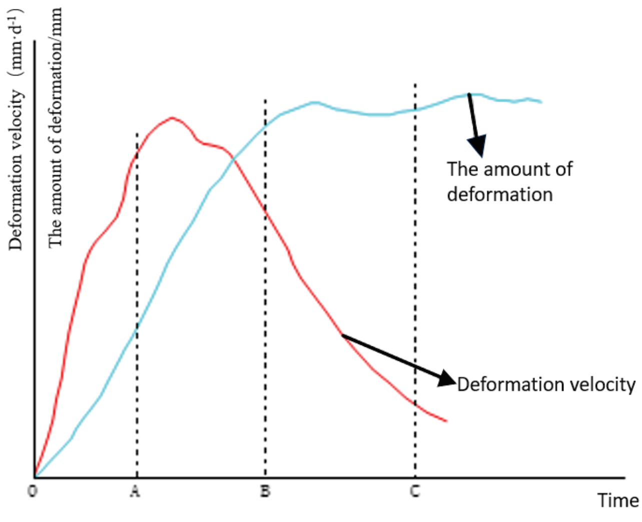


Figure 9. Diagram of the failure process of the surrounding rock

4. Efficient Undercover and Filling Process

4.1. Undercover for the comprehensive excavator

At this stage,the undercover method is mostly manual,but there are problems such as slow speed,poor efficiency,high labor intensity,and difficulty in managing the roof in time.Based on this, it is proposed to use a comprehensive excavator for efficient undercover, which is directly transferred to the coal transportation system of the working face after the coal is discharged, and at the same time, it is proposed to cooperate with the anchor net cable to strengthen and maintain the bare surrounding rock. During the initial mining period, the loose surrounding rock in the construction area was manually removed with the drilling slope of -10° of the working face,and then the EBZ200/160 comprehensive excavator was used for mechanized undercover,and a 40T scraper conveyor was equipped to transport coal.The simplified diagram of the undercover process is shown in Figure 10.

The preliminary setting process sequence is as follows: 20m undercover for the return air lane ahead of the working face→ 15m for the top filling of the 15m advance working face for the undercover rear support →; 90m undercover → undercover rear support → 70m top filling of the advanced working face of the transportation lane. The undercover process of the two lanes is shown in Figure 11 and 12 respectively

