Investigating the destressing mechanism of roof deep-hole blasting for mitigating rock bursts in underground coal mines

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ABSTRACT
Rock burst is one of the challenging problems restricting the safe and high-efficiency mining of deep-buried coal seams, and roof deep-hole blasting (RDHB) is commonly practiced in highly burst-prone zones in coal mines to control rock bursts. This research focuses on the interpretation of the destressing mechanisms of RDHB and its application for mitigating rock bursts. The theoretical analysis results show that RDHB reduces the local stress and stiffness of roof rock, thereby reducing the potential of coal failure and changing the energy release mode of the roof-coal-floor system. Then, a strain-softening numerical model based on the Hoek-Brown failure criterion was developed to evaluate the destressing efficiency of RDHB in selected underground coal mines. Further, RDHB was applied to an isolated coal pillar in a coal mine for mitigating rock bursts, and the stress variation of the isolated coal pillar before and after RDHB was detected using active seismic velocity tomography. The results illustrate that RDHB can effectively reduce stress concentration in the target area and drive stress to the surrounding areas. Moreover, due to the high correlation between in situ measurement results and numerical results, the numerical evaluation of RDHB efficiency may be successfully applied to regular use. Finally, a method for evaluating the destressing effect of RDHB based on blasting seismic energy was discussed and applied.

1. Introduction
Rock burst, a typical dynamic disaster induced by elastic strain energy emitted in a sudden, rapid and violent way from coal and rock masses in both coal and rock mines, is posing a serious threat to the production and miners’ safety (Cook 1965, Bräuner 1994, Mark and Gauna 2016). Particularly, rock burst is becoming a challenging problem in
ultradeep coal seams due to high static stress and severe perturbation stress. Related studies have revealed that the thick and hard roof is one of the contributing factors to rock bursts in coal mines (Li et al. 2015). For rock bursts governed by the thick and hard roof, traditional destress techniques such as large diameter borehole destressing and water injection to weaken coal seam may have a non-ideal performance in mitigating risks. At this moment, roof deep-hole blasting (RDHB) was generally used to reduce stress concentration and excessive elastic energy storage in coal and rock, and then to alleviate rock bursts in coal mines (Konicek et al. 2011, Ouyang et al. 2015, Dou et al. 2022). This technology has been chosen as a necessary destress measure in rock burst-prone coal mines for its efficiency, technical maturity, economy and adaptability to complex underground conditions.

Destress blasting practices were first conducted for mitigating rock bursts at mining faces in the South African gold mines in the 1950s. The aim is to prevent the occurrence of rock bursts or to mitigate their effects in underground openings. The rock burst and subsequent accidents at mining faces presented a considerable reduction, then this measure was considered an effective way to mitigate stress concentration and excessive strain energy stored in the rock mass (Roux et al. 1958, Hill and Plewman 1958). Then, destress blasting was successively applied to control rock bursts in coal and hard rock mines in Canada, Poland, the Czech Republic, China and other mining countries. For example, Vennes et al. (2020a) conducted a large-scale destress blasting at Vale’s Copper Cliff Mine (CCM) in Canada, and verified the destressing effect by measuring mining-induced stress changes in the pillar. Further, they pointed out the efficiency of destress blasting was influenced by the orientation of principal stress based on the analysis of Phase 3 destress blasting results at the mine (Vennes et al. 2020b). The occurrence mechanism of rock bursts in coal mines and hard rock mines is similar, and both are caused by stress concentration and energy release in the surrounding rock system. However, for rock bursts in coal mines, the superficial phenomenon is the ejection or throwing of the coal seam, but the energy released from the roof and floor strata also contributes greatly to rock bursts. For this reason, destress blasting was commonly designed in the roof strata above coal seams in coal mines, and destress blasting in coal mines is also known as RDHB. Wojtecki et al. (2022), Konicek et al. (2013) and Konicek and Waclawik (2018) conducted RDHB for rock burst prevention in the upper Silesian region in Poland and the Czech Republic. Then, the seismic effect parameters and seismic focal mechanisms were developed to evaluate the effectiveness of RDHB based on microseismic monitoring, and the results showed that most RDHBs have at least a very good destressing effect. Fuławka et al. (2022) developed a novel method of destress blasting efficiency evaluation based on in situ seismic measurements, and the novel method was successfully applied in the control of rock bursts in underground coal mines. Qi et al. (2007), Zhao (2021), Chen and Liu (2018), and Dou et al. (2020) performed RDHB on the hard roof to reduce the potential of rock bursts in coal mines, China. The destressing effectiveness was verified by detecting the change of stress and microseismic activity in coal seams. Above engineering practices illustrated successful destress blasting ultimately markedly reduces the stress of coal and rock in the effective range of blasting, drives stress peak to deep areas of coal and rock and changes the rock failure mode from brittle elastic to ductile plastic.
Moreover, numerical simulations have been used to further study the process and mechanism of destress blasting. Tang (2000) adopted FLAC3D software to investigate the destressing mechanism of blasting in hard rock and proposed rock fragmentation factor $\alpha$ and stress reduction factor $\beta$ to evaluate its effectiveness. Sainoki et al. (2017) optimized the traditional model proposed by Tang and obtained more reliable results. Mu et al. (2006) simulated the stress distribution in coal and rock before and after RDHB, and a considerable stress drop in the coal seam was presented after blasting. Correspondingly, the destressing effectiveness was examined by electromagnetic radiation technology in a coal mine, in China. Drover et al. (2018) investigated the destress blasting design for deep tunneling by blasting numerical models and discussed the effect of in situ stress on the blasting-induced cracks. As a result, they proposed a destress blasting design concept considering the effect of in situ stress. Similarly, Saadatmand Hashemi and Katsabanis (2021) studied numerically blasting-induced damage zones and the stress change during blasting, and discussed the effect of in situ stresses and their orientation on the blasting-induced damage zones. Yardimci and Karakus (2020) discussed the effectiveness of destress blasting based on the simulations of an underground longwall mine located in Australia. Moreover, Zhao (2021) and Zuo et al. (2019) studied numerically the effect of blasting layouts on blasting-induced damage zones, including three types of blasting layouts: ‘straight-line’, ‘triangular-fractal’ and ‘deep-shallow-fractal’. The results show ‘deep-shallow-fractal’ blasting layout causes more developed plastic damage zones and more uniform rock fragmentation.

