Study on the Coal Pillar Weakening Technology in Close Distance Multi-Coal Seam Goaf

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Abstract: The pressure relief of coal pillars in close-distance multi-coal seam goaf is a complex engineering problem with the characteristics of “dynamic mine pressure”. Hence, this paper studies such problems. First, the influence factors of the coal pillar in the goaf on the mine pressure of the mining face of the lower coal seam under this condition were theoretically analyzed, and it was concluded that vertical stress is the most important element, followed by horizontal stress. Next, a physical similarity simulation experiment was designed to study the stress distribution law of the coal pillar floor in the goaf before and after pressure release and the damage depth. Finally, a technology and monitoring method for coal pillar blasting pressure alleviation in goaf were introduced and implemented in engineering practice. After the pressure is alleviated, the surrounding rock stress of the lower coal seam mining face is redistributed, and the vertical stress is decreased by 20%. The adjacent rock’s deformation is improved. This technology’s cost and safety advantages are extraordinary and helpful for mining coal seams over close distances.

Keywords: close-distance multi-coal seam; coal pillar in the goaf; physical similarity simulation; deep hole blasting; borehole stress monitoring

1. Introduction

China is the number one coal production and consumption country all over the world. Coal represents around 70% of China’s main energy consumption structure [1]. The consumption structure with coal as the dominant energy source is not expected to alter dramatically during the next 10 years. The occurrence circumstances of coal seams have gotten increasingly complicated in recent years as mining intensity and depth have increased, and the mining environment has deteriorated. For close-distance coal seam mining, the lower coal seam is impacted by the mining stress and tectonic stress of the coal seam. Additionally, it bears the impact of the residual coal pillar stress transfer in the higher coal seam, resulting in a more severe occurrence of intense rock pressure in the coal seam’s working face. The strata behavior of the coal seam working face is more severe. Over the years, experts from both China and overseas have undertaken extensive studies on coal pillar pressure relief technology to overcome the pressure relief issue of residual coal pillars in the overlaying goaf. Numerous outcomes have been attained by using high-pressure water injection, deep-hole blasting, drilling pressure relief, and other physical pressure relief techniques. Among them, the deep hole blasting pressure relief within the coal pillar is an active destress measure that may decrease the coal body’s stress concentration and strength. Bi Huijie et al. [2] regulated the deformation of the rock around the roadway via deep hole blasting, causing the high-stress section of the roof to become the deepest portion. Pang Lining et al. [3] researched hard coal body hydraulic fracturing coupled with deep hole blasting weakening technology to address the issue of hard coal body and low lump coal rate in the Jurassic coal seam in northern Shaanxi. Yang Jingxuan et al. [4] conducted...
a study on the development and technology of liquid explosives for deep hole blasting problems in coal rock bodies and achieved the effect of the directed blast with a rigid ceiling and great spacing in deep holes. Kong Lingqiang modified the technological parameters to enhance the roof’s blasting impact [5]. Liu proposed a three-zone pre-splitting method for thick and hard roof without coal pillar mining based on the principle of rock mechanics to solve the problem of a large overhanging span of medium-thick and hard roof [6]. Wang Fangtian studied the effect of deep hole pre-cracking blasting of coal pillars by theoretical calculation and numerical simulation and gave the effect of high-energy blast stress wave on rock stress field and rupture range [7]. Some specialists have also investigated liquid CO₂ fracturing technology for forced roof caving or roof pressure alleviation of the working face [8–10]. This method has high safety but an insufficient blasting effect compared with explosive blasting. Currently, it is mostly used for pre-splitting and pressure relief of a coal seam or gas permeability enhancement. In this paper, the coal pillar in the mining void area of Ganzhuang mine in Shanxi Province is taken as the research object. Theoretical analysis, similar simulation, and field stress measurement analysis are adopted to study the stress analysis law of the coal pillar bottom plate in the coal mining void area overlying the close coal seam. The coal pillar pre-blast treatment plan is proposed to alleviate the influence of mine pressure in the surrounding rock of the lower coal seam roadway. It is of reference significance to similar working face mining.

