Research Article

Blasting for Fracturing and Improving the Permeability of Deep, Soft, Outburst Prone Coal Seams Using Blasthole and Relief Hole Drilled into the Underlying Stratum: Optimal Hole Distance

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1. Introduction

With the increasing operation depth of coal mines in China, some coal mines are gradually developing from low gas mines to high outburst mines [1, 2]. According to the statistics of coal mining accidents, gas outburst remains the major type of coal mining accident in China, and the prevention and control of gas hazards remain severely challenging [3–5]. Current outburst-prevention measures adopted at coal mines include protective seam mining [6, 7], large-area gas drainage [8, 9], coal seam water infusion [10, 11], hydraulic fracturing [12, 13], and presplit blasting [14–16]. Notably, methane in deep coal is also an important unconventional natural gas resources. Successful extraction of this natural gas could both improve energy supply and reduce the dangers of underground coal mining [17]. For low-permeability coal seam, deep-hole presplit blasting remains an effective approach for improving the gas drainage efficiency of low-permeability coal seams.

Many scholars have studied the deep-hole presplit blasting in coal seam. Liu et al. [18] found that the effect of drilling blasting with borehole cross the seam was better than that down the seam. Chu et al. [19] found that the blasting crack damage of coal body is the result of the joint action of explosion stress wave, detonation gas, and coal mine methane. Guo et al. [20] showed that the displacement compensation space provided by the relief hole and the surface properties of the hole wall enhanced the tangential tensile...
stress between the coal particles on and around the relief hole wall, which promoted the development and expansion of the radial cracks around the control hole. Cai [21] carried out a long hole loose blasting test on the -610MW2EB4 working face of Liyi Coal Mine in China and found that the antipermeability of coal seam was effectively improved after blasting, and the loose radius reached 3.3-5.5 m. Shang et al. [22] designed the drilling, charging, and hole sealing processes of coal seam blasting. Zhang [23] conducted gas extraction after presplitting blasting in C1201 working face of Chiyu Coal Mine in China and found that the concentration of gas extraction after blasting reached 1.93-3.2 times of the original, the flow rate of gas extraction reached 1.4-2.6 times of the original, and the influence radius reached 4.5 m. The above research has promoted the development of coal seam blasting to increase permeability. However, deep-hole presplit blasting does not produce the desired effect for soft, high-gas-content coal seams deeply buried between hard underlying and overlying strata. The problems include the rapid closing of blasting-induced fractures and a short duration of the permeability improvement effect. To address the above problems, a method for blasting soft, outburst-prone coal seams using a blasthole and a relief hole drilled into the underlying stratum was proposed. This outburst prevention method involves blasting the underlying stratum and controlling the direction of blasting-induced fracturing using a relief hole, and it realizes directional gas drainage from soft, high-gas-content coal seams.

Therefore, in this study, a theoretical model was established for describing the stress wave propagation of blasting intended to fracture and improve the permeability of coal seams using a blasthole and a relief hole drilled into the underlying stratum. Furthermore, the working mechanisms of the relief hole and the transmission characteristics of the blasting-induced stress were analyzed. The fracturing induced by blasting using different blasthole–relief hole distances under different in situ stress conditions was investigated using an implicit–explicit coupled method and the ANSYS/LS-DYNA software. Our results have certain reference value for the blasting of soft, outburst-prone coal masses for outburst prevention.

2. Theoretical Mechanisms

2.1. Stress Wave Propagation Characteristics. Blasting-induced stress waves may be reflected from or transmitted through blasting media, depending on their wave impedance [24]. For blasting intended to fracture and improve the permeability of soft, outburst-prone coal seams using blastholes drilled into the underlying stratum, the blasting-induced stress wave propagates from the underlying stratum to the soft coal seam. The process of propagation of the blasting stress wave from the underlying stratum to the soft coal seam was modeled, as shown in Figure 1. In the figure, \( F \) is the longitudinal incident wave induced by blasting, which propagates from the underlying stratum to the soft coal seam, \( P_i \) is the longitudinal reflected wave, \( S_i \) is the horizontal reflected wave, \( P_t \) is the longitudinal transmitted wave, and \( S_t \) is the horizontal transmitted wave.

