Research Article

Study on Pressure Relief Zone Formed Inside Roadway Rib by Rotary Cutting with Pressurized Water Jet for Preventing Rock Burst

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Based on the engineering geological conditions of mining area 2502 in Yanbei coal mine as a background, the comprehensive index method is used to evaluate the rock burst risk of mining area 2502, and the burst tendency of coal seam 5# under natural and water-saturated conditions is also tested in laboratory. Then the pressurized water jet technology is proposed as an effective and feasible method to prevent rock burst. By means of theoretical analysis, the coal-rock mass system has three strain-stress states under the influence of static load, and it has another strain-stress state under the influence of static and dynamic combined load; meanwhile, the greater the horizontal tectonic stress caused by fold structure is, the more prone to instability and failure and induce rock burst coal mass composed of countless cell cubes is; when a pressure relief zone is formed inside the roadway rib by using pressurized water jet technology, the initial single peak stress curve will change to a final bimodal stress curve inside the roadway rib. A “strong-weak-strong” prevention structure can be formed inside the roadway rib and this prevention structure can well prevent the occurrence of rock burst. By means of numerical simulation analysis, the optimal diameter, length, and interval of rotary cutting sections are identified as 400 mm, 3.0 m, and 15.0 m, respectively. By means of on-site industrial application analysis, all these monitoring results verify that when the pressurized water jet is used to conduct rotary cutting of roadway rib, a large-scale pressure relief zone will be formed in the coal mass inside the roadway rib, and it can achieve the purpose of preventing rock burst in the roadway rib. The research results can effectively provide theoretical foundation and a new guidance for preventing rock burst with similar engineering geological conditions.

1. Introduction

Rock burst is a strong dynamic phenomenon in which the coal and rock mass around the driving or mining space suddenly destroys and releases a lot of energy. It is one of the most destructive dynamic disasters in coal mine, which is easy to cause heavy casualties [1–3]. In recent years, due to the large-scale and high-intensity mining of coal mines in China, rock burst disaster has become one of the main disaster forms restricting the safe and efficient production of coal mines. With the increasing mining intensity of coal mines in China, the distribution range of mines with rock burst is becoming wider and wider. At present, rock burst has occurred in more than 100 coal mines in China, which seriously affects the normal production of coal mines. The research on the occurrence mechanism, monitoring, early
warning, and prevention of rock burst is not only a worldwide problem but also an urgent scientific and technological problem to be solved [4–6].

The engineering practices show that the natural geological conditions and mining technical conditions of coal and rock mass have a great impact on the occurrence of rock burst. For example, in geological structure areas such as fault area, fold area, and sudden change area of coal seam thickness, often due to unreasonable roadway layout or mining layout of working face, the high energy accumulation in coal and rock caused by mining poses a threat to rock burst. These areas are prone to induce rock burst [7–10]. Fold structure is a geological phenomenon left by crustal movement. The special shape of fold strata formed by crustal movement, the distribution of tectonic stress, and the local stress concentration caused by mining result in the characteristics of rock burst disaster in fold area, which is different from other types of rock burst disasters and general structural rock burst. Tan et al. [11] analyzed the distribution characteristics of structural stress field and rock burst in Mentougou mine field and discussed that, in the syncline structure, the structural principal stress is concentrated, which is easy to accumulate a large amount of elastic energy and rock burst triggered by mining and other factors, so as to put forward preventive ways such as reasonable roadway layout and mining procedure. Qi et al. [12–14] analyzed a large number of measured data samples in Taozhuang coal mine, explored the relationship between the occurrence of rock burst and tectonic stress and the source of rock burst pressure in Taozhuang coal mine from the perspective of tectonic stress, and considered that attention should be paid to the relationship between the advancing direction of working face and the direction of tectonic stress in mining. Lee et al. [15] believed that there is high residual tectonic stress in the fold area, which will lead to stress release and rock burst during mining. Gao et al. [16] established the catastrophe model of rock burst of different media contact type in the fold area, and it is shown that the movement path in the control plane determined by horizontal force and vertical force has an obvious impact on the stability of roof and floor and coal seam system.

At present, the research on the prevention and control of rock burst mainly depends on the pressure relief methods, such as coal seam pressure relief blasting, coal seam high-pressure water injection, and large-diameter borehole pressure relief [17–19]. However, these methods have the problems that the pressure relief effect is not obvious or the pressure relief is excessive. The pressurized water jet technology is a new pressure relief method, and this method has the advantages of little bit wear, no spark, no dust, and so on [20, 21], so it can be used to study the pressure relief effect of roadway rib in the area affected by fold structure.

2. The Engineering and Geological Conditions

2.1. The Typical Geological Conditions. Huayan coal field, located in the north of Huating County, Gansu Province, has typical geological structural characteristics. The whole coal fields are mainly affected by fold geological structure, and several syncline and anticline axes pass through them. The plan of whole coal fields is shown in detail in Figure 1.

According to Figure 1, the whole coal fields are located in the southwest edge of Ordos block or the fault fold belt on the west edge of Ordos block, which belong to Shaanxi-Gansu-Ningxia basin. All these coal fields have long been affected by the typical geological conditions of this area, especially the compressive tectonic stress in the southwest-northeast and east-west. Under the influence of this compressive tectonic stress for a long time, the movement of tectonic plates in this area is very intense, and a typical fold geological structure is formed in this area. Among all these coal fields, Huayan coal field is located in the middle of the whole coal field and its division area is relatively large. A syncline structure axis and an anticline structure axis pass through this coal field longitudinally from north to south, resulting in the serious influence of tectonic stress during the mining activities in this coal field.

