INTRODUCTION

China is the biggest consumer and producer of coal in the world, accounting for 49% of global coal consumption and 46% of global coal production, almost as much as the rest of the world combined.1 With the depletion of the shallow-buried coal, the cover depth of the coal mines increases year by year.2,3 In comparison with the underground mines abroad, much more rockburst hazards have occurred in China’s underground mines, among which 75% ~ 85% occurred in roadways.4

Rock burst is an unpredictable and violent dynamic disaster caused by the sudden rupture of rock and is always related with release of vast seismic energy.5-7 Rock burst accidents usually occur locally without influencing the general stability of the mine but threaten the safety of the personnel's lives in this local area. A number of deaths and considerable economic loss have been caused by rock burst hazards.8-10 At present, over 180 coal mines in China have been identified with rock burst tendency, among which over 50 mines are operating at a depth of over 800 m (ultradeep mines).11,12 As

Abstract

With the increase in cover depth of the coal seam, more and more rock bursts have occurred in deep coal roadways. The conventionally used passive preventive measures are not sufficient to reduce rock burst hazards in deep coal mines. To address this problem, this study firstly evaluated the abutment stress applied to the roadway surrounding rock by constructing an abutment-stress-transfer model after roadway excavation. Then, according to different positions of roadways in the coal seam, roadways are classified into three types: shallow-buried roadway with hard surrounding coal/rock; deep-buried roadway with soft surrounding coal/rock; and medium-deep–buried roadway with medium-hard surrounding coal/rock. Stress criterion and energy criterion for rock burst occurrence were proposed according to the three types of roadways. The rock bursts that occurred in stope 32 of Gucheng coal mine in China agree well with the calculation results. Finally, an active rock burst preventive approach (long borehole destress drilling) is suggested before intense supporting measures are adopted during roadway construction to prefracture the coal seam or competent strata. The efficiency of this method at different excavation stages is evaluated by examining borehole drill cuttings. The whole roadway was excavated without any rock burst after the borehole destress drilling method was implemented.

KEYWORDS

abutment stress, destress borehole drilling, dynamic hazard, roadway construction, rock burst
the cover depth of coal mines increases, high-stress–induced rock burst hazards occur more and more frequently and seriously in China.

To predict rock bursts, a number of rock burst-predicting theories have been proposed, such as rock burst proneness theory, stiffness theory, strength theory, and instability theory.\textsuperscript{13-15} However, most of the previously prosed theories are empirical and assumptive. To predict rock burst hazards quantitatively, researchers have proposed new rock burst-occurrence theories and rock burst-prevention measures. For example, Guo et al. proposed the evaluation criterion for the location of potential hazards induced by the surrounding rock of the roadway based on characteristic radii.\textsuperscript{16} Wang et al analyzed the coupling effect of hard-thick strata and faults using UDEC numerical simulation and field observation to understand the mechanism of rock burst.\textsuperscript{17} Besides the rock burst-occurrence mechanism, passive rock burst preventive measures (eg, rock bolt, cable bolt, steel net, and steel belt) are usually adopted during roadway excavation.\textsuperscript{18} However, in rock burst-prone roadways (roadways driven in thick and hard coal seam), although different types of supports are used, rock bursts still take place frequently.

After excavation, the load-bearing zone and abutment stress in the surrounding rock redistribute and finally the plastic zone and elastic zone are formed. The time taken in this process depends on the cover depth, characteristics of coal/rock, and excavation disturbance.\textsuperscript{19-21} After the roadway is supported, a rock-bolt-support zone and a rock-cable-support zone are formed, as shown in Figure 1. The bolt cable is connected to the rock mass in the elastic area, leading to the formation of prestress bundles. This restrains the surrounding coal/rock cracking and reduces its plasticity degree. As a result, the area of the plastic zone decreases and the elastic zone gets closer to the roadway wall. Consequently, more elastic strain energy will be stored in the elastic zone and less energy is dissipated in the plastic zone, increasing the chance of rock bursts.\textsuperscript{22}

To address this issue, this study uses active measures (long borehole drilling) together with passive measures (supporting) to prevent rock bursts; that is, after excavation, the roadway surrounding rock is destressed first by drilling long destress boreholes to transfer the high abutment stress into the deep coal body before the roadway is supported. Long-hole destress drilling has been used in underground mines with high rock burst risks.\textsuperscript{23} This active destress measure can significantly reduce the overall elastic modulus of the coal/rock body, density, and strength, which will lead to the increase of the plastic area near the roadway.\textsuperscript{24} Besides, destress can dissipate the elastic energy accumulated in the coal seam and increase resistance. The “low stress” and “low density” protection zone supported by rock bolts and cable bolts can be regarded as an anchoring structure. This structure together with long cable bolts and individual props will provide active support force for the roadway.