Despite the effectiveness of destress blasting for alleviating rock bursts being verified by considerable engineering practices and the consensus of qualitative explanation of the destressing mechanism being well-received, there is a general lack of detailed explanation of the destressing mechanism. Especially, different from hard rock mines, the burst object (coal seam) and destress object (roof rock mass) are not consistent for rock burst in coal mines, resulting in a more complex interpretation of the destressing mechanism of RDHB. However, there is little literature that makes systematic research on RDHB for preventing rock bursts in coal mines. In this research, based on the energy transfer in the roof-coal-floor system during the coal failure process, the mechanisms of RDHB for mitigating rock bursts in coal mines were analyzed theoretically first. Afterwards, a simplified rock blasting numerical model was developed to evaluate the destressing effect of RDHB. Meanwhile, the influence of blasting intensity and rock medium properties on the destressing effect was discussed using this numerical model. RDHB was applied to a complex isolated coal pillar in a coal mine to mitigate the risk of rock burst, and the stress variation of the isolated coal pillar before and after RDHB was measured using active seismic velocity tomography. The in-situ measurement results are compared with the numerical results to verify the efficiency of the numerical model. Finally, a method for evaluating the destressing effect of RDHB based on blasting seismic energy was discussed and applied.

2. Theoretical analysis of destressing mechanisms of RDHB for mitigating rock bursts

During underground coal mining, three objects of roof, coal and floor constitute the underground stope space. As shown in Figure 1, the removal of coal leads to the loss
of confining stress at the mining boundary and the stress gradually transfers into
deeper solid as mining progresses. Zones of vertical stress that exceed the in-situ
stress are known as abutment zones, and elevated stress is known as abutment stress,
which progresses along with the coal face advancement. The abutment stress gener-
ates a static loading on the roof-coal-floor system. Moreover, the overburden in the
goaf will fracture and collapse subjected to mining stress, and mining disturbance
may induce far-field fault slip, which will generate dynamic stress. Dou et al. (2015)
pointed out that there is a high risk of rock bursts when static stress $\sigma_s$ in coal and
dynamic stress $\sigma_d$ superimposes and exceeds the critical stress $\sigma_{bmin}$. To summarize,
the static stress in coal is the foundation of rock burst, and the dynamic stress is the
trigger. Therefore, the high static stress areas (orange areas) are the potential risk
areas of rock bursts.

However, the above stress criterion only can judge whether the coal failure occurs
but not answer whether rock bursts are likely to occur. This means that stress condi-
tion is a necessary but not sufficient condition for the occurrence of rock bursts. For
this reason, Cai et al. (2019, 2020) further investigated the energy transfer in the
roof-coal-floor system during the coal failure process under static loading conditions,
as shown in Figure 2. In this conceptual model, a strain increment ($\Delta e_2$) in coal will
simultaneously cause a strain change ($\Delta e_1$) in the roof and floor under the static
stress $\sigma_s$. As a result, the ratio of coal strain ($\Delta e_2$) to total strain ($\Delta e$) can be
expressed as:

$$\frac{\Delta e_2}{\Delta e} = \frac{\Delta e_2}{\Delta e_1 + \Delta e_2} = \frac{1}{1 + \frac{k_2}{k_1}} \tag{1}$$

where $k_1$ is the loading elastic modulus of rock, and $k_2$ is the unloading elastic modu-
lus of coal.

Generally, the stiffness and strength of the roof and floor (purple curve) are much
greater than that of coal (red curve), when $k_1 + k_2 = 0$, $\Delta e_2/\Delta e \to \infty$, corresponding to

Figure 1. Rock burst induced by static and dynamic stress superposition.
point S1, the roof-coal-floor system reaches an extremely unstable state. The small dynamical disturbance will be greatly amplified and induced the whole dynamic failure. Then, the roof-coal-floor system will reach a stable state, corresponding to point S, which indicates the end of rock bursts. The elastic strain energy $U_1$ accumulated in the surrounding rock system will be released, which induces a rock burst. $U_2$ is the energy dissipated from $U_1$ during the rock burst and $U_3$ is the energy released from the roof-coal-floor system. For more details on the derivations and graphical explanation, please refer to Cai et al. (2020). Therefore, the elastic strain energy accumulated in the roof rock was the fundamental factor contributing to rock bursts. However, when the stiffness and strength of the roof rock are less than that of coal (blue curve), a similar dynamic failure of the surrounding rock system will occur when $k_3 + k_4 = 0$. Among them, $k_3$ is the loading elastic modulus of coal, and $k_4$ is the unloading elastic modulus of rock. Instead, the roof rock occurs failure before the coal under the compressive load in the roof-coal-floor system. At this moment, the elastic strain energy $U_4$ accumulated in the coal will be released and transferred to the roof rock, resulting in the failure extension of the roof rock. $U_5$ is the energy dissipated during the failure of roof rock. Because the roof mass is in an enclosed space, there is no space for its ejection and throwing. Therefore, the energy from the surrounding rock system is released in the form of seismic events, rather than rock bursts.

