Research Article

Study on Rock Burst Event Disaster and Prevention Mechanisms of Hard Roof

Nan Zhou, Hengfeng Liu, Jixiong Zhang, and Hao Yan

State Key Laboratory of Coal Resources and Safe Mining, China University of Mining & Technology, Xuzhou 221116, China

Correspondence should be addressed to Hengfeng Liu; cumtlhfeng@163.com

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Coal mining under hard roofs is jeopardized by rock burst-induced hazards. In this paper, mechanisms of hard roof rock burst events and key techniques for their prevention are analyzed from the standpoint of energy evolution within geological conditions typical of the hard roofs found in Chinese coal mines. Equations used to calculate the total strain energy densities of the coal-rock mass and hard roof working face are derived. Moreover, several failure-causing energy evolution rules are analyzed under various conditions. Various rock roof and coal mass thicknesses and strengths are considered, and a method of preventing hard roof rock burst events is proposed. The results obtained show that rock burst events can be facilitated by high stress concentrations, significant accumulation of strain energy in the coal-rock mass, and rapid energy release during roof breakage. The above conditions are subdivided into two classes: energy accumulation and energy release. The total strain energies of the coal mass and working faces in the roof are positively correlated with the roof thickness, roof strength, and coal mass strength. The coal mass strength primarily influences the overall accumulation of energy in the working face, and it also has the largest effect on the total energy release (i.e., the earthquake magnitude).

1. Introduction

Hard roof conditions in Chinese coal mines vary significantly. Their thicknesses can vary from several dozen to several hundred meters. The coal resources under hard roofs occupy about one-third of their total volume. Currently, about 40% of fully mechanized working faces are hard roof working faces, and problems related to mining hard roof working faces are present in more than 50% of national mines. A hard roof is the rock stratum located above a coal seam or a relatively thin immediate roof. It is relatively thick, contains poorly developed joints, and is made up of strong rock with a large bearing capacity. The presence of a hard roof is one of the primary causes of rock burst in working faces [1, 2]. The hard roof does not cave spontaneously during mining. Instead, it is suspended above the goaf. When the area of the suspended roof reaches a certain value, the load carried by hard roof exceeds its ultimate strength, and caving occurs across a large area. This leads to a fast release of the energy accumulated inside the roof and coal seams and causes mining disasters such as rock burst events, severe damage to equipment, and significant casualties and injuries.

Extensive studies of hard roof rock burst events have been conducted worldwide since the 1950s. Bieniawski et al. explored burst tendency indices experimentally, constructed the burst tendency theory, and suggested that the occurrence of rock burst events in the coal seam depends largely on its mechanical properties [3]. Petukhov classified rock burst hazards [4]. Nawrocki simulated the roof using a shear beam and obtained the stress distribution of the deformed region in the coal seam. His results implied that dynamic failure occurs when destabilization in the static equilibrium state exceeds the stress limit of the coal-rock mass, resulting in a rock burst event [5]. Goszcz elaborated a geological model of roof-type rock burst events [6]. Their model envisaged that sudden dislocation and destabilization of the hard roof is the primary prerequisite for rock burst events. Huwe reported that factors other than the stress state influence
the occurrence of rock burst events [7]. These factors include the composition of the surrounding rock mass, the rock mass thickness, and the distance between the rock mass and the coal seam. Mu et al. proposed the catastrophe model for hard roof collapse, revealed mechanisms for hard roof collapse and development, and recommended measures intended to prevent and control hard roof rock burst events [8]. Fang et al. investigated rock burst event mechanisms using several methods including field analysis, indoor testing, and numerical simulation [9]. Yao et al. used theoretical analyses and field measurements to prove that sudden hard roof collapses result from unstable development of coal mass cracks, which can cause burst events [10–12]. Wang et al. found the mechanism that induces rock bursts can be divided into two stages: the accumulation of strain energy prior to HTS failure and the release of strain energy because of the coupling effect of HTS failure and fault slipping [13]. Zhou et al. summarized rock burst classification and its varying definitions, examined and compared different rock burst assessment methods, and listed current achievements, limitations, and some promising directions for future research [14–16]. Jiang et al. deduced the fracture failure step distance of the hard and thick igneous rock and the calculation method of the fracture failure of hard and thick key strata and obtained the hard, thick critical layer collapse mechanism and the calculation expression of fracture step distance [17]. Saharan and Mitri found the key of improving the technique of destress blasting, which includes proper understanding of fractures growth under the influence of polycrystalline stress regime and proper utilization of explosive energy for effective fracture creation and growth [18]. Briefly, methods of preventing and controlling hard roof rock burst events can be classified into four types: the coal pillar support, stress reduction, forced caving, and backfilling methods [19, 20].

Thus, comprehensive studies have been performed worldwide that consider the laws that govern motion of a hard roof, rock burst control theories, and hard roof control techniques. A series of theoretical and practical achievements have been obtained. However, there are few results on energy evolution during hard roof deformation, energy accumulation laws, hard roof energy conversion and release caused by exposure to breakage, and relationships between rock burst events and coal-rock strength, burial depth, and roof thickness. This paper considers the aforementioned issues in detail and proposes hard roof rock burst control and prevention strategies.