Based on the continuity of stress wave propagation, the particle velocities and stresses on the two sides of the rock–coal interface during blasting can be expressed as

\[
\sigma_i + \sigma_r = \sigma_i, \quad (1)
\]

\[
\nu_i - \nu_r = \nu_i, \quad (2)
\]

where \( \sigma_i, \sigma_r, \) and \( \nu_t \) are the stresses (MPa) induced by the incident, reflected, and transmitted waves, respectively, and \( \nu_i, \nu_r, \) and \( \nu_t \) are the particle vibration velocities (m/s) of the incident, reflected, and transmitted waves, respectively.

For a longitudinally propagating stress wave,

\[
\begin{aligned}
\sigma_i &= \frac{\rho_1 C_p}{\rho_1 C_p} \nu_i, \quad \nu_i = \frac{\sigma_i}{\rho_1 C_p} \\
\sigma_r &= \frac{\rho_1 C_p}{\rho_1 C_p} \nu_r, \quad \nu_r = \frac{\sigma_r}{\rho_1 C_p} \\
\sigma_t &= \frac{\rho_2 C_p}{\rho_2 C_p} \nu_t, \quad \nu_t = \frac{\sigma_t}{\rho_2 C_p}
\end{aligned}
\]

By combining Equations (1), (2), and (3), \( \sigma_i \) and \( \sigma_t \) can be expressed as

\[
\sigma_i = \frac{\rho_2 C_p \sigma_r - \rho_1 C_p \sigma_i}{\rho_2 C_p + \rho_1 C_p}, \quad (4)
\]

\[
\sigma_t = \frac{2\rho_2 C_p \sigma_r}{\rho_2 C_p + \rho_1 C_p}, \quad (5)
\]

where \( \rho_1 \) and \( \rho_2 \) are the densities (kg/m\(^3\)) of the rock stratum and coal seam, respectively, and \( C_{p1} \) and \( C_{p2} \) are the velocities (m/s) of the longitudinal waves propagating in the rock stratum and coal seam, respectively.

Then, the reflection coefficient \( F \) can be expressed as

\[
F = \frac{\rho_2 C_p - \rho_1 C_p}{\rho_2 C_p + \rho_1 C_p}. \quad (6)
\]
Different signs, resulting in a tensile strength, the tensile strength of the rock is much lower than its compressive strength. When the blasting stress wave is transmitted from the coal to the underlying stratum to the soft coal seam, because \( \rho_s \sigma_{r} < \rho_1 \sigma_{r1} \), \( F < 0 \), and the incident and reflected waves have different signs, resulting in a tensile reflected wave on the coal–rock interface that reacts on the rock mass. As the tensile strength of the rock is much lower than its compressive strength, the tensile reflected wave can better fracture the rock side of the coal–rock interface. The weakness plane in the rock resulting from the fracturing disposes the blasting stress wave to propagate in that direction and, as the coal mass has a lower strength than the rock mass, finally leads to the formation of massive fractures in the soft coal mass, thereby improving its permeability.

2.2. Working Mechanisms of Relief Hole. The blasting produces a free surface effect at the relief hole, and the effect disposes the stress wave to propagate in the direction toward the relief hole. In addition, the blasting stress wave is reflected from the relief hole and transformed into a tensile stress wave, resulting in a stress concentration around the relief hole, which promotes further fracturing. Figure 2 illustrates the range of fracturing induced by blasting using a relief hole [25].

In Figure 2, the blasthole (on the left) and relief hole (on the right) are designed to have the same radius and are located on the same horizontal line to simplify the model. After blasting, the stress wave is attenuated with distance; thus, the stress in any particle \( A \) can be expressed as [26]

\[
\sigma_r = P_0 \left( \frac{L}{r} \right)^{-\alpha},
\]

\[
\sigma_\theta = \frac{\mu}{1 - \mu} \sigma_r,
\]

where \( p_0 \) is the initial pressure after blasting (MPa), \( \sigma_r \) is the radial stress (MPa), \( \sigma_\theta \) is the tangential stress (MPa), \( L \) is the distance from the particle to the center of the blasthole (m), \( r \) is the radius of the blasthole (m), \( \alpha \) is the attenuation coefficient, and \( \mu \) is the Poisson’s ratio of the material.