2.2. The Typical Engineering Conditions. Yanbei coal mine mainly exploits Huayan coal field, and the strong mine pressure events and even rock burst events in mining area 2502 of Yanbei coal mine appear continuously during the mining stage. Therefore, it is necessary to further analyze the occurrence mechanism of strong mine pressure events or rock burst events during these working faces driving and mining stages. The north side of mining area 2502 is mining area 1504 and Shanzhai coal mine, and the south side of mining area 2502 is a coal pillar with 100 m width, the second level auxiliary transportation roadway, belt roadway, and return air roadway in turn. Mining area 2502 is bounded by coal pillar with mining area 2501. Mining area 2502 is adjacent to mining area 1503 in the east and mining area 2503 in the west. The plan of mining area 2502 of Yanbei coal mine is shown in detail in Figure 2.

According to Figure 2, mining area 2502 is located at the level of +860 m∼+1175 m, and its corresponding ground level elevation is +1350 m∼+1425 m. The average burial depth of mining area 2502 is about 390 m, which means that the burial depth of this mining area is not large.

Consider that the static load in the coal and rock mass around the driving and mining space is composed of ground stress and bearing stress, as shown in the following equation:

\[ \sigma_s = \sigma_{s1} + \sigma_{s2}, \]  

(1)

where \( \sigma_{s1} \) is the ground stress, MPa; \( \sigma_{s2} \) is the bearing stress, MPa.

It is assumed that the average unit weight of the overlying strata of minable coal seam 5# in this mining area is determined; the corresponding self-weight stress can be calculated out, as shown in the following equation:

\[ \sigma_{s1} = \gamma_1 \cdot H_1, \]  

(2)

where \( \gamma_1 \) is the average unit weight of the overlying strata, kN/m³; \( H_1 \) is the burial depth of minable coal seam 5#, m.
Due to the fact that the burial depth of this mining area is not large, the corresponding self-weight stress value is also small.

The tectonic stress can also be calculated out, as shown in the following equation:

$$\sigma_{s1} = \lambda_2 \cdot \gamma_1 \cdot H_1 = \lambda_2 \cdot \sigma_{s1},$$  \hspace{1cm} (3)

where $\lambda_2$ is the horizontal stress coefficient.

As can be seen from equation (3), when the self-weight stress is fixed, the tectonic stress is positively correlated with horizontal stress coefficient. However, due to the serious influence of tectonic stress, the corresponding horizontal stress coefficient is relatively large, so the tectonic stress value is also relatively large. According to equations (2) and (3), the tectonic stress plays a dominant role in static load. Namely, even though the buried depth of coal seam 5# is small, the static load can still reach a large value under the serious influence of tectonic stress.

2.3. The Risk Assessment of Rock Burst. The thickness of minable coal seam 5# in mining area 2502 is 17.6 m~50.4 m, and its average thickness is about 32.0 m. Considering the great thickness of coal seam 5#, the layered mining method can be used to mine this coal seam. The coal seam 5# can be divided into two layers: upper layer and lower layer. The direct mining height of this layer is 3.0 m, and the corresponding top coal caving height is 9.0 m. The ratio between direct mining height and top coal caving height is about 1:3. The coal seam dip is 8°~17°, and its average dip is about 12°. According to Figure 2, it can be seen that panel 250206 is in
the mining stage, and panel 250204 is in the driving stage at the same time. During the mining stage of panel 250206, the strong mine pressure events and even rock burst events appear continuously in this stage. Therefore, the comprehensive index method is used to evaluate the rock burst risk of mining area 2502. The evaluation indicators of comprehensive index method can be divided into two parts, and they are the mining geological indicator and the mining technology indicator, respectively [22].

The evaluation result of mining geological indicators is shown in Table 1.

According to Table 1, the evaluation result of mining geological indicators \((W_{t1})\) can be determined as 0.76.

The evaluation result of mining technology indicators is shown in Table 2.

According to Table 2, the evaluation result of mining technology indicators \((W_{t2})\) can be determined as 0.57.

The final evaluation result of rock burst risk of mining area 2502 can be determined by the combined evaluation results of mining geological indicators and mining technology indicators, as shown in the following equation:

\[
W_t = \max\{W_{t1}, W_{t2}\}.
\] (4)

According to equation (4), the final evaluation result of rock burst risk of mining area 2502 can be determined as 0.76. The final evaluation results and the corresponding dangerous levels are shown in Table 3.

According to the evaluation result of the above comprehensive index method and in combination with Table 3, it can be seen that the danger level of mining area 2502 is strong burst danger. Meanwhile, the coal samples were drilled from the coal seam 5# in mining area 2502 and tested in the laboratory with relevant qualifications, and the test results of burst tendency can be obtained in Table 4.

It can be seen from Table 4 that the coal seam 5# has strong burst tendency under natural condition; however, the burst tendency of coal seam 5# will change into weak burst tendency under water-saturated condition.
feasible method. A pressurized water jet for prevention of rock burst is an effective and practical tool. It can be seen that pressurized water can effectively cut coal mass, and the cutting capacity can be adjusted by changing the water pressure. Meanwhile, the pressurized water jet can also cut the coal mass around driving or mining space, and it will aggravate the instability and failure of coal mass. It is necessary to take some effective measures to prevent rock burst in the subsequent mining process. kX._he water jet is an effective and practical tool for prevention of rock burst in mining area 2502. Taking the roadway with relatively weak support system as an example, it can be seen that the occurrence mechanism of rock burst in the roadway rib is shown in Figure 3.

According to Figure 3(a), it can be seen that the high stress concentration area can be formed in the ribs when a roadway is excavated. The coal mass in the high stress concentration area is in a relatively unstable state. When the concentrated stress is large enough, the coal mass in this area will be unstable and damaged. According to Figure 3(b), it is assumed that the high stress concentration area is coal pillar A, and the vertical concentrated stress along the x-axis applied to coal pillar A can be expressed as equation (5), which is analyzed from the perspective of plane strain.

\[
p_{vc} = f(y_1, t),
\]

where \(y_1\) is the displacement of coal pillar A, \(m\); \(t\) is the time required for displacement of coal-rock mass system (composition of immediate roof and coal pillar A), \(s\).

