Study on the disaster reduction mechanism of presplitting blasting and reasonable blasting parameters for shallowly buried remnant pillars

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Abstract
Disasters such as support crushing easily occur at the mining face under a shallowly buried remnant pillar; therefore, a method for presplitting blasting remnant pillars to relieve stress concentrations and reduce disaster occurrence is proposed. Based on the geological conditions of Nanliang Mine, combining theoretical analysis and numerical simulation to study the bearing characteristics and stress distribution characteristics of remnant pillars, the influence mechanism of presplitting blasting on remnant pillars in terms of the uncoupling factor, axial charge factor, blast hole arrangement, and detonation method is analyzed, revealing the presplitting blasting and disaster reduction mechanism of shallowly buried remnant pillars; furthermore, reasonable blasting parameters are determined. The results show that the rock is affected by the concentrated load within 40 m below the coal pillar; the reasonable blasting parameters are the uncoupled charge coefficient of 1.5, the axial charge factor of 1.0, and the simultaneous detonation of staggered holes. Blasting coal pillars can reduce the transmission effect of the overlying load and ensure the safe use of low-resistance hydraulic support.

1 | INTRODUCTION

Western China hosts a large number of shallowly buried and closely spaced coal seams.1 As the coal resources of the upper group are gradually depleted, some mines gradually progress to the lower coal seam. Some mines use the room-and-pillar mining method and the interval mining method for mining the upper coal seam, causing numerous remnant coal pillars in the goaf. Stress concentration is likely to occur near a remnant pillar2-4 and remaining protective coal pillar.5 When the lower coal seam passes through the remnant pillar of the upper coal seam, support crushing problems such as abnormal pressure of the support, rib spalling, end-face collapse, and roof cutting6-8 occur, especially when the working face of the lower coal seam is cut out of a remnant pillar on the upper coal seam.9 Figure 1 is a schematic diagram of the working face out of a remnant pillar, and Table 1 summarizes the abnormal pressure of a...
YUAN et al. working face as it passes through the coal pillars in a coal mine. The main effects are the sharp increase in the support shrinkage, the serious damage due to rib spalling, and the end roof collapse; additionally, the dynamic load factor increases.

Aiming at the influence of the remnant pillars in the upper coal seam on the advancement of the working face in the lower coal, many scholars have studied the support crushing mechanism, the stress evolution and propagation mechanism, the bearing characteristics of the roof and the fully mechanized roof control technology of the contiguous seams under the room-and-pillar mining gob. Thus, prevention methods of support crushing, such as predigging a roadway, preblasting the coal pillars, repairing the size of the coal pillars, ground drilling, sand filling the coal room, and controlling the mining height, have been proposed.

Although the above scholars revealed the support crushing and disaster-causing mechanism when the working face passes through the remnant pillar from different aspects, and proposed that blasting and weakening the remnant pillar can reduce the transmission effect on the load and ensure safe mining of the lower coal seam. However, there are few studies on the reasonable parameters and blasting effects of coal pillar presplitting blasting. Inspired by the abovementioned works and studies, theoretical analysis and numerical simulation were used to evaluate the transfer effect of load by coal pillars, reasonable blasting parameters and the blasting effect of coal pillars.

2 | ENGINEERING SITUATION

2.1 | Geological conditions

The Nanliang Mine focuses on the 2−2 coal and 3−1 coal seams with a layer spacing that ranges from 20 to 40 m (average layer spacing of 35 m). The 2−2 coal seam has a thickness of 2.2 m, a burial depth of 0-177 m (an average burial depth of 83 m), and a bedrock thickness of 40-65 m. The topsoil is mainly red clay and loess with a thickness of 0-104.5 m. The 20 113 working face adopts the long-wall interval coal mining method. Each working face is 50 m and reserves a coal pillar with 10 m. The thickness of the 3−1 coal seam is 2.6 m, and its average burial depth is 117 m. The 30 117 working face is located directly below the 20 113 working face (Figure 2). The 30 117 working face needs to pass through the 20 113 working face residual interval coal pillar several times during advancement, likely inducing abnormal pressure on the support and thus support crushing accidents.

