Fracture failure analysis of hard–thick sandstone roof and its controlling effect on gas emission in underground ultra-thick coal extraction

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A B S T R A C T

In underground coal extraction of fully mechanized caving, the overlying hard–thick sandstone main roof could control the failure extent and the movement evolution of the entire overburden strata. The instantaneous failure of the hard–thick sandstone main roof possibly causes strata pressure behaviors, rock-bursts and abnormal gas emissions, which may result in equipment damages and casualties. Tashan coalmine was chosen as a field case study base because of its super great mining height (SGMH) and the overlying hard–thick sandstone roof (HTSR). This mine has experienced a great deal of damaging hydraulic support and abnormal gas emission accidents caused by strata pressure behavior. The fracture failure analysis was analyzed based on “Key Strata Theory” and numerical simulation results. The hard–thick sandstone main roof could perform as a very large double-sided embedded rock beam in the primary fracture and as a cantilever-articulated rock beam in periodic fracture, simultaneously generates a huge hanging space in the gob. The fracture failure of the hard–thick sandstone main roof causes a permeability enhancement in the adjacent rock-coal strata and near face coal seam. The substantial amounts of gas stored in the remaining coal, surrounding rock strata and adjacent coal seams rush out and aggregate in the caved and fissure zone of the gob, thereby forming a huge gas cloud. The disasters due to coupled strata pressure behavior and abnormal gas emissions, which primarily occurred after primary and periodic fracture failure, are predominantly caused by the instantly fracture of main roof. When the main roof reached the ultimate broken span and underwent, rotation and collapse, substantial amounts of gas accumulating in the gob escaped to the working face under the extrusion and impaction of the caving rock strata, which easily produced abnormal gas emissions, some of which exceeded the statutory limit. Shortening the length of the HTSR failure span using hydraulic presplitting and decreasing the gas content of the coal seam using gas drainage technology are recognized as two effective approaches to solve this issue.

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1. Introduction

With the rapid development of mechanical mining equipment, the fully mechanized caving with super great mining height (SGMH) technique has been extensively used in China as an advanced underground coal extraction method with high recovery and efficiency [28]. Hard–thick roof is usually the main factor that causes excessive stress concentration and induces dynamic strata behaviors [12]. Dynamic stressing produced by key strata strenuous movement and fracture in roof caving zone can easily trigger the strong mining tremor (even rock-burst disaster) [16]. The mining-induced rock-bursts and earthquakes could indirectly trigger unusual gas emissions, and gas bursts could also trigger rock-bursts, which are referred to as gas outburst rock-bursts [15]. Due to the super great mining height and the hard–thick sandstone roof, the mining-induced strata deformation and failure area extended, the span and value of peak abutment stress enlarged, the height of caving and fissure zone increased. A series of health and safety hazards has been reported, such as hydraulic support damage, rock-burst and mine earthquake, coal and gas outburst, and abnormal gas emission. Under fully mechanized caving of the overlying hard–thick sandstone roof, the roof does not cave in a continuous manner respective to time and consequently forms a large overhang rock beam containing induced stress and concentrated energy. The accompanying dynamic load can cause rock burst or mine earthquakes. Furthermore, the roof rock debris falls at a high velocity due to the great height of the hanging space, and the temperature increase or spark which caused by the vibration and collision of roof rock debris may induce gas ignition even explosion [26,35,37]. Rock bursts easily lead to other mine disasters, the most serious of which is abnormal gas emissions. There have been numerous reports of such events. In Germany, Hauula Trask coalmine (1955) and Ruhr Coalfield (1981) were the sites of abnormal gas emission incidents accompanied by mine earthquakes [2,3]. In China, abnormal gas emissions following rock-burst occurred in the Huaibei coalfield and Fuxin coalfield [25]. Those accidents had the same characteristics of abnormal gas emission and rapid gas accumulation following rock-bursts, which were subsequently detonated by open lights or sparks from the mining.

In recent years, an increasing number of studies of mining-induced stress redistribution, overlying rock strata failure and gas emission have been performed with consistent results [34,24,33]. Nie et al. [19] developed a computerized tomography (CT) scanning experimental system to observe the meso-structure and fracturing in coal. Cai et al. [5] reported that the large quantities of gas emitted after weighting are the result of the joint action of abutment stress redistribution and main roof fracturing. Li et al. (2008) concluded that the large quantities of gas emission after weighting are caused by a flow increase effect induced by pressure relief. Weng et al. [27] concluded that the predominant destruction of the coal–rock mass occurs slightly earlier than the period of weighting, whereas the peak gas emissions occurs slightly later than the period of weighting. However, there has been little investigation of the interactional influence of coal–rock strata movement, strata pressure behavior and gas emissions in fully mechanized caving with SGMH.