Given the above, the core purposes of all destress measures are to change the stress state of the surrounding rock system and weaken the ability of coal and rock to accumulate elastic strain energy. In other words, destress measures change the strength, stiffness and energy conditions of rock bursts. RDHB begins with drilling directional holes in the roof strata, then executing charging, hole sealing and detonating successively. In general, the hole length and explosive weight for the RDHBs are not less than 25 m and 30 Kg, respectively (Qi et al. 2007). Related studies have illustrated that blasting in rock causes rock structure failure within a certain range around the blasting hole (Donze et al. 1997, Sanchidrián et al. 2007). To be specific, when the explosive detonates in the confining stress rock, the rock around the hole wall is crushed by the explosive shock wave, forming a crushing zone. Then, the explosive shock wave rapidly decays into the stress wave, rock cracks are generated and
extended under the coupling action of stress waves and high-temperature gas, forming a crack zone. At this time, the elastic strain energy accumulated in the rock is released transiently along the crack surface, coupled with explosion residual energy, causing the elastic vibration of the rock. The failure characteristic of rock subjected to blasting load was shown in Figure 3.

By analyzing the occurrence conditions of rock bursts and the rock blasting damaged characteristic, the mechanism of RDHB for mitigating rock bursts can be described in two aspects. On the one hand, blasting in rock destroys rock structure along with the release of elastic strain energy accumulated in rock. Since the damaged rock has poor loading capacity and energy storage capacity, the local stress of rock around the blasting hole was effectively reduced and the stress peak was transferred to the surrounding areas after blasting. As a result, the potential of rock bursts is markedly reduced by changing the stress condition. On the other hand, the stiffness condition of the roof-coal-floor system changes fundamentally due to the blasting damage of roof rock, it will influence the energy transfer in the surrounding rock system. To be specific, when RDHB has an ideal effect, that is to say, the strength and stiffness of roof rock are weakened to be less than that of coal. As discussed earlier, rock burst is less likely to occur under such conditions.

3. Numerical evaluation of the destressing effect of RDHB

3.1. Numerical method

In order to evaluate the destressing effect of RDHB for mitigating rock bursts in coal mines, a simplified rock blasting numerical model was developed to investigate the rock damage effect subjected to blasting load, stress re-distribution and elastic strain energy evolution in the rock after blasting. The numerical calculation process in FLAC\textsuperscript{3D} was shown in Figure 4. It mainly includes geometrical model construction, rock mass constitutive and parameters selection, rock prestressing, blasting dynamic analysis and stress re-distribution analysis.
3.1.1. Geometry of the model
First, the RDHB model was simplified to a regular cylindrical hole blasting in the homogeneous and isotropic rock mass. Considering the numerical computing capability, the geometry size of the model was finally determined to be 20 m x 10 m x 20 m. Generally, the diameter of blasting hole in coal mines was 75 to 150 mm. A regular cylindrical hole with a diameter of 100 mm and a length of 2.0 m was designed in the center of the model. The mesh size of the zone near the blasting hole was 0.02 m and gradually increases outward, reaching 0.5 m at the boundary of the model, as shown in Figure 5.

3.1.2. Constitutive model
The strain-softening model based on the Hoek-Brown failure criterion was selected as the rock mass constitutive model in this numerical model. To be specific, three Hoek-Brown strength envelopes were defined in the strain-softening model to characterize the strength softening behavior of rock mass, including one peak strength envelope (black curve) and two residual yield envelopes (red and blue curves), as shown in Figure 6. The stress response to the strain of rock mass was shown in Figure 7. The first residual strength envelope (red curve) represents the phase from peak strength to post-peak strength (A-B phase) of the rock. In this phase, the continuous accumulation of plastic shear strain $e_p^P$ leads to the fracture of rock, but the rock is still interlocked. The first phase will end when the plastic shear strain reaches the critical value $e_p^{crit}$. The second residual strength envelope (blue curve) represents the phase from post-peak strength to ultimate strength (B-C phase). In this phase, the continuous accumulation of volumetric strain results in strength loss. This phase will end when the volumetric strain reaches the maximum volume strain $e_u$, and the rock mass has been fully destroyed (Itasca Consulting Group Inc 2019).
Formulas (2) to (5) give the expression of the Hoek-Brown failure criterion and related parameters (Hoek and Brown 1980, Hoek et al. 2002).

\[
\delta_1 = \delta_3 + \delta_{ci} \left( m_b \frac{\delta_2}{\delta_{ci}} + s \right)^x
\]  

(2)

where, \(\sigma_1\) and \(\sigma_3\) are the maximum and minimum principal stresses of rock mass failure, respectively; \(\sigma_{ci}\) is the uniaxial compressive strength of rock; \(m_b\), \(s\) and \(x\) are Hoek-Brown constants, the calculation formula of each parameter is as follows:

\[
m_b = m_i \exp \left( \frac{\text{GSI}-100}{28} \right)
\]  

(3)
where, $m_i$ is a constant representing rock strength, and GSI is a geological strength index representing the integrity of rock mass.

In this constitutive model, the input parameters of rock mass mainly include density, elastic modulus, uniaxial compressive strength, geological strength index GSI, material constant $m_i$ and critical plastic shear strain coefficient. The GSI parameter can be estimated according to the classification system of the geological strength index proposed by Hoek-Brown (Hoek et al. 2005). The material constant $m_i$ can be estimated by the following formula, and the others parameters can be measured in the laboratory.

$$m_i \approx \frac{\sigma_c}{\sigma_t}$$

where, $\sigma_c$ and $\sigma_t$ are the uniaxial compressive strength and tensile strength of rock mass, respectively.