2. Hard Roof Energy Evolution and Failure Mechanisms

In the working faces used in cave mining, the stress in the coal-rock mass is at equilibrium before mining of the coal seams. The roof is gradually exposed during mining, and the system is no longer in its initial equilibrium state. Since the roof is suspended and exposed above the mining space, bending deformation occurs due to the weight of the roof and pressure from the overlying weak rock strata. This causes stress redistribution in the mining space. The coal mass is subject to clamping from the roof and floor, which generates significant abutment stress in the working faces, as shown in Figure 1.

Rock strata motion and strata-pressure behavior in the working face are characterized by large caving step distances, apparent strata-pressure behavior, and sudden pressure [21–23]. The process of exposing the roof to loads, bending deformation, and breakage is also a process of energy accumulation, conversion, and release. It primarily involves strain, gravitational potential, kinetic, surface, and vibration energies [24–27]. Object energy changes can be divided into dissipation and accumulation. Energy can be dissipated via object deformation or work on external objects. Figure 2 depicts a failure analysis of a hard roof during the process that leads to breakage.

When mining of coal seam begins, bending subsidence occurs on the roof due to the absence of support at its bottom. Thus, the gravitational potential energies of the hard roof and its overlying weak rock strata are converted into roof strain energy. Strain energy is also stored in the coal mass due to compression of the coal wall caused by roof deformation.

The roof curvature and strain energy increase with the distance mined. Cracks develop in both the roof and the coal mass. Thus, some of the strain energy is converted into surface and vibration energies and is released. Phenomena such as coal wall spalling, coal burst events, and slab burst events occur when the release rate reaches certain values. If the strain energy is relatively large and the surface and vibration energy release rates fail to match the strain energy conversion rate, extra strain energy is converted into coal-rock mass kinetic energy, and its subsequent release can cause a rock burst event.

The roof breaks when the bending deformation limit is reached. The gravitational potential and strain energies of the roof and coal seam can be rapidly released and converted into kinetic, surface, and vibration energies. Thus, pressure appears across a large area of the roof. At the same time, the roof and coal mass in front of the working face rebind due to their varied loads, which causes a large portion of the strain energy accumulated in the roof and coal mass to be converted into surface and vibration energies and then rapidly released. The risk of a rock burst event and collapse is high if significant strain energy is released.

The hard roof rebinds and vibrates after breakage. Thus, some gravitational potential energy and strain energy is converted into vibration energy, which is then transferred to the front roof and coal mass in the form of a vibration wave. The disturbance induced by this vibration energy causes destabilization and can cause a rock burst event if the strain energy accumulation in the front roof and coal mass reaches a critical value.

3. Mine Energy Variation before and after Hard Roof Breakage

3.1. Analysis of Roof Deformation before the Initial Breakage Event. Before the initial roof breakage event, the two sides of the roof are supported by the coal mass but its middle portion is suspended and exposed to bending deformation, as shown in Figure 3.
The propulsion length of the coal mine working face is much longer than the working face itself. A cross section of the working face along the advancing direction was selected for a mechanical analysis in which the roof was treated as a beam structure. Assuming that \( q_0 \) is the in situ stress and \( k_d \) is the coefficient of the advanced concentrated stress, the loads of the suspended and exposed roof (including the weight of the roof itself and the weak rock strata that subside during its deformation) are reduced to a uniform load \( q_e \). If the coal mass is treated as an elastic foundation, a beam model can be constructed on a semi-infinite elastic foundation to simulate the roof loads before the initial breakage event. The side along the advancing direction of the working face is selected from the middle of the model to be analyzed due to the lateral symmetry shown in Figure 4. Typically, the maximum advanced abutment stress is located between 3 m and 10 m from the coal wall, which is much smaller than the affected region. Therefore, the stress concentration area can be disregarded in the remainder of the analysis [28].

The Cartesian coordinate axes are constructed as follows: the middle of the roof immediately above the coal wall is taken as the origin, the advancing direction along the working face is used as the \( x \)-axis, and the vertical downward direction is the \( \omega \)-axis. As shown in Figure 4, \( L \) is the length of the suspended and exposed roof, \( L_0 \) is the range affected by the advanced concentrated stress, \( k_c \) is the elasticity coefficient of the coal mass, \( \omega_1 (x) \) is the deflection coefficient of the roof above the coal mass, and \( M_0 \) is the bending moment at the middle of the suspended and exposed roof. The load on the roof \( q(x) \) and the bottom supporting load \( p(x) \) can be expressed as follows:

\[
\begin{align*}
q(x) &= k_d q_0 - \frac{(k_d - 1)q_0}{L_0} x, \\
p(x) &= k_c \omega_1 (x), \quad 0 \leq x \leq L_0, \\
q(x) &= q_e, \\
q(x) &= 0, \quad \frac{L}{2} \leq x < 0.
\end{align*}
\]