2. Project Overview

Ganzhuang Coal Mine is one of the principal mines of Shanxi Datong Coal Mine Group. The mining technology of multi-seam downstream mining is implemented in accordance with the mine’s geological circumstances and after careful analysis of the economic benefits, along with the coal mining strategy of moving forward and backward while using the collapse method to manage the goaf. The surface of the mine area is covered by the fourth series loess, and loess gullies are developed. There are no large surface water bodies or rivers in the area. The coal seams in the well field are gently produced, the faults are not developed, there are no trap columns, and the tectonic type is simple. All coal seams have an inclination angle of 1 to 2 degrees and are almost horizontal. The coal seams n° 2, n° 3, and n° 7 have already been mined, while the coal seams n° 8, n° 11, and n° 14 are prepared for mining. Some part of the n° 8 coal seam has been excavated, and the unmined part is located above the n° 11 coal seam gob, which is about to be mined. The n° 11 coal seam is located in the lower part of the Datong Formation, with a buried depth of 378 m and an average thickness of 3.51 m. The coal seam is spaced 31 m from the gob area of the n° 8 coal seam above. The average thickness of the n° 14 coal seam is 1.53 m, and the interval between the coal seam and the n° 11 coal seam above is 8.8 m. Among them, the coal pillar of the n° 8 coal seam is 20 m wide, and the horizontal distance from the lower n° 11 coal seam is 19 m. The width of the n° 11 coal seam is 25 m, and the horizontal distance from the lower n° 14 coal seam 14,203 working face is 7 m. The histogram of the coal seam is shown in Figure 1, and the spatial position of the coal rock mass is shown in Figure 2. After the n° 11 coal seam was mined, many coal pillars were left in the gob. Therefore, when the mining roadway is arranged in the n° 14 coal seam, the mining roadway is affected by the coal pillar of the upper n° 11 coal seam, and the surrounding rock is seriously deformed, which is not conducive to the safe and efficient production of the mine.
3. Theoretical Analysis

The coal pillars have been a shielding method for controlling mine pressure in coal mining. The coal pillar can improve the stability of the surrounding rock of the underground roadway and ensure the smooth operation of the underground ventilation, transportation, drainage, and other systems [11–13]. The residual coal pillar transmits high stress to the surrounding rock and coal seam below, but at the same time diffuses and decays according to certain laws. At the mining depth of about 400 m, the influence of high stress of the coal pillar has been greatly weakened at the depth of 60 m from the coal seam. The lower coal seam mining working face location can be designed according to the residual coal column under the high stress transmission diffusion attenuation law. However, due to the layout of the well field and the restriction of the three machines supporting the coal mining working face, the residual coal column and the working face of the coal seam below must have a certain staggered distance. However, when the working face is back mined, the coal pillar support pressure propagates to the bottom plate, affecting the mining of adjacent coal seams as well as the stability of the underlying coal seam roadway. As a result, the magnitude of the vertical stress, horizontal stress, and shear stress, as well as the influence
range, can be judged by mastering the distribution law of the stress below caused by the residual coal column. The residual coal pillar and bottom plate are simplified as elastomers based on elastic mechanics theory. The force calculation model of the residual coal pillar can be established [14,15], as shown in Figure 3.

\[
\begin{align*}
\text{Figure 3. Force model of bottom plate under residual coal column pressure.}
\end{align*}
\]

The stress induced by the microset load \( dF = \lambda q d\xi \) at any point \( M \) in the half-plane body is Equation (1), and integration over the range \(-b\) to \(a\) yields Equation (2), where \( \xi \) is the scale on the \( y \)-axis and \( \lambda \) is the stress concentration factor.

\[
\begin{align*}
\sigma_x &= \frac{2\lambda q d\xi}{\pi} \left[ \frac{x^3}{x^2 + (y-\xi)^2} \right]^2 \\
\sigma_y &= \frac{2\lambda q d\xi}{\pi} \left[ \frac{x(y-\xi)^2}{x^2 + (y-\xi)^2} \right]^2 \\
\tau_{xy} &= \frac{2\lambda q d\xi}{\pi} \left[ \frac{x^2(y-\xi)^2}{x^2 + (y-\xi)^2} \right]^2 \\
\end{align*}
\]

\( \int_{-b}^{a} \)

For the uniform load \( \lambda q \) applied above the residual coal pillar, the integral of Equation (2) can be obtained:

\[
\begin{align*}
\sigma_x &= -\frac{\lambda q}{\pi} \left[ \frac{x(a-y)}{x^2 + (a-y)^2} + \frac{x(b+y)}{x^2 + (b+y)^2} + \arctan \frac{a-y}{x} + \arctan \frac{b+y}{x} \right] \\
\sigma_y &= -\frac{\lambda q}{\pi} \left[ -\frac{x(a-y)}{x^2 + (a-y)^2} - \frac{x(b+y)}{x^2 + (b+y)^2} + \arctan \frac{a-y}{x} + \arctan \frac{b+y}{x} \right] \\
\tau_{xy} &= \frac{\lambda q}{\pi} \left[ \frac{x^2}{x^2 + (a-y)^2} - \frac{x^2}{x^2 + (b+y)^2} \right] \\
\end{align*}
\]

The stress situation at the bottom plate of the residual coal column can be analyzed by Equation (3). Because \( \lambda q \) is a constant, the interrelationship between horizontal stress, vertical stress, and shear stress and stress concentration coefficient of the residual coal column at different depths under uniform load is obtained using GeoGebra data processing software, as shown in Figure 4 below.
column at different depths under uniform load is obtained using GeoGebra data processing software, as shown in Figure 4 below.