### 3.1.3. Blasting stress loading

In order to simulate cylindrical borehole blasting, a radial blasting stress wave is generally applied to the elements on the borehole surface in the FLAC$^{3D}$ dynamic analysis program. At present, the blasting stress wave can be obtained by theoretical calculation, field measurement or related dynamic program calculation. In this study, the following general form of a pulse function is used to represent the blasting stress wave (Cho and Kaneko 2004, Ma and An 2008).

$$P = P_0 \left[ e^{-\alpha t} - e^{-\beta t} \right]$$
where, $P$ is the pressure at time $t$, $P_0$ is the peak pressure on the borehole surface, $\alpha$ and $\beta$ are constants.

Constants $\zeta$ and $t_0$ are defined to describe the rise and fall of the stress wave function:

$$\zeta = 1/(e^{-\alpha t_0} - e^{-\beta t_0})$$  \hspace{1cm} (8)

$$t_0 = (1/(\beta - \alpha)) \ln (\beta / \alpha)$$ \hspace{1cm} (9)

Formula (7) is modified as:

$$P = P_0 \zeta [e^{-\alpha t} - e^{-\beta t}]$$  \hspace{1cm} (10)

According to the blasting theory, the peak pressure of blasting on the borehole surface with the cylindrical uncoupled charge can be calculated by the following formula (Dai 2001):

$$P_0 = \frac{1}{2} \rho_0 D_b^2 \frac{2}{1 + \gamma} K^{-2/\gamma} l_e n$$ \hspace{1cm} (11)

where, $P_0$ is the peak pressure, $\rho_0$ is the explosive density, $\gamma$ is the expansion adiabatic index of detonation product, $\gamma = 3$, $D_b$ is detonation velocity, $K$ is the radial decoupling coefficient of charge, $l_e$ is the axial coefficient of charge, $n$ is the pressure increase coefficient, which is generally 10.

### 3.1.4. Boundary conditions and mechanical damping

Three stages constitute the numerical simulation of RDHB, including prestressing loading, blasting stress loading and rock stress redistribution. Among them, the stages of rock prestress loading and stress redistribution belong to statics analysis, while the stage of blasting stress loading is dynamic analysis. In the statics analysis, three-dimensional equal confining pressure of 30 MPa is applied to the model face and all the free faces are fixed, then the initial stress field is equilibrated. In the dynamic analysis, the initial fixed boundary is replaced by a static boundary, which can absorb wave energy reflected from the model boundary as much as possible. Damping characteristics exist in all propagation media. It is related to material internal friction and the possible contact surface, which can reduce the amplitude of the natural vibration of the system. Therefore, damping characteristics should be reappeared in numerical simulation to ensure high reliability. Multiple damping schemes are provided in FLAC$^{3D}$, and local damping is widely applied in rock engineering. Generally, the critical damping ratio can be set as 2% to 5%. In this paper, it was set as 5%, then the local damping coefficient is 0.1571.

### 3.1.5. Simulation scheme

In order to study the influence of blasting intensity and rock mass properties on the destressing effect, two groups of six cases were investigated in this research. The first group of three cases were designed to analyze the influence of blasting intensity. The
Table 1. Borehole and explosive performance parameters of RDHB.

| Case | Coal mine | Explosive types      | Explosive density (kg/m³) | Explosive velocity (m/s) | Borehole diameter (mm) | Explosive diameter (mm) | Radial decoupling coefficient | Peak impulse pressure (GPa) |
|------|-----------|----------------------|---------------------------|--------------------------|------------------------|--------------------------|-------------------------------|---------------------------|
| Case 1 | YT       | Emulsion explosive   | 1100                      | 3600                     | 108                    | 80                       | 1.35                          | 2.94                      |
| Case 2 | GJP      | Emulsion explosive   | 1300                      | 3600                     | 85                     | 60                       | 1.42                          | 2.57                      |
| Case 3 | XZ       | Emulsion explosive   | 1250                      | 3600                     | 94                     | 64                       | 1.47                          | 2.00                      |
borehole and explosive performance parameters of RDHB used in three coal mines in China were investigated first, and then the peak impulse pressure was calculated according to Formula (11), as shown in Table 1. Among which, the detonation velocity of explosive is influenced by many factors, such as explosive composition, the type of sensitizer, the type and size of the booster, the temperature of the explosive and ambient temperature, the critical diameter and density of the explosive (Mertuszka et al. 2019a, 2019b). Particularly, the emulsion explosives used in the selected underground coal mines are different in composition and proportion. Therefore, the detonation velocity of explosive is extremely difficult to determine accurately, then a certain simplification must be made for the value of explosive velocity in the numerical calculation. In this research, the velocity of explosive is declared by the explosive manufacturer, and the value is 3600 m/s. In future works, the velocity of emulsion explosives needs to be further accurately determined.

According to formulas (7) to (10), it is assumed that impulse wave rise time $t_0$ is 100 $\mu$s, $\beta/\alpha = 1.5$, and the peak impulse pressure is 2.0 GPa, 2.5 GPa and 3.0 GPa, respectively. The function curves of the pulse waves are shown in Figure 8. Moreover, the rock type in the three simulation cases was selected as moderately hard rock, and its physical and mechanical parameters can be found in Case 4 in Table 2.

The second group of three cases was designed to investigate the influence of rock mass properties. The rock mass simulated in this study is sandstone, and the related physical and mechanical parameters of rock mass are shown in Table 2. Moreover, the borehole and explosive performance parameters of RDHB used in the three simulation cases can be found in Case 3 in Table 1 and the peak impulse pressure is 2.0 GPa.