Relationships between the deflection \( \omega(x) \) and loads \( q(x) \) and \( p(x) \) are derived via the Winkler assumption [29–31]:

\[
E_I I_1 \frac{d^4 \omega(x)}{dx^4} = q(x) - p(x),
\]

where \( E_1, I_1, \) and \( E_I I_1 \) are the elastic modulus, moment of inertia \( I_1 = h_1^3/12 \), where \( h_1 \) is the roof height, and bending rigidity of the hard roof. Differential equations of deflection can be obtained by substituting equation (1) into equation (2):

\[
\begin{align*}
E_I I_1 \frac{d^4 \omega_1 (x)}{dx^4} &= k_d q_0 - \frac{(k_d - 1)q_0}{L_0} x - k_c \omega_1 (x), \\
E_I I_1 \frac{d^4 \omega_2 (x)}{dx^4} &= q_e,
\end{align*}
\]

where \( \omega_2 (x) \) is the deflection of the roof above the goaf.

Equation (3) can be reduced to the following form by assuming that the characteristic coefficient is \( \beta = \sqrt{k_c/(4E_I I_1)} \):

\[
\begin{align*}
\frac{d^4 \omega_1 (x)}{dx^4} + 4\beta^4 \omega_1 (x) &= E_I I_1 \left( k_d q_0 - \frac{(k_d - 1)q_0 x}{L_0} \right), \\
\frac{d^4 \omega_2 (x)}{dx^4} &= \frac{q_e}{E_I I_1}.
\end{align*}
\]

A qualitative analysis of the above equations shows that deflection of the roof above the coal seam \( \omega_1 (x) \) approaches a constant value when \( x \) approaches infinity \( (x \rightarrow +\infty) \). This can be used to solve equation (4) to obtain a general solution for hard roof deflection:

\[
\begin{align*}
\omega_1 (x) &= \frac{k_d q_0 - ((k_d - 1)q_0 x)/L_0}{k_c} \\
&\quad + A_1 e^{-\beta x} \cos (\beta x) + A_2 e^{-\beta x} \sin (\beta x), \\
\omega_2 (x) &= \frac{q_e}{24E_I I_1} x^4 + A_3 x^3 + A_4 x^2 + A_5 x + A_6,
\end{align*}
\]

where \( A_1 \sim A_6 \) are undetermined parameters.

The relationships between the bending moment \( M(x) \), rotation angle \( \theta(x) \), shear force \( Q(x) \), and deflection \( \omega(x) \) of any roof cross section are as follows:

\[
\begin{align*}
\theta(x) &= \frac{d\omega(x)}{dx}, \\
M(x) &= -E_I I_1 \frac{d^2 \omega(x)}{dx^2}, \\
Q(x) &= -E_I I_1 \frac{d^3 \omega(x)}{dx^3}.
\end{align*}
\]

The boundary conditions at \( x = -L/2 \) are

\[
\begin{align*}
\theta_1 \left( -\frac{L}{2} \right) &= 0, \\
Q_1 \left( -\frac{L}{2} \right) &= 0,
\end{align*}
\]

and the continuity conditions at \( x = 0 \) are

\[
\begin{align*}
\omega_1 (0) &= \omega_2 (0), \\
M_1 (0) &= M_2 (0), \\
\theta_1 (0) &= \theta_2 (0), \\
Q_1 (0) &= Q_2 (0),
\end{align*}
\]
where $M_1$ and $M_2$ are the roof bending moments above the coal seam and goaf, respectively; $\theta_1$ and $\theta_2$ are the roof rotation angles above the coal seam and goaf, respectively; and $Q_1$ and $Q_2$ are the roof shear forces above the coal seam and goaf, respectively.

The six undetermined parameters can be determined by substituting equations (5) and (6) into the boundary and continuity conditions:

$$
egin{align*}
A_1 &= -\frac{(24q_0k_dE_iL_0\beta^2 - 24q_0E_iL_0\beta^2 - q_cL_0k_c\beta^2 - 6L_0q_kL_2\beta - 6L_0^2k_cq_cL)}{24(\beta^3E_iI_0k_c(2 + \beta L))}, \\
A_2 &= \frac{(24q_0k_dE_iL_0\beta^2 - 24q_0E_iL_0\beta^2 - q_cL_0k_c\beta^2 + 6L_0q_kL_2\beta)}{24(\beta^3E_iI_0k_c(2 + \beta L))}, \\
A_3 &= \frac{q_cL}{2E_iI_0}, \\
A_4 &= -\frac{(24q_0k_dE_iL_0\beta^2 - 24q_0E_iL_0\beta^2 - q_cL_0k_c\beta^2 + 6L_0q_kL_2\beta)}{12(\beta^3E_iI_0k_c(2 + \beta L))}, \\
A_5 &= \frac{(L(2q_0k_dE_iL_0\beta + 12q_0k_dE_iL_0\beta - 12q_0E_iL_0\beta^2 + 3L_0q_kL))}{12(\beta^3E_iI_0k_c(2 + \beta L))}, \\
A_6 &= A_1 + \frac{(48q_0k_dE_iL_0q_kL + 24q_0k_dE_iL_0)L}{24(\beta^3E_iI_0k_c(2 + \beta L))}.
\end{align*}
$$
3.2. Roof Energy Distribution before the Initial Breakage Event

3.2.1. Calculation of the Roof Bending Deformation Energy.
Bending deformation is the primary type of deformation in the roof before the initial breakage event (Figure 5). Microelements of the bending beam subject to lateral forces are also exposed to actions of the bending moment \( M(x) \) and shearing force \( Q(x) \). Unlike the bending strain energy, the strain energy generated by the relatively small shearing force can be ignored \[32\].