The following of this paper will firstly describe a rock burst that occurred in Gucheng coal mine, Shandong province, China, and then establish the abutment-stress-transfer model of the roadway surrounding rock. Based on this model, criteria for rock burst occurrence are proposed in section 4. Then, long-hole destress drilling together with cuttings-examining methods for rock burst prevention is discussed in section 5. Finally, the application effect of the proposed rock burst criteria and preventive measures are evaluated during roadway construction in Gucheng coal mine.

\section*{2 ROCKBURSTS IN GUCHENG COAL MINE}

\subsection{2.1 Geological setting of the longwall panel}

Gucheng coal mine is located in Shandong Province, China. Mining in this mine began on January 2001 with a production capacity of 2 200 000 ton per year. At present, coal seam 3 in stope 32 is being mined by a fully mechanized longwall caving method. The immediate roof is 3.7-m mudstone, while the main roof is medium sandstone with thickness of 18.7 m. A layer of 5.7-m siltstone formed the floor. The cover depth of the coal seam is about 1200 m, and the average thickness of the coal seam is 9 m with an inclination ranging from 8° to 20°. The overlying strata are mainly composed of competent sandstones. From laboratory tests, the uniaxial compressive strength, the dynamic failure time, burst energy index, and elastic energy index for coal seam 3 are 18.5 MPa, 42 mm, 6.5, and 4.8, respectively.\textsuperscript{25}
2.2 Description of the rock burst hazards

Several rock bursts occurred in the roadway of stope 32 on 30 August 2015. The rock bursts were mainly located behind the working face in the intake and return air roadways of stope 32, as shown in Figure 2. A number of people were injured, and pieces of equipment (anchor net, cable bolt, supports, conveyor, and transformer) were damaged (see Figure 3). Roof subsidence was also observed from the site (Figure 3A). The actual damage lengths of roadways for areas A, B, C, and D in Figure 2 are 150 m, 90 m, 70 m, and 30 m, respectively. This rock burst disaster caused a temporary suspension in production. Recovery of production was extremely difficult for this mine.

3 ABUTMENT-STRESS-TRANSFER MODEL AFTER ROADWAY EXCAVATION

After excavation, the in situ geostress (including horizontal and vertical stress) will be redistributed. The horizontal stress transfers to the roof and floor strata, while the vertical stress caused by the weight of the overlying strata above the roadway is transferred to the coal/rock on both sides of the roadway. The transferred stress is superimposed with the pre-existing in situ stress in the surrounding rock, as shown in Figure 4A,B, increasing significantly the chance of rock bursts.

To construct the abutment-stress-transfer model, the overlying strata are regarded as homogeneous and isotropic without considering special geological factors such as faults, folds, and changes of strata strength and thickness. Assuming the height and width of the roadway are \( a \) and \( b \), respectively, and the thickness of coal seam is \( m(a \leq m) \). The load \( (P_1) \) acting on the roof of the roadway before excavation is expressed as (assuming the roadway is within the coal seam).

\[
P_1 = [h \gamma_0 + (m - a) \gamma_m] b \tag{1}
\]

where \( h \) is the height from the roadway roof to the surface; \( \gamma_0 \) is the average unit weight of the overlying strata above the coal seam; and \( \gamma_m \) is the average unit weight of the coal seam.

If we create a coordinate system with the origin on the wall of the roadway (see Figure 5) and assume the influential range of the abutment stress on one side of the roadway is \( l \), the self-weight \( (P_2) \) of the overlying strata within the influential range can be calculated as follows:

\[
P_2 = [h \gamma_0 + (m - a) \gamma_m] l \tag{2}
\]

The load \( (P_0) \) acting on one side of the roadway within the influential range is superimposed by \( P_2 \) and half of \( P_1 \):

\[
P_0 = \frac{P_1}{2} + P_2 = [h \gamma_0 + (m - a) \gamma_m] \left( \frac{b}{2} + l \right) \tag{3}
\]