When the blasting stress propagates to around the relief hole, the following equations can be derived based on the theory of elasticity [27]:

\[
\sigma_{rr} = \frac{1}{2} \left[ \left( 1 - k^2 \right) \left( \sigma_\theta - \sigma_r \right) + \left( 1 - 4k^2 + 3k^4 \right) \left( \sigma_r + \sigma_\theta \right) \cos 2\theta \right],
\]

\[
\sigma_{r\theta} = \frac{1}{2} \left[ \left( 1 - k^2 \right) \left( \sigma_\theta - \sigma_r \right) - \left( 1 + 3k^4 \right) \left( \sigma_r + \sigma_\theta \right) \cos 2\theta \right],
\]

\[
\tau_{rr} = \frac{1}{2} \left[ \left( 1 + 2k^2 - 3k^4 \right) \left( \sigma_\theta + \sigma_r \right) \cos 2\theta \right],
\]

where \( \sigma_{rr} \) is the radial stress in the relief hole (MPa), \( \sigma_{r\theta} \) is the tangential stress in the relief hole (MPa), \( k = r_y/L \) is an equation coefficient, and \( \theta \) is the angle between the two lines connecting the particle to the blasthole and relief holes.

According to blasting theory, the tensile strength of a rock mass is much lower than its compressive strength, and blasting fragments a rock mass mainly through tensile-stress-induced circumferential fractures. In the far field of blasting, the blasting stress wave is not sufficiently strong to fracture the rock mass. Thus, only the tangential tensile stress needs to be discussed here, and the minimum stress required to fracture the zone around the relief hole can be expressed as

\[
\sigma_{r\theta} = \left[ \left( 1 + \frac{1}{2} k^2 + \frac{3}{2} k^4 \right) \sigma_\theta + \left( \frac{3}{2} k^4 - \frac{1}{2} k^2 \right) \sigma_r \right].
\]

2.3. Effect of In Situ Stress on Relief Hole. The state of relief holes under in situ stress is shown in Figure 3. The stress...
stress in the relief hole (MPa), \(a\) is the distance to the relief hole (cm), and \(r\) is the radius of relief hole (cm).

(1) When \(\theta = 0\), \(a = r\), \(\tau_{r\theta} = 0\), \(\sigma_{rr} = 0\), and \(\sigma_{\theta\theta} = -P_x + 3P_y\)

(2) When \(\theta = \pi/2\), \(a = r\), \(\tau_{r\theta} = 0\), \(\sigma_{rr} = 0\), and \(\sigma_{\theta\theta} = 3P_x - P_y\)

Under the condition of in situ stress, the rock is subjected to tangential tensile stress in the direction of maximum principal stress and tangential compressive stress in the direction of minimum principal stress. The tensile strength of rock is less than the compressive strength. Therefore, the cracks will develop in the direction of the maximum principal stress.

### 3. Numerical Simulation Calculation Scheme

#### 3.1. Numerical Modeling

A numerical calculation model was established for the C13-1 coal seam in the Huainan mining area using the ANSYS/LS-DYNA software. The C13-1 coal seam has an average thickness of 4.5 m and horizontal bedding. It mainly consists of lumpy durain mixed with strips of bright vitrain and thus is semidark and semibright coal. The C13-1 coal seam had a measured gas pressure of 3.8 MPa, an initial gas emission rate of 5×10^2 m^3/day, and a comprehensive index of 16.

### 4. GeoFluids

![Figure 4: Numerical calculation model: (a) blasting model without relief hole; (b) blasting model with relief hole.](image-url)
Figure 5: Continued.
and a $K$ value of 6.6. The coal seam was predicted to have a risk of gas outburst. The coal seam had a high gas content and low permeability and was underlain by a fine sandstone stratum [29].