The strain energy of a microelement with a distance of \( y \) between its cross section and the central axis is

\[
\Delta U = \frac{M^2(x)}{2EI} y^2. \tag{10}
\]

Thus, the strain energy density of a roof of unit width is

\[
u_{1t} = \int_{-h/2}^{h/2} \frac{M^2(x)}{2EI_1} y^2 dy. \tag{11}\]

The bending strain energy is reduced to

\[V_{1t} = \int_0^x \frac{M^2(x)}{2EI_1} dx. \tag{12}\]

The strain energy densities and strain energies of the roofs above the coal seam and goaf can be obtained by separately substituting terms \( M_1(x) \) and \( M_2(x) \) into equations (11) and (12).

3.2.2. Calculation of the Coal Mass Compressive Strain Energy. When the coal seam is simulated as an elastic mass, its compressive deformation can be treated as uniaxial compression of this elastic mass. The compressive deformation of the coal seam is the same as the deflection of the roof above the coal seam \( \omega_1(x) \). Therefore, the equation for the strain energy density of a coal seam of unit width is

\[
\Delta U_{1c} = \frac{1}{2} k_c \omega_1^2(x), \tag{13}\]

where \( \Delta U_{1c} \) is the strain energy density of the coal seam. It can be determined by substituting the previously solved deflection of the roof above the coal seam \( \omega_1(x) \) into equation (13):

\[
u_{1c} = \frac{1}{2k_c} \left( E \left( A_1 \cos(\beta x) + A_2 \sin(\beta x) \right) e^{\beta x} + \frac{k_c \phi L_0 - (k_2 - 1) \phi x}{k_0 L_0} \right)^2. \tag{14}\]

Thus, the compressive strain energy of the coal seam is

\[V_{1c} = \int_0^{L_0} \nu_{1c} dx. \tag{15}\]

3.3. Mine Energy Distribution after the Initial Breakage Event. A hard roof is characterized by high strength, thickness, and integrity. Therefore, a large coal seam area can be suspended and exposed in the goaf after mining. Spontaneous caving does not occur easily. Thus, a significant hazard is present after the initial hard roof breakage event \[1, 33\]. Hence, this paper considers only energy variations in the mine before and after the initial breakage event. Breakage occurs when the roof bending deformation reaches its limit. It is important to examine the initial breakage distance, deformation and energy distributions after breakage, and the amount of energy released by the breakage event.

3.3.1. Initial Hard Roof Breakage Distance. According to the maximum tensile stress criterion, which is also referred to as the first strength theory, the roof will not break if the following condition is satisfied:

\[
\sigma_{\text{max}} \leq [\sigma_1], \tag{16}\]

\[
\sigma_{\text{max}} = \frac{M_2(x)}{W_0}. \tag{17}\]
where \( \sigma_{\text{max}} \) and \([\sigma_i]\) are the maximum and allowable hard roof tensile stress.

The maximum tensile stress in a rectangular beam is

\[
\sigma_{\text{max}} = \frac{6M_{\text{max}}}{h_i^2}.
\]  

(17)

Substituting equation (17) into equation (16) produces

\[
M_{\text{max}} \leq \frac{[\sigma_i] \cdot h_i^2}{6}.
\]  

(18)

The final length of the suspended and exposed roof \( L_{\text{max}} \) and the distance from the location of the roof breakage to the coal wall \( x_0 \) can be derived by substituting the above bending moment relationship into equation (18).

### 3.3.2. Energy Distribution after Hard Roof Breakage.

The initial breakage event occurs when the length of the portion of the roof that is suspended and exposed exceeds its ultimate length \( L_{\text{max}} \). The roof above the goaf breaks in the middle and contacts the floor, while the roof above the coal seam breaks at a distance of \( x_0 \) from the coal wall. Thus, the roof becomes simply supported from both sides, as is shown in Figure 6. A mechanical model of a hard roof after the initial breakage event can thus be constructed (Figure 7).