2.2 | The bearing characteristics of the remnant pillar

When shallow coal seams are intermittently mined, the immediate roof will follow the mining and fall, and the basic roof will be suspended after mining. The loading of the coal pillars is shown in Figure 3, and the actual stress of the remnant pillars can be calculated by Equation (1)\(^\text{17,18}\)

$$\sigma_s = \frac{c + b}{c} \gamma H$$  

where \(b\) is the spacing distance of the interval coal pillars (m), \(c\) is the width of the coal pillar (m), \(\sigma_s\) is the bearing load of the

| Mine | Working face | D/m | L/m | Support shrinkage/m | Characteristic |
|------|--------------|-----|-----|---------------------|---------------|
| Daliuta | 22 103 | 23.3 | 1.7 | 1.5 | A large number of support safety valves open, and some supports are crushed |
| | 21 304 | 19.2 | 3.4 | 1.2 | The rib spalling width is 1.1 m |
| | 21 305 | 18.9 | 4-5 | 0.7-0.8 | Dynamic load factor is 1.8 |
| | 21 306 | 21.3 | 5 | 1.1-1.3 | Dynamic load factor is 1.9 |
| Huojitu | 12 102 | 5.2 | 2.2 | 1.2 | A large number of support safety valves open, triggering a spray state |
| | 12 103 | 0.6-5 | 3 | 0.5-0.8 | The rib spalling width is 0.5 m |
| | 12 105 | 5.3-13.6 | -0.4 | 0.6 | Cutting top, and a large number of support safety valves open |
| Shigetai | 1601 | 13.0 | -5.7 | 1.7 | The support safety valve is violently opened, and multiple supports are crushed |

FIGURE 1 Schematic diagram of the working face out of a coal pillar, where \(L\) is the distance from the support crushing to the coal pillar boundary and \(D\) is the interburden thickness.
remnant pillar (MPa), $\gamma$ is the average unit weight of the overburden (kN/m$^3$), and $H$ is the total thickness of the overburden (m).

The 20 113 working face has a maximum depth of 110 m, including 50 m of topsoil and 60 m of bedrock. The topsoil has a unit weight of 18 kN/m$^3$, and the bedrock has an average unit weight of 25 kN/m$^3$. According to the calculation of the maximum burial depth, the actual load of the remnant pillar is 14.4 MPa.

### 2.3 Stress distribution characteristics of the remnant pillar at the floor

The stress at any point of the floor around the remnant pillar mainly depends on its load and the distance from the point to both sides of the remnant pillar. Using the semi-infinite plane theory of elastic mechanics, the load on the remnant pillar is simplified to a uniform load, as shown in Figure 4.

Assuming that the coal pillar is subjected to a uniform load of size $q$, the vertical stress of $M(x_0, y_0)$ at any point on the floor around the coal pillar can be calculated according to Equation (2).\cite{2,5,17}

$$\sigma_y = \sum_{i=1}^{n} \int_{-c/2}^{c/2} \frac{-2qy^3}{\pi(y^2 + (x + (n-1)(b + c) - \xi)^2)^2} d\xi$$

where $q$ is the load collection degree of the bearing by the coal pillar (MPa), and $n$ is the number of coal pillars.

According to the specific parameters of the 20 113 working face, where $\sigma_y = 14.4$ MPa, $c = 10$ and $b = 50$ m, the distribution of the floor vertical stress concentration factor ($K$) of the three remnant pillars can be obtained, as shown in Figure 5.

As shown in Figure 5, the distribution of the vertical stress concentration factor under the remnant pillar is bubble-like and spreads from both sides of the coal pillar into the surrounding rock. The vertical stress concentration factor under the coal pillar is the greatest at any depth and gradually decreases from under the coal pillar to the gob on either side of the coal pillar. The vertical stress concentration factor directly below the coal pillar decreases gradually and increases with the distance from the coal pillar, from 6.0 at 0 m below the coal pillar to 1.0 at 40 m. The stress concentration factor remains unchanged after 40 m, and the vertical stress concentration factor at 20 m below the coal pillar reaches approximately 1.8. The spacing between the 30 117 working face and the 20 113 working face is 20-40 m; when the 30 117 working face passes through the remnant pillar of the 20 113 working face, it will be affected by the concentrated stress of the floor under the coal pillar.