![Figure 8. Function curves of pulse waves.](image-url)

**Table 2. Physical and mechanical parameters of different types of sandstone.**

| Case   | Rock types           | Density (kg/m$^3$) | Elastic modulus (GPa) | Uniaxial compressive strength (MPa) | Uniaxial tensile strength (MPa) | GSI | Critical plastic shear strain coefficient | $m_i$ |
|--------|----------------------|--------------------|-----------------------|-------------------------------------|---------------------------------|-----|------------------------------------------|-------|
| Case 4 | Moderately hard rock | 2621               | 16.4                  | 60.9                                | 3.5                             | 75  | 17.4                                     | 0.01  |
| Case 5 | Hard rock            | 2630               | 20.5                  | 92.6                                | 5.0                             | 75  | 18.5                                     | 0.01  |
| Case 6 | Extremely hard rock  | 2647               | 26.4                  | 122.4                               | 7.4                             | 75  | 16.5                                     | 0.01  |
3.2. Results and analysis

3.2.1. Rock damage and weakening of rock mechanical parameters after blasting

Fish language in FLAC$^3$D was used to program for extracting rock mechanical parameters after blasting, including strength, elastic modulus and cohesion. As shown in Figure 9(a), the measuring plane was designed as the YZ plane at $X = 5.0$ m, and the measuring line was designed parallel to the Z-axis along the center of the model ($X = 5.0$ m, $Y = 10.0$ m).

Limited by the length of the paper, this paper only gives detailed simulation results of Case 3. Figure 9(b)–(d) shows the imagery of strength loss indicator, elastic modulus and cohesion of rock after blasting in the measuring plane, as well as the fluctuation curve of the above parameters on the measuring line. The strength loss indicator in Figure 9(b) ranges from $-1.0$ to $1.0$. Before the rock mass reaches the yield state, the value is $1.0$ and decreases to $0$ with the increase of plastic shear strain. Subsequently, this value continues to decrease to $-1.0$ with the increase of volumetric strain, which represents that the rock mass has been fully destroyed. The result shows
that the strength loss indicator of rock mass reaches $-1$ within 0.58 m around the blasting hole, indicating that rock mass in this zone has been fully destroyed, corresponding to the crushing zone of rock. Then, the strength loss indicator is less than 0 in the range of 0.58 m to 3.00 m outside the blasting hole, indicating that rock mass in this zone is damaged but not fully destroyed, corresponding to the crack zone of rock. In the range of 3.00 m to 4.55 m outside the blasting hole, the rock mass is in the plastic deformation state without failure. Similarly, it can be seen from Figure 9(c) and (d) that elastic modulus and cohesion of rock mass also perform the same softening characteristics. To be specific, the elastic modulus and cohesion are weakened to 0 after blasting within the radius of 0.64 m and 1.64 m around the blasting hole, respectively. Moreover, the above parameters gradually return to the initial level with the increase of distance from the blasting hole. This suggests that rock failure occurs subjected to blasting load, resulting in the weakening of rock mechanical parameters, including strength, elastic modulus and cohesion.

3.2.2. Stress and energy evolution in rock after blasting

Figure 10(a) shows the imagery of the maximum principal stress of rock on the measuring plane after blasting and the corresponding fluctuation curve on the measuring line in Case 3. Further, the elastic strain energy accumulated in rock can be calculated by the following formula (Shen et al. 2020):

$$ U = \frac{1}{2E} \left[ \sigma_1^2 + \sigma_2^2 + \sigma_3^2 - 2\mu(\sigma_1\sigma_2 + \sigma_2\sigma_3 + \sigma_1\sigma_3) \right] $$  \hspace{1cm} (12)

where, $\sigma_1$, $\sigma_2$, and $\sigma_3$ are triaxial principal stresses respectively, $E$ is elastic modulus, and $\mu$ is Poisson’s ratio.

Fish language in FLAC$^{3D}$ was used to program for extracting the elastic strain energy in rock before and after blasting, and then $\Delta U$ was obtained by difference processing. When $\Delta U > 0$, it represents energy accumulation, while $\Delta U < 0$ represents energy release. The imagery of elastic strain energy difference $\Delta U$ of rock before and after blasting and the corresponding fluctuation curve on the measuring line were shown in Figure 10(b).
Figure 10 shows that zones around the blasting hole can be divided into three symmetric zones: A- destressing zone (energy release zone), B- stress increase zone (energy accumulation zone) and C- initial stress zone (energy stable zone). First, within 2.31 m around the blasting hole, the maximum principal stress after blasting is lower than the initial stress and the elastic strain energy difference is negative, corresponding to the destressing zone (energy release zone). Then, the maximum principal stress gradually increases with the distance from the blasting hole in a certain range and reaches the peak at 3.15 m away from the blasting hole. Subsequently, the maximum principal stress gradually decreases to the initial stress and keeps the balance. Correspondingly, in the range of 2.31 m to 6.25 m away from the blasting hole, the elastic strain energy difference is a positive value. The corresponding zone belongs to the stress increase zone (energy accumulation zone). Finally, the maximum principal stress and elastic strain energy difference approach the initial stress and zero, respectively, corresponding to the initial stress zone (energy stable zone). The results suggest that blasting can effectively reduce the stress level and energy accumulation of the rock around the blasting hole. However, it also can be seen that new energy accumulation zones will be formed outside the energy release zone. As described above, the rock around the blasting hole will fail subjected to blasting load, accompanied by strength loss, elastic modulus softening and cohesion softening. For this reason, the blasting damaged rock loses its loading ability and energy accumulation ability. As a result, the stress loading onto the damaged rock is transferred to the intact rock, and the stress reduction zone and stress increase zone are formed, respectively. Some stress detection techniques are practiced in the newly formed stress increase zones to determine the risk of rock bursts. If there is a potential of rock burst, a new round of destress blasting will be practiced in these zones. The mining operation can be restarted until these areas have no risk of rock bursts.