Assuming \( \sigma_x \) is the abutment stress on one side of the roadway within the influential range, the following equation can be obtained:

\[
\int_0^l \sigma_x dx = P_0 x \in [0, l] \tag{4}
\]

To estimate \( \sigma_x \), the abutment stress curve in Figure 5 can be simplified to be a “linear” piecewise function with the midpoint of the roadway wall as the original point (Figure 6):

\[
\sigma_x = \begin{cases} 
\frac{kT}{s-l} x & x \in [0, s] \\
\frac{kT-x}{s-l} x + \frac{\gamma_0-kT}{s-l} & x \in [s, l]
\end{cases} \tag{5}
\]

where \( T \) is the initial geostress of the coal and rock mass on one side of the roadway, which can be expressed as

\[
T = \sigma_i = h \gamma_0 + (m - a) \gamma_m \tag{6}
\]

and \( k \) is the concentration coefficient of peak abutment stress; \( s \) is the horizontal distance between the peak value of abutment stress and the edge of roadway.

Substituting Equation (3), (5), and (6) into (4), \( k \) can be obtained as

\[
k = \frac{l + x + b}{l} \tag{7}
\]

It should be noted that to accurately evaluate the value of “k,” we need to obtain complex geological conditions and mining parameters. Also, the “k” values vary with the cover depth of the roadway. Therefore, Equation (7) provides a fast approach to approximately evaluate the concentration coefficient of peak abutment stress without measurement.
However, it is recommended to modify the calculated value if geostress measurement is available in the field.

By substituting \(k\) and \(T\) into (4), the abutment stress \(\sigma_x\) in the surrounding rock of the roadway can be obtained as follows:

\[
\sigma_x = \begin{cases} 
\frac{(l+b)(k+T+m-a)y}{l(s-l)}x & x \in [0,s) \\
\frac{(l+b)(k+T+m-a)y}{s-l} & x \in [s,l]
\end{cases}
\]  

(8)

According to Equation (8), the abutment stress \(\sigma_x\) is related to the physical characteristics of the coal seam (thickness and strength of the coal seam) under the condition that the cover depth of the coal seam and the size of the roadway are fixed. The influential range of the roadway abutment stress is influenced significantly by the thickness of the coal seam. After excavation, the development of the plastic zone above the roadway in a thinner coal seam is limited by the hard rock roof and floor. Therefore, the horizontal plastic zone is larger than that in the vertical direction. The deformed development of the plastic zone may cause instability of the roadway, which may increase the
chance of rock bursts. The strength of the coal seam influences the shape of the abutment stress curve. The larger the strength of the coal seam is, the larger the peak value of the abutment stress will be. On the other hand, when the strength of coal seam is lower than a critical value (such as soft coal without rock burst tendency), the risk of rock burst is relatively small. Generally speaking, rock burst hazards occur frequently in the coal seam with uniaxial compressive strength between 5 and 15 MPa.

4 | ROCKBURST CRITERIA FOR DIFFERENT ROADWAY TYPES

4.1 | Overall compressive strength of roadway

The roadway can be classified into three types according to its size and position relative to the coal seam, that is, type I: roadway within the coal seam; type II: roadway with its roof in the rock and floor in the coal seam; and type III: roadway with its roof in the coal seam and floor in the rock, as shown in Figure 7. For type II and type III roadways, the overall compressive strength of its surrounding rock can be approximately estimated as follows:

\[ [\sigma_c] = \sum_{i=1}^{N} [\sigma_i] h_i / \sum_{i=1}^{N} h_i \]  

(9)

where \([\sigma_c]\) is the overall compressive strength of the surrounding rock of the roadway, \([\sigma_i]\) and \(h_i\) are the uniaxial compressive strength and height of the \(i^{th}\) coal (rock) layer where the roadway is located.

4.2 | Overall load-bearing strength of roadway surrounding rock

As mentioned above, the purpose of analyzing the overall load-bearing strength of the roadway is to estimate the overall load-bearing strength of the roadway surrounding rock. As is known, roadway burst occurs when abutment stress in the surrounding rock is larger than its load-bearing strength. From Figure 8, it can be seen that along the axial direction from the edge of the roadway to the deep coal body, there exist three zones: crack or plastic zone, elastic zone, and original-rock zone. Correspondingly, the coal/rock in these zones is in no or uniaxial stress state, biaxial stress state, and triaxial stress state. Previous study indicates that the load-bearing strength of the surrounding rock is linearly related to its overall compressive strength before failure.7,27 Therefore, the load-bearing strength \((R)\) of the roadway surrounding rock is given by

\[ R = \eta [\sigma_c] = \eta \sum_{i=1}^{N} [\sigma_i] h_i / \sum_{i=1}^{N} h_i \]  