Most naturally occurring underground rock masses associated with coal measure strata consist of intact rock with discontinuities that include fissures, fractures, joints, faults, bedding planes and shear zones (Wang et al., 2013). The distinct element method [8] and discontinuous deformation analysis have the capability of simulating large displacements and rotations and are suited to discontinuum analysis of underground fully mechanized caving coal extraction. In this paper, a three-dimensional numerical model developed by 3DEC was used to investigate the fracture failure process of the hard–thick sandstone roof and its controlling effect on the adjacent coal seams and rock strata. In the middle of 1990s, Qian et al. [21] proposed the “Key Strata Theory” in strata control, the stratum which controls the movement of the whole or partial overburden strata is defined as the key stratum (KS), when the KS breaks, the whole or part of overburden strata above KS will subside simultaneously. Based on “Key Strata Theory” and numerical simulation results, the structural characteristics of the very large double-sided embedded rock beam performed in primary fracture and the cantilever-articulated rock beam performed in periodic fracture are analyzed. This paper studies the abutment stress field distribution, the large scale disturbance and permeability increase in the adjacent rock-coal strata and ahead self-coal induced by the instantaneous fracture failure of the hard–thick sandstone main roof. Then, the formation mechanism of the huge hanging space and gas warehouse stored in the gob, the coupling disaster-causing mechanisms of the instantaneous fracture failure of HTSR and concomitant abnormal gas emission are described. Finally, two effective approaches are proposed to mitigate and eliminate disasters associated with these process.

2. Geological setting and mining conditions

Datong coalfield, is situated in a northeast-trending syncline in the northern of the North China Block and the Shanxi Platform. Many thick coal seams in the Datong coalfield are of hardness higher than 2.5, and their roofs are of a hardness higher than 7.0, i.e., a configuration referred to as “double hard coal seam” [28]. Tashan Coalmine (TSCM) is a typical example, the hard–thick sandstone main roof produces an extremely large space and has a strong impact on the gas migration field. There have been many hydraulic support damage incidents and abnormal gas emission accidents (shown in Fig. 1) caused by the fracture failure of main roof. For example, the main roof of the 8210 working face ruptured and fell on November 30, 2012, and three gas over-limit accidents occurred over the subsequent three days.

Tashan coalmine (TSCM), is one of the largest underground coalmines in the world. It is located on the mid-east edge region of the Datong coalfield and was constructed by the Datong Coal Mine Group Co. Ltd. Its length (east–west) and width (north–south) are 24.3 km and 11.7 km, respectively. The mine produced more than 24 Mt of coal in 2013. The 8212 working
The face is situated in the northwest region of the second mining district, 2614 m along the strike and 230 m along the dip. The fully mechanized caving with super great mining height (SGMH) technique employed by this working face are shown in Fig. 2(a) and (b). The generalized stratigraphic column of this working face is shown in Fig. 2(c). The working face is in the #3–5 coal seam, which is a typical extra-thick and coalescent coal seam whose thickness varies from 7.25 m to 20.19 m with an average of 11.17 m. The mining height of the working face is 3.5 m, and the average caving height is 7.67 m; the top-coal recovery rate is 85%. The compressive strength of the coal seam is exceeds 25 MPa, because of the influence of the invasion of lamprophyre in the overlying rock, the upper part of the coal seam has been thermally metamorphose and is silicified, thereby creating a fabric that is friable and fragile. The intact layered hard and thick sandstone main roof (HTSR) is approximately 35.5 m thickness lies 11.8 m above the coal seam, and has compressive strength of approximately 110 MPa.
3. Mining-induced overburden movement and collapse law

3.1. Structural characteristics of coal measures stratum

The stratified rock mass structure is the significant characteristic of coal measures stratum. The tensile and shear strength of the roof rock is very low compared to its compressive strength. The fracture characteristics are thus primarily due to tensile failure or shear failure. Under normal circumstances, the length is longer than twice that of the primary fracturing in the working face, and the rock beam model was used in the calculation. The structural representation of HTSR in Tashan Coalmine could be simplified as Fig. 3. According to the “Key Strata Theory” proposed by Qian et al. [21] in the middle of 1990s, the stratified rock mass structure is primarily composed of every hard stratum, while the overlying soft stratum could be regarded as static load.