Figure 4. Stress distribution at the bottom of the coal seam. (a) Vertical stress distribution; (b) Horizontal stress distribution; (c) Shear stress distribution.
From Figure 4 above, the stress distribution pattern of the bottom plate of the residual coal pillar can be seen: (1) The vertical stress of the bottom plate of the residual coal pillar is distributed as a “single arch”, symmetrically distributed by the central axis of the coal pillar, and the peak stress appears in the central axis of the coal pillar. The vertical stress in the bottom plate of the residual coal pillar decreases with the increase of depth, but the influence range of the stress continues to expand, and the vertical stress concentrates the influence depth up to about 60 m. In the horizontal direction, the vertical stress decreases continuously with the increase of distance from the center axis of the coal pillar. (2) The horizontal stress of the bottom plate of the residual coal pillar is also symmetrically distributed with the central axis of the coal pillar, but in the vertical direction, the stress curve changes from single-peaked to double-peaked, and the trend of stress changes tends to moderate, and the stress influence range also expands. In the horizontal direction, the trend of horizontal stress is the same as that in the vertical direction, and the stress decreases as the distance from the center axis of the coal pillar increases. (3) The shear stress of the bottom plate of the residual coal pillar becomes a central symmetric “single peak”, and the shear stress value at the central axis of the coal pillar is 0. The peak stress is located at the edge of the coal pillar. The shear stress tends to increase with the distance from the central axis of the coal pillar and then decreases. According to the relationship between vertical stress, horizontal stress, shear stress, and stress concentration, the depth of influence of vertical stress in the residual coal pillar is about 60 m, the depth of influence of horizontal stress is about 10 m, and the depth of influence of shear stress is smaller. That is, the influence range of vertical stress of the residual coal column is greater than that of horizontal stress and shear stress. It can be concluded that the vertical of the bottom plate of the residual coal pillar stress plays a dominant role in the stress transfer process of the bottom plate of the residual coal pillar.

4. Experimental Setup

4.1. The Physical and Mechanical Parameters of Rock Samples

Coal and rock blocks were collected from the mine engineering site and processed into standard rock specimens in the laboratory to prepare for a rock mechanics test. A TGM-U204 wave parameter tester was used to detect the integrity of rock specimens and eliminate specimens containing joints and cracks. The rock mechanics test allows for the simultaneous determination of the physical and mechanical characteristics of the coal and rock on the roof and floor of the roadway as well as the evolution mechanism of the coal and rock mass structure in mine mining [16,17]. These initiatives can be used as a research basis for creating mine roadway supports, classifying the stability of the surrounding rock, coordinating hydraulic support designs for working faces, and more [18–20]. The n° 14 coal seam core was drilled vertically upward from the center of the road to the top coal in the unmined area of the n° 11 coal seam to collect a complete rock sample. Similar to this, vertical cores were taken from the middle of the n° 11 coal seam’s roadway to the top coal in the unmined section of the n° 8 coal seam. The rock samples were gathered to provide parameters for later physical similarity simulation and the construction of coal pillar blasting holes. The rock samples and the experimental procedure are depicted in Figure 5, and the mechanics parameters of the rock cores collected at the two locations were analyzed in a lab. Table 1 displays the primary measures of the coal rock mass’s compressive strength, tensile strength, elastic modulus, and Poisson’s ratio.

| Rock Name     | Bulk Modulus (GPa) | Tensile Strength (MPa) | Shear Modulus (GPa) | Density (kg/m³) | Angle of Internal Friction (°) | Cohesive Force (MPa) |
|---------------|--------------------|------------------------|--------------------|----------------|-----------------------------|---------------------|
| Coal          | 1.0                | 0.6                    | 0.46               | 1430           | 28                          | 1.2                 |
| Fine Sandstone| 2.7                | 1.0                    | 1.6                | 2500           | 35                          | 2.0                 |
| Siltstone     | 2.8                | 2.5                    | 3.8                | 2650           | 35                          | 6.0                 |
| Sandstone     | 5.1                | 1.17                   | 1.3                | 2490           | 31                          | 6.2                 |