Figure 4 illustrates the model of the coal seam. The model had the dimensions of $600 \times 600$ cm. It was configured with an arc-roof roadway (60 cm in width and 90 cm in height) located 60 cm above the coal–rock interface and 60 cm from the right boundary of the model, and a blasthole in the underlying stratum of the roadway (360 cm (210 cm for blasting and 150 cm for stemming) in length and 60 cm in diameter) inclined at an angle of 25° to the horizontal line.
The blasting of the coal seam was simulated using the blast-hole alone and a relief hole with a length of 360 cm and different distances (180, 300, and 420 cm, or 3, 5, and 7 times the hole diameter, respectively) to the blasthole. A horizontal in situ stress $\sigma_x$ was applied to the right boundary of the model, a vertical in situ stress $\sigma_y$ was applied to the upper boundary, and displacement constraints were applied to the left and bottom boundaries. The boundaries were defined as nonreflecting to reproduce the actual situation better, thus eliminating their potential influence on the tensile reflected wave.

3.2. Coal, Rock, and Stemming Models. The coal, rock, and blasthole stemming were defined using the *MAT_JOHNSON_HOLQUIST_COOK(HJC) model and the *MAT_ADD_EROSION failure criterion was added for describing the tensile failure of the coal, rock, and blasthole stemming during the blasting process. The Holmquist–Johnson–Cook (HJC) model, an improved version of the Johnson–Cook model, expresses the equivalent strength of a rock mass as the function of its pressure, strain rate, and damage and considers the combined effects of strain rate, hydrostatic pressure, and cumulative damage on rock strength. This model has been widely used in the simulation calculation of impact-induced large rock deformations [30]. Table 1 shows the physiomechanical parameters of the coal, rock, and blasthole stemming selected for the simulation.

3.3. Explosive Material and State Equation. The explosive material was defined using *MAT_HIGHEXPLOSIVE_BURN. The Jones–Wilkins–Lee (JWL) state equation was used to describe the process of the external work of blasting and the expansion and driving of explosion products [31]. The JWL state equation is expressed as

$$p = C_0 + C_1 \mu + C_2 \mu^2 + C_3 \mu^3 + (C_4 + C_5 \mu + C_6 \mu^2) E_i,$$  \hspace{1cm} (17)

where $p$ is the pressure (Pa), $\mu$ is the specific volume (dimensionless), $C_0$–$C_6$ are the coefficients of the polynomial, and $E_i$ is the internal energy per unit volume ($J/m^3$). Table 3 shows the configuration of the air parameters.

3.5. In Situ Stress Loading Scheme. Three mining depths (0, 1000, and 1500 m) were simulated, and the corresponding in situ stresses were calculated using the formula proposed by Heidbach et al. [33] and applied in the X and Y directions. Table 4 shows the in situ stresses for the three different mining depths simulated.

4. Results and Analysis

4.1. Analysis of Fracturing. Figure 5 shows the blasting outputs at different mining depths and blasthole–relief hole distances. The ratio of the fractured area to the total area of coal mass, $n$, was calculated by performing image processing using MATLAB. Table 5 shows the corresponding results. Figure 6 shows the explosion crack box dimension fitting curve. Table 6 shows the fractal dimension of each case. The $n$ of 5 times hole spacing increased by 3.49%, 1.05%, and 1.00%, and the fractal dimension increased by 1.89%, 0.29%, and 0.05%, compared with that of no relief hole, 3 times hole spacing, and 7 times hole spacing. When the buried depth is 0 m, the development of 5 times hole spacing crack is better than other conditions. Figure 4(a) shows the blasting outputs at the mining depth of 0 m and different blasthole–relief hole distances. At the mining depth of 0 m, blasting using different blasthole–relief hole distances fractured a large area of the coal mass, confirming the feasibility of blasting to fracture and improve the permeability of soft coal masses using a blasthole and a relief hole drilled into the underlying stratum. A further analysis of the blasting outputs showed that all the blasting configurations produced several fractures at the coal–rock interface and near the roadway. The reason behind the formation of massive fractures at the coal–rock interface has been analyzed in Section 1 and is not elaborated here. The reason behind the formation of massive fractures along the roadway is that the roadway can be considered as an air-filled cavity that produces a free surface effect and promotes fracturing in that direction. Therefore, the supporting structure of the roadway should be reinforced during floor blasting to avoid excessive damage to the roadway due to blasting vibration and affect the stability of the roadway. Figure 4(b) shows the blasting outputs at the mining depth of 1000 m and different blasthole–relief hole distances. The coal mass was significantly less fractured when compared with that at the mining depth of 0 m, indicating that the in situ stress inhibited the blasting-induced fracturing. The $n$ of 3 times hole spacing increased by 8.93%, 4.15%, and 7.68%, and the fractal dimension increased by 19.12%, 4.08%, and 10.95%, compared with that at no relief hole, 5 times hole spacing, and 7 times hole spacing. Figure 4(c) shows the blasting outputs at the mining...
Figure 6: Continued.