The primary fractures of mined working faces were statistically analyzed, and the average primary fracture length of the main roof in the First panel and the second panel were 45.7 and 53.9 m, respectively. The distance between the immediate roof and the main roof after the collapse of the immediate roof is assigned the variable $D$, and the maximum rotation of the broken block of the main roof is assigned the variable $D_{\text{max}}$. When $D > D_{\text{max}}$, the main roof has the potential to hang over the caved zone and behave as a cantilever structure [14]. According to Eq. (1) below [14],

$$ l = \frac{28.7}{\text{m}}; \quad D = 8.22 \text{ m} > D_{\text{max}} = 6.35 \text{ m}, $$

and thus there is a certain length at which the HTSR has the potential to hang over the caved zone and behave as a cantilever structure. The HTSR could be regarded as the “Key Strata” that controls the extent of failure and the evolution of movement in all or a majority of the overburden strata.

\[
\begin{align*}
 l &= h_m \sqrt{2\sigma_t / q} \\
 M - \sum_{i=1}^{m} h_i (K'_i - 1) &= \Delta > \Delta_{\text{max}} = h_m - \sqrt{2q_l^2 / \sigma_c}
\end{align*}
\]  

(1)

where $q$ is the load imposed by the main roof, 12.5 MPa; $h_m$ is the thickness of the main roof, 11.17 m; $\sigma_t$ is the tensile strength of main roof, 9.3 MPa; $\sigma_c$ is the compressive strength of the main roof, 70 MPa; $M$ is the mining height, 11.76 m; $K'_i$ is the bulking factor of strata, 1.25; $h_i$ is the thickness of strata between the coal and the main roof, presents in Table 1; and $l$ is the length of the broken step of the main roof, m.

3.2. Analysis of fracture failure processed in the main roof

Numerical simulations which can overcome some complicated problems in the analytic method are widely used in engineering and theoretical analysis [7,23]. 3DEC, is developed by Itasca Consulting Group Inc., which based on discrete element method to describe the discrete media behavior [13]. 3DEC is one of the most important numerical software in current rock mechanics calculations. The physical and numerical model construction work-flow for the establishment of the fracture failure process analysis of the hard–thick sandstone roof shown in Fig. 4.

The model was developed based on the geological stratigraphy condition of the 8212 working face, the length in the $x$-direction and the height in the $z$-direction are 200 m and 122 m, the thickness in the $y$-direction is set as 3 m for simplified calculation. The horizontal displacement at the lateral and the vertical displacement at the base are constrained. A vertical load ($q = \sum \rho gh = 10.6$ MPa) is applied on the top to simulate the 434 m overburden weight. There are 40 m border influence areas on the both sides, open-off cut on the left and advancing to the right with level modeling. The model used elastic block model and joint area contact Coulomb slip model. Properties assigned to blocks and joints are conventionally derived from laboratory testing programs [13]. The laboratory results for the mechanical properties of the rock and coal samples mainly cited from (Geological Report for Preliminary Construction of Tashan Coalmine, 2007). Based on the primary works of [18,29], the parameters of the main rock stratum and coal seams employed in the model have been scaled and discounted, which shown in Table 1.
The numerical simulation model present an opportunity to nearly realistically simulate the response of the coal-rock mass to the fully mechanized caving with super great height mining. The initial equilibrium state and motion states of the numerical simulation model are shown in Fig. 5. Significant coal-rock mass fracturing and sequential bed separation were observed, and the caved coal-rock mass consisted of compressible, granulated material and an accumulation unconsolidated material in the gob before the fracture of the HTSR. The #2 coal seam was situated in the caving zone, which collapsed with the collapse of the immediate roof. Abundant mining-induced fractures were generated and gradually propagated, and a huge fracture-pore space developed in the gob. Many fractures developed in the overlying rock stratum and even extended

<table>
<thead>
<tr>
<th>Rock strata</th>
<th>Density (kg/m³)</th>
<th>Bulk modulus (GPa)</th>
<th>Shear modulus (GPa)</th>
<th>Cohesion (MPa)</th>
<th>Friction angle (°)</th>
<th>Tensile strength (MPa)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Fine sandstone</td>
<td>2800</td>
<td>65</td>
<td>54.4</td>
<td>14.2</td>
<td>54</td>
<td>9.3</td>
</tr>
<tr>
<td>Medium sandstone</td>
<td>2700</td>
<td>44.1</td>
<td>41.6</td>
<td>12.3</td>
<td>48</td>
<td>5.2</td>
</tr>
<tr>
<td>Siltstone</td>
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<td>54.4</td>
<td>51.2</td>
<td>11.8</td>
<td>50</td>
<td>5.41</td>
</tr>
<tr>
<td>Pebby sandstone</td>
<td>2578</td>
<td>36.5</td>
<td>32.2</td>
<td>7.55</td>
<td>40</td>
<td>4.27</td>
</tr>
<tr>
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<td>1.2</td>
<td>2.65</td>
<td>30</td>
<td>2.39</td>
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<tr>
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<td>26.5</td>
<td>21.2</td>
<td>4.57</td>
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<table>
<thead>
<tr>
<th>Rock strata</th>
<th>Normal stiffness (GPa)</th>
<th>Shear stiffness (GPa)</th>
<th>Cohesion (MPa)</th>
<th>Friction angle (°)</th>
<th>Tensile strength (MPa)</th>
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</thead>
<tbody>
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<td>86</td>
<td>0.96</td>
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<td>17</td>
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<td>0.63</td>
<td>18</td>
<td>14.2</td>
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<td>85</td>
<td>0.71</td>
<td>23</td>
<td>15.4</td>
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<tr>
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<td>54.2</td>
<td>0.46</td>
<td>13</td>
<td>5.5</td>
</tr>
<tr>
<td>Coal seam</td>
<td>3.2</td>
<td>3.4</td>
<td>0.13</td>
<td>10</td>
<td>2.3</td>
</tr>
<tr>
<td>Mudstone</td>
<td>32</td>
<td>30</td>
<td>0.26</td>
<td>12</td>
<td>7.6</td>
</tr>
</tbody>
</table>