Table 1. Physical and mechanical parameters of rock mass.
The similarity theory, the geological circumstances of the mined coal seams, and the physical and mechanical characteristics of each coal seam serve as the foundation for the physical similarity simulation. The structure of the surrounding rock under the influence of actual mining operations, the law of rock pressure distribution, and even specific changes to the surrounding rock structure such as rock grouting, sand collapse, and gas protrusion are inferred by creating a similar model of the seam in the laboratory and analyzing the changes in mechanistic parameters and structural changes of the model [21–23]. The experimental model is shown in Figure 6. Through this similar simulation experiment, the focus is on the fracture law of the overlying strata after mining in the goaf of n° 8 and n° 11 coal seams, the structure formed by the basic roof, and the instability law developed to caving and rotation, and its control on the roof of the working face influence. When the n° 14 coal seam working face passes through the residual coal pillar in the n° 11 coal seam goaf, the blasting pressure relief process is adopted, and the stability of the coal pillar before and after mining and the failure movement law of the lower rock layer are studied. It also provides a reasonable evaluation of the stability of the roof plate of the n° 14 coal seam roadway. It provides a basis for proposing a scientific and reasonable blasting pre-cracking unloading scheme for coal pillars in the mining area to ensure the safety of the residual coal pillars weakening unloading mining during the pushing of the working face in the coal mine.

**Figure 5.** Acquisition of cores and mechanical experiments. (a) Cores drilled on site; (b) Different experimental procedures.

**4.2. Scheme of Physical Modelling**

The similarity theory, the geological circumstances of the mined coal seams, and the physical and mechanical characteristics of each coal seam serve as the foundation for the physical similarity simulation. The structure of the surrounding rock under the influence of actual mining operations, the law of rock pressure distribution, and even specific changes to the surrounding rock structure such as rock grouting, sand collapse, and gas protrusion are inferred by creating a similar model of the seam in the laboratory and analyzing the changes in mechanistic parameters and structural changes of the model [21–23]. The experimental model is shown in Figure 6. Through this similar simulation experiment, the focus is on the fracture law of the overlying strata after mining in the goaf of n° 8 and n° 11 coal seams, the structure formed by the basic roof, and the instability law developed to caving and rotation, and its control on the roof of the working face influence. When the n° 14 coal seam working face passes through the residual coal pillar in the n° 11 coal seam goaf, the blasting pressure relief process is adopted, and the stability of the coal pillar before and after mining and the failure movement law of the lower rock layer are studied. It also provides a reasonable evaluation of the stability of the roof plate of the n° 14 coal seam roadway. It provides a basis for proposing a scientific and reasonable blasting pre-cracking unloading scheme for coal pillars in the mining area to ensure the safety of the residual coal pillars weakening unloading mining during the pushing of the working face in the coal mine.

**Figure 6.** Similar simulation experimental model (similarity ratio of 1:100).
The detailed column diagram of the coal seam served as the basis for the model’s design. The Xi’an University of Science and Technology produced a 3 m long and 0.2 m wide planar stress model frame. The model’s real laying height is 1.16 m. By applying a stress-similar ratio and loading the upper portion of the model, the simulation of the remaining rock mass can be transformed into lead bricks with an identical weight load. In order to study the pressure distribution law of the coal pillar both before and after the mining of each coal seam working face, twelve CL-YB-114WX pressure sensors were put at the bottom of the pre-set residual coal pillar in the n° 8, n° 11, and n° 14 coal seams [24]. At the same time, DNS113-type pressure sensors were pre-buried at 10 cm, 20 cm, 30 cm, 40 cm, and 50 cm in the rock beneath the bottom plate of each coal pillar. The horizontal distance between each sensor was set to 6 cm in order to track and examine the rock’s stress distribution at various points beneath each coal pillar. The major similarity coefficients were chosen as follows: geometric similarity constant \( \alpha_1 \) is 100, bulk density ratio \( \alpha_2 \) is 1.56, time similarity constant \( \alpha_t = \sqrt{\alpha_1} \), Strength, modulus of elasticity, adhesion similarity constant \( \alpha_R = \alpha_E = \alpha_C = \alpha_1 \times \alpha_2 = 156 \), respectively. The model materials were created by layering and compacting river sand and loess as the aggregate, gypsum and calcium carbonate as the cementing materials, and water in a certain ratio. Mica powder was then sprinkled between the layers to imitate layering. Table 2 displays the model filling size, order, and material distribution.