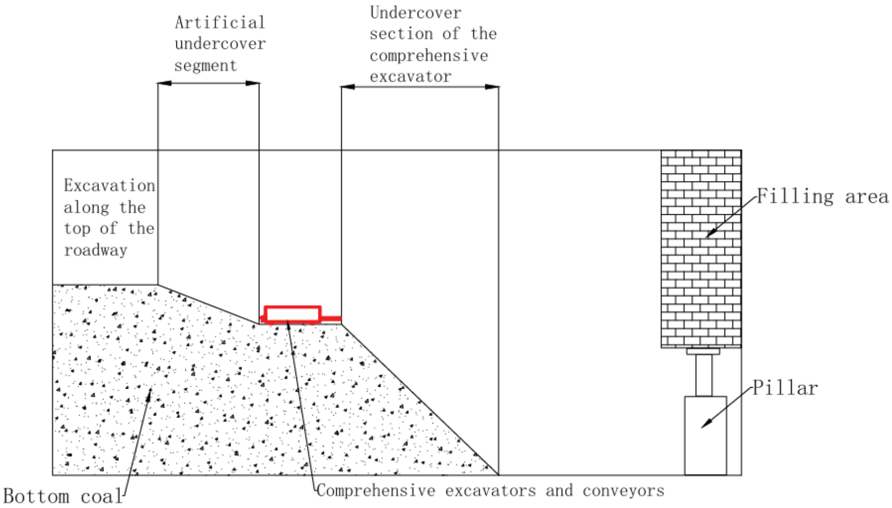


Figure 10. Undercover process diagram

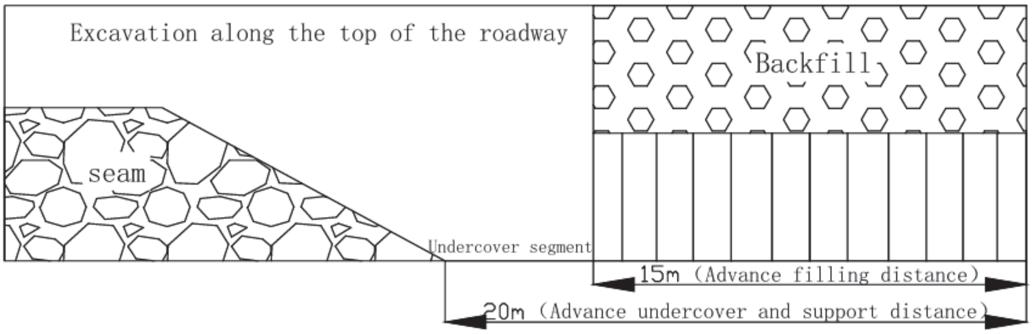


Figure 11. Undercover process diagram of return air lane

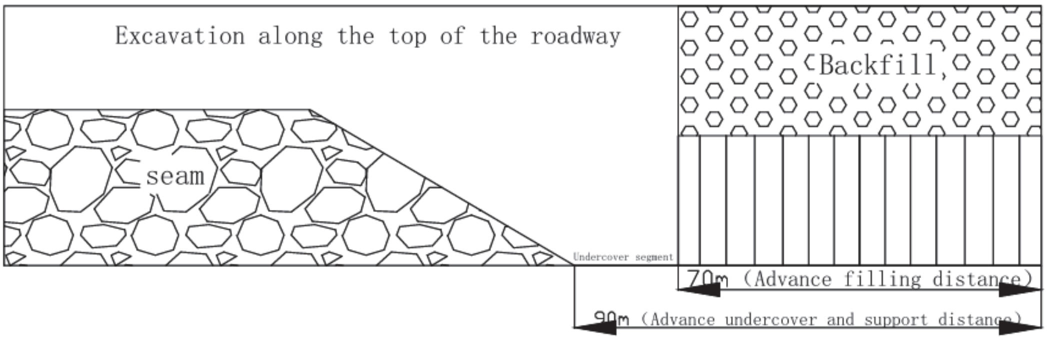


Figure 12. Undercover process diagram of transportation lane

4.2. Monorail hoisting automatic filling process.

At present, most of the traditional filling systems have many shortcomings, such as low filling efficiency, high labor intensity, short conveying distance, and continuous outward movement with the need to follow the working face. The 1613 working face adopts a new set of intelligent pulping + long-distance conveying and filling system, which can achieve the purpose of reducing the labor intensity of workers, efficient pulping and long-distance transportation, so as to meet the requirements of safe, efficient and rapid advancement of the working face.

The 1613 working face is about 1.5m per day, the transportation lane and the return air lane are filled work, and the total consumption of two 12m long filling bags along the groove is consumed in 4 days, and the filling class is designed according to a filling class every day, then the filling amount of 13 needs to be completed along the groove every day, and the filling of two cycles of two roadways can be completed within 3 days, and the mining footage of 4 days is satisfied. The volume of a single filling cycle is 125m³, and the material consumption of each filling bag is about 38t, of which 19t is for material A and 19t. Therefore, according to the above design, the belt along the groove and the return air along the groove each consume 41.7m³ of material, and a total of 25 tons of materials are required. The working time of each filling shift is calculated according to 6h, and the grouting capacity requirement is 83.4m³/6h=13.9m³/h. After calculation, it can be seen that the pulping capacity of the 1613 working face is 20m³/h, and the conveying capacity of the pumping station is designed to meet the grouting demand according to 15m³/h.

The pulping and filling equipment is mainly composed of 2 tracks, 2 pneumatic hoists, 2 pneumatic screw feeders, 2 high-speed pulping machines, 2 low-speed mixers, 1 high-flow grouting pump, supporting automatic control systems, and supporting pipelines. The whole pulping process is automatically completed by the PLC control system in accordance with the established procedures, except for the manual assistance required for the mixing machine blanking. The equipment layout is shown in Figure 13.