The numerical simulation model present an opportunity to nearly realistically simulate the response of the coal-rock mass to the fully mechanized caving with super great height mining. The initial equilibrium state and motion states of the numerical simulation model are shown in Fig. 5. Significant coal-rock mass fracturing and sequential bed separation were observed, and the caved coal-rock mass consisted of compressible, granulated material and an accumulation unconsolidated material in the gob before the fracture of the HTSR. The #2 coal seam was situated in the caving zone, which collapsed with the collapse of the immediate roof. Abundant mining-induced fractures were generated and gradually propagated, and a huge fracture-pore space developed in the gob. Many fractures developed in the overlying rock stratum and even extended

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**Fig. 4.** Model construction work-flow adopted for analysis of fracture failure process.
to the top of the model. The fracture-pore space in the gob and the fissure zone could be regarded as the gas migration channel and the agglomeration space of the pressure relief gas.

With the advance of the working face, the top coal behind the supports are caved and mined out, the false roof and the immediate roof irregularly collapse, dilate, accumulate and fill the gob. But it is insufficient to fill the space left behind after the fully mechanized caving of the extremely thick coal seam. Without the support of the coal and the immediate roof, the THMR supports the entire overburden and prevents the free movement of the overlying conditional stratum, which forms a temporary equilibrium rock beam. Then, the HTSR forms a double-sided embedded beam (Fig. 3(a)) with one side supported by the coal wall in the front and the other side supported by the coal wall in the back. Consistent with masonry beam theory, when the ultimate broken span is reached, the HTSR would undergo spallation and break up at the beginning of the middle region in the lower part of the main roof. The HTSR would then squeeze together during the subsidence and rotation, produce a powerful horizontal thrust, rub against itself, and as appear two destabilizing forms undergoing slipping and rotation. After the primary fracture, with the advancing working face, the main roof gradually enters a period fracture stage. Given the hardness and thickness of the main roof, the hanging rock beam span is generally long, with one side interlocking with the neighboring main roof block and being supported by the coal wall in the front, whereas the other side hanging above the gob. It acts as a temporary equilibrium cantilever-articulated rock beam (Fig. 3(b)). The main roof presented periodic breakdown movement as the transferring rock beam maintained the transferring force connection in the advancing direction.

Comprehensive analysis the theoretical analysis of the structural characteristics and the numerical simulation. The HTSR controls the failure extent and the movement evolution of the entire overburden strata, could perform as a very large double-sided embedded rock beam in the initial mining period and as a cantilever-articulated rock beam in the normal mining period. It simultaneously forms a huge hanging space in the gob. The roof structure is composed of top coal, the immediate roof and the main roof. The movement and caving span of every rock strata is vary depending on the different rock mass conditions of strength, thickness and the development of bedding and joints. The movement of the overlying rock stratum does not simply develop upward one layer; it occurs by individual composite structures and moves regularly as a group.

3.3. Pressure relief effect of the main roof

The gas stored by free and adsorbed forms into the coal and the adsorbed gas typically accounts for over 90%, while no such relationship is known to exist for rocks where the gas is free gas in pores [11,9,31]. Under the influence of mining
activities, the gas existing in the adjacent coal seams and rock strata desorb gradually and release into the gob primarily through mining-induced fractures. The degree of degassing is particularly dependent on the physical properties and geometry of the strata, the thickness and distance between gas sources, the shape and extent of the relaxed zone, and existing differential pressure between in situ gas and mine ventilation system [17]. The recognized worldwide discontinuous maximum vertically boundary lines of the relaxed zone in the roof and floor strata are 200 m above and 100 m below [17,20,1].