Then the cumulative value of vertical concentrated stress applied to coal pillar A can be determined, as shown in the following equation:

\[
P_i = \int_{x_i}^{x_2} p_{vc} = \int_{x_i}^{x_f} f(y_1, t),
\]
To determine the stress of the immediate roof, the interval $x_1$ to $x_3$ is the width of coal pillar $A$, $m$.

The stress of the overlying strata along the $x$-axis applied to immediate roof can be expressed as follows:

$$ P_y = M \frac{d^2 y_2}{dt^2} + k(y_2 - y_1). \quad (7) $$

Then the cumulative value of overlying strata stress applied to corresponding immediate roof above coal pillar $A$ can be determined, as shown in the following equation:

$$ P_2 = \int_{x_1}^{x_2} P_y = \int_{x_1}^{x_2} M \frac{d^2 y_2}{dt^2} + k(y_2 - y_1), \quad (8) $$

where $M$ is the average mass of immediate roof, $\text{kg/m}$; $k$ is the stiffness of immediate roof, $\text{kN/m}^2$; $y_2$ is the displacement of immediate roof, $m$.

From the perspective of energy, it can be seen that, in order to maintain the balance of coal-rock mass system, the energy accumulated in the immediate roof above coal pillar $A$ ($E_2$) must be less than that accumulated in the coal pillar $A$ ($E_1$), as shown in the following equation:

$$ E_1 \geq E_2. \quad (9) $$

### Table 2: The evaluation result of mining technology indicators.

| Number | Indicators | Mining technology indicators | Indicator classification | Indicator scores | Evaluation scores |
|--------|------------|-------------------------------|-------------------------|-----------------|------------------|
| 1      | $W_1$     | Relationship between working face and adjacent gob ($A_2$/side) | $A_2 = 0$ | 0 | 1 |
|        |           |                               | $A_2 = 1$ | 1 | |
|        |           |                               | $A_2 = 2$ | 2 | |
|        |           |                               | $A_2 \geq 3$ | 3 | |
| 2      | $W_2$     | Width of working face ($B_2$/m) | $B_2 > 300$ | 0 | 1 |
|        |           |                               | $150 \leq B_2 < 300$ | 1 | |
|        |           |                               | $100 \leq B_2 < 150$ | 2 | |
|        |           |                               | $B_2 \leq 100$ | 3 | |
| 3      | $W_3$     | Width of stage coal pillar ($C_2$/m) | $C_2 \leq 6$ | 0 | 1 |
|        |           |                               | $6 \leq C_2 < 10$ | 1 | |
|        |           |                               | $10 \leq C_2 \leq 50$ | 2 | |
| 4      | $W_4$     | Thickness of bottom coal ($D_2$/m) | $D_2 = 0$ | 0 | 1 |
|        |           |                               | $0 < D_2 \leq 1$ | 1 | |
|        |           |                               | $1 < D_2 < 2$ | 2 | |
|        |           |                               | $D_2 \geq 2$ | 3 | |
| 5      | $W_5$     | Distance between the mining face or driving face and the syncline or anticline with a sharp change of dip angle ($E_2$/m) | $E_2 \geq 50$ | 0 | 1 |
|        |           |                               | $20 \leq E_2 < 50$ | 1 | |
|        |           |                               | $10 \leq E_2 < 20$ | 2 | |
|        |           |                               | $E_2 < 10$ | 3 | |
| 6      | $W_6$     | Distance between the mining face or driving face and the faults with vertical drop greater than 3 m ($F_2$/m) | $F_2 \geq 50$ | 0 | 1 |
|        |           |                               | $20 \leq F_2 < 50$ | 1 | |
|        |           |                               | $10 \leq F_2 < 20$ | 2 | |
|        |           |                               | $F_2 < 10$ | 3 | |
| 7      | $W_7$     | Distance between the mining face or driving face and the location of coal seam erosion, coalescence, or thickness change ($G_2$/m) | $G_2 \geq 50$ | 0 | 1 |
|        |           |                               | $20 \leq G_2 < 50$ | 1 | |
|        |           |                               | $10 \leq G_2 < 20$ | 2 | |
|        |           |                               | $G_2 < 10$ | 3 | |
|        |           |                               | $W_{12} = (nW_i/nW_{max})$ | 0.57 | |

Note. $W_i$ represents the evaluation score of number $i$; $W_{max}$ represents the maximum indicator score of number $i$.

### Table 3: The final evaluation results and the corresponding dangerous levels.

| Final evaluation results | $W_i \leq 0.25$ | $0.25 < W_i \leq 0.5$ | $0.5 < W_i \leq 0.75$ | $W_i > 0.75$ |
|--------------------------|------------------|-----------------------|-----------------------|-------------|
| Danger levels            | No burst danger (A) | Weak burst danger (B) | Medium burst danger (C) | Strong burst danger (D) |

### Table 4: The test results of burst tendency of coal seam 5#.

| State                      | $D_f$/ms | $W_{ET}$ | $K_E$ | $R_s$/MPa | Burst tendency results |
|----------------------------|----------|----------|-------|-----------|------------------------|
| Natural condition          | 152      | 4.58     | 11.26 | 20.1      | Strong burst tendency  |
| Water-saturated condition  | 473      | 2.08     | 6.8   | 9.8       | Weak burst tendency    |

Note. $D_f$ represents the dynamic failure time; $W_{ET}$ represents the energy index; $K_E$ represents the burst energy index; $R_s$ represents the uniaxial compressive strength.
3.1.1. The Influence of Static Load. It is assumed that the coal pillar \( A \) is mainly affected by static load, and the acceleration of immediate roof movement is zero, as shown in the following equation:

\[
\frac{d^2 y}{dt^2} = 0. \tag{10}
\]

Assuming that the displacement of immediate roof is zero and the displacement increment in coal pillar \( A \) increases by \( \Delta y_1 \), both the cumulative value of vertical concentrated stress applied to coal pillar \( A \) \( (P_1) \) and the cumulative value of overlying strata stress applied to corresponding immediate roof above coal pillar \( A \) \( (P_2) \) will change, and their corresponding stress increments are shown in the following equation:

\[
\begin{align*}
\Delta P_1 &= f'(y_1, t) \cdot \Delta y_1 = \frac{df(y_1, t)}{dy_1} \cdot \Delta y_1, \\
\Delta P_2 &= -k \cdot \Delta y_1,
\end{align*} \tag{11}
\]

where \( \Delta P_1 \) is the stress increment of cumulative value of vertical concentrated stress applied to coal pillar \( A \) under static load, MPa; \( \Delta P_2 \) is the stress increment of cumulative value of overlying strata stress applied to corresponding immediate roof above coal pillar \( A \) under static load, MPa.