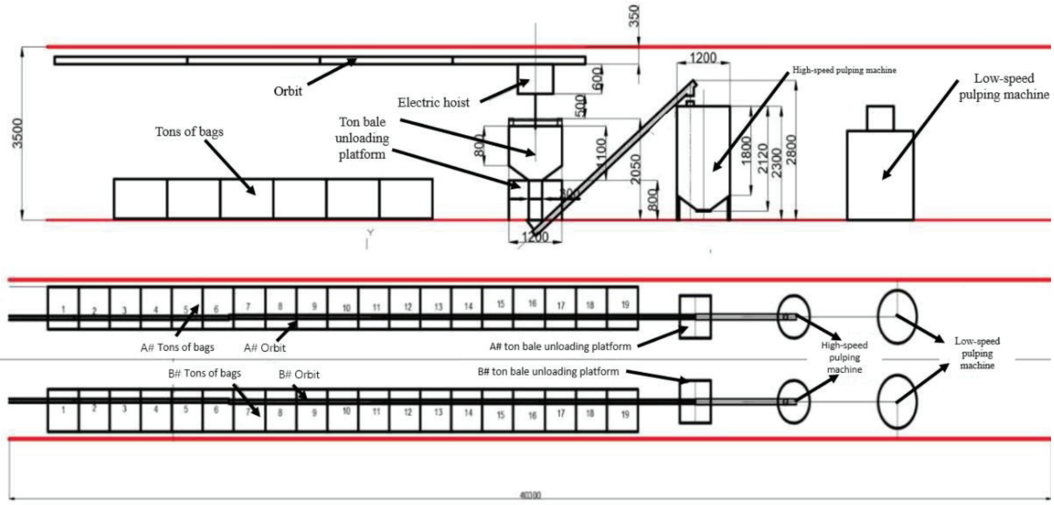


Figure 13. Equipment layout drawing

5. Filling Effect Test

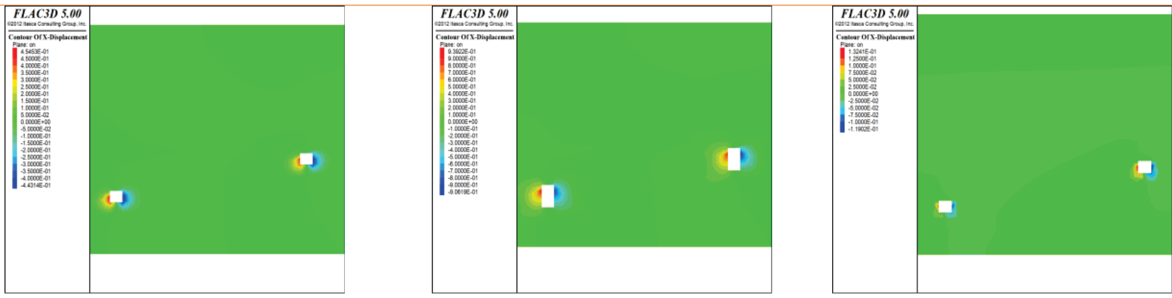
5.1. Numerical simulation verification.

In order to verify the on-site filling effect, the numerical simulation software FLAC3d was used to establish the stress model of the overfill at the top of the ultra-high roadway, and the support effect after filling was simulated and analyzed. Some of the surrounding rock mechanical parameters refer to the mechanical test results of the surrounding rock near the working face of the mining roadway, as shown in Table 3. The horizontal displacement contours of the two sides of the roadway at each stage and the subsidence clouds of the top and bottom plates are shown in Figs. 14 and 15 respectively, and the data curves are shown in 16 and 17.

After comparing and analyzing the proximity of the two sides at 20m from the working face at different stages, it can be seen that the maximum horizontal displacement of the two sides of the roadway increases from 465mm (excavation period) to 895mm (undercover period), and finally decreases to 132mm (after filling). The maximum roof subsidence is increased from 290 mm (excavation period) to 910 mm (undercover period) and finally to 98 mm (after filling). The maximum subsidence and the maximum displacement peak are located at 5-8m of the advanced working face, and the surrounding rock of the roadway is under the action of the advanced supporting pressure, the development degree of the internal cracks in the rock mass is improved, the damage is more serious, and the two gangs shrink more violently in the roadway, so the reinforcement support should be focused on when undercover in the area.

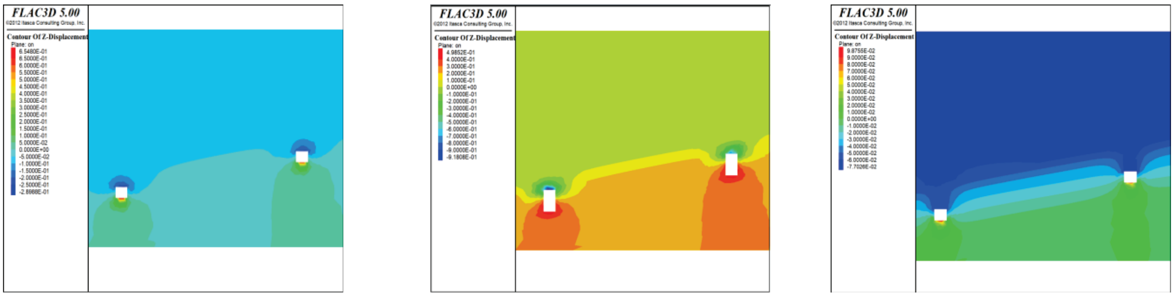
Table 3. Partial coal and rock mechanical parameters

