

Article

A Method and Engineering Practice for a Fully Mechanized Caving Coalface to Rapidly Pass Through a Large Fault

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Abstract: One of the technical problems that must be solved in coal mine production is when the coalface rapidly crosses the fault. Based on the occurrence characteristics of the F_6 fault (maximum throw: 13.5 m) in the #3_{up} 1101 fully mechanized caving coalface at Gaozhuang Coal Mine, two different solutions allowing the coalface to pass through this fault were proposed, and the solution of pre-driven roadways with rock pillars was optimally determined. The main implementation steps of the method include designing the layout parameters of the pre-driven roadways, determining the width for rock pillars between the adjacent pre-driven roadways, construction of pre-driven roadways by smooth wall blasting, and controlling the surrounding rock deformation of the pre-driven roadways. The results of engineering practice show that it took only 23 days for this coalface to pass through fault F_6 , about one month shorter than the time required by traditional methods (e.g., proactively taking a detour). Moreover, this method helped achieve stable coal production (an 8.5×10^4 t increase), prevented much gangue from mixing with coal, reduced wear and tear on the mining equipment, and enhanced safety. The economic benefits delivered totaled about CNY 71.1 million. Therefore, this method can ensure continuous, safe, and efficient mining at the coalface, alleviating the tight situation of mine production succession. The results of this study can provide a good reference to help coalfaces rapidly move across faults under similar geological conditions in other mines.

Keywords: fully mechanized caving coalface; large dip angles; large-throw fault; pre-driven roadways; continuous advance



Academic Editor: Yosoon Choi

Received: 22 November 2024

Revised: 5 January 2025

Accepted: 10 January 2025

Published: 13 January 2025

Citation: Zhang, W.; Yang, F.; Chang, J.; Zhao, S.; Xu, B.; Xiang, J. A Method and Engineering Practice for a Fully Mechanized Caving Coalface to Rapidly Pass Through a Large Fault. *Appl. Sci.* **2025**, *15*, 731. <https://doi.org/10.3390/app15020731>

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1. Introduction

In China, coal will maintain its status as the principal energy source for quite a long time [1]. As China's strategic coal reserve plan is gradually improving and high-quality coal resources occurring in good conditions are being depleted, there is currently a growing demand for thick coal seams buried in complex geological conditions. Coal seams with large dip angles account for an estimated 15–20% of China's proven recoverable coal reserves and their annual production makes up 10% of the country's total coal production [2]. Coal seams with large dip angles form in complex conditions and are unstable [3–5]. Coalfaces in such coal seams often encounter faults and other difficult geological structures as they advance. The rock masses around faults are normally fragmented due to tectonic stresses [6,7], adding to the difficulty of controlling the roof and posing a safety threat as a coalface advances through a fault [8,9]. Other problems associated with faults include the slow

advance of a coalface, the large volume of waste rock produced, and the low recovery rate, which have a strong adverse impact on production efficiency [10,11]. Therefore, the technical difficulty of moving past faults in front of coalfaces quickly must be tackled in order to achieve safe and efficient production.

Currently, ensuring safety while achieving rapid fault pass through at the coalface has become a focus of research for many scholars. Zhao et al. [12] developed a mechanical model for coal pillar protection in fault zones, analyzed tectonic stress distribution and stability factors, and proposed a perimeter rock control method combining “slant mining” and “grouting”. Han et al. [13] used the UDEC Voronoi numerical model to study the progressive damage of surrounding rock structures when mining through a fault and proposed a damage index suitable for engineering applications. Das et al. [14] studied the impact of fault dip and drop on coal pillar strength and quarry enclosure stability using DEM and statistical analysis. Ma et al. [15] analyzed stress distribution in the quarry under different fault structures through similar simulation experiments and improved perimeter rock stability by enhancing support. Fu et al. [16] explored the fault crossing method using FLAC3D and validated its effectiveness in crossing the fault zone with the “portal bracket + top-cutting and shielding bracket” support method. Guo et al. [17] used a machine learning algorithm, preferring the gradient boosting decision tree method, to predict surrounding rock deformation in the fault area, providing guidance for precise control of the surrounding rock in the quarry over the fault. Li and Yang [18] applied loose blasting technology to fault penetration in the comprehensive mining face, achieving non-repetitive cutting by optimizing blasting parameters and processes. Zhang et al. [19] proposed pre-digging single lane over fault technology, using reasonable “replacement” of excavated gangue and coal to achieve rapid, gangue-free coal mining over the fault. Guo and Peng [20] proposed adjusting the coal cutting height for small drop faults by leaving top or bottom coal to pass through the fault, then gradually mining the top and bottom to restore the original mining height. Weiming and Debing [21] proposed two methods, “roof broken” and “floor broken”, for the coalface pass through fault. Through FLAC simulation analysis, they concluded that both methods require focus on area protection and applicable conditions. Wang and Zhang [22] proposed an “avoidance and concession” method for the coalface, adjusting the two alleys and the position of the coal mining face according to the fault’s spatial pattern to minimize or eliminate rock cutting.

The traditional approaches mentioned above are normally applicable only under certain geological conditions, and possess some unavoidable weaknesses. Therefore, it is of great practical significance and engineering value to explore a method of coalface crossing fault with strong applicability, good safety, and good continuity. Based on the engineering background of the Gaozhuang Coal Mine (GCM) in the Zaozhuang Mining Area, a method for quickly passing through large faults using pre-driven roadways (PDRs) with rock pillars is explored in view of the occurrence characteristics of the F_6 fault in front of the #3_{up}1101 coalface, which has a maximum throw of 13.5 m. The method includes key technological processes such as designing the PDRs layout parameters, determining the width of rock pillars between adjacent PDRs, construction of PDRs by smooth wall blasting (SWB), and controlling surrounding rock deformation of the PDRs. After this new method was implemented at the #3_{up}1101 coalface, the coalface rapidly passed through fault F_6 and continuous, safe, and efficient mining was achieved, which guaranteed the coal production and quality, and realized good technical and economic benefits. The results of this study can provide both a method reference and experiential guidance for rapid passage of coalfaces across faults under similar geological conditions in other mines.

2. Engineering Background

2.1. Overview of GCM

GCM (As shown in Figure 1), one of the main mines run by Zaozhuang Mining Group, is situated in the southwest of Tengnan Coalfield, with its main shaft located in Weishan County, Shandong Province, China. Covering an area of 32 km², GCM stretches 8.5 km long in the north–south direction and 2.5 to 5.5 km in the east–west direction. It borders Fucun Coal Mine to the east, Xuzhuang Coal Mine to the west, Jiangzhuang Coal Mine to the north, and Kongzhuang Coal Mine to the south. The topography of the coalfield is characterized by a gently southeastward-sloping alluvial plain formed in a marginal lacustrine environment, which ranges in elevation from +33 m to +34 m. GCM was opened in 1997 with a designed production capacity of 0.9 Mt per year. After several technical upgrades, this mine currently has an approved production capacity of 3.5 Mt per year and has a remaining service life of about 25 years, which belongs to a typical resource-depleted aging mine. It was developed by vertical shafts at multiple levels. Parallel central ventilation shafts serve to draw air out of the mine. The main coal seams being worked are seams #3_{up} and #3_{down}, and the elevations of the corresponding mining levels are −430 m and −660 m, respectively.