| Rock Formation Number | Rock Formation Name | Rock Thickness (m) | Model Thickness (m) | Proportion (Sand: Gypsum: Calcium Carbonate) | Cumulative Height (m) |
|-----------------------|---------------------|-------------------|--------------------|---------------------------------------------|-----------------------|
| 14                    | Siltstone           | 8.53              | 17                 | 837 8.53 0.32 0.75                          | 116                   |
| 13                    | n° 8 coal seam      | 1.70              | 3                  | 2.1 0.10 0.52 2.10                          | 99                    |
| 12                    | Siltstone           | 2.71              | 6                  | 837 8.53 0.32 0.75                          | 96                    |
| 11                    | Medium-grained sandstone | 2.55          | 5                  | 837 8.53 0.32 0.75                          | 90                    |
| 10                    | Siltstone           | 6.13              | 12                 | 837 8.53 0.32 0.75                          | 85                    |
| 9                     | Fine-grained sandstone | 3.28            | 6                  | 837 8.53 0.32 0.75                          | 73                    |
| 8                     | Medium-grained sandstone | 2.30            | 5                  | 837 8.53 0.32 0.75                          | 67                    |
| 7                     | Fine-grained sandstone | 5.80            | 12                 | 837 8.53 0.32 0.75                          | 63                    |
| 6                     | Siltstone           | 7.65              | 16                 | 837 8.53 0.32 0.75                          | 51                    |
| 5                     | n° 11 coal seam     | 3.51              | 7                  | 2.1 0.1 0.52 2.1                            | 35                    |
| 4                     | Fine-grained sandstone | 5.22            | 10                 | 837 8.53 0.32 0.75                          | 28                    |
| 3                     | Siltstone           | 2.41              | 6                  | 837 8.53 0.32 0.75                          | 18                    |
| 2                     | n° 14 coal seam     | 1.60              | 3                  | 2.1 0.1 0.52 2.1                            | 13                    |
| 1                     | Siltstone           | 6.26              | 10                 | 837 8.53 0.32 0.75                          | 10                    |

5. Results of Physically Similar Simulation Experiments

The experiment is comprised of three phases. In the first phase, extracting the n° 8 coal seam and leaving a 20 m-wide coal pillar is simulated. The process of mining an n° 11 coal seam and leaving a 25 m coal pillar is simulated in the second stage, and the process of mining an n° 14 coal seam and weakening an n° 11 coal pillar is simulated in the third stage. Figure 7 displays the experiment’s final morphological outcomes. There is a specific stress similar constant ratio because the simulated experimental monitoring data and the actual data in the field are similar. This study is transformed to the actual field stress for expression in order to correlate to the field outcomes.
In Figure 8, a graph shows the stress equivalence curves measured by each stress sensor before weakening the n° 11 coal pillar. Figure 8b shows the stress equivalence curves measured by the stress sensors under the three coal pillars after the weakening of the n° 11 residual coal pillar. Firstly, the width of each coal pillar is less than 25 m, which is basically equal to the influence range of overrun support pressure at the working face, so the support stress of the coal pillar becomes a complete tortoise-back-shaped distribution, and the stress increase coefficient at the highest place is three for all. The stress value on the n° 8 residual coal pillar is 26 MPa, and the stress on the n° 11 residual coal pillar is 28 MPa. Secondly, the n° 8 coal seam was mined first, and then the overlying rock activity formed a new articulation form when the n° 11 coal seam was mined so that the stress generated by the residual coal pillar of the upper n° 8 coal seam was dispersed and played a pressure-bearing role. Therefore, during the mining process of the n° 14 coal seam, the residual coal column of the n° 8 coal seam has no stress influence on it. The n° 14 coal seam is only affected by the concentrated stress of the residual coal column of the n° 11 coal seam, and the influence range covers the comprehensive mining working face of the n° 14 coal seam and its adjacent back mining roadway below the residual coal column of the n° 11 coal seam; that is, the residual coal column of the n° 11 coal seam brings more significant dynamic pressure influence on the comprehensive mining working face and back mining roadway of the n° 14 coal seam. Thirdly, to ensure the working face’s safe production, the n° 11 coal pillar must be depressurized and weakened, thus reducing the dynamic pressure impact on the lower coal seam roadway and the working face. As shown in Figure 8b, the legacy coal pillar of the n° 11 coal seam is weakened, and the weakening location is 2 m upward from the bottom plate of the coal pillar body. After weakening, the coal pillar body stress redistributed, the coal pillar body is broken to produce longitudinal and transverse fractures, interspersed with some tiny fractures, but it still maintains a certain pressure-bearing state [25–27]. The roof rock above the coal pillar body will also be loosened by blasting, and breakage, rotation, and sinking will occur. The resulting load will be transferred downward and act on the coal pillar body with a certain support capacity. A portion of the roof rock load is consumed and transferred in pressure-bearing by the coal pillar body. As a result, the stress on the bottom plate of the n° 11 residual coal pillar is dispersed, and the load sent down to the bottom plate by the broken coal pillar body is reduced. The stress effect on the n° 14 coal recovery face and the roadway is significantly reduced as a result of the contraction of the stress nucleus and the reduction of the stress influence range, which favors the regular pushing of the working face. The recovery roadway does not need reinforcement support.
Therefore, the spacing between the bottom of the shell holes was set at 6500 mm.