The name of the rock	Bulk(Gpa)	Shear(Gpa)	Cohesion(MPa)	Internal friction angle(°)	Density (kg/m³)
Medium-grained sandstone	12.10	9.15	2.67	37	1800
Siltstone	8.42	6.52	2.88	41	2600
Carbonaceous mudstone	6.55	5.95	2.55	40	2570
II. 1 coal seam	4.26	2.81	1.27	31	1640
Limestone	7.71	3.60	1.42	28	2130
Backfill	2.12	2.92	0.97	30	1650



Roadway excavation period Roadway undercover period After the top is filled

Figure 14. Diagram of the approaching volume of two gangs at a distance of 20 m from the working face at each stage



Roadway excavation period Roadway undercover period After the top is filled

Figure 15. The subsidence of the roof at each stage 20m away from the working face

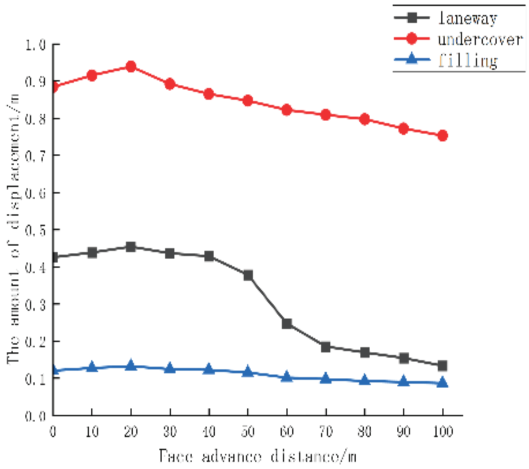


Figure 16. Diagram of the approach of the two gangs at each stag

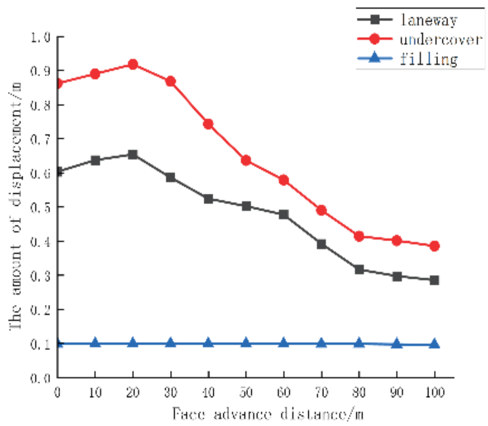


Figure 17. Curve diagram of roof subsidence at each stage

Through 15, it can be seen that the subsidence of the roof increases significantly after undercover, but the amount of the bottom drum decreases from 650mm to 490mm, which is due to the secondary reinforcement support of the roof and the two sets of supplementary anchor cables after undercover in this paper (3-1), and the stress is not transmitted to the bottom plate. It can be seen from Fig. 16 and 17 that the subsidence of the roof plate is reduced by nearly 800mm after filling, accounting for 88% of the total displacement of the roof and bottom plate, and the proximity of the two gangs is reduced from 940mm to 130mm, which is reduced by 86%. And there is no large-scale deformation and failure of the backfill body after filling, and only a small amount of compression is found at the top and bottom corners, so the deformation of the roadway of the 1613 working face can be effectively controlled by using high-water materials for the top belt filling management roof.

5.2. Field measurements.

The displacement measurement points were arranged by the cross distribution method on the 1613 working face to monitor the displacement of the roadway surface at each stage. Each measuring point is spaced 100m, and the monitoring frequency is collected once every 10m advanced by the working face, and the measurement point layout is shown in Figure 18, and the monitoring results are shown in Figure 19, 20, 21, and 22.