To explore the influence scope and extent of the fully mechanized caving with SGMH on the lower adjacent rock-coal strata, the #8 coal seam and its surrounding rock strata were added to the new numerical model (Fig. 6) established by FLAC 3D. The model covers an area of 440 m by 360 m and 172.4 m in height, the Mohr–Coulomb failure criterion was selected for the simulation, the initial properties of the rock mass are listed in Table 1.

According to the numerical model results, the #2 coal seam located 9.4 m above the mined coal seam, the #Shan4 coal seam located 35.3 m above the mined coal seam, and the #8 coal seam located approximately 34.8 m below the mined coal seam should be regarded as the dominating gas emission sources. And the primary surrounding rock strata should be considered; these include immediate roof, main roof, immediate floor, main floor, and the sandy mudstone between main floor and #8 coal seam.

Fig. 6. Z-stress redistribution of the adjacent coal seams after the ultra-thick coal seam extracted.
4. Gas emission sources and process analysis

4.1. Gas emission from the coal wall and coal seam ahead

The gas content of the #3–5 coal seam in the 8212 working face were measured using the determination method of the gas desorption index with drill cuttings [10]. The gas content is generally too low to less than 4.0 m$^3$/t, and the maximum measured gas content is 3.23 m$^3$/t. The sources of gas emission in fully mechanized caving with SGMH are similar to those associated with traditional long-wall mining. These sources are primarily the coal walls, the mined and caving coal body and the gob, although the proportions of each component are very different. In the process of mining and caving, the massive coal body is broken into granular coal, and the desorption intensity increases with the decrease in the coal particle size. With the advancing of the working face, the exposure of freshly cut ribs is fast and ceaseless, abundant pore-cracks are generated in the distressed zone ahead.

The main influence factors of abutment stress are the roof features, coal seam strength, coal seam thickness and recovery ratio of top coal. The distribution range of the abutment stress, the width of the plastic and the distance between the abutment peak and the coal wall in fully mechanized caving with SGMH is greater. Xue et al. [30] proposed there is a rapid increment of permeability after roof failure of coal in front of the mining face. Relieved methane flow is affected by the rock mass permeability, which is significantly affected by the abutment stress redistribution induced by extraction [6,32]. The mining-induced variation in the abutment stress in front of the coal wall destroys the original structure of the coal and the roof, accompanying with the destabilizing and caving of the main roof. The abutment stress loading on the coal seam gradually transfers to the deep regions, and the range of the stress relief zone expands. Fig. 7 depicts the structure partition of the top coal and abutment stress. The top-coal structure in fully mechanized caving mining could be divided into original status zone, compacted zone, disturbed zone, fractured zone and caving zone. And the abutment stress could be divided into original stress zone, stress concentration zone and stress relief zone. In the stress relief zone of abutment stress, corresponding to the disturbed zone, fractured zone and caving zone of coal seam, the original fractures open and expand, mining-induced fractures generate and propagate, the coal body is broken up and caved, the strength is reduced and permeability is increased, the gas migration channels are unblocked and gas desorption process is aggravated. Coal gas rush into the working face through fractures under the gas pressure gradient between in situ gas pressures and the mine atmosphere. This referred to as the “pressure relief and flow increase effect”.

4.2. Gob gas emission sources and process analysis

Under the condition of thick-hard sandstone overlying, the fully mechanized caving with SGMH causes large scale disturbance of the coal measure strata, the disturbance could increase the permeability of the coal-rock strata through a reduction on the stress as well as the generation and propagation of mining-induced fractures. Then the gas contained in the disturbed coal-rock strata can migrate towards the low pressure gob. Due to the gas pressure gradient and the development of surrounding rock-coal mass fractures, a large amount of gas rushes into the gob along the fractures and accumulates in the upper part of the gob. The caved zone and fissure zone of the gob exhibits porous-connectedness spatial characteristics and forms a huge gas warehouse. The gas of the gob ($Q_g$), substantially originates from the adjacent coal seams ($Q_c$), the surrounding rocks strata ($Q_r$) and the remaining coal ($Q_{rc}$). The gob gas emission sources and process analysis is based on the predicted method by different gas source (PMDS) elaborated by [1,9,17], and appropriate modification referenced the research results of [20,22,36].