The working face has been arranged. Above the n° 14 coal seam are the partially mined n° 11 coal seam and the thoroughly mined n° 8 coal seam. The residual coal pillar between the n° 11 coal seam and the n° 8 coal seam has no stress influence on it. The n° 14 coal seam is only affected by the concentrated stress of the residual coal column of the n° 11 coal seam, and the influence range covers the comprehensive mining working face because weakening; (b) n° 11 residual coal pillar stress curve distribution after weakening.

6. Coal Pillar Weakening Scheme in Goaf

6.1. Coal Pillar Weakening Scheme

Ganzhuang Coal Mine is preparing to extract the n° 14 coal seam, and the 14,203 working face has been arranged. Above the n° 14 coal seam are the partially mined n° 11 coal seam and the thoroughly mined n° 8 coal seam. The residual coal pillar between the 8509 working face and the 8507 working face of n° 11 coal was taken as the blasting pressure relief object to study the blasting scheme and blasting effect. According to different blasting types, the loose blasting type was selected for this blasting type design. Under the intense compression of the explosive blast shock wave, the coal was divided into blast cavity zone, crushing zone, fracture zone, and non-destructive disturbance zone in order from the location of the coal source outward, as shown in Figure 9. The radius $R_2$ of the fragmentation zone and the radius $R_3$ of the fracture zone directly affected the choice of blast hole spacing, and the value of blasting hole spacing was $2R_2-2kR_3$; the compensation coefficient $k$ was 1–2. According to the blasting energy requirements of the drilling rig and the hole forming ability of the drill bit, the diameter of the blasting hole was determined to be 94 mm. The blasting explosive was a coal mine tertiary emulsion explosive with a density of 1.2 g/cm$^3$, an explosive detonation speed of 6000 m/s, a coil length of 500 mm, a diameter of 70 mm, a single-hole charge of 4 kg, and a millisecond delay detonator for blasting. According to the following formulas, combined with the mechanical parameters of the coal pillar in the gob area of Ganzhuang Coal Mine and the performance parameters of the explosive, the radius of the fracture area was preliminarily calculated to be 1570 mm. In coal pillar blasting, the blasting pressure relief object is the residual coal pillar in the goaf. In order to prevent the coal pillar from being too broken during the blasting process, the compensation coefficient $k$ was set to two, and the blasting hole spacing was set to 6280 mm. Therefore, the spacing between the bottom of the shell holes was set at 6500 mm.

\[
R_2 = \left[ \frac{\rho_0 D^2 n K_1^2 l_1 B}{8 \sqrt{2} \sigma_{cd}} \right]^{\frac{1}{2}} r_b \tag{4}
\]

\[
R_3 = \left[ \frac{\sigma_{cd}}{\sigma_{cd}} \right]^{\frac{1}{2}} \left[ \frac{\rho_0 D^2 n K_1^2 l_1 B}{8 \sqrt{2} \sigma_{cd}} \right]^{\frac{1}{2}} r_b \tag{5}
\]

\[
B = \left[ (1 + b)^2 + (1 + b^2) - 2 \mu_d (1 - \mu_d) (1 - b)^2 \right]^{\frac{1}{2}} \tag{6}
\]
8509 working face and the 8507 working face of n° 11 coal was taken as the blasting pressure relief object to study the blasting scheme and blasting effect. According to different blasting types, the loose blasting type was selected for this blasting type design. Under the intense compression of the explosive blast shock wave, the coal was divided into blast and non-blast zones. To ensure the blasting effect and avoid joint damage, the blasting pressure relief object was set at 6500 mm. Therefore, the spacing between the bottom of the shell holes was set to 6280 m. The drilling layout is shown in Figure 10.

\[
\beta = \frac{2 - \mu_d}{1 - \mu_d} \quad (7)
\]
\[
b = \frac{\mu_d}{1 - \mu_d} \quad (8)
\]
\[
\alpha = 2 \pm \frac{\mu_d}{1 - \mu_d} \quad (9)
\]

Figure 9. Drilling and blasting diagram.