(10)

where \(\eta\) is the stress coefficient of the surrounding rock, which increases with the cover depth of the coal seam according to previous literature.7,27 From plastic zone to elastic-plastic transition zone, \(\eta\) gradually increases from 0 to 1; from elastic-plastic transition area to elastic-original-rock transition zone, \(\eta\) increases from 1 to 3~5 (dependent on the coal/rock strength); and from the elastic-original-rock zone further, \(\eta\approx3~5\). An average value \((\eta_{\text{max}}\approx4)\) is usually selected in practice, as shown in Figure 9. According to the above analysis results, the minimum and maximum values of \(\eta\) can be selected as 1 and 4, respectively. The load-bearing strength of roadway surrounding rock can be written as

\[ R = \eta [\sigma_c] = \begin{cases} \frac{x \sum_{i=1}^{N} [\sigma_i] h_i}{N \sum_{i=1}^{N} h_i}, & x \in [0, s) \\ \frac{\sum_{i=1}^{N} [\sigma_i] h_i}{(3x^2-4x) \sum_{i=1}^{N} h_i}, & x \in [s, l] \end{cases} \]  

(11)

\[ \sum_{i=1}^{N} h_i = l - s \]

- Crack/plastic zone
- Elastic zone
- Original-rock zone

FIGURE 7 Types of roadways

FIGURE 8 Stress state of coal seam in different zones
4.3 Predicting rock burst hazards based on load-bearing strength and abutment stress of surrounding rock

4.3.1 Classification of the roadway

The roadway can be classified according to the relationship between its abutment stress and load-bearing strength, as well as the relationship between cover depth and surrounding rock/coal strength: Type I’: shallow-buried roadway with hard surrounding coal/rock; Type II’: deep-buried roadway with soft surrounding coal/rock; and Type III’: medium-deep–buried roadway with medium-hard surrounding coal/rock (see Figure 9). For type I’ roadway, the abutment stress of the surrounding rock is lower than its load-bearing strength at any position. Therefore, Type I’ roadway can keep stable for a long time (Figure 10). For type II’ roadway, although the abutment stress of the surrounding rock is lower than its load-bearing strength, elastic energy cannot be stored in soft surrounding rock/coal as large deformation usually occurs in this case, and hence, “creep” or “rheological” hazards usually take place rather than rock bursts. For type III’ roadway, the abutment stress of the surrounding rock near the roadway is higher than its load-bearing strength, which may lead to accumulation of elastic energy in the medium-hard surrounding coal/rock. As a result, rock burst is very likely to occur.

4.3.2 Rock burst criteria

This study proposes energy criterion and stress criterion for rock bursts. The roadway will be at rock burst risk if either energy criterion or stress criterion is satisfied.

1. Stress criterion

When the abutment stress of the surrounding rock is larger than its load-bearing strength, the surrounding rock near the roadway will firstly fail and form a plastic zone which gradually expands to the deep rock mass (Figure 11). Further, the rock in elastic stress state will form an elastic zone. Only when the abutment stress ($\sigma_x$) acting on the surrounding rock in the elastic zone exceeds its load-bearing strength ($R_x$), can the conditions for rock burst occurrence be satisfied (stress criterion). The stress criterion can then be expressed as

$$\sigma_x \geq R_x \in [s,l-s]$$  \hspace{1cm} (12)

1. Energy criterion. When the accumulated elastic energy of the coal/rock mass in the elastic zone is larger than the dissipated energy (energy dissipated to overcome frictional and support resistance during rock burst ($U_h$),

![Stress coefficient versus distance from roadway edge](image1)

**FIGURE 9** Stress coefficient versus distance from roadway edge

![Abutment stress and load-bearing strength](image2)