Regarding the influence of remnant pillars on the working face of the lower coal seams, it is generally believed that\cite{11,19,20} blasting coal pillars can reduce the impact of coal pillars on lower coal seams. That is, advancing the working face of the lower coal seam by a certain distance using the blasting method to destroy the integrity of the remnant coal pillars and reduce the transmission effect of the overburden load. Drilling methods include subsurface inclined drilling (method 1) and vertical
boreholes from the ground (method 2). A schematic diagram of coal pillar blasting is shown in Figure 6.

Due to the particularity of coal pillars and the complexity of blasting construction, and considering the factors of blasting cost, it is important to study and determine reasonable blasting parameters.

3 | STUDY ON REASONABLE PARAMETERS OF REMNANT PILLAR PRESPLITTING BLASTING

3.1 | Blasting fracture range of a single hole

The effect of a blasting dynamic load on a coal and rock mass is mainly defined by the generated shock wave and stress wave. After the explosion of the explosive, the coal and rock mass is subjected to the shock wave and the stress wave, and the crushing zone, the crack area and the elastic zone sequentially form around the blasthole, as shown in Figure 7.

A coal mine generally adopts a column charge to presplit blasting and can adopt a coupled charge or an uncoupled charge scheme. With a coupled charge, there is no gap between the explosive and the wall of the blasthole; with an uncoupled charge, there is a gap between the blast and the wall of the blasthole. The calculation formulas of the crushing zone and the crack area are shown in Table 2.

In Table 2, \( \rho_0 \) is the charge density (kg/m³); \( D \) is the detonation velocity (m/s); \( r_1 \) is the borehole diameter (mm); \( K_\sigma \) is the increase factor of coal strength under dynamic load; \( \sigma_0 \) is the uniaxial compressive strength of the coal and rock mass under static load (MPa); \( A \) is the correlation coefficient; \( B \) is a constant, where \( B = \sqrt{6a^2 - 2a^3 + 2} \); \( \lambda \) is the lateral stress coefficient, where \( \lambda = \mu_d / (1 - \mu_d) \); \( \mu_d \) is the dynamic Poisson’s ratio of coal, where \( \mu_d = 0.8 \mu \); \( K \) is the uncoupling factor, where \( K = r_1/r_0 \); \( r_0 \) is the cartridge diameter (mm); \( l_e \) is the axial charge factor; \( \alpha \) is the attenuation index of load propagation, where \( \alpha = (2 - \mu_d) / (1 - \mu_d) \); \( \sigma_R \) is the radial stress at the interface between the crushing zone and the crack area, where \( \sigma_R = \sqrt{2K_\sigma \sigma_0 / B} \); \( \sigma_{id} \) is the tensile strength of the coal under dynamic load, where \( \sigma_{id} = K_\sigma \sigma_t \); \( \beta \) is the attenuation index of the stress wave in the crack area, where \( \beta = (2 - 3\mu_d) / (1 - \mu_d) \); \( r_2 \) and \( r_2' \) are the crushing zone radii of the coupled and uncoupled charges (mm); and \( r_1 \) and \( r_3' \) are the crack area radii of coupled and uncoupled charges (mm).

According to the Poisson’s ratio of coal \( \mu = 0.29 \), it can be calculated that \( \alpha = 2.3 \), \( B = 1.58 \), \( \sigma_R = 27.4 \) MPa, \( \sigma_{id} = 2.18 \) MPa, and \( \beta = 1.697 \). Taking parameters \( A = 0.86 \), \( \rho_0 = 1200 \) kg/m³, \( \sigma_0 = 20.43 \) MPa, \( l_e = 1 \), \( D = 3600 \) m/s and \( K_\sigma = 1.5 \) into account, \( r_1 = 70 \) mm and \( K = 1 \) when the charge is coupled, and \( r_1 = 105 \) mm and \( K = 1.5 \) when the charge is not coupled. The blasting effect of a single hole with a coupled and uncoupled charge is shown in Table 3.