Similarly, the equations that describe deflection of a hard roof after the initial breakage event can be solved as follows:

\[
\begin{align*}
\omega_3(x) &= \frac{k_d q_0 (L - x_0 - (k_d - 1) q_0 x)}{k_i (L - x_0)} + \frac{q_1 (L + 2x_0)}{8EI_i \beta^3} \cos(\beta x), \\
\omega_4(x) &= \frac{q_1 (x + L/2)(L + 2x_0)^3 - 4h_c^2}{192EI_i (L + 2x_0)^2} \\
& \quad \cdot \left[ (L + 2x_0)^3 - 8(L + 2x_0)(x + L/2)^2 + 8(x + L/2)^3 \right],
\end{align*}
\]

(19)

where \( \omega_3(x) \) and \( \omega_4(x) \) are roof deflections above the coal seam and goaf, respectively, after the initial breakage event. The strain energy density of the unit width roof is derived as follows:

\[
\begin{align*}
\frac{V_{21}}{2} &= \frac{E_1 I_i \beta^3 A_2}{4} \left[ e^{-2\beta (L-x_0)} e^{2\beta (L-x_0)} + \cos(2\beta L_0 - 2\beta x_0) - \sin(2\beta L_0 - 2\beta x_0) - 2 \right], \\
V_{21} &= \frac{q_1^2}{7680E_1 I_i (L + 2x_0)^3} \left( L^4 + 8L^3 x_0 + 24L^2 x_0^2 + 32L x_0^3 + 496x_0^4 \right)(L^2 + 4Lx_0 + 4x_0^2 + 4h_c^2)^2, \\
& \quad \text{if } L/2 - x_0 \leq x < 0.
\end{align*}
\]

(20)

The strain energy of the roof is

\[
\begin{align*}
\frac{V_{2c}}{2} &= \frac{E_0 E_1 \omega_3^2(x)}{2h_c} \left[ q_0 k_d (L_0 - x_0) - (k_d - 1) q_0 x \right] + \frac{q_1 (L + 2x_0) \cos(\beta x) e^{-\beta x}}{8EI_i \beta^3},
\end{align*}
\]

(22)

Then, the strain energy of the coal seam after the initial roof breakage event is reduced to

\[
V_{2c} = \int_0^{x_0 - x_1} v_{2c} \, dx.
\]

(23)

The equations for the coal seam strain energy density and strain energy can be obtained by substituting \( \omega_3(x) \) into equations (22) and (23).
where $y$ is the average specific weight of the roof.

Then, the energy release in the goaf during roof breakage can be calculated as follows:

$$\Delta V_2 = \Delta V_s + V_{1t} - V_{2t}. \quad (26)$$

### 4. Factors That Influence Energy Evolution-Driven Failure

The strain energy densities of the roof and coal mass are related to the roof thickness and strength, coal mass strength, and working face burial depth. However, in practical projects, the only variables are the roof thickness and the roof and coal mass strengths. Therefore, this paper considers relationships between the roof thickness, roof strength, coal mass strength, and roof and coal mass strain energy densities in front of the working face using a simple variable method.

The No. 6304-1 working face of the Jisan coal mine is used as a case study for this analysis. The hard roof of this working face has a height $h_t$ of 41.6 m, a burial depth of 660 m, and an elastic modulus $E_t$ of 17 GPa. The coal seam of this working face has a height $h_c$ of 3.5 m, an elastic modulus $E_c$ of 5.25 GPa, an allowable tensile strength $[\sigma_t]$ of 13.5 MPa, and an elastic foundation coefficient $k_c$ of 1.5 GPa. The influence range $L_0$ of the advanced concentrated stress is 100 m. The in situ stress $q_0$ is 16.5 MPa. The total pressure $q_c$ from the hard roof itself and the weak rock stratum that deform with it is 2.27 MPa. The coefficient $k_d$ is 2.5, and other parameters are listed in Table 1.

#### 4.1. Roof Thickness

The effect of the roof thickness on the strain energy density was studied by increasing it from 10 m to 100 m in increments of 10 m while keeping all other parameters unchanged. Figures 8 and 9 show the calculated strain energy densities of the coal mass, roof, and overall total working face before and after the initial roof breakage event.

Before the initial roof breakage event, the maximum strain energy densities of the roof rock and coal masses in the roof gradually increase with the roof thickness. However, their distributions vary as one advances to the working face. As the distance to the coal wall increases, the coal mass strain energy density gradually decreases until it reaches zero. At the same time, the rock mass strain energy density first increases and then drops to zero. The strain energy density of the coal mass significantly exceeds that of the rock mass, and thus, the variation in the overall strain energy density of the working face follows that of the coal mass.

A larger coal mass strain energy density is observed, and more strain energy is accumulated as one approaches the coal wall. When the distance to the coal wall is between 20.1 m and 66.8 m, increasing the roof thickness changes the relationship between the coal mass strain energy density and the roof thickness from positive to negative. The maximum strain energy density of the roof rock mass is reached when the distance to the coal wall ranges from 3 m to 19 m and gradually far from the coal when the roof thickness increases.

After roof breakage, the location of the break in the roof gradually moves further from the coal wall as the roof thickness increases. Distances range from 4.46 m to 13.54 m. The amplitudes of the variations in the coal mass, roof rock mass, and working face strain energy densities increase at the same time. Figure 9(a) shows that the coal mass strain energy density variation is primarily concentrated in the area 0–60 m from the coal wall, and the strain energy density of the roof rock mass has almost been thoroughly released. The amplitude of the strain energy density variation in the roof is much lower than that of the coal mass, which implies that the strain energy density variation trend in the working face is driven primarily by that of the coal mass.