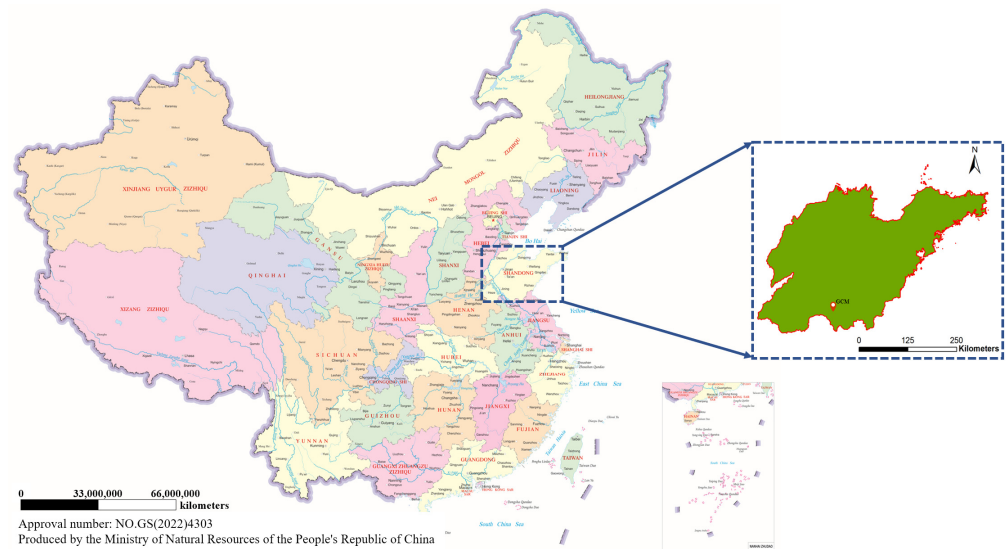


Figure 1. Geographical location of Gaozhuang Coal Mine (GCM).

2.2. Mining Technical Conditions of the Study Area

The #3_{up}1101 coalface is the first coalface of the western eleventh mining district at the −600 m level in GCM. This coalface is 1110 m to 1246 m long (with average of 1178 m) in the direction of advance and 223 m wide, covering an area of 262,694 m². Its elevation is between −330 m and −456 m and the elevation of the surface above it falls within the range of +33.04 m to +34.02 m. The #3_{up} coal seam extracted at the coalface belongs to the Shanxi Formation of Lower Permian Series in Paleozoic Era. The coal seam is 4.4 m to 6.4 m thick, 5.1 m on average. Its dip angle varies widely (from 4° to 27°), with an average of about 16°, which belongs to a typical large dip coal seam. The false roof of the coal seam is mudstone with an average thickness of about 0.3 m. The immediate roof is siltstone with an average thickness of about 10.4 m. The main roof is mainly composed of coarse and medium sandstone with an average thickness of about 77 m. The immediate bottom is silty mudstone with an average thickness of about 4.4 m. The coal is extracted from this coalface by a fully-mechanized caving method, with a cutting height of 2.9 m and caving height of 2.2 m. The cycle progress is 0.8 m, and the roof behind the coalface is allowed to

completely collapse into the goaf. Figure 2 shows the generalized stratigraphic column of this study area.

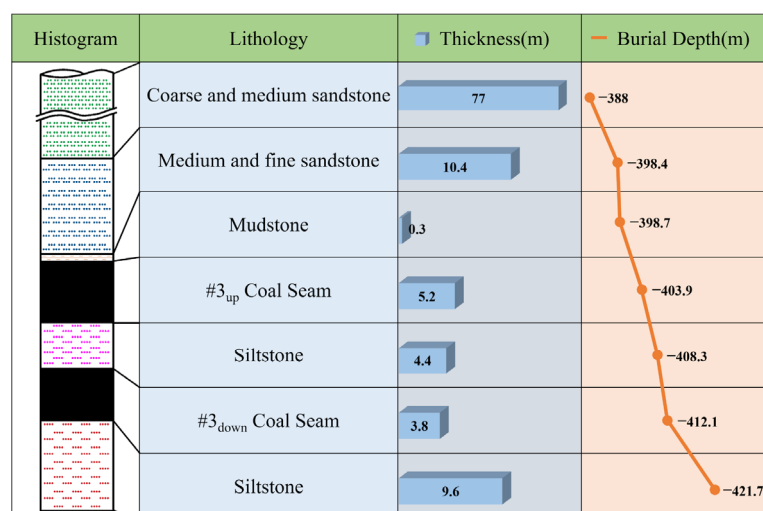


Figure 2. Generalized stratigraphic column of this study area.

The overall structure of the #3_{up}1101 coalface is a single-dip structure. There are 26 exposed faults within the coalface, with 8 faults having a drop greater than 2 m and the remainder being less than 2 m. Among them, the F₆ fault, which was actually exposed during the excavation process of the headgate of the coalface, has a maximum drop of 13.5 m, a full rock section length of 23 m, and is 543 m away from the coalface opening. The drop gradually decreases along the direction of the tailgate. In order to further investigate the occurrence characteristics of the F₆ fault, three derivative faults were explored 28 m ahead of the exposed fault location by two survey roadways, and the final determination of the F₆ fault showed a strike direction of 18°, a dip direction of 76°, a dip angle of 55°, which formed an 87° angle with the coalface advance direction, and would locally have a significant impact on the mining process. The actual occurrence characteristics of the F₆ fault are shown in Figure 3.

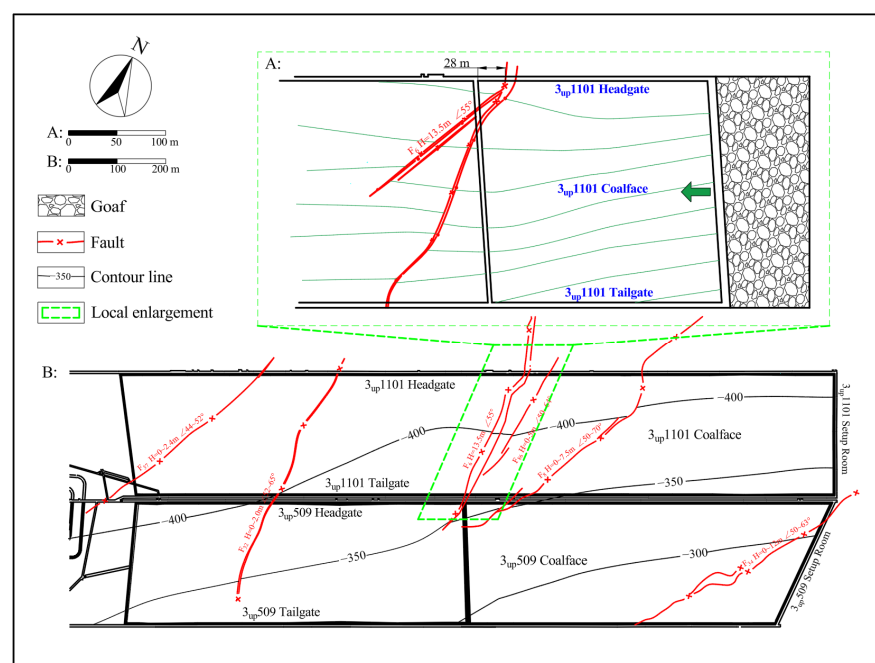


Figure 3. Actual occurrence characteristics of F₆ fault.

3. Determination of a Reasonable Solution to Pass Through Fault F₆

3.1. Proposed Two Solutions for Crossing the Fault

Considering the conditions of the #3_{up} coal seam and the actual occurrence characteristics of the F₆ fault, if using the forcible passage method, the range of rock breaking would be large, the propulsion speed would be slow, and it would be easy to cause serious damage to the mining machines and generate a high coal–gangue mixture. If using the detour method, the coalface would need to be partially dismantled and reinstalled, the on-site construction time would be long, the implementation process would be complex, and the fault protection coal pillar may easily experience spontaneous combustion in the mined-out area, increasing the safety production risk of the coalface. In view of the above reasons, in order to achieve safe and efficient production of the coalface, two solutions for crossing the F₆ fault were proposed. Solution 1 uses PDRs with rock pillars, while solution 2 uses a single PDR without rock pillars.

(1) Solution 1: PDRs with rock pillars

Figure 4 shows the schematic of solution 1. In this solution, starting from the F₆ fault exposed in the headgate, according to the mining level of the coalface, a group of roadways with rectangular cross-section are pre-driven into the panel parallel to the main investigation roadway, with each roadway width of 3 m and height of 3 m. Each pre-driven roadway intersects with the secondary investigation roadway, and a certain width of rock pillar is left between adjacent pre-driven roadway. The surrounding rocks of the PDR are supported by a combined system of “anchor–net–ladder–beam–cable”.