3.2 | Uncoupling coefficient optimization

To study the effect of the uncoupled charge coefficient on blasting-induced cracking, LS-DYNA software was used to establish the blasting-induced cracking model of a single hole, and the model radius was determined to be 2.5 m, according to the results in Table 3. Considering the symmetry of the model and simplifying the calculation, the model was built as shown in Figure 8.
The MAT_PLASTIC_KINEMATIC material model in the LS-DYNA software was used to simulate the coal and rock mass, and the MAT_HIGH_EXPLOSIVE_BURN material model was used to simulate the explosive. The physical and mechanical parameters of the 2−2 coal seam and explosive are shown in Table 4.

The model failure criterion is set to 1.5 times the static tensile strength, and to ensure the full development of the fracture, the calculation time is set to 4.0 ms. The measurement zone and the fracture zone radii of the two charge structures were measured using the “measure” tool in the postprocessor of the software, and the measurement results are shown in Figures 9 and 10.

An analysis of Figures 9 and 10 shows that when the uncoupling coefficient is <1.5, the number of radial cracks increases with the uncoupling coefficient. When the uncoupling coefficient is >1.5, the length and number of radial cracks increase with the uncoupling coefficient, and the interaction of the radial cracks is more extensive when the coupling coefficient is not 1.5. When the uncoupling coefficient is increased from 1.0 to 2.5, the radii of the crushing zone are 28, 49, 42, 35, and 32 cm, and the lengths of the radial cracks are 190, 178, 180, 173, and 135 cm, respectively. The cracking radii are 218, 227, 222, 208, and 167 cm, respectively. Considering the radius of the blasting-induced cracks and the interaction among radial cracks, a reasonable uncoupling coefficient is determined to be 1.5.

### 3.3 Optimization of the axial charge coefficient

The axial charge coefficient is the ratio of the length of the charge section to the length of the hole, which mainly reflects the amount of charge. To study the influence of the axial charge coefficient on the blasting-induced cracking range and to ensure the full development of the blasting-induced fractures, combined with the cracking radius (2.22 m) when the uncoupling coefficient is 1.5, a model with a size of 3.0 m (x) × 3.0 m (y) × 2.0 m (z) was established, and the blasting range with axial charge coefficients of 0.5 and 1.0 was studied. The model section is shown in Figure 11.

The crack expansion after blasting is shown in Figure 12 considering the two axial charge coefficients.

As shown in Figure 12, when the charge coefficient is 0.5, the radial cracks mainly form in the middle of the model, and the stress wave is superimposed between the cartridge, resulting in tensile stress. The coal pillar fractures in the direction of the vertical stress wave, the maximum radial crack is 2.4 m, and the development range is approximately 0.5 m. When the charge coefficient is 1.0, the development range of the cracks is spherical around the blasthole, and the radial cracks mainly form in the middle of the model and develop to the model boundary. The radial crack length exceeds 3.0 m, and the development range is approximately 0.8 m.

To further study the blasting-induced cracking effect of different axial charge coefficients, monitoring points are arranged in the model to monitor the change in unit stress after coal pillar blasting. The coordinates of the three measuring points A, B, and C are (1, 2, 1), (2, 2, 1), and (3, 2, 1), respectively.

### Table 2 Calculation formulas of the crushing zone and crack area

| Charge structure | Crushing radius/m | Crack area radius/m |
|------------------|-------------------|---------------------|
| Coupled charge   | \(r = \left(\frac{\sqrt{2\mu^3/\alpha k}}{8k\alpha}\right)^{1/2} r_1\) | \(r = \left(\frac{\sqrt{2\mu^3/\alpha k}}{2\alpha}\right)^{1/2} r_2\) |
| Uncoupled charge | \(r_f = \left(\frac{\sqrt{2\mu^3/\alpha k}}{18k\alpha}\right)^{1/2} r_1\) | \(r_f = \left(\frac{\sqrt{2\mu^3/\alpha k}}{2\alpha}\right)^{1/2} r_2\) |

### Table 3 Blasting-induced cracking range of a single hole

| Charge structure | Crushing zone radius/mm | Crack area radius/mm | Effective damage radius of single hole blasting/m |
|------------------|-------------------------|----------------------|-----------------------------------------------|
| Coupled charge   | 292                     | 1384                 | 1.68                                           |
| Uncoupled charge | 354                     | 1677                 | 2.03                                           |