Equation (4) is used to calculate the radius of blast hole fragmentation area, Equation (5) is used to calculate the radius of blast hole fracture area, Equation (6) is used to calculate a factor in Equation (4), Equation (7) is used to calculate the stress wave attenuation index, Equation (8) is used to calculate the lateral stress factor, and Equation (9) is used to calculate the load propagation attenuation index. The above equation \(\sigma_{e,cd}\) is the dynamic compressive strength of the rock, MPa; \(\sigma_{td}\) is the dynamic tensile strength of the rock, MPa; \(\beta\) is the stress wave decay index; \(\mu_d\) is the dynamic Poisson’s ratio of the rock; \(\rho_0\) is the explosive density of explosives, kg/m\(^3\); \(D_v\) is the explosive blast velocity of the coal body, m/s; \(n\) is the pressure increase coefficient when the explosive blast product expands and collides with the gun hole wall, generally take \(n = 10\); \(K\) is charge radial uncoupling coefficient, \(K = \frac{d_b}{d_c}\); \(d_b\) and \(d_c\) for the radius of the gun hole and the radius of the package, mm; \(\eta\) for the expansion of the blast products adiabatic index, generally taken as 3; \(l_c\) for the charge axial coefficient, take \(l_c = 1\); \(B\) and dynamic Poisson’s ratio \(\mu_b\) related; \(\beta\) is the lateral stress coefficient; and \(\alpha\) is the load propagation attenuation index, positive and negative signs corresponding to the shock wave area zone and stress wave area, respectively. \(\gamma_b\) is the radius of the borehole, m.

6.2. Coal Pillar Weakening Release Drilling Construction Plan

The length of n° 14 coal 14,203 working face return tunnel is 1050 m. In order to have contrast, no blast hole is constructed within 100 m from the cutting eye of 14,203 working face outward, and the blast hole is built from 100 m outward. The angle of the blast hole construction is +24°, and the hole depth is 22 m, through the upper n° 11 coal floor, into the coal column body perpendicular to the floor upward 2 m before the final hole. The horizontal distance from the hole mouth to the bottom of the hole is 20 m, and the vertical distance is 9 m. The drilling layout is shown in Figure 10.
Since the distance between the upper residual coal pillar edge and the roadway side is 7 m, the installed borehole stress gauge can completely measure the stress change within the coal pillar stress influence range. In order to have comparability and check the redistribution of stress after blasting off the overlying residual coal column, one stress sensor was set in the blasting area and one in the non-blasting area, and the distance between the two stress sensors was 80 m. Each monitoring point was monitored from near 70 m from the working face, and the data was collected every 5 min.

Figure 10. Construction drawing of coal pillar weakening release drilling.

7. Stress Monitoring and Analysis of Fully Mechanized Mining Face
7.1. Stress Monitoring Program

The stress sensor was used to measure the change of the lateral stress field of the surrounding rock section of the 14,203 transport roadway below the coal pillar and to test the blasting pressure relief effect of the upper residual coal pillar [30–33]. This time, the YHY30 mine intrinsically safe borehole stress sensor was used. It adopts an infrared communication method to collect data to an FCH64/2 mine intrinsically safe handheld collector (Hereinafter referred to as collector), and the collector transmits the data to the computer for data monitoring. The equipment used is shown in Figure 11. The layout of the stress sensor is shown in Figure 12. The diameter of the borehole is 42 mm, and the depth of the borehole is 8 m. Since the distance between the upper residual coal pillar edge and the roadway side is 7 m, the installed borehole stress gauge can completely measure the stress change within the coal pillar stress influence range. In order to have comparability and check the redistribution of stress after blasting off the overlying residual coal column, one stress sensor was set in the blasting area and one in the non-blasting area, and the distance between the two stress sensors was 80 m. Each monitoring point was monitored from near 70 m from the working face, and the data was collected every 5 min.

Figure 11. Borehole Stress Gauge Kit. (a) stress monitor; (b) borehole dynamometer; (c) stress collector; (d) pressurized oil pump.