(a) and (b) show the relationship between the logarithm of the number of explosions $N(\delta)$ and the logarithm of the diameter $\delta$, for different hole spacings and depths $H$. The graphs illustrate the impact of single blast holes compared to multiple blast holes at different spacings.

The data points are represented as follows:
- Single blast hole
- 5 times hole spacing
- 3 times hole spacing
- 7 times hole spacing

The curves are labeled with the corresponding depth $H$ values:
- $H = 0 \text{ m}$
- $H = 1000 \text{ m}$

The coefficients $D_k$ for different hole spacings and depths are:

For $H = 0 \text{ m}$:
- $D_1 = 1.5492$
- $D_2 = 1.5740$
- $D_3 = 1.5785$
- $D_4 = 1.5777$

For $H = 1000 \text{ m}$:
- $D_5 = 1.3066$
- $D_6 = 1.5565$
- $D_7 = 1.4955$
- $D_8 = 1.4029$
depth of 1500 m and different blasthole–relief hole distances. At this mining depth, the blasting failed to fracture a large area of the coal mass. The $n$ of 3 times hole spacing increased by 1.92%, 1.32%, and 1.37%, and the fractal dimension increased by 13.59%, 6.78%, and 6.81%, compared with that at no relief hole, 5 times hole spacing, and 7 times hole spacing. The fracturing results showed that the in situ stress inhibited the blasting-induced fracturing in the coal mass. Thus, for the blasting of deeply buried coal masses for permeability improvement, the blasthole–relief hole distance should be reduced to achieve better fracturing.

### 4.2. Analysis of Stress Transmission

Temporal variations in the stresses at two points (points 1 and 2, which were located 100 and 300 cm, respectively, above the central line of the coal–rock interface) were measured. Figure 7 shows the stress–time curves. For a given mining depth, the maximum stress produced by blasting using a relief hole was larger than that produced by blasting using no relief hole, indicating that the addition of the relief hole decreased the stress wave attenuation rate and increased the range of stress wave propagation. A larger initial in situ stress led to a larger effective stress sustained after blasting, indicating that the magnitude of the initial in situ stress was a major factor affecting the postblasting stress state in the coal mass. Thus, for the blasting of deeply buried coal masses for permeability improvement, the effect of the in situ stress should be fully considered, and the effects of the in situ stress and blasting should be synergized to achieve the optimal blasting output. At the mining depth of 0 m, the maximum effective stress at the blasthole–relief hole distance of five times the hole diameter was larger than that for the blasthole–relief hole distances of three and seven times the hole diameter. At the mining depths of 1000 and 1500 m, the maximum stress at the blasthole–relief hole distance of three times the hole diameter was larger than that at the other blasthole–relief hole distances. The stress–time curves further indicate that, for the blasting of deeply buried coal masses for permeability improvement, the blasthole–relief hole distance should be reduced to achieve better blasting output.

### 4.3. Analysis of Optimal Blasthole–Relief Hole Distance

The maximum loads induced by the blasting stress wave yielded by the series of simulations were analyzed to identify the optimal blasthole–relief hole distance at different mining depths. Figure 8 shows the results. As the mining depth increased, the maximum stress loads at both points 1 and 2 decreased, indicating that the in situ stress inhibited the propagation of the blasting stress wave. At the mining depth of 0 m, the maximum stress load at the blasthole–relief hole distance of five times the hole diameter was significantly larger than that at the blasthole–relief hole distances of three and seven times the hole diameter, and the maximum stress loads at the blasthole–relief hole distances of three and seven times the hole diameter were approximately equal. At the

<table>
<thead>
<tr>
<th>Mining depth</th>
<th>Single blast hole</th>
<th>3 times hole spacing</th>
<th>5 times hole spacing</th>
<th>7 times hole spacing</th>
</tr>
</thead>
<tbody>
<tr>
<td>0 m</td>
<td>1.5492</td>
<td>1.5740</td>
<td>1.5785</td>
<td>1.5777</td>
</tr>
<tr>
<td>1000 m</td>
<td>1.3066</td>
<td>1.5565</td>
<td>1.4955</td>
<td>1.4029</td>
</tr>
<tr>
<td>1500 m</td>
<td>1.1959</td>
<td>1.3584</td>
<td>1.2722</td>
<td>1.2718</td>
</tr>
</tbody>
</table>

Figure 6: Explosion crack box dimension fitting curve.