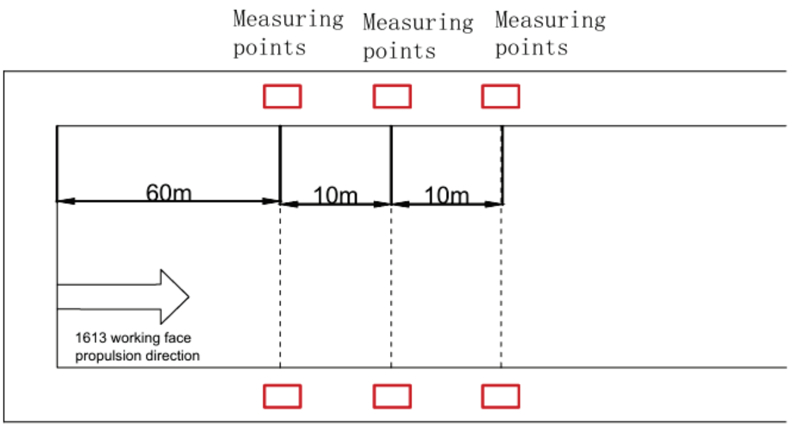


Figure 18. Measurement point layout drawing

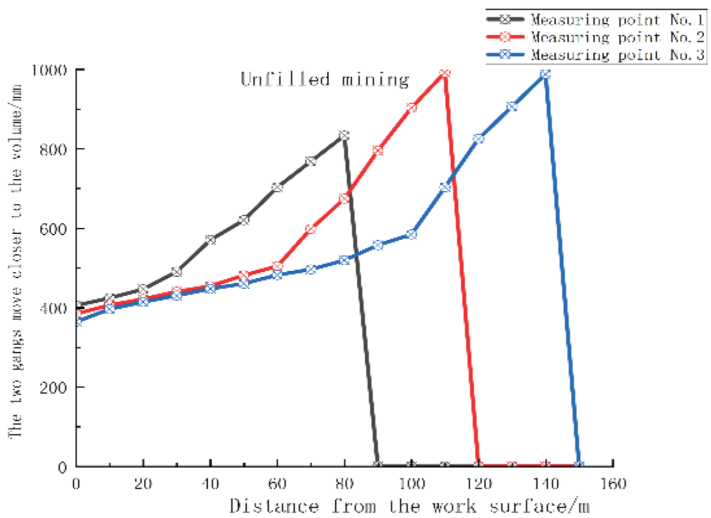


Figure 19. The monitoring results of the two gangs were not filled

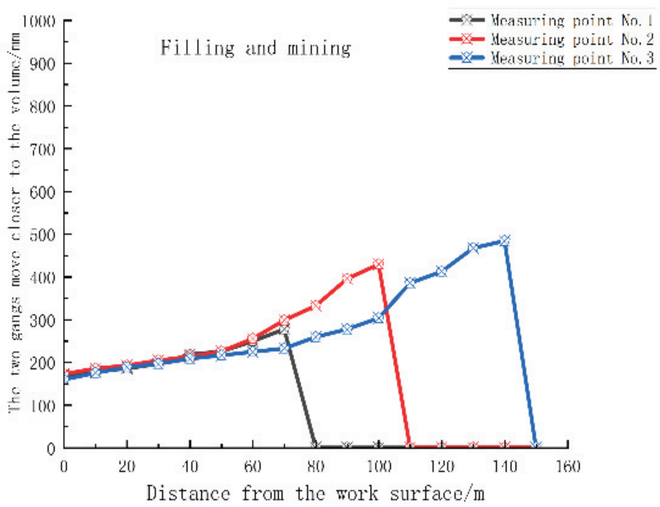


Figure 20. Diagram of the monitoring results of the approaching volume of the two gangs after filling