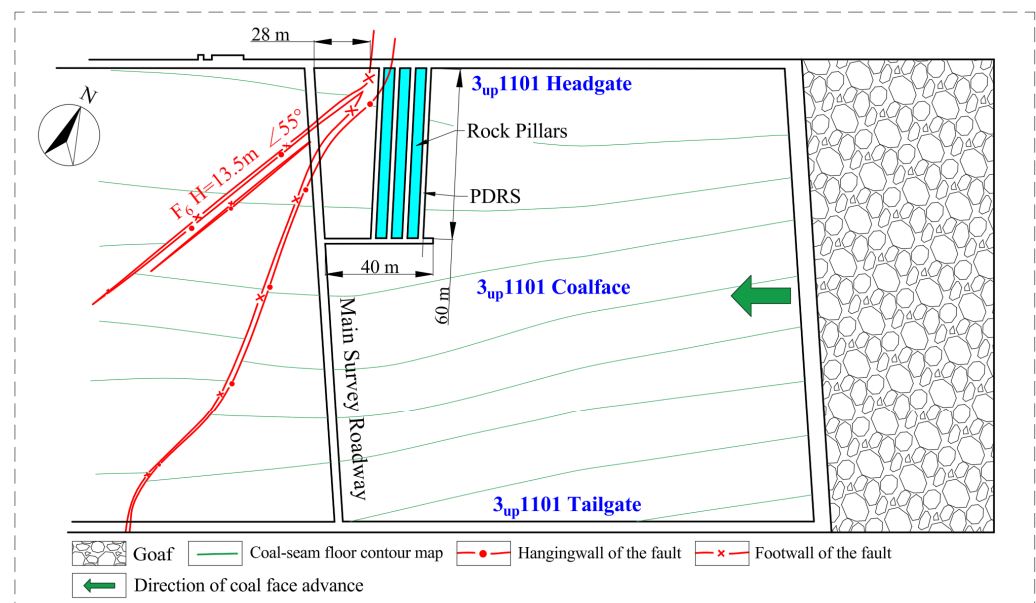


Figure 4. Solution of pre-driven roadways (PDRs) with rock pillars.

(2) Solution 2: A single PDR without rock pillars

Figure 5 shows the schematic of solution 2. In this method, a single roadway is pre-driven into the panel from the intersection between fault F₆ and the headgate. With a width of 12 m and a height of 3 m, this roadway has an extra-large cross-section. This scheme will form a 12 m-wide rockless area. In the rockless area, a joint support method of “anchor–net–ladder–cable” is used, as well as a composite support method of “individual hydraulic props and beams”.

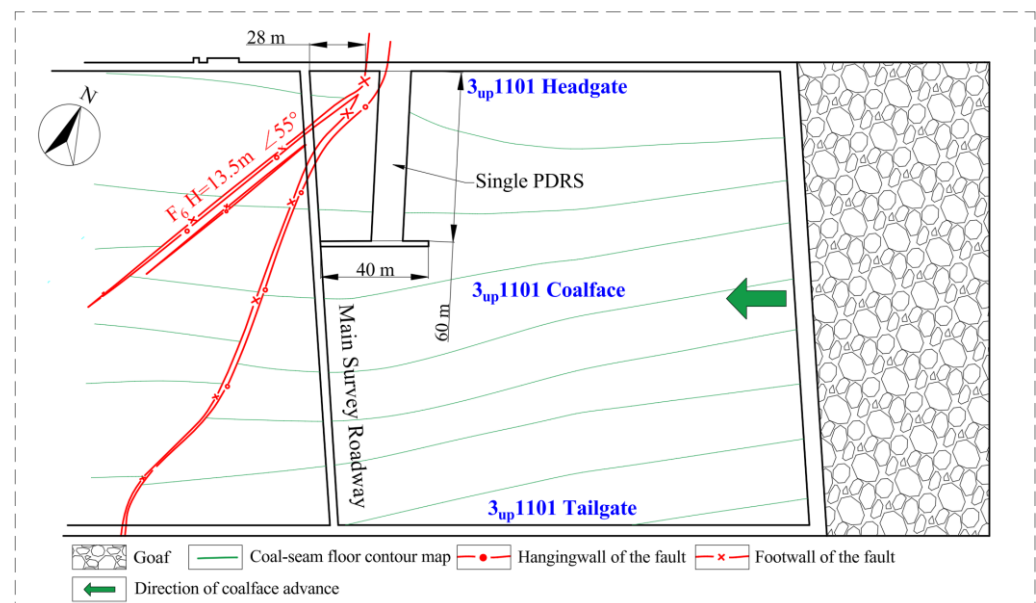


Figure 5. Solution of a single pre-driven roadway (PDR) without rock pillars.

3.2. Determination of Reasonable Solution

Table 1 summarizes the advantages and disadvantages of the two solutions of the #3_{up}1101 coalface over the F₆ fault, the solution of PDRs with rock pillars and the solution of a single PDR without rock pillars.

Table 1. Advantages and disadvantages of two solutions.

Methodology	PDRs with Rock Pillars	A Single PDR Without Rock Pillars
Advantages	✓ Placed rock columns disperse ground pressure and prevent regional stress concentration. ✓ Set up rock pillars to disperse roof pressure and prevent regional stress concentration. ✓ Small cross-section tunnel supports low technical requirements, simple construction.	✓ Maximize resource recovery, improve coal quality, and reduce resource waste. ✓ Coalface can be advanced more quickly, increasing production in the short term. ✓ No rock pillars are left, reducing the risk of spontaneous combustion due to poor ventilation.
Disadvantages	✧ Retention of rock pillars may result in some coal resources not being mined. ✧ Multiple pre-dug roadways complicate ventilation, raising safety risks with poor air flow. ✧ Controlling layer levels becomes more challenging when crossing the F6 fault.	✧ Absence of rock pillars in roadways and coalface demands enhanced support. ✧ High-standard support materials and techniques are required, increasing costs. ✧ Roof slab is prone to accidents such as slab collapse, and safety risk is higher.

In summary, the solution of PDRs with rock pillars provides superior control over the stability of the top and bottom plates and effectively mitigates the influence of mining pressure. This approach is particularly suited to scenarios involving complex geological conditions and stringent safety requirements. Conversely, the solution of a single PDR without rock pillars is better suited to stable geological conditions where improving mining efficiency is a priority, although it entails higher support costs and greater technical

investment. For the #3_{up}1101 coalface, which crosses the F₆ fault—a complex geological situation—the primary focus must be on ensuring the safety and stability of both the coalface and the roadway during the advancement process. As such, solution 1 is deemed more appropriate for the #3_{up}1101 coalface crossing the F₆ fault.

4. Key Technologies for Rapidly Passing Through Large Fault

4.1. Parameters Design of the PDRs Layout

(1) Layout pattern of the PDRs

Based on the characteristics of fault F₆ revealed by the main and secondary survey roadways, to minimize the impact of fault rocks on the normal mining of the coalface, six PDRs parallel to the strike of the fault plane were designed, including PDRs #1, #2, #3, and #4, and opposite PDRs #1 and #2 (which are driven in the opposite direction to the first four PDRs). The openings of all roadways are located along secondary survey roadway. As the cutting height is 2.9 m, the fault has little influence on the mining operations where the fault throw is smaller than 3 m. Therefore, each roadway should be driven to a position where the fault throw is less than 3 m. The total length of the six PDRs is calculated at 189 m. Figure 6 shows the layout pattern of the PDRs. Figure 7 presents a sectional view of the continuous passage of the coalface through the PDRs.