### Table 4 Physical and mechanical parameters of the 2−2 coal seam and explosive

| Coal and rock mass | Lithology | Density/(kg m\(^{-3}\)) | Elastic modulus/GPa | Compressive strength/MPa | Tensile strength/MPa | Explosive | Density/(kg m\(^{-3}\)) | Detonation velocity/(m s\(^{-1}\)) |
|-------------------|-----------|--------------------------|---------------------|--------------------------|---------------------|-----------|--------------------------|----------------------------------|
| 2−2 coal          | 1321      | 2.6                      | 20.43               | 1.45                     | 1300                | 3600      |                          |                                  |
respectively, as shown in Figure 13. The stress-time history curve for each measuring point is shown in Figure 14.

As shown in Figure 14, when the charge coefficient is 0.5, the peak stresses of the A, B, and C points are 10, 5, and 3 MPa, respectively. When the charge coefficient is 1.0, the peak stresses are 21, 10, and 6 MPa, respectively. The peak value of the stress at the measuring point increases with the charge weight; when the charge weight is doubled, the peak value of the measured point is doubled. A comprehensive comparison of Figures 12 and 14 shows that a reasonable axial charge coefficient is 1.0.

### 3.4 Optimization of blasthole arrangement and initiation method

According to the above research, the effective crack range of a single hole with a reasonable charge structure is approximately 5 m, and the remnant pillar width of the 20113 working face is 10 m; thus, a single-row linear hole layout will lead to poor coal pillar weakening. At the same time, considering the blasting sequence of the holes, four blasting schemes are proposed: (a) staggered holes simultaneously detonate; (b) staggered holes have a delayed initiation; (c) symmetric holes are detonated simultaneously; and (d) symmetric hole have a delayed initiation.

The model size for staggered holes is 14 × 10 × 2.2 m (length × width × thickness), and the model size for square holes is 15 × 10 × 2.2 m (length × width × thickness). The diameters of the blasthole and cartridge are 105 and 70 mm, the axial charge coefficient is 1.0, and the blasthole arrangement is shown in Figure 15. In Figure 15, ① to ⑥ are the hole numbers, and the calculation times are 6 and 4 ms for the simultaneous initiation and delayed detonation interval, respectively, so the total calculation time is 10 ms. The cracks that developed with the different hole arrangements are shown in Figure 16.

Figure 16 shows that when the staggered holes are simultaneously detonated, the range of the development of the cracks is generally centered at the blasthole, the cracks between the blastholes are mutually crosscut, and a large area
of damage occurs in the elastic area of the coal pillar. When the staggered holes have a delayed detonation, the cracks are mainly concentrated around the blastholes, the cracks between the blastholes are poorly developed, and the scope of damage in the elastic zones of the coal pillars is small. When simultaneous initiation and delayed detonation of symmetric holes are adopted, the cracks in the rock around the borehole are well developed, and those between the boreholes crosscut each other, but there are few cracks in the elastic zones of the coal pillars, and the coal pillars still exhibit a high bearing capacity. Based on the bearing characteristics of coal pillars and the development of blasting-induced cracks, the optimal method of simultaneous blasting with staggered holes is determined.

4 | ANALYSIS OF THE PRESPLITTING BLASTING EFFECT OF A PILLAR

The finite difference numerical simulation software (FLAC2D) was used to analyze the influence of coal pillars before and after blasting on the advance of the lower coal seam. A model with dimensions of 250 m (length) × 69.7 m
(height) was established, and 40 m boundary coal pillars are added to both sides of the model to eliminate the influence of the boundary effect on the result. The horizontal displacement is fixed on the left and right sides of the model, the vertical and horizontal displacements are fixed on the bottom of the model, and a stress boundary condition is used on the top of the model to simulate the self-weight of the overlying strata. Five monitoring lines were set in the model to monitor the change in the floor stress of the 20 113 working face before and after the blasting pillar. The model size and monitoring line distribution are shown in Figure 17. Before blasting, the coal and rock mass adopts the Mohr-Coulomb constitutive model, and the physical and mechanical parameters of the coal and rock mass are shown in Table 5.