At this moment, the energy accumulated in the coal pillar \( A \) and immediate roof above coal pillar \( A \) will change, respectively, and the final energy value is shown in the following equation:

\[
\begin{align*}
E_{11} &= \left( P_1 + \frac{1}{2} \Delta P_1 \right) \cdot \Delta y_1, \\
E_{22} &= \left( P_2 + \frac{1}{2} \Delta P_2 \right) \cdot \Delta y_1,
\end{align*} \tag{12}
\]

where \( E_{11} \) is the final energy accumulated in coal pillar \( A \) under static load, kJ; \( E_{22} \) is the final energy accumulated in immediate roof above coal pillar \( A \) under static load, kJ.

Combining equations (9), (11), and (12), the balance equation of coal-rock mass system can be obtained as follows:

\[
k + f'(y_1, t) \geq 0. \tag{13}
\]

According to equation (13), it can be seen that there are three states for coal-rock mass system:

(1) When the coal-rock mass system meets the following conditions (as shown in equation (14)), the coal pillar \( A \) is in an elastic state, and the coal-rock mass system is stable.

\[
\begin{align*}
&k + f'(y_1, t) > 0, \\
&\frac{df(y_1, t)}{dy_1} > f'(y_1, t), \quad k > 0.
\end{align*} \tag{14}
\]

(2) When the coal-rock mass system meets the following conditions (as shown in equation (15)), the coal pillar \( A \) is in a residual strength state (I). At this time, the coal pillar \( A \) is gradually damaged and its strength is gradually reduced. The coal-rock mass system is metastable, and the failure process of coal pillar \( A \) is static failure.

\[
\begin{align*}
&k + f'(y_1, t) > 0, \\
&\frac{df(y_1, t)}{dy_1} = f'(y_1, t), \quad k > 0.
\end{align*} \tag{15}
\]

(3) When the coal-rock mass system meets the following conditions (as shown in equation (16)), the coal pillar \( A \) is in a residual strength state (II). In such case, however, the coal-rock mass system has a sudden dynamic failure. The failure process of coal pillar \( A \) is brittle failure and it has a sudden change in intensity.

\[
\begin{align*}
&k + f'(y_1, t) < 0, \\
&\frac{df(y_1, t)}{dy_1} = f'(y_1, t), \quad k > 0.
\end{align*} \tag{16}
\]

The failure process of coal pillar \( A \) is accompanied by the sudden release of energy, that is, the amount of energy.
released in the rock burst start-up, as shown in the following equation:

\[
\Delta E = E_{11} - E_{22} = \frac{1}{2} \Delta y_1^2 \cdot \left( f' \left( y_1, t \right) + k \right).
\] (17)

The corresponding strain-stress curves of coal-rock mass system are shown in Figure 4.

3.1.2. The Influence of Static and Dynamic Combined Load. It is assumed that the coal pillar \( A \) is mainly affected by static and dynamic combined load, and the acceleration of immediate roof movement is not zero, as shown in the following equation:

\[
\frac{d^2 y_2}{dt^2} \neq 0.
\] (18)

Assuming that the displacement of immediate roof is zero, the displacement increment in coal pillar \( A \) also increases by \( \Delta y_1 \), and the immediate roof has an accelerated movement, in such case, both the cumulative value of vertical concentrated stress applied to coal pillar \( A \) (\( P_1 \)) and the cumulative value of overlying strata stress applied to corresponding immediate roof above coal pillar \( A \) (\( P_2 \)) will change, and their corresponding stress increments are shown in the following equation:

\[
\begin{align*}
\Delta P'_1 &= f' \left( y_1, t \right) \cdot \Delta y_1 = \frac{d f_1 \left( y_1, t \right)}{dy_1} \cdot \Delta y_1, \\
\Delta P'_2 &= -k \cdot \Delta y_1 - M_1 \cdot \frac{d^2 y_2}{dt^2},
\end{align*}
\] (19)

where \( \Delta P'_1 \) is the stress increment of cumulative value of vertical concentrated stress applied to coal pillar \( A \) under static and dynamic combined load, MPa; \( \Delta P'_2 \) is the stress increment of cumulative value of overlying strata stress applied to corresponding immediate roof above coal pillar \( A \) under static and dynamic combined load, MPa.

At this moment, the energy accumulated in the coal pillar \( A \) and immediate roof above coal pillar \( A \) will change, respectively, and the final energy value is shown in the following equation:

\[
\begin{align*}
E'_{11} &= \left( P_1 + \frac{1}{2} \Delta P'_1 \right) \cdot \Delta y_1, \\
E'_{22} &= \left( P_2 + \frac{1}{2} \Delta P'_2 \right) \cdot \Delta y_1,
\end{align*}
\] (20)

where \( E'_{11} \) is the final energy accumulated in coal pillar \( A \) under static and dynamic combined load, kJ; \( E'_{22} \) is the final energy accumulated in immediate roof above coal pillar \( A \) under static and dynamic combined load, kJ.