**FIGURE 10** Relationship between abutment stress and load-bearing strength of surrounding rock for type I’ (A), type II’ (B), and type III’ (C) roadways
and energy dissipated by failing the rock mass during rock burst \( (U_d) \), the accumulated elastic energy \( (U) \) of coal/rock mass in elastic zone is

\[
U = \sum_{i=1}^{n} \frac{(1-2u_i)(1+u_i)^2 h_i}{6E_i(-u_i)^2} \cdot \left( \int \sigma_i d_i \right)^2
\]

\[= \sum_{i=1}^{n} \frac{(1-2u_i)(1+u_i)^2 h_i}{6E_i(1-u_i)^2} \cdot \left\{ \int \frac{(s+b)(h_{\gamma_0}+(m-a)\gamma_0)}{(s-l)} \right\}^2\]

\[
\int_{s-l}^{s+b} \frac{h_{\gamma_0}+(m-a)\gamma_0}{(s-l)} dx = \frac{(1+b)(h_{\gamma_0}+(m-a)\gamma_0)}{s-l} dx
\]

\[
U_h = \varphi l + \left[ 2c + \frac{akT(1+\cos^2 \alpha)}{2} \right] l
\]

\[
U_d = \frac{\left( \int \sigma_i d_i \right)^2}{2E_i} = \frac{\left( \sum_{i=1}^{N} \sigma_i h_i \right)^2}{\sum_{i=1}^{N} h_i} \cdot \frac{\left( \int_{s-l}^{s+b} \frac{3s+2l-4s}{l-s} ds \right)^2}{2E_i}
\]

where \( \mu_i \) is the Poisson's ratio of coal/rock mass and \( E_i \) is the elastic modulus of coal/rock. \( U_h \) and \( U_d \) can be expressed as

\[U_h = \varphi l + \left[ 2c + \frac{akT(1+\cos^2 \alpha)}{2} \right] l\]

\[U_d = \frac{\left( \int \sigma_i d_i \right)^2}{2E_i} = \frac{\left( \sum_{i=1}^{N} \sigma_i h_i \right)^2}{\sum_{i=1}^{N} h_i} \cdot \frac{\left( \int_{s-l}^{s+b} \frac{3s+2l-4s}{l-s} ds \right)^2}{2E_i}\]

where \( c \) is the cohesion between the coal seam and the roof/floor; \( w \) is the friction coefficient between layers; \( \alpha \) is the compression angle of the coal seam; and \( \varphi \) is the support resistance of the of the roadway. If no support is implemented, \( \varphi = 0 \), the following energy criterion for rock burst occurrence should be satisfied without considering the energy accumulation and dissipation during shape deformation of coal and rock mass:

\[U \geq U_h + U_d\]

5 | LONG-HOLE DESTRESS DRILLING FOR ROCKBURST PREVENTION

To reduce rock bursts, this study used active and passive measures for rock burst prevention; that is, after excavation, the roadway surrounding rock is destressed first by drilling long boreholes to transfer the high abutment stress into the deep coal body, and then, supports were implemented. Intense destress measures during roadway construction can significantly reduce the overall elastic modulus of the coal/rock body, density, and strength, which will lead to the increase in the plastic area near the roadway. The high abutment stress will then transfer to the deep coal/rock body, as shown in Figure 12. The destress
The degree can be determined by examining borehole drill cuttings. Besides, destress can dissipate the elastic energy accumulated in the coal seam and increase resistance. The “low stress” and “low density” protection zone supported by rock bolts and cable bolts can be regarded as an anchoring structure. This structure together with long cable bolt and individual props will provide active support force for the roadway to keep the roadway stable.

In detail, a 20-m borehole was drilled in the working face, and from within 5 ~ 10 m behind the face, long boreholes were drilled at both walls of the roadway. The diameter of these boreholes was 150 mm, and the spacing varied from 1 ~ 3 m (see Figure 13). The boreholes were drilled vertical to the roadway wall at the heights between 0.8 and 1.6 m above the bottom. After drilling the destress boreholes, the destress effect was checked daily by examining the borehole drill cuttings. The boreholes for cuttings examination were drilled parallel to and between the destress boreholes at the heights between 0.8 and 1.6 m above the bottom (see Figure 14). One 17-m cuttings-examining borehole was drilled in the working face, and from behind the face within 5 ~ 50 m, at least three 17-m cuttings-examining boreholes with spacing from 20 to 25 m were drilled in each wall of the roadway. The nearest boreholes were located 5 ~ 10 m from the working face. The diameter of the cutting-examining boreholes was 42 mm. If dynamic phenomena such as coal ejection and slight coal burst in the roadway occur, the number of cuttings-examining boreholes should be increased.