According to the spatial relationship between the gob and remnant pillar after mining of the 2-2 coal seam, first, the interval gob (50 m) of the 2-2 coal seam is mined, while the coal pillar (10 m) is left, and the vertical stress along the five monitoring lines is extracted after running a certain number of steps (10 000 steps), to represent the stress distribution in the floor before blasting the remnant pillar. For the stress distribution in the floor after blasting the coal pillar, to determine the application of the mechanical parameters of the coal pillar after blasting in FLAC2D software, it is assumed that the complete failure of the coal pillar within the effective action range of the hole and the strength of the coal pillar outside the action range of the blasting are not affected. From the results of the crack development of the staggered holes that were simultaneously detonated in Section 3.4, the crack extensions along the I-I and II-II profiles (Figure 15(A)) are extracted, as shown in Figure 18.

When calculating the blasting range of the coal pillar, only the cracks around the blasthole are considered. The blasting range D under the three burial depths of 40, 80, and 120 m were measured by the special postprocessor of the software, and the measuring result is shown in Table 6.

As shown in Table 6, under different burial depths, the blasting range of the coal pillars is roughly the same, approximately 5 m. Therefore, the blasting-induced crack range of a coal pillar is 5 m.

To determine the influence of the coal pillar blasting effect on the mining of the lower coal seam, combined with the assumptions mentioned above and the blasting range of the coal pillar, the coal pillar is partitioned in the FLAC2D software. A certain range (blasting range D) from the coal pillar is set to indicate the overbroken rock; furthermore, by adopting a double yield constitutive model, the strength of the rock outside of this range is not affected. After blasting the coal pillar, the treatment method is as follows. First, the 2-2 coal seam interval gob (50 m) is extracted and the coal pillar (10 m) is left. Then, the coal pillar is partitioned. Finally, after running the model for a certain number of steps (10 000 steps), the vertical stress (syy) along the five monitoring lines

FIGURE 17 Model size and monitoring line distribution
was extracted to describe the stress state of the floor after blasting the coal pillar.

To visually compare the change in the coal pillar floor stress (syy) before and after blasting, the stress variation coefficient (SVC) is defined, that is, the ratio of the floor stress at the same position before and after blasting. On the basis of the stress along the five monitoring lines extracted before and after blasting, the curves of SVC at different depths are established, as shown in Figure 19.

As shown in Figure 19, the SVCs of the monitoring lines are different at different burial depths. With increasing distance from the pillar, the SVC gradually increases under the coal pillar and decreases under the gob, indicating that the stress transfer effect on the load after blasting the coal pillar. Comparing Figure 19(A) to Figure 19(C), it can be seen that with the increase in the burial depth, SVC gradually decreases under the coal pillar and decreases under the gob, indicating that the greater the burial depth is, the smaller the stress transfer effect on the load after blasting the coal pillar.

Based on the theoretical analysis of the coal pillar bearing characteristics and the stress distribution law of the coal pillar floor, combined with the stratum distribution of the 30107 working face, when the support is below the coal pillar, under the action of the periodic pressure and the concentrated stress of the coal pillar, the load acting on the support is given in Equation (3).

\[ p = \frac{k_0 p_0}{\mu} \]

where \( p \) is the load acting on the support by the overburden, kN; \( k_0 \) is the nonuniform coefficient of stress transfer caused by the coal pillar (calculated from the above theoretical results, this value of 1.8); \( \mu \) is the support efficiency, 0.9; and \( p_0 \) is the average support resistance of the supports, kN, which can be calculated according to Equations (4).