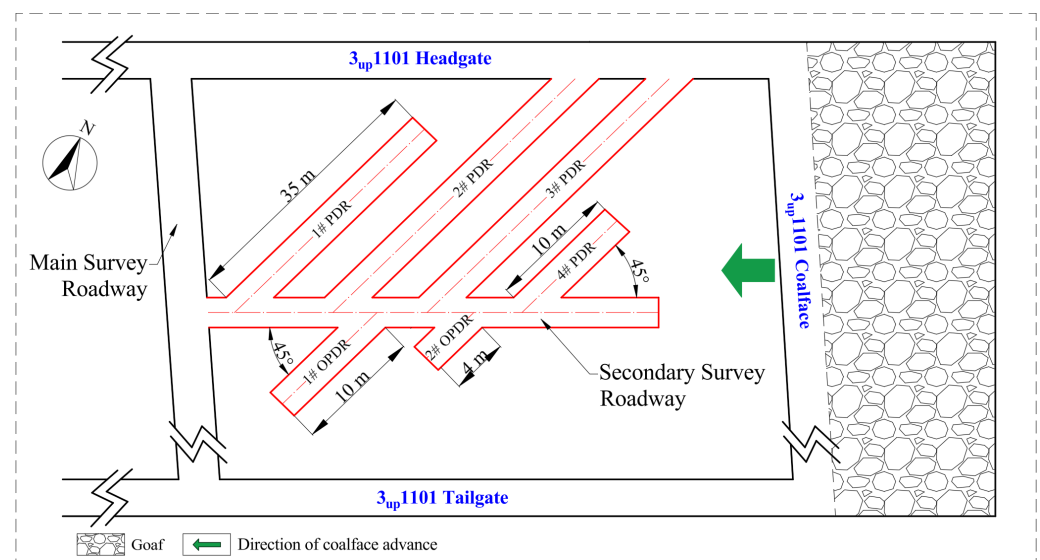


Figure 6. Layout pattern of PDRs.

(2) Slopes design of the PDRs

After careful consideration of the large throw of fault F₆ and local geological conditions, the desired position of the #3_{up}1101 coalface during passage through the fault was designed to slope at 15°. Therefore, properly designing the slopes of the PDRs is key to achieving rapid passage through the fault. If the PDRs are significantly higher or lower than the desired position, the progress of the coalface will be seriously hindered. To avoid this problem, the slopes of the PDRs were determined through elevation difference calculation. The six PDRs were designed with different slopes: PDRs #1 and #2, 14°; PDRs #3 and #4, 13°; and opposite PDRs #1 and #2, 11°.

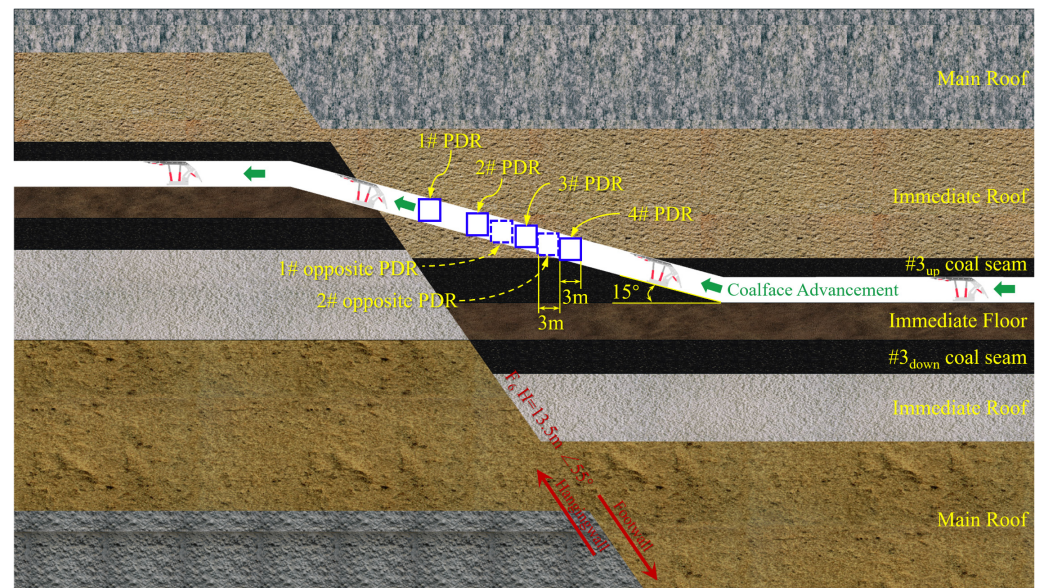


Figure 7. Sectional view of continuous passage of coalface through PDRs.

4.2. Determination of Rock Pillar Reasonable Width

As the coalface moves across the PDRs, the load from the overlying strata will be carried largely by the rock pillars between the PDRs [23,24]. A rock pillar of a proper width, under the front abutment pressure, should lie entirely within a plastic zone in order to facilitate continuous cutting by the coal cutter and reduce damage to mining equipment. Moreover, the residual bearing capacity of the rock pillars, combined with the roadway supports, is required to provide effective surrounding rock deformation control of PDRs and ensure rapid passage through the large fault.

(1) Construction of numerical models

Under natural conditions, coal rock bodies exhibit discontinuous structural characteristics due to the presence of weak planes such as bedding and joint surfaces [25]. Compared to commonly used methods like the Finite Element Method (FEM) and the Finite Difference Method (FDM), the Discrete Element Method (DEM) offers significant advantages in addressing geometric and material nonlinearity [26–28]. The numerical computation program UDEC (Universal Distinct Element Code), based on the DEM for discontinuities, is primarily used for simulating the mechanical behavior of discontinuous media under static and dynamic loads [29]. It accurately reproduces the dynamic processes of rock deformation and failure during tunnel excavation and effectively assesses the stability of surrounding rock structures under various operating conditions [30].

In this study, based on the specific mining geological conditions of the #3_{up} coalface, numerical simulation was performed using UDEC software to analyze the distribution of plastic zones in rock pillars and roof-to-floor convergence in the PDRs at different rock pillar widths (2.0 m, 3.0 m, and 4.0 m). A 2D planar computational model was built based on the mining geological conditions of the #3_{up} coalface, as shown in Figure 8. The model had a size of 300 m × 200 m (length × height). The side pressure coefficient was 0.4, the left and right boundaries were horizontally fixed, and the lower boundary was vertically fixed. The weights of overlying strata parts were treated as equivalent stress acting on the upper boundary. The coal and rock bodies utilized the Mohr–Coulomb model, and the contact surfaces employed the Mohr–Coulomb sliding contact model. The main physical and mechanical parameters of rocks mass and structural plane are listed in Tables 2 and 3.

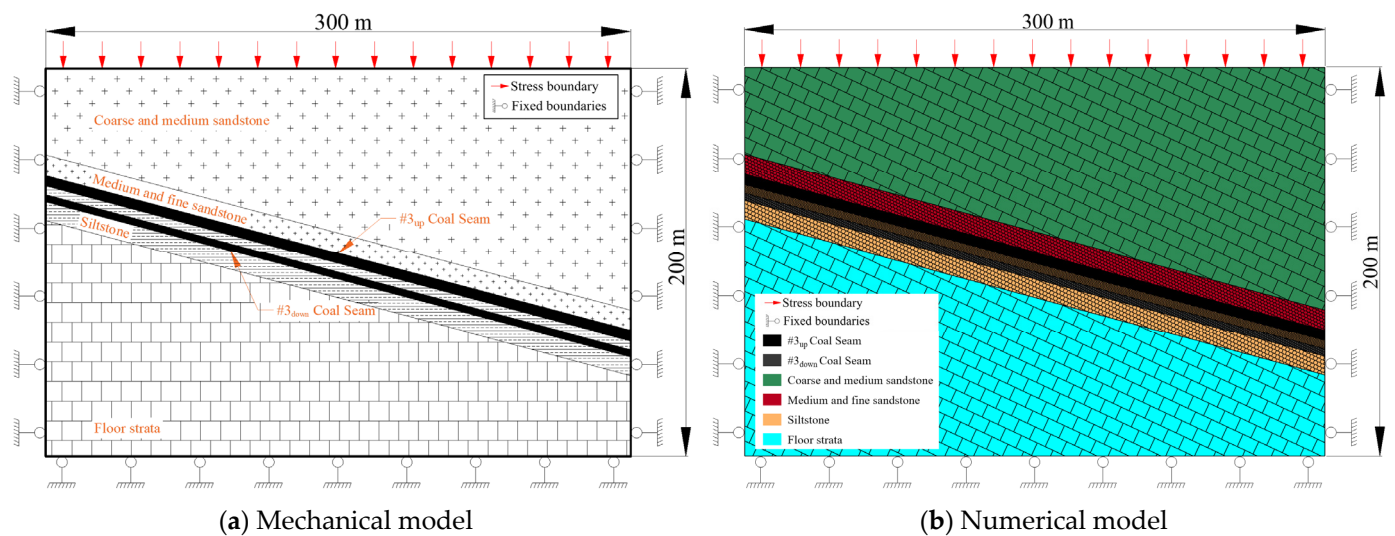


Figure 8. A 2D planar computational model.