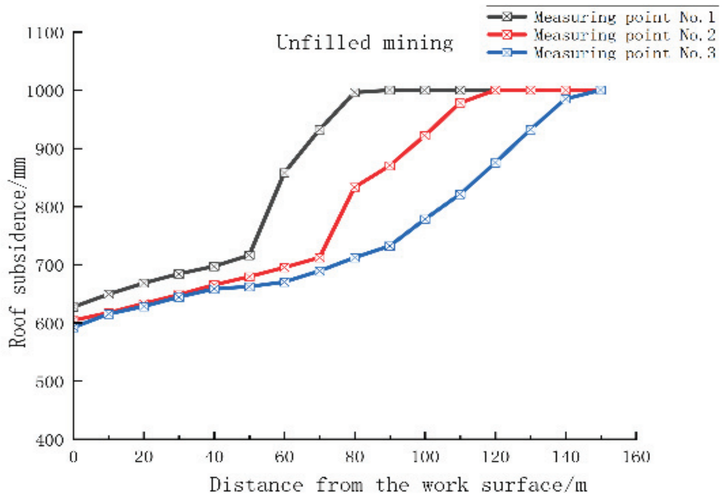


Figure 21. The result of monitoring the subsidence of the unfilled roof

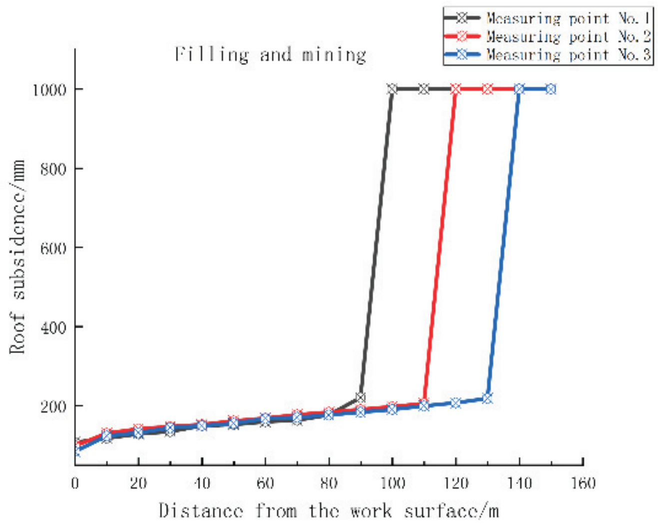


Figure 22. Monitoring results of roof subsidence after filling

Analysis figures 21 and 22 show that because the on-site roof management adopts the mining and falling, the roof sinks completely when the working face is pushed past the measurement point. Comparing the subsidence of the roof of the roadway before and after filling, it can be seen that the subsidence of the roof after filling is sharply reduced to 105mm, which is nearly 5 times lower than that of 626mm before filling. The amount of movement between the two gangs has been reduced from 410mm before filling to 160mm, a decrease of nearly 60%. When the working face is about 80-100m away from the measuring point, the subsidence of the roof of each measuring point and the approach of the two gangs reach the peak and tend to be stable with the complete collapse of the roof, which is basically consistent with the numerical simulation results. Through comparative analysis, it can be clearly concluded that the combined support system of bolt anchor net cable + backfill formed after the roadway filling of the 1613 working face has a significant effect on the deformation control of the surrounding rock, which can ensure the safe and efficient mining of the working face.

6. Economic Benefit Analysis

1613 working face two grooves according to the average height of 3m, roadway width 4.5m, filling length of 510m, then 82,273 tons can be recovered, the price of raw coal is calculated according to 1,000 yuan/ton, can increase the profit of 82.27 million yuan; The cost of filling materials is about 6.4 million yuan, and the net profit is 75.87 million yuan.

There are about 15 hydraulic supports on the working face to press coal, the length of coal pressing is 22m, the average coal pressing of the two lanes is 9m, and the total coal pressing is 22+9=33m; The average thickness of pressed coal per meter is calculated according to 3m, and the recoverable coal resources are: 510m×31m×3m×1640kg/m^3=77785.2t, and the price per ton of coal is calculated at 800 yuan, which can increase the profit by 56 million yuan; The cost of filling materials is calculated according to 12,000 yuan/m, the cost of filling materials is about 6.1 million yuan, and the profit can be increased by about 49.88 million yuan.

7. Conclusion

(1) Through comprehensive analysis and comparison, it is determined that the site adopts the full filling method to maintain the stability of the surrounding rock of the roadway; It is clarified that the length of the secondary support anchor cable after undercover is 6.2m, and the strength of the on-site backfill should be greater than 1.68MPa.

(2) It is proposed to use the undercover of the comprehensive excavator and the belt conveyor to carry out high-efficiency undercover, and the supporting monorail hoisting intelligent automatic filling process can realize continuous and efficient long-distance transportation; The pulping capacity can be measured to fully meet the on-site filling needs.

(3) The numerical simulation analysis results show that with the advancement of the working face, the maximum subsidence of the roadway roof is reduced from 910mm to 98mm, and the maximum proximity of the two gangs is reduced from 895mm to 132mm. After the industrial experiment, the monitoring results showed that the maximum subsidence of the roadway roof was 114mm, and the maximum proximity of the two gangs was 185mm. After filling, the surrounding rock control effect is good, and the backfill body does not have obvious compression deformation, which can ensure the safe use of the roadway during production and mining. Provide experience for the control of surrounding rock in the same type of mine roadway.
(4) It is estimated that the income of the two troughs and working faces after the undercover can increase by nearly 160,000 tons of raw coal, and the direct economic benefits of income generation are nearly 100 million yuan, which greatly improves the resource recovery rate.