Figure 9(c) shows the strain energy density of the overall working face before and after the initial roof breakage event.
The strain energy densities released by the total working face gradually increase with the roof thickness. The strain energy densities released by the working faces with various roof thicknesses gradually decrease upon approaching the coal wall, eventually reaching zero. We can use roofs with thicknesses of 10 m and 100 m as examples. When the hard roof breaks, the energies released at a working face with unit width are 1.9 MJ and 29.02 MJ, respectively. When the width is 80 m, the energies released at the working face are 152 MJ and 2321.6 MJ, respectively. These are equivalent to 2.3- and 3.04-magnitude earthquakes, respectively, according to the relationship between the energy and the earthquake magnitude.

4.2. Roof Strength. The roof strength was increased from 4.5 MPa to 18 MPa in increments of 1.5 MPa to investigate its influence on the strain energy density. All other parameters remained unchanged. Figures 10 and 11 show the calculated coal mass, roof, and total working face strain energy density distributions before and after the initial roof breakage event.

Before roof breakage, the strain energy density increases with the immediate roof strength when the distance to the coal wall is less than 23.5 m but decreases when this distance is greater. The strain energy stored in the coal mass is much higher than that stored in the roof rock mass. Therefore, the strain energy density distribution at the working face depends primarily on the former.

Figure 11 shows that the postroof breakage strain energy densities of the coal mass and overall working face vary significantly when the distance to the coal wall is between 0 and 32.2 m. The strain energy density of the roof rock mass

### Table 1: Summary of parameters.

<table>
<thead>
<tr>
<th>Object under study</th>
<th>Number of programme groups</th>
</tr>
</thead>
<tbody>
<tr>
<td>Roof thickness (m)</td>
<td>Programme I</td>
</tr>
<tr>
<td>10, 20, ..., 100 (interval 10)</td>
<td>41.6</td>
</tr>
<tr>
<td>Tensile strength of roof (MPa)</td>
<td>13.5</td>
</tr>
<tr>
<td>Tensile strength of coal mass (MPa)</td>
<td>15.99</td>
</tr>
<tr>
<td>Number of groups</td>
<td>10</td>
</tr>
</tbody>
</table>
varies significantly when the distance to the coal wall is between 0 and 50 m. The strain energy density of the coal mass increases with the roof strength at points 0 to 32.2 m from the coal wall. The amplitude of the strain energy density variation becomes negative when the distance to the coal wall exceeds 32.2 m. This occurs because the advanced abutment stress moves forward after the roof breaks, causing a slight increase in the coal mass strain energy density in such cases.

Similarly, the nearer the coal, the greater the change of strain energy density. The fracture position of the roof is closer to the coal wall as the strength of the roof is added. And the range is 7.06–16.08 m. For sample roofs with strengths of 4.5 MPa and 18 MPa, when the width is 80 m, the energies released at the working face are equal to those of 2.57 and 2.82 magnitude earthquakes.

4.3. Coal Mass Strength. The coal mass strength was increased from 1.25 MPa to 19.25 MPa in increments of 2 MPa in order to investigate its influence on the strain energy density. All other parameters remained unchanged. Figures 12 and 13 show the calculated coal mass, roof, and overall working face strain energy density distributions before and after the initial roof breakage event.

The strain energy density of the coal mass gradually increases with its strength before the initial roof breakage event. However, the roof rock mass strain energy density is only slightly affected by the strength of the coal mass. This is primarily because the loads from the rocks above the coal mass are transferred to the coal mass. The distribution of strain energy density of coal and rock mass decreases with the distance from coal wall to coal wall. The strain energy density of the working face is concentrated primarily in the area less than 80 m from the coal wall. The strain energy density becomes smaller as the distance to the coal wall increases.

Roof breakage occurs 9.06 m from the coal wall. The strain energy density of the coal mass varies significantly in the region 0 m–35.2 m from the coal wall. The strain energy density of the roof rock mass varies between 0 and 45 m from the wall, and that of the rock mass has been almost completely released. The overall working face strain energy density trend depends only on the coal mass variation.

After roof breakage, the strain energy density released in the overall working face has similar relationships to the coal and rock mass strengths. The strain energy density released
in the working face is concentrated primarily within 0 to 35.2 m of the coal wall. The strain energy density after breakage is larger than that before breakage when the distance to the coal wall exceeds 35.2 m. For sample coal masses with strengths of 1.25 MPa and 19.25 MPa, the energies released at the working face are equal to those of 2.38- and 3.00-magnitude earthquakes when the initial roof breakage event occurs and the width extends to 80 m.

5. Prevention and Control of Hard Roof Rock Burst Events

5.1. Analysis of Causes. Hard roof rock burst events can be subdivided into three types based on their causes: compressed coal mass, rebounded coal mass, and broken roof events [34].

5.1.1. Compressed Coal Mass. Before the initial roof breakage event, the coal mass is compressed by bending deformation of the hard roof. Therefore, significant strain energy is stored in the coal mass. Some of the strain energy stored in the coal mass is converted slowly into surface and vibration energies before being released. Fast energy release occurs when sufficient strain energy accumulates in the coal mass. Some of this strain energy is converted into kinetic energy of the coal (rock) mass. Thus, the compressed coal mass causes a rock burst.