Figure 12. The location of the stress sensor.
7.2. Stress Monitoring Analysis

Boreholes were drilled, and stress sensors were installed according to the stress monitoring program. The monitoring and data collection started from the working face of 14,203 at 70 m from each measurement point. First of all, the trend of stress curve changes at the two monitoring points can be seen in Figure 13 to show a similar pattern. The stress trend in the non-blasting and blasting areas of the roadway surrounding rock remained the same overall during the mining process. The stress values of both holes are in the stable change stage without any big jump at the location of 45–70 m from the respective measurement points in the 14,203 working face. From 40 m onwards, the stress value is in the stage of pre-mining stress, which increases and then decreases, showing a parabolic change. Secondly, before and after the blasting of the residual coal pillar above the 14,203 working face, the vertical stress values of the roadway surrounding rock had significant changes in the mining influence zone. Especially at the peak of the mining influence zone, the stress peak was 24.4 MPa when non-blasting and 19.5 MPa when blasting, and the stress value decreased by 20%. The results show that blasting of the residual coal column above can have a significant stress redistribution effect. Thirdly, from the deformation state of the channel surrounding rock, after the blasting pre-cracking of the residual coal column of no. 11 coal, the lower 14,203 comprehensive mining face can be pushed in an orderly and smooth manner, and the phenomenon of the hydraulic bracket being “crushed” and unable to push when the pressure comes from the working face does not occur during the pushing period, and the working face pressure is effectively relieved. At the same time, the convergence and deformation state of the back mining tunnel enclosure is good. There is no bulging, anchor pallet fall, or other phenomena that affect the failure of the roadway support system, so there is no need to strengthen the support further.

![Figure 13. Coal pillar internal stress gauge monitoring curve.](image)

7.3. Comparative Analysis of Economic Benefits

The fully mechanized mining face adjacent to the 14,203 working face has encountered various problems during the mining process. The overlying residual coal pillar was not pre-cracked and blasted, causing intense pressure on the working face, and the hydraulic support was “crushed” and could not be pushed. Hence, production had to be stopped for maintenance and replacement of the hydraulic cylinder. This situation happened every month, and each time the production was stopped for nearly two days, the monthly raw coal production could not be completed. The workforce succession plan was disrupted, resulting in economic losses of up to RMB 8,000,000. At the same time, the overlying residual coal pillar pressure affected the return roadway, and the deformation and convergence of the roadway surrounding rock were severe, which affected the safety of the return roadway personnel and the smooth flow of the belt transportation system. After pre-cracking blasting of the overlying residual coal pillars, the 144,202 working face can usually be...
back mined, and the monthly raw coal production can be completed on schedule. The convergence of the deformation of the surrounding rock of the back mining roadway is greatly alleviated, and no further roadway reinforcement support is required, which can save about RMB 8,000,000 and significantly improve the safety benefits of the mine.

8. Conclusions

In this work, a residual coal pillar weakening technology in the near coal seam mining area was proposed to solve the problem of back mining pressure manifestation at the working face under these types of geological conditions. Theoretically, the stress distribution law of the bottom plate caused by the residual coal column was analyzed, and the magnitude of the stress, the influence range, and the relationship with the stress concentration coefficient were derived. A directional blasting drilling technique was designed for residual coal pillars from the lower coal seam to the upper coal seam to weaken the degree of influence of residual coal pillar stresses. The field stress monitoring analysis shows that the roadway surrounding rock convergence deformation state is good to ensure the safe and orderly mining of the lower coal seam. The specific research process is shown below. Firstly, the half-plane limit theory was used to analyze the stress distribution law below the residual coal pillar. GeoGebra data processing software explores the relationship between coal seam burial depth, stress value, and stress concentration factor. The results show that vertical stress plays a dominant role. Next, physically similar simulation experiments were conducted to simulate the mining and residual coal pillar pre-cracking process of each coal seam. It is concluded that the coal pillar body is destabilized and broken after the coal pillar weakens but still maintains a certain support state, which slows down the sinking of the roof above, reducing the downward transfer of roof load and dispersing the vertical stress. Finally, a residual coal pillar blasting pre-cracking technology in the mining area was introduced, and the stress distribution of the roadway surrounding rock was collected and analyzed by stress gauges on-site. The stressed environment of the roadway surrounding rock was improved. The following conclusions were finally reached: (1) Construction of directional blasting borehole can weaken coal column and form large and small structural type buffer body to bear and disperse the load. It decreases the heavy straight stress of the lower coal seam roadway surrounding rock by 20%. The deformation of the roadway surrounding rock is effectively improved, and the comprehensive mining face is pushed smoothly without the hydraulic bracket “crushing” phenomenon. (2) Nowadays, there are many close coal seams in coal mines. This technology can significantly improve the mine pressure environment of the working face under the coal pillar and the mining roadway. It can realize safe, efficient, and low-consumption production in mines, with obvious economic and social benefits worth promoting and applying.

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