Combining equations (9), (19), and (20), the balance equation of coal-rock mass system can be obtained as follows:

\[
k - M_1 \cdot \frac{d^2 y_2}{dt^2} \cdot \frac{1}{\left( \Delta y_1 \right)^2} + f' \left( y_1, t \right) \geq 0.
\] (21)

Because the immediate roof has an accelerated movement, the stiffness of immediate roof is reduced from \( k \) to \( k' \). At this time, the stiffness of immediate roof is as follows:

\[
k' = k - M_1 \cdot \frac{d^2 y_2}{dt^2} \cdot \frac{1}{\left( \Delta y_1 \right)^2}
\] (22)

In this case, compared with the case when the acceleration of immediate roof is zero, the coal pillar \( A \) is more likely to be in an unstable state; that is,

\[
k' + f' \left( y_1, t \right) < 0.
\] (23)

At this time, a rock burst event is more likely to occur, and its strength is more violent. The corresponding amount of energy released in the rock burst start-up is about \((1/2)M_1 \cdot (d^2 y_2/dt^2)\) more than that calculated by equation (17).

The corresponding strain-stress curves of coal-rock mass system are shown in Figure 5.

3.1.3. The Influence of Horizontal Tectonic Stress. Based on the analysis in Sections 3.1.1 and 3.1.2, it can be seen that coal mass in high stress concentration area may induce rock burst under the influence of static load or static and dynamic combined load. Meanwhile, the horizontal tectonic stress can further promote the instability and failure of coal mass. The influence of horizontal tectonic stress caused by folded structure on coal seam is shown in Figure 6.

As can be seen from Figure 6, the coal seam can be divided into three zones under the influence of horizontal tectonic stress caused by folded structure, namely, syncline structure influence zone (I), wing structure influence zone (II), and anticline structure influence zone (III). Any cell cube \( A \) in zone I is subject to vertical compressive stress and horizontal tensile stress, and this zone can be defined as a zone of maximum mine ground pressure distribution, which is most prone to roof caving or rock burst; any cell cube \( B \) in zone II is subject to vertical compressive stress and horizontal compressive stress, which is prone to rock burst; any cell cube \( C \) in zone III is subject to vertical tensile stress and horizontal compressive stress, and this zone can also be defined as a zone of maximum mine ground pressure distribution [24]. Considering the stress state of coal mass in zones I and III, the coal mass is broken and some elastic energy is released in advance due to the action of tensile stress, resulting in the weak appearance degree of rock burst compared with coal mass in zone II. However, the occurrence probability of rock burst is higher than that in zone II.

Because the vertical stress and horizontal stress of coal mass in zone II are compressive stress, the stress concentration degree is high, and there is high energy accumulated in the corresponding coal mass. Under the influence of the disturbance of mining activities, the energy accumulated in the coal mass is mainly released to the mining space in the form of kinetic energy, which is the maximum rock burst dangerous zone. Taking cell cube \( B \) in zone II as the research object, its stress state is three-dimensional compression. Xie et al. [25] put forward that when the energy release rate of
any cell cube B in zone II reaches its critical maximum, this cell cube B will be damaged as a whole, and the energy release rate can be expressed as follows:

$$R_E = \frac{1}{2E} \cdot (\sigma_1 - \sigma_3) \cdot \left[ \sigma_1^2 + \sigma_2^2 + \sigma_3^2 - 2\mu(\sigma_1\sigma_2 + \sigma_2\sigma_3 + \sigma_3\sigma_1) \right],$$

(24)

where $E$ is the elasticity modulus of cell cube B, MPa; $\mu$ is the Poisson ratio of cell cube B; $\sigma_1$, $\sigma_2$, and $\sigma_3$ are maximum principal stress, intermediate principal stress, and minimum principal stress, respectively, MPa.

According to the principle of minimum energy for dynamic failure of cell cube B proposed by Zhao et al. [26], the energy consumed in the failure of cell cube B is the same as that required in the failure under unidirectional stress. Therefore, the energy release rate of unidirectional stress compression of cell cube B can be used to replace the energy release rate of three-dimensional stress compression of cell cube B. Therefore, from the perspective of energy equivalence, the following equation can be obtained:

$$\frac{1}{2E} \cdot (\sigma_1 - \sigma_3) \cdot \left[ \sigma_1^2 + \sigma_2^2 + \sigma_3^2 - 2\mu(\sigma_1\sigma_2 + \sigma_2\sigma_3 + \sigma_3\sigma_1) \right] = \frac{\sigma_r^3}{2E},$$

(25)

where $\sigma_r$ is the uniaxial compressive strength of cell cube B, MPa.

According to Figure 7, assuming that $\sigma_x$, $\sigma_y$, and $\sigma_z$ correspond to $\sigma_1$, $\sigma_2$, and $\sigma_3$ respectively, according to the generalized Hooke’s law, the strain along the $y$-axis satisfies the following equation from the perspective of plane strain:

$$\varepsilon_y = \frac{1}{E} \cdot \left[ \sigma_y - \mu(\sigma_x + \sigma_z) \right] = 0.$$

(26)

The ratio of $\sigma_x$ and $\sigma_z$ is defined as coefficient $\lambda$, and it satisfies $\lambda = (\sigma_x/\sigma_z)$. Because the horizontal compressive stress is usually higher than that in the vertical direction under the influence of horizontal tectonic stress, the coefficient $\lambda$ meets $0 < \lambda < 1$. 
The critical maximum horizontal stress \( \sigma_r \) calculated according to equation (27), as shown in Figure 7. The variation curves of critical maximum horizontal stress required for the overall failure of any cell cube B in zone II under different parameter conditions.

Combining equations (25) and (26), the critical maximum horizontal stress required for the overall failure of any cell cube B in zone II can be calculated as follows:

\[
\sigma_{x_{\text{max}}} = \frac{\sigma_r}{\sqrt{(1 - \lambda) \cdot \left[ \left(1 - \mu^2\right) \cdot \lambda^2 - 2\mu \cdot (1 + \mu) \cdot \lambda + 1 - \mu^2 \right]}}.
\]

(27)

The variation curves of critical maximum horizontal stress required for the overall failure of any cell cube B in zone II under different parameter conditions can be calculated according to equation (27), as shown in Figure 7.