5.1.2. Rebounded Coal Mass. When the roof breaks, the forces acting on the coal mass in front of the working face decrease, and the strain energy accumulated by the compressed coal mass is released instantaneously. Some portion of this is converted into surface and vibration energies, while another portion is converted into kinetic energy of the coal mass. Spalling occurs when this kinetic energy is relatively small, but a rock burst occurs when this kinetic energy is relatively large. Thus, the rebounded coal mass causes a rock burst.
5.1.3. Broken Roof. Vibrations occur when the roof breaks. These vibrations propagate into coal seams when the roof contacts the floor. If the coal seam is in a high stress-concentration region when this occurs, the accumulated strain energy reaches its limit. Additional vibration energy is generated simultaneously due to vibration during roof breakage. Thus, the original ultimate equilibrium limit state is violated, and the roof breakage event causes a rock burst.

Mining under a hard roof has potential to cause dynamic hazards such as rock burst events. Rock burst events are typically caused by high stress concentrations, accumulation of a large amount of strain energy inside the coal-rock mass, or a fast release of energy when the roof breaks. Therefore, dynamic hard roof hazards can be prevented by reducing the stress concentration, decreasing the extent of strain energy accumulation in the coal-rock mass, and reducing the amount of strain energy released when the roof breaks. Dynamic hazards can be classified into those caused by energy accumulation and those caused by energy release, as shown in Table 2.

5.2. Analysis of Influencing Factors. According to Figure 14 and previous analyses of factors that influence failure-causing energy evolution, the total strain energy of the working face is affected primarily by the strain energy of the coal mass, and the effect of mining on the strain energy is exerted primarily on the coal mass.

The total strain energy of the working face is positively correlated with the roof thickness, roof strength, and coal mass strength. The gradients that describe how these affect the overall working face strain energy before breakage are 0.348 MJ/m, 0.9 MJ/MPa, and 6.88 MJ/MPa, respectively. This indicates that the accumulation of the total strain energy in the working face before roof breakage is affected primarily by the coal mass strength, and this trend remains unchanged after the roof breaks. Therefore, reducing the coal mass strength can significantly reduce the risk of rock burst events. The gradients that describe how the roof thickness, roof strength, and coal mass strength affect the total energy released from the working face after breakage are 0.301 MJ/m, 0.57 MJ/MPa, and 1.23 MJ/MPa, respectively. This indicates that the total energy release (earthquake magnitude) from the working face is controlled primarily by the coal mass strength. The effect of the roof strength on the total strain energy of the working face is less than those of the roof thickness and coal mass strength (Figure 14). Thus, the primary strategy for preventing hard roof rock bursts is to change the characteristics of the coal-rock mass itself.

5.3. Rock Burst Prevention and Control Methods. Based on the above analysis of hard roof rock burst causes and influencing factors, prevention and control of rock bursts can be separated into global and local strategies.

5.3.1. Global Strategies. The risk and potential locations of rock bursts in the coal seam should be predicted accurately. Then, appropriate measures should be taken based on the specific characteristics of these risky areas and the coal seam mining conditions and geological structures present. The goal of these measures should be to reduce the risk of rock burst events in regions affected by geological structures and energy accumulation. Mining and exploration procedures should be comprehensively designed to reduce these risks. Based on the preexamination of coal seams and regions prone to rock burst hazards, a reasonable selection of mining and exploration techniques can radically reduce the risk of rock bursts both in specific areas and throughout the coal seam. Whenever possible, the direction of mining and exploration should be parallel to the direction of the maximum horizontal stress. The mining and exploration sequences and rates of advance of several coal seams should be coordinated. The protection layer and stress-relieving roadway should be optimized to avoid the formation of isolated or peninsula-type coal pillars. Formation of regions with high stress concentrations and drilling of roadways inside high stress coal pillars should be avoided as much as possible.

5.3.2. Local Strategies. Local strategies focus on measures that can mitigate the hazard-inducing features of a coal-rock mass. A hard roof can be treated via (i) the release of energy accumulated in the coal-rock mass via controlled hydraulic fracturing, high-pressure water jet cutting, roof fracture blasting, or transverse presplitting or (ii) reduction of the proportion of energy accumulated in the roof and the stress concentration in coal and rock layers via controlled roof caving. After the coal mass strength is reduced, the strength and ability to accumulate elastic energy in the total coal seam (particularly the coal mass near the coal wall) can be reduced via pressure release blasting, water infusion into the coal seam, pressure release from drilling or deep-hole blasting, etc. The space available for deformation of the floor should be increased by breaking the continuity of horizontal stress. This can be done by cutting grooves in the roadway floor, which can also enhance energy release in the floor, as shown in Figure 15. In addition, this should be reinforced by using yieldable supports, advanced reinforcements, roadway protection, etc.