![Figure 6](image_url)
Figure 7: Continued.
mining depth of 1000 m, the maximum stress load at the blasthole–relief hole distance of five times the hole diameter decreased rapidly with time. At point 1, the maximum stress load at the blasthole–relief hole distance of five times the hole diameter was larger than that at the blasthole–relief hole distances of three and seven times the hole diameter. In contrast, at point 2, the maximum stress at the blasthole–relief hole distance of three times the hole diameter was larger than that at the blasthole–relief hole distances of five and seven times the hole diameter. These results indicate that, under these operating conditions, a blasthole–relief hole distance of five times the hole diameter better disposed the blasting energy to be distributed in the horizontal near field between the blasthole and the relief hole and promoted the fracturing in the horizontal near field but did not effectively direct the fracturing in the far field when compared with a blasthole–relief hole distance of three times the hole diameter. By contrast, a blasthole–relief hole distance of three times the hole diameter led to lower energy dissipation in the near field of blasting and slower attenuation and longer-distance transmission of the blasting stress wave and promoted the fracturing in the far field.
depth of 1500 m, a blasthole–relief hole distance of three times the hole diameter performed much better in terms of the maximum stress load when compared with the other blasthole–empty hole distances.

5. Conclusion

We draw the following conclusions from this study.

Owing to the different wave impedances of coal and rock, the blasting stress wave was reflected from the coal–rock interface and transformed into a tensile wave, which promoted the fracturing on the rock side. The fracturing-induced weakness plane in the rock directed the blasting stress wave to develop in that direction, thereby better fracturing the soft coal mass and realizing the intended permeability improvement effect.

A free surface effect was produced near the relief hole during the blasting process, which directed the blasting stress wave to propagate in the direction of the relief hole, thus promoting fracturing in that direction. In deep rock, in situ stress will produce stress concentration at the relief hole. After blasting, cracks will develop along the direction of maximum principal stress.

The in situ stress inhibited blasting-induced fracturing in the coal mass. For the blasting of deeply buried coal masses for permeability improvement, the blasthole–relief hole distance should be reduced for better fracturing.

The supporting structure of the roadway should be reinforced during underlying stratum blasting to avoid excessive damage to the roadway due to blasting vibration and affect the stability of the roadway.

An appropriate blasthole–relief hole distance should be selected for blasting. An inappropriate blasthole–relief hole distance will dispose the blasting energy to be distributed in the horizontal near field between the blasthole and the relief hole and promote fracturing in the horizontal near field but fail to direct fracturing in the far field effectively. An excessively large blasthole–relief hole distance will also fail to direct the stress wave propagation effectively.

Data Availability

The numerical simulation data used to support the findings of this study are included within the article.

Conflicts of Interest

The authors declare that they have no conflicts of interest.

Authors’ Contributions

J.Y. Zhang contributed to the conceptualization. J.Y. Zhang and X.G. Zhao contributed to the methodology. J.Y. Zhang and X. Zhang contributed to the software. G.D. Qiao and S.G. Fu contributed to the validation. J.Y. Zhang contributed to the formal analysis. J.Y. Zhang contributed to the investigation. J.Y. Zhang contributed to the resources. J.Y. Zhang and S. Yang contributed to the data curation. J.Y. Zhang contributed to the writing—original draft preparation. J.Y. Zhang contributed to the writing—review and editing. J.Y. Zhang contributed to the visualization. J.Y. Zhang contributed to the supervision. J.Y. Zhang contributed to the project administration. J.Y. Zhang contributed to the funding acquisition. All authors have read and agreed to the published version of the manuscript.

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Geofluids