As described in Chapter 2, rock bursts may occur only when the stress exceeds rock strength, which means the stress conditions should be primarily satisfied. The simulated results show that when blasting was conducted in the roof rock, the stress of rock around the blasting hole can be effectively reduced, accompanied by the
release of elastic energy accumulated in the rock. Therefore, it can be concluded that the rock burst risk will be effectively reduced after RDHB because the stress conditions of rock bursts are difficult to meet. Furthermore, the stiffness softening of blasting damaged roof rock also changes the energy transfer in the surrounding rock system. When the strength and stiffness of roof rock are weakened to less than that of coal, the roof rock will damage before the coal in the roof-coal-floor system and elastic energy accumulated in the coal will be fed into the roof rock. In this condition, there is less potential for rock bursts to occur.

3.2.3. Rock vibration effect induced by blasting

According to the theoretical analysis of the rock blasting process, part of the explosive energy is transmitted and dissipated in the rock in the form of seismic waves, which induces rock vibration. Existing studies illustrated that blasting vibration may cause damage to the roadway and its support structure in coal mines (Kan et al. 2022). Therefore, the control of blasting vibration should not be ignored in the application of RDHB. In the numerical simulation, the rock vibration velocity during blasting dynamic analysis was recorded. The measuring points (A-G) were arranged on the left-top of the model, and the results are shown in Figure 11. As can be seen, blasting in rock can induce instantaneous vibration of rock, and the peak vibration velocity of measuring point G can reach 0.72 m/s. With the increase of propagation distance, the peak vibration velocity of rock decreases nonlinearly and the attenuation formulas can be obtained by fitting. Generally, the safety critical peak vibration velocity for roadway in coal mines is 0.05 m/s (Mutke et al. 2015). Then, the safety critical distance of blasting is calculated as 31 m by fitting formulas.

3.2.4. Effect of blasting intensity and rock medium properties

To provide quantitative insights into the effect of blasting intensity and rock medium properties, the elastic strain energy difference before and after blasting and peak vibration velocity in rock were statistically analyzed, and the results of the first group of three cases (Case 1 to Case 3) and the second group of three cases (Case 4 to Case 6) were shown in Figures 12 and 13, respectively.

According to the interpretation of Figure 10, the results suggest that both the radiuses of the energy release zone and the energy accumulation zone increase as the blasting intensity. To be specific, when the peak pressure of the blasting load is
2.0 GPa, 2.5 GPa and 3.0 GPa, the radius of the energy release zone is 2.31 m, 3.10 m and 3.90 m, respectively. With the increase of blasting intensity, the blasting impact on the rock becomes more intense, resulting in a greater damage effect. The energy difference peak in the energy accumulation zone and its radius increase as the blasting intensity, which means more stress in damaged rock is transferred to intact rock. Therefore, in order to obtain greater destressing effects, the blasting intensity should be improved as high as possible. However, the peak vibration velocity in rock during blasting has a positive correlation with blasting intensity. It can be seen from Figure 12(b) that the critical safety distance of RDHB increases from 31 m to 80 m with the blasting intensity. Therefore, the undesirable vibration effect should be considered in the application of RDHB.

In contrast, the radius of the energy release zone decreases as the increase of rock hardness. The harder the rock, the more explosive energy will be needed for damaging the rock. When the blasting intensity is the same, the radius of the blasting damage zone decreases with the increase of rock hardness, resulting in a smaller destressing zone. It can also be found that the peak vibration velocity in rock during blasting has a negative correlation with rock hardness. However, the energy difference peak in the energy accumulation zone increases with rock hardness. The main reason is that extremely hard rock has a better loading capacity, and the stress in rock is higher. This suggests that the rock medium properties have a close relationship with the blasting destressing effect. For the extremely hard rock, a higher blasting intensity is needed to achieve a greater destressing effect.

In conclusion, both the blasting intensity and rock medium properties have great influences on RDHB. Blasting vibration has a positive correlation with the destressing effect, therefore, the value can be used to evaluate the effectiveness of RDHB.

4. Field applications of RDHB

4.1. Condition of the 1210 D isolated coal pillar

The 1210 D isolated coal pillar in a coal mine was taken as engineering background to verify the destressing effect of RDHB for mitigating rock bursts in this research. The coal seam with 37.3 m thickness was divided into four slices for mining. The coarse sandstone and medium sandstone are distributed above the coal seam, and
their strength parameters range from 60 MPa to 80 MPa. As shown in Figure 14, the 1210 D isolated coal pillar is in the middle of the northern part. The northern, eastern and western parts of the 1210 D isolated coal pillar are all gob areas (including 1210 D, 1210, 1112, 1212, 1208 and 1108 panels). Due to the high expected risk of rock burst in 1210 D isolated coal pillar, some pre-destress measures including large-diameter drilling and coal seam blasting had been implemented in advance.

4.2. Identification of risk zones of rock burst in 1210 D isolated coal pillar

After the pre-destress measures are implemented in the 1210 D isolated coal pillar, the stress field distribution of the coal pillar should be detected before the formal production. Active seismic velocity tomography is widely used in the detection of stress distribution in coal and rock and its working principle is to use the travel time of induced seismic waves to invert the longitudinal seismic wave velocity distribution in the target area. According to the positive correlation between the stress and the longitudinal seismic wave velocity in the coal and rock mass, the higher the stress, the faster the longitudinal seismic wave propagation (Cai et al. 2014, He et al. 2011). Moreover, Gong et al. (2012) obtained the fitting formula of the correlation between wave velocity and stress from laboratory tests on rock samples.

\[ V_p = 2368.3(\sigma)^{0.1264} \]  \hspace{1cm} (13)

where, \( V_p \) is the longitudinal seismic wave velocity, and \( \sigma \) is the stress.

It is important to emphasize that the numerical relationship between wave velocity and stress might not be the same as that obtained from in situ measurement, which means the wave velocity obtained from active seismic velocity tomography was not converted to a specific stress value. However, a similar positive relationship exists and
can be used to detect the area of in-situ stress concentration. Therefore, the wave velocity anomaly coefficient can be calculated by the following formula:

$$A_n = \frac{V_p - V_p^a}{V_p^a}$$

where, $V_p^a$ is the average of longitudinal seismic wave velocity of all points in the model.