6. Conclusions

The results obtained lead to the following conclusions:

(1) Rock burst events can be facilitated by high stress concentrations, significant accumulation of strain energy in the coal-rock mass, and rapid energy release during roof breakage, and it can be subdivided into two classes: energy accumulation and energy release.

(2) Equations for the strain energy density, strain energy, and equivalent earthquake magnitude of the working face before and after roof breakage were derived. The case study showed that substantial energy was accumulated in the roof and coal mass in front of the working face before roof breakage. The largest portions of the roof strain and gravitational
potential energies were released instantaneously when hard roof breakage occurred. Such energy accumulation and release can cause rock bursts. The quantities of energy accumulated and released in the roof before and after the initial breakage event were larger than those before and after periodic breaking, resulting in higher rock burst risk.

(3) The total strain energy of the working face is controlled primarily by the coal mass strain energy. The effect of mining on the strain energy in the mine is exerted primarily on the coal mass. The total strain energy of the working face is positively correlated with the roof thickness, roof strength, and coal mass strength. Accumulation of the strain energy in the

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**Table 2: Hard roof failure types, causes, and prevention methods.**

<table>
<thead>
<tr>
<th>Type</th>
<th>Cause</th>
<th>Prevention and control</th>
</tr>
</thead>
<tbody>
<tr>
<td>Accumulated energy</td>
<td>Excessive stress concentration in the working face. Accumulated strain energy in the coal-rock mass exceeds the limit</td>
<td>Reduce stress concentration and decrease the extent of strain energy accumulation</td>
</tr>
<tr>
<td>Energy release</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Strain energy release</td>
<td>The strain energy in the coal-rock mass in front of the working face is rapidly released when the hard roof breaks</td>
<td>Reduce the amount of strain energy released</td>
</tr>
<tr>
<td>Gravitational potential energy release</td>
<td>Some part of the gravitational potential energy of the roof in the goaf is converted into vibrational energy, which is then transferred to a coal-rock mass with critical energy accumulation</td>
<td>Reduce the amount of gravitational potential energy released in the goaf</td>
</tr>
</tbody>
</table>

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**Figure 14: Variation in the total strain energy of the working face before and after the initial roof breakage event.**

**Figure 15: Prevention measures.** (a) Blasting broken roof. (b) Hydraulic fracturing of coal. (c) Floor blasting.
working face is controlled primarily by the coal mass strength (gradients: 6.88 MJ/MPa), while the overall energy release (earthquake magnitude) depends primarily on the coal mass strength (gradients: 1.23 MJ/MPa).

(4) Key methods of preventing and controlling hard roof rock bursts were proposed based on the above analysis rock burst causes and factors. Both global and local strategies were considered. Global strategies focus on rock burst risk reduction in regions affected by geological structures and energy accumulation, as well as avoiding formation of regions with high stress concentrations. Local strategies focus on mitigation of hazardous coal-rock mass features via a series of locally applied techniques.

**Notations**

- \(q_0\): In situ stress
- \(k_d\): Coefficient of the advanced concentrated stress
- \(q_c\): Loads of the suspended and exposed roof
- \(L\): Length of the suspended and exposed roof
- \(L_0\): Range affected by the advanced concentrated stress
- \(k_s\): Elasticity coefficient of the coal mass
- \(M_0\): Bending moment at the middle of the suspended and exposed roof
- \(E_0\): Elastic modulus of the roof
- \(I_0\): Moment of inertia
- \(h_c\): Roof height
- \(\beta\): Characteristic coefficient
- \(M_1\): Roof bending moments above the coal seam
- \(M_2\): Roof bending moments above the goaf
- \(\theta_1\): Roof rotation angles above the coal seam
- \(\theta_2\): Roof rotation angles above the goaf
- \(Q_1\): Roof shear forces above the coal seam
- \(Q_2\): Roof shear forces above the goaf
- \(u_{11}\): Strain energy density of a roof of unit width (before the initial breakage event)
- \(V_{11}\): Bending strain energy of the roof (before the initial breakage event)
- \(\Delta u_{1c}\): Strain energy density of the coal seam (before the initial breakage event)
- \(V_{1c}\): Compressive strain energy of the coal seam (before the initial breakage event)
- \(\sigma_{1\text{max}}\): Maximum hard roof tensile stress
- \([\sigma_1]\): Allowable hard roof tensile stress
- \(L_{\text{max}}\): Final length
- \(x_0\): Location of the roof breakage to the coal wall
- \(u_{21}\): Strain energy density of a roof of unit width (after the initial breakage event)
- \(V_{21}\): Bending strain energy of the roof (after the initial breakage event)
- \(u_{2c}\): Strain energy density of the coal seam (after the initial breakage event)
- \(V_{2c}\): Compressive strain energy of the coal seam (after the initial breakage event)
- \(\Delta V_1\): Strain energy variations of the roof
- \(\Delta V_2\): Energy release in the goaf
- \(\gamma\): Average specific weight of the roof.

**Data Availability**

The data used to support the findings of this study are available from the corresponding author upon request.

**Disclosure**

The authors are solely responsible for the content.

**Conflicts of Interest**

The authors declare no conflicts of interest.

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