![Fig. 7. Partition schematic diagram of the top-coal and abutment stress in fully mechanized caving with SGMH.](image-url)
The gas existing in the gob is complicated and difficult to monitor, because the gob is a relatively sealed space that is deficient in oxygen and enriched in gas. In recent years, there has been a great deal of research into gob gas emissions. Herein, the estimation of the quantity of gas emitted by the gob is based on Eq. (2) [20,17,36,1]. The contributions of strata-relaxation gas emission from roof and floor gas sources and the adjacent coal seams are a function of the distance from the mined coal seam [4]. Noack [20] concluded that the surrounding rock strata could be considered as fictitious coal seams for which reduced gas contents are assumed for the purpose of prediction. The corresponding reduction factors are 0.019 for mudstone, 0.058 for sandy shale and 0.096 for sandstone. The gas emission ratio (Table 3) of the surrounding strata and the adjacent coal seams depend substantially on the empirical degassing coefficients (Fig. 8), and certain modifications based on the degree of strata relaxation were developed using the FLAC finite difference program (Fig. 6). The amount of gas in the adjacent coal seams and the surrounding rock strata are estimated based on the gas content in the #3–5 coal seam and the relationship between the burial depth and the gas content.

\[
Q_g = Q_c + Q_r + Q_{9m}
\]

\[
Q_c = \frac{A}{1440} \sum_{i=1}^{n} (X - X_i) \cdot \frac{m_i}{X_i} \cdot K_i
\]

\[
Q_r = \frac{A}{1440} \sum_{j=1}^{n} X_j \cdot N_j \cdot \frac{m_j}{X_j} \cdot K_j
\]

\[
Q_{9m} = \eta_m (1 - \eta_f) \left( k X - X_j \right) + Q_1
\]

\[
X_t = X_c \left( \frac{100 - \text{Mad} \%}{100} \right) A_{ad} \%
\]

where \( A \) is the daily production, 18.4 kt/d; \( K_i \) and \( K_j \) are the emission ratio of the adjacent coal seams and surrounding rock strata, respectively; \( m_i \) and \( m_j \) are the thickness of the adjacent coal seams and surrounding rock strata, respectively; \( M \) is the mining height, 11.17 m; \( X_t \) is the fictitious gas content of the surrounding rock strata; \( N_j \) is the reduction factors of the surrounding rock strata; \( \eta_m \) is the average efficiency of mining and caving coal, 25.1 t/min; \( \eta_f \) is the average remaining coal rate of mining and caving coal, 10.3%; \( Q_1 \) is the gas emission quantity emitted from the coal walls of the gob, which could be ignored because of the working face employing retreating mining method; \( X_t \) is the gas content of the coal bulk when the gas pressure is 0.1 MPa; \( X_c \) is the dry ash-free basis residual gas content of the coal bulk according to Table 2.

The calculated gas emission quantity of gas emitted from the adjacent coal seams and the surrounding rock strata of 8212 working face is shown in Table 3. The total gas emission of the gob is shown in Table 4. On the basis of the prediction, the gob gas emission quantity of the 8212 working face is approximately 31.26 m³/min during the period of regular advance. Note that this prediction assumes that no gas drainage or roof presplitting measures are applied to the working face. The
gas emission before the main roof primary fracture is regarded as dominating emitted from self-coal seam (coal face and mined and caved coal body) without the gas emitted from the gob.

5. Coupling disaster-causing mechanisms

Strata pressure behavior and abnormal gas emission coupling disasters mainly include two situations: one occurs in the primary fracture of the main roof, and the other occurs in the periodic fracture. The former is worse because the length of the broken span of the main roof in the primary fracture is approximately twice that of the periodic fracture, whereas the latter is the most common disaster because of the periodic fractures that accompany the entire mining process of the working face. Fig. 9 shows the coupling disaster-causing mechanism schematic diagram of the strata pressure behavior and abnormal gas emission.

Table 2
Residual gas content of coal bulk with different volatile content [1].

<table>
<thead>
<tr>
<th>Volatile (Vdaf) %</th>
<th>6–8</th>
<th>8–12</th>
<th>12–18</th>
<th>18–26</th>
<th>26–35</th>
<th>35–42</th>
<th>42–56</th>
</tr>
</thead>
<tbody>
<tr>
<td>$X_c$ (m³/t•r⁻¹)</td>
<td>9–6</td>
<td>6–4</td>
<td>4–3</td>
<td>3–2</td>
<td>2</td>
<td>2</td>
<td>2</td>
</tr>
</tbody>
</table>

Notice: the residual gas content could be taken as the gas adsorption quantity under 0.1 MPa.

Table 3
Gas emission emitted from adjacent coal seams and surrounding rock strata.