Table 2. Main physical and mechanical parameters of rocks.

Lithology	Densities /kg·m ⁻³	Tensile Strength /MPa	Elastic Modulus /GPa	Internal Friction Angle /°	Compressive Strength /MPa
Coarse and medium sandstone	2580	1.78	2.78	31.4°	25.99
Medium and fine sandstone	2540	2.93	3.14	34.5°	28.51
Mudstone	2650	1.62	2.16	29.8°	20.16
#3 _{up} Coal	1420	0.99	1.82	36.4°	12.79
Siltstone	2460	2.45	3.13	28.7°	33.22

Table 3. Main physical and mechanical parameters of structural plane.

Lithology	Normal Stiffness /GPa	Tangential Stiffness/GPa	Coherence /MPa	Internal Friction Angle /°	Tensile Strength/MPa
Coarse and medium sandstone	1.8	1.3	0	23°	0
Medium and fine sandstone	1.6	1.1	0	25°	0
Mudstone	1.2	1.0	0	21°	0
#3 _{up} Coal	1.0	0.7	0	18°	0
Siltstone	1.1	0.9	0	20°	0

(2) Analysis of simulation results

Figure 9 shows the distribution of plastic zones in the rock pillars with 2.0 m to 4.0 m width. Figure 10 illustrates the roof-to-floor convergence in the PDRs.

According to Figures 9 and 10, the extent of plastic zones in the rock pillars varied with rock pillar width, and the rock pillar width also affected the surrounding rock deformation. When the rock pillars were 2.0 m wide, plastic deformation occurred throughout the rock pillars, the slope damage of the rock column was mainly dominated by shear damage, and there was tension damage in the right position. When the rock pillars were 3.0 m wide, there was a certain amount of deformation on both sides of the roadway, plastic zones made up about 95% of the rock pillars, and the damage to the rock column was mainly dominated

by shear damage. When the rock pillars width was 4.0 m, elastic zones were observed around the center lines of the pillars and plastic zones accounted for only about 70% of the rock pillars. When the rock pillars were 2.0 m wide, the roof-to-floor convergence in the PDRs was 450 mm, compared to 260 mm for the rock pillars with a width of 3.0 m and 240 mm for the rock pillars with a width of 4.0 m.

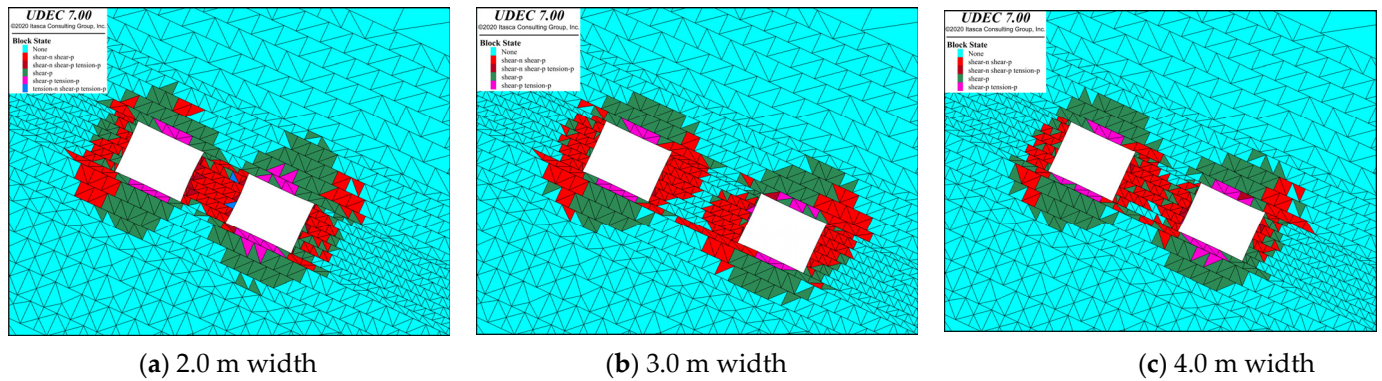


Figure 9. Distribution of plastic zones in rock pillars.

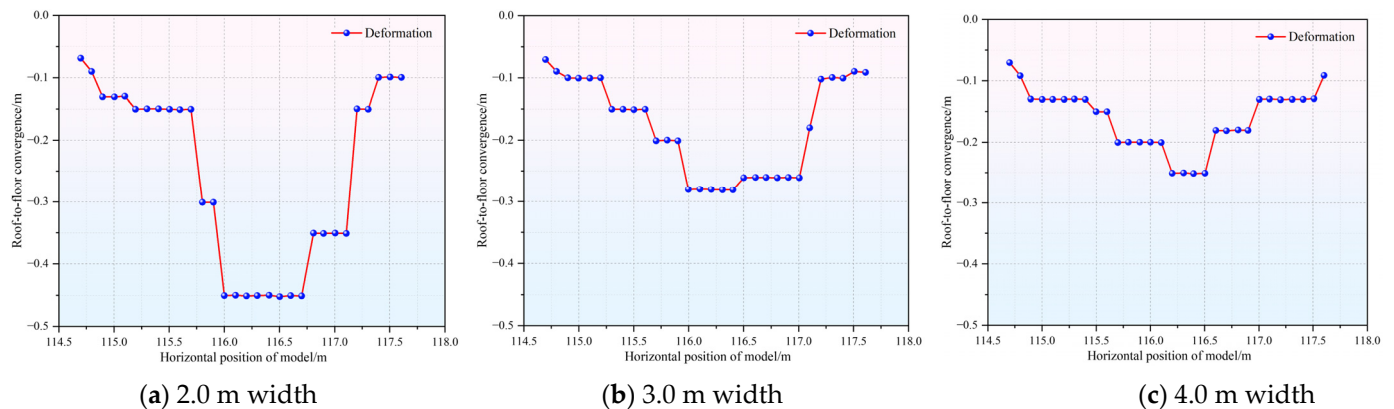


Figure 10. Roof-to-floor convergence in PDRs.

After taking into account the results above, it can be found that when the setting width of the rock pillar is greater than 3 m, the control effect on the roof-to-floor convergence in the PDRs is not significant. Therefore, due to the cost of subsequent construction and support, and the real demand for rapid passage through the fault, the proper rock pillars width was set at 3.0 m.

4.3. Construction of PDRs by SWB

SWB is a controlled millisecond blasting technique to create a smooth contour. It requires correct blasting parameters and a proper construction method [31,32]. In order to ensure the PDRs had the desired contours and could prevent severe damage to the surrounding rocks, SWB was adopted to construct the PDRs.

(1) Design of blasting parameters

In order to facilitate the parameters design of SWB, the actual driving width of the roadway was set at 3200 mm, and the height remains unchanged. Then, the blasting parameters were preliminarily determined based on local technical and geological conditions, the cross-sectional dimensions of the PDRs mentioned above, and relevant design requirements. The cutting holes are inclined. A charge structure with a primary explosive being near the blast-hole mouth and stemming length (over 500 mm) was designed for each blast hole. A

class 2 emulsion explosive permitted for use in coal mines was selected for SWB. In each blasting process, the explosive charges in all blast-holes should be connected in series and then detonated by one initiation. A delay electric detonator was used and the powerful MFD-100 initiator was chosen as the trigger for the detonating cord attached to it. The blast-hole layout is shown in Figure 11. The blast-hole parameters are provided in Table 4.