Figure 15 shows the wave velocity anomaly coefficient in the 1210 D isolated coal pillar. The results show that the 1210 D isolated coal pillar does not show the wave velocity anomaly in all regions, indicating that not the whole 1210 D coal pillar was in high-stress concentration. There are six zones that show wave velocity anomaly in the 1210 D isolated coal pillar which are marked A to F. These wave velocity anomaly zones are distributed in the boundary area of the coal pillar and have high risks of rock bursts. It is necessary to take some destress measures to ensure the safety of mining.

### 4.3. Scheme of RDHB for rock burst prevention and effect evaluation

Considering the site conditions and the feasibility of destress measures, RDHB was taken in zones A and B, coal seam destress blasting was implemented in zones C, D
and E, while no destress measures are designed for zone F, as shown in Figure 16. The detailed technology parameters of RDHB and coal seam destress blasting are shown in Tables 3 and 4, respectively. The explosive used in the coal mine is the emulsion explosive, and its performance parameters are listed in Case 3 in Table 1.

After the destress measures were implemented in wave velocity anomaly zones, the active seismic velocity tomography was performed again in the 1210 D isolated coal pillar. The distribution of wave velocity in 1210 D isolated coal pillar after the implementation of destress measures was obtained, as shown in Figure 17. Compared with the first active seismic velocity tomography test result shown in Figure 15, the second test result shows considerable variation in wave velocity anomaly distribution. To be specific, an obvious pressure relief area is formed in the middle of target area A. However, the stress concentration in area A is not completely eliminated, but driven to both sides of the target area to form two independent smaller stress concentration areas, which are consistent with the numerical simulation results. Then, the stress concentration in zone B and zone D was significantly weakened and driven away to the area further ahead, and the coal burst risk of the target area was thus lowered. The stress concentration in zone C and E did not change significantly, indicating that the pressure relief effect of destress measures in this area was not ideal. In zone F, the stress concentration did not change because no destress measure was conducted. The results show that

| Table 3. Technology parameters of RDHB for zones A and B. |
|---------------------------------|------------|-------------|-------------|----------------|-----------------|
| NO. | Length (m) | Angle (°) | Diameter (mm) | Charge (kg) | Sealing length (m) |
|-----|------------|---------|---------------|-------------|-------------------|
| A1  | 49         | 57      | 94            | 42          | >15               |
| A2  | 44         | 68      | 94            | 36          | >14               |
| A3  | 52         | 53      | 94            | 42          | >15               |
| B1  | 48         | 64      | 94            | 40          | >14               |
| B2  | 48         | 64      | 94            | 40          | >14               |
| B3  | 48         | 64      | 94            | 40          | >14               |

| Table 4. Technology parameters of coal seam destress blasting for zones C, D and E. |
|---------------------------------|------------|-------------|-------------|----------------|-----------------|
| NO. | Length (m) | Diameter (mm) | Charge (kg) | Sealing length (m) |
|-----|------------|---------------|-------------|-------------------|
| C   | 10         | 42            | 3           | >5                |
| D   | 10         | 42            | 5           | >5                |
| E   | 10         | 42            | 5           | >5                |

Figure 17. Distribution of wave velocity anomaly coefficient in 1210 D isolated coal pillar after destress measures.
RDHB can effectively release the elastic strain energy and lower the degree of the stress concentration in the target area. Moreover, compared with coal seam destress blasting, RDHB shows better performance in mitigating rock bursts.

5. Discussion

Destressing effect evaluation is an essential part of the application of RDHB for mitigating rock bursts. As discussed in the introduction, traditional methods, such as hydraulic pressure sensors, active seismic velocity tomography, drilling cuttings and electromagnetic radiation were successfully used to evaluate the destress effect of RDHB. However, the implementation of these programs requires a lot of mechanical equipment and technicians. Therefore, these methods have the disadvantages of complex operation, time-consuming and high cost. More importantly, these engineering operations will seriously interfere with mining production activities. It is necessary to propose more applicable and operable evaluation methods. As described in Chapter 3, the numerical simulation results show that blasting in rock will induce rock vibration, and there is a positive correlation between vibration intensity and the destressing effect. Therefore, blasting seismic parameters can be used to evaluate the destressing effect of RDHB. Moreover, since the microseismic monitoring system was widely applied in coal mines, the blasting seismic parameters are easy to obtain. This makes it possible to evaluate the destressing effect of RDHB based on blasting seismic parameters.

Sanchidrián et al. (2007) proposed the following energy balance equation for rock blasting:

\[ E_e = E_f + E_s + E_k + E_{NM} \]  

where, \( E_e \) is the explosive energy, \( E_f \) is the fragmentation energy, \( E_s \) is the seismic energy, \( E_k \) is the kinetic energy and \( E_{NM} \) is other energy forms not measured (all in J).

However, RDHB is practiced in the prestressed rock, and the rock will accumulate a certain amount of elastic strain energy under prestress. For blasting in prestressed rock, the energy balance equation can be given as follows:

\[ U_1 + E_e = U_2 + E_f + E_s + E_k + E_{NM} \]  

where, \( U_1 \) is the elastic strain energy in prestressed rock before blasting, \( U_2 \) is the elastic strain energy in the rock after blasting.
In general, the rock displacement is nearly zero in RDHB because there is no room for the broken rocks to eject, therefore, it is assumed that kinetic energy \((E_k)\) approximates zero (Konicek et al. 2013). Seismic energy is thus given as:

\[
E_s = E_e + \Delta U - E_f - E_{NM}
\]

where, \(\Delta U = U_1 - U_2\) is change of elastic strain energy in rock. Knotek et al. (1985) define a coefficient \(K\), which is related to natural conditions of the rock mass, to describe the relationship between explosive energy \(E_e\), fragmentation energy \(E_f\) and other energy forms not measured \(E_{NM}\).

\[
E_f + E_{NM} = KE_e
\]

Then, Formula (17) is modified as:

\[
E_s = (1 - K)E_e + \Delta U
\]

It can be concluded that the seismic energy is positively correlated with the change of elastic strain energy under constant blasting parameters and rock mass conditions. This suggests that using the blasting seismic parameters to evaluate the destressing effect is scientific and feasible in coal mines.