References

[1] Kang Hongpu, Fan Mingjian, Gao Fuqiang, et al. Chinese Journal of Rock Mechanics and Engineering, 2015, 34 (11): 2227- 2241.

[2] HE Manchao, XIE Heping, PENG Suping, et al. Chinese Journal of Rock Mechanics and Engineering,2005,24(16) : 2803-2813.

[3] Nie Baisheng, Lu Hongqi, Li Xiangchun, et al. Journal of China Coal Society,2015,40(4):754-759.).

[4] WANG Zhe. Study on the Appearance Law of Mining Pressure in the Comprehensive Release of Ultra-long Working Face[J]. Safety in Coal Mines, 2017, 48(2): 188-191

[5] WANG Yuhe, CHEN Wenguo, ZHANG Pu, et al. Simulation study on the reasonable position of the final mining line under the influence of mining[J]. Coal Engineering, 2016, 48(5): 65-67,70

[6] Xie J, Xu J, Wang F. Mining-induced stress distribution of the working face in a kilometer-deep coal mine-a case study in Tangshan coal mine-a case study in Tangshan coal mine. Journal of Geophysics and Engineering, 2018, 15 (5): 2060- 70

[7] FANG Xinqiu,XU Ruiqiang,ZHAO Junjie. Research on the instability mechanism and control of triangular coal in goaf fully mechanized working face[J].Journal of China University of Mining and Technology,2011,40(5):678-683.

[8] B Yu, J Zhao, T J Kuang. In situ investigations into overburden failures of a super-thick coal seam for longwall top coal caving[J]. International Journal of Rock Mechanics and Mining Sciences, 2015, 8(78). 155-162.

[9] J Zhang, L Liu. Asymmetrical Support Technology for Dynamic Pressure Roadway: A Case Study from Guotun Coal Mine in China[J]. Geotechnical and Geological Engineering, 2018, 37(2):823-832.

[10] S C Li, Q Wang, H T Wang. Model test study on surrounding rock deformation and failure mechanisms of deep roadways with thick top coal[J]. Tunnelling and Underground Space Technology incorporating Trenchless Technology Research, 2015, 9(47). 153-167.

[11] Li Jixing, Xu Lei, Jiang Hongjun. Stability Mechanism and Support Technology of Jugao Roadway[J]. Safety in Coal Mines, 2012, 43(10): 60-63.

[12] CHEN Qiuyu. Research on filling materials and filling technology of 3122 fully mechanized discharge surface empty roadway of Jingxin Coal Industry[D]. China University of Mining and Technology, 2018.

[13] HE Xiangning, CHEN Yong, QIN Zhengyuan. Research and application of hollow roadway technology in fully mechanized working face[J]. Coal Science and Technology, 2017, 45(6): 124-130

[14] Wang Cheng, Ding Ziwen, Xiong Zuqiang, et al. Deformation law and control technology of surrounding rock of large-section dynamic pressure mining roadway[J].China Safety Science Journal, 2018, 28(5): 135-140.

[15] Xiong Zuqiang,FAN Chuanhe. Coal Science and Technology, 2015, 43(5): 13-16.)

[16] SUN Ruyi. Research on bag filling technology at the top of ultra-high mining roadway in fully mechanized working face[D].Jiaozuo:Henan Polytechnic University, 2020.

[17] X D Zhang, D Y X. Support Technology of Ultra-high Roadway in Thick Coal Seam[J]. Information Technology Journal,2013,12(17):4095-4100.

[18] Hu Fangjian, Fang Yongjun. 107 Shandong Coal Science & Technology,2011,21(04):88+90.

[19] ZHANG Baolin. Shandong Coal Science & Technology, 2020, 26(01): 44-45.

[20] Cao Qijia, Chen Yong, Song Kun. Research and application of high-water material filling technology for deep working face through empty roadway[J]. Coal Technology, 2017,36(6):55-57.

[21] JIA Junfei. Research on the instability mechanism of surrounding rock and top bag filling control of ultra-high roadway in extremely soft and thick coal seam[D].China University of Mining and Technology,2022.