It can be seen from Figure 7 that the critical maximum horizontal stress required for the overall failure of any cell cube B in zone II decreases exponentially with the decrease of coefficient \( \lambda \); that is, the increase of horizontal compressive stress will lead to the decrease of the critical maximum horizontal stress required for the overall failure of any cell cube B in zone II. This means that the increase of horizontal compressive stress caused by strong tectonic stress will further reduce the critical maximum horizontal stress required for the overall failure of any cell cube B in zone II, and then the coal mass composed of countless cell cubes B is more prone to instability and failure and induce rock burst. Meanwhile, with the increase of Poisson ratio of cell cube B, the critical maximum horizontal stress required for the overall failure of any cell cube B in zone II will also increase.

In a word, the greater the horizontal tectonic stress caused by fold structure, the smaller the corresponding coefficient \( \lambda \). Under the smaller coefficient \( \lambda \), the coal mass composed of countless cell cubes B is more prone to instability and failure and induce rock burst.

3.2. The Prevention Mechanism of Rock Burst

3.2.1. The Analysis from the Change of Stress.

Considering that the pressurized water jet for prevention of rock burst is an effective and feasible method, the pressurized water jet is used to conduct rotary cutting of roadway rib and pressure relief zone is formed inside the roadway rib. The prevention mechanism of rock burst in roadway rib is shown in Figure 8.

According to Figure 8, it can be seen that when a pressure relief zone is formed inside the roadway rib, the initial single peak stress curve will change to a final bimodal stress curve inside the roadway rib. The initial peak stress \( \sigma_a \) at point \( a \) also changes to final peak stresses \( \sigma_b \) and \( \sigma_c \) at points \( b \) and \( c \), and it satisfies that peak stress \( \sigma_b \) is less than peak stress \( \sigma_a \) [27]. Considering the transfer and release effect of highly concentrated stress in the pressure relief zone, the final peak stresses \( \sigma_b \) and \( \sigma_c \) at points \( b \) and \( c \) are both less than peak stress \( \sigma_a \) at point \( a \). Therefore, the corresponding cumulative value of vertical concentrated stress near point \( c \) applied to coal pillar \( m \) \( (P_m) \) will not easily lead to the instability and failure of coal pillar \( m \) and induce rock burst. At this time, the coal-rock mass system is more likely to be in the state shown in Figures 4(a) and 4(b). It can be seen that this method has a good preventive effect on the influence of static load, and the coal-rock mass system is not easy to be in the state shown in Figure 4(c). When
severely affected by mining activities, the coal pillar $m$ will be affected by the static and dynamic combined load, and the coal-rock mass system is more likely to be in the state shown in Figure 5.

3.2.2. The Analysis from the Change of Energy. Whether under the influence of static load or static and dynamic combined load, the occurrence process of rock burst in roadway rib is shown in Figure 9.

As can be seen from Figure 9, when the energy accumulated in coal pillar $m$ ($E_{pm}$) is larger than the critical energy required for the instantaneous failure of coal pillar $m$ ($E_{n}^{\prime}$), the coal pillar $m$ is activated and then, in the start-up stage, the excess energy released in the start-up stage ($E_{pm}^{\prime}$) will propagate towards the roadway space with the elastic stress wave as the carrier; the pressure relief zone can consume some energy ($E_a$) and can be seen as a prevention stage (I), and the excess energy released in the prevention stage (I) ($E_{pm}^{\prime}$) is superimposed with the energy accumulated in coal pillar $n$; if the energy accumulated in coal pillar $n$ ($E_{pn}$) is greater than the critical energy required for the instantaneous failure of coal pillar $n$ ($E_{n}^{\prime}$) and the critical energy that the surrounding rock supporting structure can bear ($U_s$), a rock burst event will take place in the roadway rib. This stage can be seen as a prevention stage (II), and the severity of a rock burst event is determined by the final excess energy released in the occurrence stage ($\Delta E$). In the induced process of rock burst, when the excess energy released in any stage cannot meet the conditions shown in Figure 9, a rock burst event will not occur.

In a word, a "strong-weak-strong" prevention structure can be formed inside the roadway rib when the pressurized water jet is used to conduct rotary cutting of roadway rib, and this prevention structure can well prevent the occurrence of rock burst.

4. The Numerical Simulation of Pressure Relief Zone

4.1. The Forming Process of Pressure Relief Zone. In order to establish a 3D model for the numerical simulation of pressurized water jet parameters, the rotary cutting process of roadway rib by pressurized water jet is expounded first. The schematic diagram of pressure relief zone forming after rotary cutting by pressurized water jet is shown in Figure 10.

The specific rotary cutting process of pressure relief zone is shown in Figure 11.

According to Figures 10 and 11, the formation process of pressure relief zone inside a roadway rib can be roughly divided into the following steps:

(a) A conventional drill bit is used to drill an ordinary mechanical borehole on the roadway rib, and the diameter and length of this ordinary mechanical borehole are defined as $d$ and $L$ respectively.

(b) The conventional drill bit is replaced by a new drill bit with three jet nozzles, and then the pressurized water jets from the three jet nozzles cut the initial borehole wall by rotation mode. The diameter and length of rotary cutting section (the pressure relief zone) are defined as $D$ and $l_2$, respectively. The length of protective section is defined as $l_1$, and the sum of $l_1$ and $l_2$ is $L$.

(c) The pressure of water supply can be adjusted through the pressure regulating valve, which leads to the pressure change of water jets. The different pressurized water jets have different cutting efficiency to the initial borehole wall, which affects the diameter of rotary cutting section ($D$).

According to Figure 10, the frequently used model of drill bit with three jet nozzles is ZJN94/3, and its matching diameters of the jet nozzles ($d_0$) are selected from 1.6 mm, 1.8 mm, 2.0 mm, and 2.5 mm, respectively. The pressure-flow test results of different jet nozzles in laboratory are shown in Figure 12.