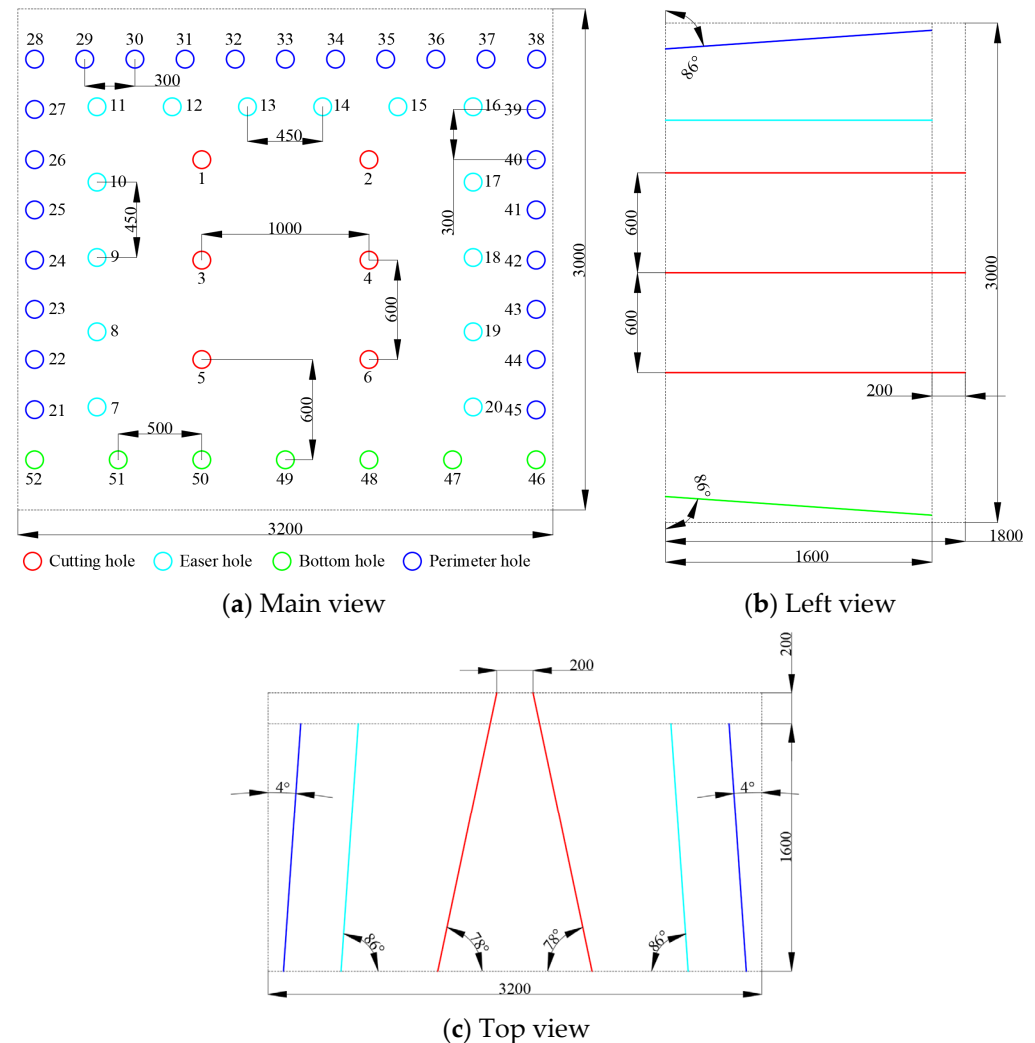


Figure 11. Blast-hole layout before optimization (units: mm).

Table 4. Blast-hole parameters before optimization.

Name	Number	Depth /m	Spacing /m	Resistance /m	Charge		Angel			
					Charge /kg	Total /kg	Level		Vertical	
							L /°	R /°	Elevation /°	Depression /°
Cutting hole	1–6	1.8	1.00	0.60	0.64	3.84	78	78	0	0
Easer hole	7–20	1.6	0.45	0.45	0.32	4.48	0	0	0	0
Perimeter hole	21–45	1.6	0.30	0.45	0.16	4.00	86	86	86	/
Bottom hole	46–52	1.6	0.50	0.60	0.48	3.36	86	86	/	86
Total	/	/	/	/	/	15.68	/	/	/	/

(2) Optimization of blasting parameters

The blast-hole parameters were optimized based on the feedback from the construction site. The results of optimization are shown in Figure 12 and Table 5. The depth of the cutting hole, originally set at 1.8 m, was increased to 2.2 m. The depth of the easer and perimeter holes was increased from 1.6 m to 2.0 m, and the explosive load was increased proportionally. A line of slicing holes (hole spacing: 150 mm) parallel to the perimeter holes were added to contour of each roadway in order to connect the blast holes into a cut plane. During the blasting operations in the field, the slicing holes were not loaded with charges because the rock pillars between them can be broken by the blast energy generated by the explosion of explosives in the perimeter holes. This can leave eyelike marks on the blasted surfaces. SWB was implemented at the PDRs based on the optimized blast-hole layout and blasting parameters. The results reveal that the slicing holes helped achieve desired roadway contours and the deepening of the holes increased the drilage advance during a cycle. Despite the increase in the number of holes drilled, the duration of drilling and the amount of work required in the next cycle were diminished and the overall drilling efficiency was improved after the optimization. The construction cost per meter of roadway was reduced by approximately 8.89%.

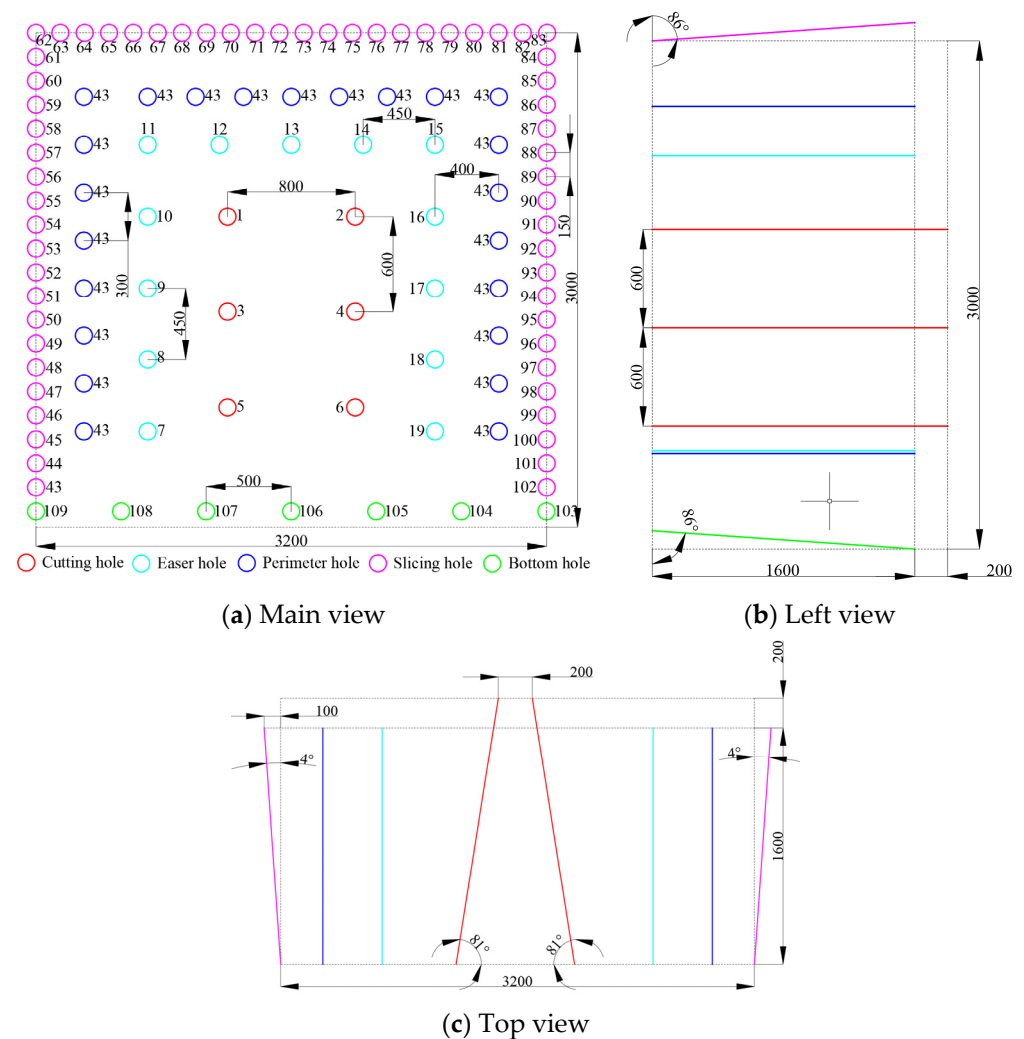


Figure 12. Blast-hole layout after optimization (units: mm).