<table>
<thead>
<tr>
<th>Location</th>
<th>Coal-rock mass</th>
<th>Thickness (m)</th>
<th>Gas content (m³/t)</th>
<th>Residual gas content (m³/t)</th>
<th>Spacing (m)</th>
<th>Reduction factors (%)</th>
<th>Emission ratio (%)</th>
<th>Gas emission (m³/min)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Upper adjacent</td>
<td>Shan4</td>
<td>1.78</td>
<td>2.4</td>
<td>1.61</td>
<td>35.3</td>
<td>78.3</td>
<td>100</td>
<td>3.07</td>
</tr>
<tr>
<td>#2</td>
<td>1.69</td>
<td>3.2</td>
<td>1.61</td>
<td>9.39</td>
<td>33.9</td>
<td>100</td>
<td>100</td>
<td>2.04</td>
</tr>
<tr>
<td>Lower adjacent</td>
<td>#8</td>
<td>6.12</td>
<td>4.6</td>
<td>1.61</td>
<td>34.8</td>
<td>38.5</td>
<td>100</td>
<td>7.41</td>
</tr>
</tbody>
</table>

The gas emission emitted from adjacent surrounding rock strata

<table>
<thead>
<tr>
<th>Location</th>
<th>Spacing (m)</th>
<th>Reduction factors (%)</th>
<th>Emission ratio (%)</th>
<th>Gas emission (m³/min)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Upper adjacent</td>
<td>3.9</td>
<td>0.019</td>
<td>100</td>
<td>2.04</td>
</tr>
<tr>
<td>Immediate floor</td>
<td>3.7</td>
<td>0.042</td>
<td>100</td>
<td>2.04</td>
</tr>
<tr>
<td>Sandy mudstone</td>
<td>3.9</td>
<td>0.058</td>
<td>45.3</td>
<td>2.30</td>
</tr>
</tbody>
</table>

Table 4
The prediction of the gas quantity emitted from the gob of the 8212 working face.

<table>
<thead>
<tr>
<th>$\eta$ (t/min)</th>
<th>A (kt/d)</th>
<th>$Q_g$ (m³/min)</th>
<th>$Q_g$ (m³/min)</th>
<th>$Q_{\infty}$ (m³/min)</th>
</tr>
</thead>
<tbody>
<tr>
<td>25.1</td>
<td>18.4</td>
<td>11.76</td>
<td>9.97</td>
<td>9.53</td>
</tr>
</tbody>
</table>

gas emission before the main roof primary fracture is regarded as dominating emitted from self-coal seam (coal face and mined and caved coal body) without the gas emitted from the gob.

Fig. 9. Coupled disaster-causing mechanism of the strata pressure behavior and the gas abnormal emission.
During the initial mining period, the gob in the working face has not yet formed, and the predominant gas emission sources are the process of mining and caving mining. The gas emission quantity is relatively low. With the advancing of the working face, the gob forms and expands gradually. The presence of the gob provides a certain freedom space for the surrounding coal-rock strata, which induces the fractures development and the permeability increase; then, the gas storing in the surrounding rock and the adjacent coal seams escape and accumulate in the gob. Because of the restrictive recovery rate of the top coal, there is a few of residual coal in the gob, and the gas bearing in the residual coal would escape and accumulate in gob as well. The hard main roof would form a huge double-sided embedded beam and create a huge hanging space in the gob. Strata pressure behaviors drive the high concentration of the gas moving outwards and increase the fractures in the roof and floor rock; then, the velocity of the gas escaping from the surrounding rock and the neighboring coal seam increased. Due to the super strength of the hard–thick sandstone main roof, it could not rupture in time with the advancing of the coal face and then performed a huge space in the gob. As a result, there is only a fractional amount of gob gas that rushes toward the working face with its gradual advance. When the main roof reached the ultimate broken span, with the slumping and breaking of the main roof, the vast majority of gas stored in the gob would rush out instantaneously with the positive pressure of the main roof fracturing failure and the negative pressure of the working face. One result in, a significant potential safety hazard to the working face.

Periodic weighting caused by mining could also have a periodicity influence on the coal seam, which changes the fractures of the coal seam correspondingly and accelerates the process of gas desorption and migration; thus gas emission increases. With the advancing of the working face, some points of the main roof ahead reach the ultimate tensile strength and generates and extends fractures; then the immediate roof begins to break up and fall down with the load transferred by the subsidence of the main roof, which forms the small periodic weighting. With the continued advancing of the working face, the length of the cantilever structure of the main roof increases continuously and reaches the ultimate broken span without the support of the immediate roof; the tensile fracture in the main roof extends and cuts through gradually. Then, the main roof begins to break apart and fall further and across a wider area, which forms the large periodic weighting. The movement process of the cantilever-articulated rock beam could be divided into separation, suspension, rotation and collapse. With the period failure and the rotation of the main roof, the abundant gas which accumulated in the gob high concentration formed gas airflow, and rushed into the working face under the comprehensive effect of extrusion and impaction caused by the caving rocks. The result is an increase in the gas volume fraction at the working face, which easily leads to the gas concentration in excess of the allowable level.