6 | CASE STUDY FOR EVALUATION OF ROCKBURST CRITERIA AND PREVENTIVE MEASURES

The values of parameters used for analysis are determined according to the geological conditions of the roadway in stope 32: $h = 1130$ m, $m = 9$ m, $E = 14$ GPa, $u = 0.25$, $c = 1$ MPa, $w = 0.1$, $a = 9^\circ$, $\varphi = 1.2$ MPa, $\gamma_0 = 25$ kN/m$^3$, $\gamma_m = 13.7$ kN/m$^3$, $a = 4$ m, $b = 4.8$ m, $[\sigma_c] = [\sigma_{coal}] = 18.5$ MPa (Type I roadway); $s = 4$ m, $l = 8$ m (determined by borehole drill cuttings examination); $\eta_{\text{min}} \approx 1$, $\eta_{\text{Max}} \approx 4$. Substituting the above parameters into Equations (8) and (11), we can obtain the abutment stress $\sigma_x$ and load-bearing strength $R$ versus distance from roadway wall, as shown in Figure 15. It can be seen that when $0 \text{ m} < x < 6.2$ m, $\sigma_x > R$, indicating the stress criterion for rock burst occurrence is satisfied. Within this area, it can be obtained that $U = 5.3 \times 10^7$ J, $U_h = 1.6 \times 10^7$ J, and $U_d \approx 3.2 \times 10^7$ J $U_v = U - U_h - U_d > 0$, suggesting that the energy criterion is satisfied. The rock burst that occurred in stope 32 agrees well with the theoretical calculation results.

To prevent such type of rock burst, the roadway was firstly intensely destressed before supported. The rock burst risk was determined by examining borehole drill cuttings using the following equations

$$G = \pi r^2 \omega \rho$$

where $G$ is the critical amount of the borehole drill cuttings; $r$ is the radius of the borehole (0.021 m in this study); $\rho$ is the average bulk density of the coal seam ($1.37 \times 10^3$ kg/m$^3$); and $\omega$ is the borehole drill cuttings ratio which is the ratio of the borehole depth (17 m in this study) to coal seam thickness. The
17-meter borehole can be divided into two parts: [0 m, 13 m] and (13 m, 17 m] to evaluate the critical amount of the borehole drill cuttings. The $\omega$ values can be calculated as [0, 1.5] and (1.5-1.89] in these two parts, and the critical amount of the borehole drill cuttings in these two parts can then be obtained as [0, 4.0 kg/m] and (4.0, 5.4 kg/m]. The actual amount of the borehole drill cuttings are shown in Figure 16. It can be seen that after destressing, the actual amount of the borehole drill cuttings is less than the critical value along the depth of the borehole. However, two mutations can be identified in this curve at 4 ~ 6 m and 14 ~ 16 m, suggesting the two positions are the elastic-plastic and elastic-original-rock intersection zones. In comparison with the distribution of plastic and elastic zones before destressing, both the plastic zone and elastic zone become larger, indicating the effectiveness of the destress-support technique. After long borehole drilling was implemented, the roadway in stope 32 was stable without rock bursts during roadway excavation, as shown in Figure 17.

The rock bolt and cable bolts were used to support the roadway roof and wall. The diameter and length of the rock bolt were 22 mm and 200 mm, respectively. The in-row spacing and the distance between rows are equal (600 mm × 600 mm). The cable bolt had a diameter and length of 21.8 mm and 7800 mm, respectively, with spacing of 1600 × 1600 mm.

7 | CONCLUSIONS

Rock bursts are very likely to be induced during roadway construction in deep coal mines. This type of rock bursts may cause serious damage to the roadway and injure or kill underground workers. This study proposed rock burst-occurrence criteria by constructing a stress-transfer model after excavation of the roadway. This model can be applied to other rock burst-prone roadways in deep coal mines.

To solve the problem of stress concentration and elastic energy accumulation caused by passive support measures during roadway construction, long borehole drilling method is proposed in this study. The engineering practice
has proved that this new technique can increase the plastic and elastic areas, which reduces rock burst risk degree dramatically.

As geological conditions vary from mine to mine, when applying the proposed rock burst criteria and long borehole drilling methods, other preventive measures such as reducing working face advancing speed, selecting appropriate extraction methods and sequences, and strategically placing supports should also be considered. Furthermore, in the process of tunneling, the bottom coal with thickness larger than 1 m should be excavated to release the accumulated elastic energy. In the future work, criteria and mechanism of roadway rock bursts during longwall mining will be studied.

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