\[ p_0 = b \rho_1 g h_z + \left( b \rho_2 g h_1 l_1 + b \rho_3 g K h_0 l_0 \right) \left( 0.54 - \frac{0.24}{l - \sin \theta} \right)^4 \]

where \( b \) is the width of the supports, 1.5 m; \( \rho_1, \rho_2, \) and \( \rho_3 \) are the densities of the immediate roof, the key stratum, and the load layer, respectively, and \( \rho_1 = \rho_2 = \rho_3 = 2500 \text{ kg/m}^3 \); \( l_1 \) and \( l_2 \) are the roof control distance and the breaking span, respectively, the values of which are 4 and 13.44 m; \( h_z, h_1, h_0 \) are the thicknesses of the immediate roof, the key stratum and the load layer, respectively, the values of which are 18.3, 8.2 and 44.5 m; \( \theta \) is the rotating angle, which is generally 8°; \( i \) is the block coefficient of the key layer, and \( i = h_1/l_1 = 0.7 \); and \( K \) is the load transfer coefficient, 0.42. These parameters are substituted into Equations (3) and (4) to obtain \( p_0 = 4262.9 \text{ kN and } P = 8525.8 \text{ kN} \).

The above calculation shows that the hydraulic support of ZY9000/14/26D used in the 30117 working face can ensure safe and smooth mining of the working face, and the practical production also proves that applying this type of hydraulic support is very effective in the field. To improve the efficiency of the 2−2 coal seam hydraulic support, the hydraulic support of ZY7200/14/28D is selected for the right

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**TABLE 6** Blasting range of coal pillars under different burial depths

| Blasting range (D) | Burial depth/m | 40 m | 80 m | 120 m |
|-------------------|---------------|------|------|-------|
| \( D_1 \)/m       |               | 5.1  | 5.0  | 5.2   |
| \( D_2 \)/m       |               | 5.0  | 5.0  | 5.0   |
panel of the 3-1 coal seam. Combined with the above theoretical calculations, the hydraulic support of ZY7200/14/28D provides a lower working resistance, and during the period when the working face passes through the coal pillar, this lower working resistance may cause support crushing accidents. To ensure that the 3-1 coal seam does not have the support crushing accident when using the hydraulic support with a lower working resistance, based on the transfer effect of the coal pillar to the overburden load, the method of blasting coal pillars is selected to ensure the safety of the application of the low-resistance hydraulic support in the lower coal seam.

According to the above numerical simulation, which determined the transfer effect of the coal pillar after blasting on the load, the maximum load transfer coefficient of the coal pillar to the lower coal seam after blasting is 0.71 (at a burial depth of 40 m), and substituting this information into Equation (3) can show that $P = 3363 \text{kN}$, far <7200 kN, that is, after blasting the coal pillar, the load transfer effect of the coal pillar is greatly reduced and can ensure the safe use of low-resistance hydraulic support.

5 | CONCLUSION

1. Theoretical analysis determined that the actual load on the coal pillar is 14.4 MPa, and the vertical stress distribution under the coal pillar is bubble-like, spreading from both sides of the coal pillar into the surrounding rock. At the same depth, the vertical stress is greatest under the coal pillar, and with increasing distance from the coal pillar, the stress gradually decreases.
2. The coal and rock mass is subjected to shock waves and stress waves, and a crushing zone and crack area is generated around the blasthole. The crushing zone radius of the coupled charge and the uncoupled charge are 292 and 354 mm, respectively, the radius of the crack area is 1384 and 1677 mm, respectively, and the effective fracture radius is 1.68 and 2.03 m, respectively.
3. Reasonable blasting parameters and the blasting effect are determined by numerical simulation. The reasonable blasting parameters are as follows: Uncoupling coefficient of 1.5, axial charge coefficient of 1.0, and the blast hole arrangement and detonation mode follow the scheme of the simultaneous detonation of staggered holes. Blasting coal pillars can reduce the transfer effect of coal pillars on the overburden load; the larger the burial depth is, the smaller the transfer effect of coal pillars on the load after blasting. The support load under the coal pillar decreases from 8525.8 kN before blasting to 3363 kN after blasting, which can ensure the safe use of low-resistance hydraulic support.

ACKNOWLEDGMENTS

This work was financially supported by the Fundamental Research Funds for Central Universities (Grant No. 2019XKQYMS61).

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How to cite this article: Yuan Y, Yuan C-F, Zhu C, Liu H-X, Wang S-Z. Study on the disaster reduction mechanism of presplitting blasting and reasonable blasting parameters for shallowly buried remnant pillars. Energy Sci Eng. 2019;7:2884–2894. https://doi.org/10.1002/ese3.468