Table 5. Blast-hole parameters after optimization.

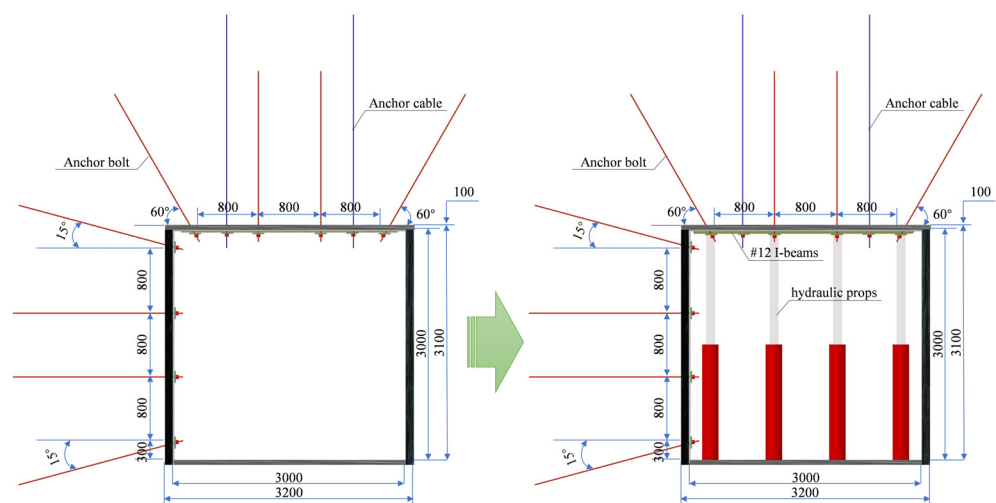
Name	Number	Depth /m	Spacing /m	Resistance /m	Charge		Angel			
					Charge /kg	Total /kg	Level		Vertical	
							L /°	R /°	Elevation /°	Depression /°
Cutting hole	1–6	2.2	0.8	0.60	0.78	4.68	81	81	0	0
Easer hole	7–19	2.0	0.45	0.45	0.40	5.20	0	0	0	0
Perimeter hole	20–42	2.0	0.30	0.40	0.20	4.60	0	0	0	0
Slicing hole	43–102	1.6	0.15	0.35	/	/	86	86	86	/
Bottom hole	103–109	1.6	0.50	0.60	0.48	3.36	86	86	/	86
Total	/	/	/	/	/	17.84	/	/	/	/

4.4. Control of Surrounding Rock Deformation in the PDRs

Controlling surrounding rock deformation in the PDRs is an important task following completion of the roadways. Effective control of surrounding rock deformation is the key to successful rapid passage through large fault F_6 . In order to ensure the coalface can move across the PDRs smoothly, the roadway support must be strong enough to enable the PDRs to maintain cross-sectional areas required by the coalface. Based on the geological conditions of the #3_{up}1101 coalface, PDR construction must be completed before the distance from the coalface to the PDRs reaches 200 m, and the PDRs should be provided with secondary reinforced support before the front abutment pressure can exert an influence on them. Therefore, according to the section size of the PDRs and the size of the rock columns between the roadways, combined with the Gaozhuang Coal Mine's anchor rod (rope) construction equipment and on-site experience, the initial support and reinforced support program for the PDRs was determined.

(1) PDRs support parameters

According to the cross-sectional dimensions of the PDRs and the size of the rock pillar left between the roadways, it was determined that the initial support for the PDRs should adopt the “anchor–net–ladder” support method. On this basis, the secondary reinforcement support method using “anchor cable beam + individual hydraulic prop” was applied. The overall supporting section of the PDRs is shown in Figure 13, and the specific supporting parameters are as follows:

**Figure 13.** Sectional view of entire support system for PDRs (units: mm).

Roof support: An array of high-strength threaded bolts ($\Phi 20 \text{ mm} \times 2200 \text{ mm}$) was installed on the roof of each PDR; both the spacing between bolts in the same row and the inter-row spacing were 800 mm. The threaded bolts had extended anchorage length (over 1337 mm). Later, two rolls of resin-based anchoring agent were applied to each bolt, with the CK2335 agent applied prior to the K2350 agent. The tray attached to each bolt was a square plate with dimensions of $150 \text{ mm} \times 150 \text{ mm} \times 8 \text{ mm}$ (length $\times$ width $\times$ thickness). The steel ladder was constructed of rebar with 16 mm diameter and measured 3000 mm by 90 mm. Each rhombic wire mesh was woven from #8 iron wires, and had dimensions of $3600 \text{ mm} \times 900 \text{ mm}$ (length $\times$ width).

Side support: An array of high-strength threaded bolts ($\Phi 18 \text{ mm} \times 1800 \text{ mm}$) was inserted into each roadway side, with the spacing between bolts in the same row and the inter-row spacing being 800 mm. Threaded bolts with extended anchorage length (over 1337 mm) were employed. Two rolls of resin-based anchoring agent were applied to each bolt, with the CK2335 agent applied prior to the K2350 agent. The tray attached to each bolt was a square plate with dimensions of $150 \text{ mm} \times 150 \text{ mm} \times 8 \text{ mm}$ (length $\times$ width $\times$ thickness). Each rhombic wire mesh was woven from #8 iron wires, and had dimensions of $2300 \text{ mm} \times 900 \text{ mm}$ (length $\times$ width).

Reinforced support: Anchor beams with a length of 2800 mm were laid transversely onto the roof of each PDR at intervals of 1600 mm. Each anchor beam was formed by welding two #12 I-beams, and then equipped with two anchor cables ($\Phi 15.24 \text{ mm} \times 6000 \text{ mm}$), which were 2400 mm apart. Later, DZ38(42)-25/100 hydraulic props were used to reinforce the supports, with four props in each row. The spacing between props in the same row and the inter-row spacing were 600 mm and 800 mm, respectively.

(2) Measurements of surrounding rock deformation

In order to verify if surrounding rock deformation was effectively controlled, a deformation observation station was set up in #4 PDR (the one closest to the coalface) to measure rocks deformation through the cross method. The measurement began when the coalface was 50 m from #4 PDR and the results are presented in Figure 14. As shown in Figure 14, the amount of surrounding rock deformation tended to increase as the coalface approached this PDR. When the coalface reached a position about 27 m from this PDR, the amount of deformation started to increase significantly. As their distance reached around 14 m, a sharp increase in the surrounding rock deformation began. During the advance of the coalface, the maximum roof-to-floor convergence measured was 251 mm, the maximum relative displacement between the two sides was 270 mm, and the average cross-sectional area of the roadway was approximately 70.25% of the original designed cross-sectional area. The field measurements demonstrate that the cross-sectional areas of the PDRs can allow the coalface to pass quickly through the large fault.