As the uniaxial compressive strength of coal seam 5# is about 16.2 MPa, the corresponding diameters of jet nozzles that can be selected are 1.6 mm or 1.8 mm. However, when the pressure of water jet is a certain value, the larger the diameter of jet nozzle is, the larger the corresponding flow is. As the larger flow is conducive to the return water in the borehole to carry the coal cinders, the final diameter of jet nozzle is 1.8 mm.
**Figure 9:** The occurrence process of rock burst in roadway rib.

**Figure 10:** The schematic diagram of pressure relief zone forming after rotary cutting by pressurized water jet.

**Figure 11:** The schematic diagrams of rotary cutting process by pressurized water jet. (a) The vertical view; (b) the front view. Note: 1, jet nozzle; 2, drill; 3, pressurized water jet; 4, initial borehole wall.
the roadway rib, compared with the peak stress position of numerical simulation result, the drilling cuttings method well verifies the rationality of this 3D model.

4.4. The Parameters Optimization of Pressure Relief Zone. When the pressurized water jet is used to conduct rotary cutting of roadway rib, a series of rotary cutting sections will be formed inside roadway rib. However, diameter \((D)\), length \((l)\), and interval \((X)\) of rotary cutting sections need to be optimized, so as to better relieve the pressure inside the roadway rib and achieve the effect of preventing rock burst.

The maximum effective cutting length of pressurized water jet is mainly related to the diameter of rotary cutting sections, and it can be calculated by the following empirical formula when the pressure of water jet is greater than 15 MPa [28]:

\[
r_e = \frac{(200 \sim 300) \cdot d_r}{2}
\]  

(28)

It can be seen from the previous study that the final diameter of jet nozzles is 1.8 mm, and the maximum effective cutting length of pressurized water jet is 180 mm–270 mm by substituting it into equation (28). Take the middle value here, which is about 225 mm. Based on the effective cutting length of pressurized water jet, it is assumed that the corresponding diameter of rotary cutting sections is taken as 200 mm, 300 mm, or 400 mm, respectively. The interval of rotary cutting sections is taken as 3.0 m, 4.0 m, and 5.0 m, respectively. The numerical simulation schemes for different diameters and intervals are shown in Table 6.

According to the numerical simulation schemes for different diameters and intervals in Table 6, the specific numerical simulation results are shown in Figure 16. It can be seen from Figure 16(a) that when the interval of adjacent rotary cutting sections is 3.0 m, the vertical stress will not be superimposed at diameter of rotary cutting sections of 200 mm; the vertical stress will be superimposed at diameter of rotary cutting sections of 300 mm, and the coal mass between adjacent rotary cutting sections bears high superimposed vertical stress; the vertical stress will be highly superimposed at diameter of rotary cutting sections of 400 mm, and the coal mass between adjacent rotary cutting sections cannot bear high superimposed vertical stress and begins to undergo a wide range of plastic failure. Under this condition, the coal mass between all rotary cutting sections will form a continuous plastic failure zone. This continuous plastic failure zone can be seen as a pressure relief zone, and it can well transfer and release the energy accumulated in coal mass inside roadway rib, so as to achieve the purpose of preventing rock burst.

It can be seen from Figures 16(b) and 16(c) that when the interval of adjacent rotary cutting sections is 4.0 m or 5.0 m, the vertical stress will not be fully superimposed and the coal mass between all rotary cutting sections will not form a continuous plastic failure zone. Under this condition, these rotary cutting sections will not transfer and release the energy accumulated in coal mass inside roadway rib well, and the effect of preventing rock burst is poor.

In summary, the optimal diameter and interval of rotary cutting sections are 400 mm and 3.0 m, respectively. Under
Table 5: The test results of physical and mechanical parameters.

| Lithology types | Thickness (m) | Density (kg·m\(^{-3}\)) | Bulk modulus (GPa) | Shear modulus (GPa) | Cohesion (MPa) | Internal friction angle (°) |
|-----------------|---------------|--------------------------|-------------------|-------------------|----------------|---------------------------|
| Siltstone       | 22            | 2550                     | 6.9               | 4.8               | 2.1            | 30                        |
| Mudstone        | 7             | 2100                     | 4.4               | 2.6               | 1.2            | 26                        |
| Coal seam 5#    | 31            | 1350                     | 2.7               | 0.85              | 0.87           | 22                        |

Figure 13: The 3D model established by FLAC\(^{3D}\) numerical software.

Figure 14: The vertical stress distribution inside the roadway rib.

Figure 15: The average value of on-site drilling cuttings weight inside the roadway ribs. (a) The left rib; (b) the right rib.
the condition of optimal diameter and interval parameters, the specific numerical simulation results of different lengths of rotary cutting sections are shown in Figure 17.

It can be seen from Figure 17 that the large-diameter borehole (110 mm) has poor pressure relief effect for coal mass inside roadway rib. However, when the pressurized water jet is used to conduct rotary cutting of roadway rib, the larger the length of rotary cutting section is, the larger the corresponding length of pressure relief zone is. At the same time, with the increase of the length of rotary cutting section, the outer peak stress away from the roadway rib also decreases accordingly, and the inner peak stress near the roadway rib remains basically unchanged. When the length of rotary cutting section is 5~20 m, the outer peak stress away from the roadway rib is about 10.7 MPa, and it reduces to be close to the original rock stress. Under this condition, the corresponding length of pressure relief zone is 13 m, and it can fully depressurize the coal mass inside the roadway rib. Finally, the length of rotary cutting section is determined to be 5 m~20 m. Accordingly, the length of protective section is determined to be 0 m~5 m. The protective section can prevent the occurrence of injury accidents caused by return water with coal cinders, and it also will not damage the surrounding rock support system at the same time.