In this study, 118 rounds of RDHB for mitigating rock bursts in GJP coal mines were analyzed. As shown in Figure 18, these RDHBs present a group of fan-shaped holes, which are composed of three inclined cylindrical holes with different lengths. Among them, the length of the blast hole ranges from 43 m to 67 m, and the corresponding explosive charge ranges from 40 Kg to 80 Kg. Moreover, multiple initiation patterns were used in the implementation of RDHB. Some RDHBs were initiated individually, and some were initiated by two blast holes simultaneously. When RDHBs were performed, the microseismic monitoring system was used to record blasting-induced seismic and calculate the seismic parameters.

Figure 19 shows the relationship between the blasting charge and corresponding seismic energy of these RDHBs. This suggests that the seismic energy increases.
nonlinearly with the explosive charge. Moreover, the seismic energy distribution is relatively discrete under the same explosion charge, which indicates that the destressing effect of RDHB is different.

Kan et al. (2022) propose a blasting efficiency index to evaluate the blasting effect based on blasting seismic energy. This research result provides simulation verification and theoretical support for the application of this index.

The blasting efficiency index \( B_e \) is defined as the elastic strain energy of rock mass released per unit mass explosive. The blasting efficiency index \( B_e \) is thus given as:

\[
B_e = \frac{\log E_s - \log E_p}{\log P_e} \tag{20}
\]

\[
E_p = E_{pi} \times P_e \times k_s \tag{21}
\]

where, \( E_s \) is blasting seismic energy; \( E_p \) is the seismic energy converted from the explosive chemical energy; \( E_{pi} \) is the theoretical chemical energy per kilogram of explosive; \( P_e \) is the explosive charge; \( k_s \) is the blasting seismic energy conversion coefficient.

Therefore, the blasting efficiency indexes of the 118 rounds of RDHB were calculated based on Formulas 20 and 21. In this research, the value of \( E_{pi} \) and \( k_s \) are 3000 KJ/kg and 0.01%, respectively. The normal probability distribution of the blasting efficiency index \( B_e \) was shown in Figure 20. According to the classification system for the blasting efficiency index developed by Kan et al. (2022) and compared to their conclusions, the results show that the blasting efficiency index in this research is generally lower than their results. To be specific, the blasting destressing effect varied from ineffective to good, with 67% being general and approximately 22% good, while the ineffective is 11%. The main reasons are that the explosive charge in this research is markedly less than that of in their study. Especially, an ineffective effect accounts

Figure 20. Normal probability distribution of blasting efficiency index \( B_e \).
for a large proportion under the condition of the 40 Kg explosive charge. The result indicated that it is more likely to obtain a good destressing effect by increasing the explosive charge.

6. Conclusions

In this paper, the destressing mechanisms of RDHB were interpreted by theoretical analysis and evaluated by numerical models. RDHB was practiced in an isolated coal pillar and the change of stress field distribution before and after RDHB was detected to investigate the destressing effectiveness. Based upon the results, the following conclusions are drawn.

1. Based on the conceptual model of rock burst, the destressing mechanisms of RDHB were interpreted from the changes in strength and stiffness conditions in the roof-coal-floor system. The theoretical analysis results show the mechanisms of RDHB for mitigating rock bursts can be described in two aspects. RDHB reduces the local stress and the stress is not sufficient to cause coal failure, thereby reducing the potential of rock bursts. Moreover, the stiffness of the roof-coal-floor system changes fundamentally due to the blasting damage of roof rock, it will influence the energy transfer in the surrounding rock system. When the stiffness of roof rock was weakened to be less than that of coal, rock burst is less likely to occur.

2. Numerical models were built to evaluate the destressing effect of RDHB. A softening model based on the Hoek-Brown failure criterion was established to investigate the failure of rock subjected to blasting load and the evolution process of stress and energy in the rock after blasting. The simulated results show that the mechanical parameters of rock were weakened after blasting, including strength, stiffness and cohesion. The local stress of rock around the blasting hole was effectively reduced after blasting, accompanied by the release of elastic energy accumulated in the rock. Moreover, both the blasting intensity and rock medium properties have great influences on destressing effect of RDHB. A greater destressing effect may be obtained under a larger blasting intensity or blasting in moderately hard rock. The results also indicated that blasting vibration has a positive correlation with destressing effect, and the value can be used to evaluate the blasting destressing effect.

3. RDHB was practiced in an isolated coal pillar in a coal mine for mitigating rock bursts. The stress variation of the isolated coal pillar before and after RDHB was detected using the active seismic velocity tomography to validate the destressing effectiveness. The results show that RDHB can effectively reduce stress concentration in the target area and drive stress to the surrounding areas, thereby reducing the potential of rock bursts. Due to the high correlation between in situ measurement results and numerical results, the numerical evaluation of RDHB efficiency may be successfully applied to regular use.

4. An effect evaluation method for RDHB based on blasting seismic energy was discussed and applied. The ineffective effect of RDHB mainly occurs when the
explosive charge is 40 Kg. Although a small explosive charge may be enough to provoke a strong tremor under some conditions, it is more likely to obtain a good destressing effect by increasing the explosive charge.

Disclosure statement

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Data availability statement

The data presented in this study are available on request from the corresponding author.

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