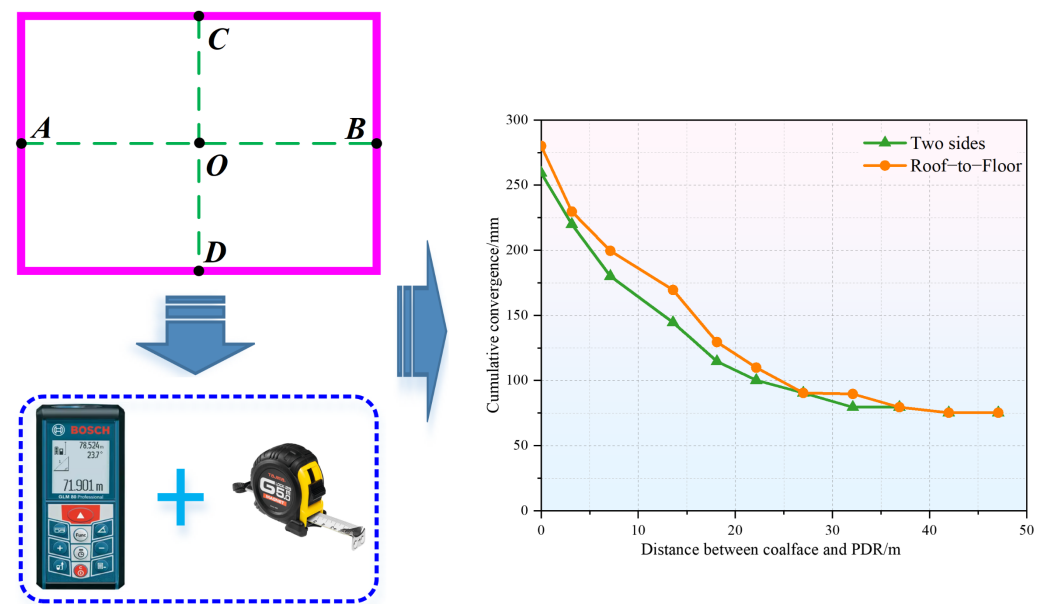


Figure 14. Field measurement results of surrounding rock deformation in #4 PDR.

5. Discussion on Engineering Application Effect

5.1. Practical Effect

The method of PDRs with rock pillars was implemented at GCM to help the #3_{up}1101 coalface to pass through a fault with a maximum throw of 13.5 m. It took only 23 days for this coalface to pass through this fault, thus achieving the aim of rapid passage. Compared to traditional methods to deal with faults (e.g., taking a detour proactively), the method proposed in this study avoids relocation of the coalface, the accompanying production suspension, and tremendous work to construct auxiliary roadways, thereby significantly decreasing the time required to pass through the fault (by about 1 month). It follows that this method can ensure continuous, efficient, and safe mining at this coalface, effectively alleviating the tight situation of mine production succession.

5.2. Comprehensive Benefits

(1) Safety benefits

During the rapid crossing of a major fault locally, most of the gangue in the fault-affected area was excavated in advance using the method of PDRs. In addition, the rock pillars between the PDRs were found to have deformed plastically in their entirety under the front abutment pressure, allowing the shearer to directly cut the narrow rock pillars. This greatly mitigated the wear and tear on the mining equipment (the cost incurred by wear and tear was approximately CNY 0.82 million) and improve coalface safety. Targeted initial support and reinforced support schemes were proposed for the PDRs, ensuring that the deformation of the surrounding rock of the roadway was within a controllable range and reducing the risk of accidents during the rapid crossing of the fault at the coalface and subsequent engineering operations.

(2) Economic benefits

After implementation of the aforementioned method, the coalface was able to maintain stable coal production (over 3000 t per day) during passage through the fault. A total of 8.5×10^4 t of coal was extracted over this period, creating direct economic returns of about CNY 34 million. As the majority of the gangue in the fault-affected region had been dug out in advance, the coal produced from the coalface during its passage through the fault showed significantly improved quality, which created an economic benefit increase of CNY

41 million. The cost of the entire PDRs was low, at about CNY 4.72 million. This method was thus estimated to create CNY 71.1 million of economic benefits in total.

5.3. Limitations and Suggestions

(1) Limitations of the study

The study determined a pillar width of 3.0 m as protective for the surrounding rock using UDEC simulation software, and this was proven to be effective. In practical engineering applications, the pillar width is influenced by factors such as geological conditions and rock mass quality. Particularly in cases of uneven coal seams or complex fault zones, a fixed width may not ensure stability, necessitating further on-site experiments and data adjustments.

Although the method achieved favorable results in the case study, its applicability is still constrained by factors such as coal seam geological characteristics, fault size, and mining equipment. Therefore, similar results may not be achieved under comparable mining conditions or face characteristics, and the method should be adjusted based on specific site conditions.

(2) Suggestions for future study

More extensive validation and optimization of the adaptability of the method in different mining areas and under different geological conditions is recommended, especially in engineering contexts with large fault falls and different lithological combinations, to collect more comprehensive data for the improvement and promotion of the method.

It is also recommended to continue to optimize the design methodology for rock pillar width under varying geological conditions. Based on numerical simulation and field test data, researches should also integrate machine learning and other advanced techniques to investigate the influence of additional variables, such as coal seam thickness, seam dip, fault displacement, and fault extension range, on the structural stability of rock pillars. This approach aims to further enhance the rationality and reliability of rock pillar design.

6. Conclusions

In this study, a new method for rapidly crossing large faults, specifically the “PDRs with Rock Pillars”, is proposed. On this basis, the study examines the specific implementation steps of key technologies for rapidly crossing large faults and evaluates the practical engineering application and overall benefits of the method. Based on the research findings, the following conclusions can be drawn:

- (1) Based on the mining geological conditions, two solutions for the #3_{up}1101 coalface at GCM to pass through the fault F_6 were proposed. A comparison of the two solutions suggests that the PDRs with rock pillars has both technical and economic advantages over the solution based on a single PDR without rock pillars.
- (2) Based on the characteristics of fault F_6 revealed, six PDRs were parallel designed in proper positions, and the distribution of plastic zones in rock pillars and roof-to-floor convergence in the PDRs at were determined at different rock pillar widths (2.0 m, 3.0 m, and 4.0 m). The results reveal that the appropriate rock pillar width is 3.0 m.
- (3) In order to ensure PDRs with desired contours can be achieved, SWB was employed to construct the PDRs, which also improved the excavation efficiency and lowered the excavation cost. On this basis, combined with the corresponding initial support and strengthened support, the surrounding rock deformation was effectively controlled in a PDR. This demonstrates that the support parameters determined can allow for rapid passage through the large fault.

- (4) The method of PDRs with rock pillars was implemented at the #3_{up}1101 coalface, GCM. The time taken to pass through the fault was significantly shortened compared to that required by traditional methods. Other benefits created by this method include stable coal production, lesser gangue rate, reduced wear and tear on the mining equipment, and considerable overall economic benefits amounting to CNY 71.1 million. In this way, it has improved the overall safety factor of the coalface, and achieved continuous, safe, and efficient mining.

Author Contributions: Conceptualization, W.Z., F.Y., and J.C.; methodology, W.Z., F.Y., and S.Z.; software, F.Y. and S.Z.; validation, W.Z. and B.X.; formal analysis, W.Z. and J.X.; investigation, F.Y. and J.X.; resources, W.Z., F.Y., S.Z., and B.X.; data curation, F.Y. and J.C.; writing—original draft preparation, W.Z., F.Y., and S.Z.; writing—review and editing, W.Z., J.C., and B.X.; visualization, J.C. and J.X.; supervision, W.Z. and F.Y.; project administration, W.Z., F.Y., and J.C.; funding acquisition, W.Z. and F.Y. All authors have read and agreed to the published version of the manuscript.

Funding: This work was financially supported by the National Natural Science Foundation of China (No. 52374141), the National Key Research and Development Program of China (No. 2024YFC3013904), the Graduate Research and Practice Innovation Program of Jiangsu Province (No. KYCX24_2882), and the Graduate Innovation Program of the China University of Mining and Technology (No. 2024WLJCRCZL021).

Institutional Review Board Statement: Not applicable.

Informed Consent Statement: Not applicable.

Data Availability Statement: The original contributions presented in the study are included in the article; further inquiries can be directed to the corresponding author.

Acknowledgments: We wish to thank the GCM for their support in conducting this important study. The authors are also grateful to the anonymous reviewers for their constructive comments and helpful suggestions.

Conflicts of Interest: The authors declare no conflicts of interest.

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