6. Comprehensive control technologies of hydraulic presplitting and gas drainage

Based on the mining-induced coal-rock mass fracture characteristics and pressure behavior and the gas migration features and the gas emission trends in the gob and fissure zone, a comprehensive strategy to control the strata pressure behavior and abnormal gas emissions was proposed, as shown in Fig. 10. Two approaches are included to address this issue: one is to shorten the length of the main roof caving span via the hydraulic presplitting of the roof, and the other is to reduce the amount of gas emissions through comprehensive gas drainage technology to reduce the gas content of the coal seam.

Hydraulic presplitting technology could weaken the mechanical properties of the roof and top-coal, and improve the cavability and permeability. This process accelerates the caving of the roof over time and reduces the degree of the primary roof fracture over a large area. The result is coal body fragmentation by the mine pressure, which reduces the amount of coal and gas remaining in the gob and enhances the communication characteristic between the roof drainage roadway and the gob in the initial mining period. The measure roadway, which was connected to the roof drainage roadway and parallels the open-off cut with a spacing of 25 m, is arranged alone on the floor of the #2 coal seam; the length was 150 m. Hydraulic presplitting boreholes included the following: those of the main roof were arranged along the roof of the measure roadway and penetrated through the main roof; those of the immediate roof and the top-coal hydraulic presplitting boreholes were arranged along the floor of the measure roadway and extended to the top-coal; and the open-off hydraulic presplitting boreholes were arranged along the roof of the open-off cut and penetrated through the main roof, perpendicular to the main roof.

The comprehensive gas drainage system primarily included the roof drainage roadway and the upper–middle–lower three-layer coal seam drainage borehole, assists with gas drainage in the upper corner and ground well drilling technology.
The roof drainage roadway is arranged along the floor of the #2 coal seam, which is situated in the roof caved zone. It directly extracted gas from the gob, released the disturbed zone of the front coal seam and diluted the gas concentration of the upper corner and rear tail roadway. Furthermore, the roof drainage roadway could be used to forecast the strata pressure behavior in the meantime because it is the location of the initial strata pressure behavior due to the suddenly increase in its gas concentration and gas emission. Considering the extra-thickness and moderate permeability of the #3–5 coal seam, one row of drainage boreholes in the bedding coal seam was far from sufficient for the required gas drainage. The upper–middle–lower three-layer bedding coal seam drainage borehole constructed in the intake airflow roadway and the return airflow roadway was proposed; a 10 m superseding length was expected in the middle of the working face.

7. Conclusions

The application and generalization of fully mechanized caving with SGMH poses disasters to the coupling of strata pressure behavior and abnormal gas emission, which have been rarely seen in traditional long wall mining. Such disasters may lead to gas explosions, equipment damage, blocking of escape routes, and resource production disruptions in coal mines. This study focused on mining-induced coal-rock strata fracture characteristics, abutment stress field distributions and gas emission and migration features of fully mechanized caving with SGMH and revealed the coupled disaster-causing mechanisms of strata pressure behavior and abnormal gas emission. Then, two approaches were proposed to address this issue.

(1) The thick-hard sandstone roof could perform as a very huge double sided embedded rock beam in the initial mining period and a cantilever-articulated rock beam in the normal mining period, which generates huge hanging space in the gob. The movement of rock stratum performed as a composite structure and moved in groups, the HTSR which bears the entire weight and prevents the subsidence of the overburden could be regard as the “Key Strata” and controls the failure extent and the movement evolution of the entire overburden strata.

(2) The gas emission in fully mechanized caving with SGMH has differ from those in traditional long wall mining in a few respects. Because the HTSR does not rupture promptly, there is only a fractional rush of gob gas to the working face with its gradual the advance, and the vast majority of the gas is stored in the gob and formed a huge gas warehouse.

(3) The instantaneous fracture failure of HTSR and concomitant abnormal gas emission coupling disasters, which primarily occurred after primary and periodic fracture of the HTSR, are predominantly caused by the instantly fracture of main roof and aggravated by the abutment stress relief. When the HTSR reached the ultimate broken span, with the rotating and collapsing of HTSR, substantial amounts of gas accumulating in the gob escaped into the working face under the comprehensive extrusion and impaction effect of the caving rocks, which easily cause a significant potential safety hazard.

(4) Disasters due to coupled rock bursts and abnormal gas emission could be mitigated and eliminated by shortening the length of main roof caving span by hydraulic presplitting roof technology and reducing gas emission quantity via gas drainage technology.

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