5. On-Site Industrial Verifications

5.1. Application in Roadway Rib of Panel 250204. According to the above numerical simulation results, the optimal diameter, length, and interval of rotary cutting

| Number | Diameter (mm) | Interval (m) | Number | Diameter (mm) | Interval (m) | Number | Diameter (mm) | Interval (m) |
|--------|---------------|--------------|--------|---------------|--------------|--------|---------------|--------------|
| 1      | 200           | 3.0          | 4      | 300           | 3.0          | 7      | 400           | 3.0          |
| 2      | 200           | 4.0          | 5      | 300           | 4.0          | 8      | 400           | 4.0          |
| 3      | 200           | 5.0          | 6      | 300           | 5.0          | 9      | 400           | 5.0          |

Figure 16: The vertical stress distribution laws under different diameters and intervals. (a) The interval is 3.0 m; (b) the interval is 4.0 m; (c) the interval is 5.0 m.
sections are 400 mm, 3.0 m, and 15.0 m, respectively. In the on-site industrial application stage, the pressurized water jet is used to conduct rotary cutting of roadway rib about 50 m behind the driving face, which is located in a wing structure influence zone (II) caused by folded structure. A total of 16 pairs of boreholes with rotary cutting sections are symmetrically implemented in coal mass inside both left and right ribs. Meanwhile, a total of 11 pairs of large-diameter boreholes are symmetrically implemented in coal mass inside both left and right ribs. The pressure relief zone formed by large-diameter boreholes is about 40 m behind the pressure relief zone formed by pressurized water jet, and the specific on-site implementation schemes are shown in Figure 18.

5.2. Verification of Pressure Relief Effect

5.2.1. The Electromagnetic Radiation Monitoring Result. As the electromagnetic radiation value is positively correlated with the stress state of coal mass inside the roadway rib, the electromagnetic radiation value is used to reflect the pressure relief effect of coal mass inside the roadway rib. The monitoring results of electromagnetic radiation value are shown in Figure 19.

It can be seen from Figure 19 that the electromagnetic radiation average value is slightly reduced from 62.2 mV to 54.4 mV in the pressure relief zone formed by large-diameter boreholes, and this shows that the pressure relief effect of large-diameter boreholes is not obvious; the electromagnetic

![Figure 17: The vertical stress distribution laws under different lengths of rotary cutting section. Note: a, the length of pressure relief zone is 5 m; b, the length of pressure relief zone is 8 m; c, the length of pressure relief zone is 13 m.](image)

![Figure 18: The specific on-site implementation schemes.](image)
radiation average value is greatly reduced from 57.4 mV to 26.2 mV in the pressure relief zone formed by pressurized water jet, and this shows that the pressure relief effect of rotary cutting sections is very significant.

5.2.2. The Microseismic Monitoring Result. Based on the on-site installed SOS microseismic equipment, the energy level of microseismic events in different zones of the roadway is monitored, as shown in Figure 20.

After more than a month of monitoring by SOS microseismic equipment, it can be seen from Figure 20 that the energy level of microseismic events is not greater than $1 \times 10^4$ J in the pressure relief zone formed by pressurized water jet, and there are microseismic events with energy level greater than $1 \times 10^5$ J or even $1 \times 10^6$ J in other zones. Meanwhile, the occurrence frequency of microseismic events is also lower in the pressure relief zone formed by pressurized water jet than that in other zones.

According to the monitoring results of electromagnetic radiation method and microseismic method, it is verified that when the pressurized water jet is used to conduct rotary cutting of roadway rib, a large-scale pressure relief zone will be formed in the coal mass inside the roadway rib, which can realize the transfer and release of highly concentrated stress accumulated in coal mass inside the roadway rib and achieve the purpose of preventing rock burst in the roadway rib.
6. Conclusions

(1) Due to the serious influence of tectonic stress, the corresponding horizontal stress coefficient is relatively large, so the tectonic stress value is also relatively large. The tectonic stress plays a dominant role in static load; namely, even though the buried depth of coal seam 5# is shallow, the static load can still reach a large value under the serious influence of tectonic stress.

(2) The coal seam 5# has strong burst tendency under natural condition; however, the burst tendency of coal seam 5# will change into weak burst tendency under water-saturated condition. Therefore, as the water jet with certain pressure can effectively cut coal mass and infiltrate coal mass to a certain extent, the pressurized water jet for prevention of rock burst is an effective and feasible method.

(3) The high stress concentration area can be formed inside the roadway rib when a roadway is excavated, and there are three states for coal-rock mass system under the influence of static load. They are as follows: the coal pillar A is in an elastic state, and the coal-rock mass system is stable; the coal-rock mass system is metastable, and the failure process of coal pillar A is static failure; and the failure process of coal pillar A is brittle failure, and it has a sudden change in intensity. A rock burst event is more likely to occur and its strength is more violent under the influence of static and dynamic combined load.

(4) When a pressure relief zone is formed inside the roadway rib, the initial single peak stress curve will change to a final bimodal stress curve inside the roadway rib. A “strong-weak-strong” prevention structure can be formed inside the roadway rib and this prevention structure can well prevent the occurrence of rock burst.

(5) According to the numerical simulation results, the optimal diameter, length, and interval of rotary cutting sections are 400 mm, 3.0 m, and 15.0 m, respectively. In the on-site industrial application stage, the electromagnetic radiation average value is greatly reduced from 57.4 mV to 26.2 mV in the pressure relief zone formed by pressurized water jet. The energy level of microseismic events is not greater than $1 \times 10^{4}$ J and the occurrence frequency of microseismic events is also lower in the pressure relief zone formed by pressurized water jet. All these monitoring results verify that when the pressurized water jet is used to conduct rotary cutting of roadway rib, a large-scale pressure relief zone will be formed in the coal mass inside the roadway rib, and it can achieve the purpose of preventing rock burst in the roadway rib.

Data Availability

All data used to support the findings of this study are included within the article, and there are not any restrictions on data access.

Conflicts of Interest

The authors declare no conflicts of interest.

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