A safe mining approach for deep outburst coal seam groups with hard-thick sandstone roof: Stepwise risk control based on gas diversion and extraction

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Abstract
Protective seam mining is the preferred method for gas control and safe mining in high-outburst coal seam groups. However, a hard-thick sandstone roof (HTSR) triggers new gas problems when it covers directly above the first-mined protective coal seam. To solve these problems, a typical case study in No. 12 Coal Mine was conducted. The simulation results indicate that the HTSR greatly prevents the plastic deformation of the upper roof formations without affecting the expansive deformation in the underlying protected seams. The pressure-relief gas in protected seams will flow into the protective seam without continuing to migrate to roof strata, thereby turning the protective seam into a new high-risk area. Therefore, roof presplitting (RPS) technology was proposed to effectively break the HTSR, create roof fracture passages, and divert the gob gas of the protective seam upward. Meanwhile, gob-side entry construction (GSEC) was conducted to enhance gas discharge in the protective seam by adopting rapid filling, roof cutting, and active supporting technologies. Through the upper protective seam mining, RPS, and GSEC, gas migration passages were constructed step-by-step to divert the gas from high-risk areas to low-risk areas. As a key first step, gas diversion not only optimized the distribution of gas sources but also significantly reduced gas hazards. Source-by-source extraction was then performed to effectively control the high-concentration gas in the newly formed gas source, further eliminating the mining risk. By applying the stepwise risk control method combined with gas diversion and extraction, the safe production was successfully achieved with excellent outburst prevention effects, and the gas extraction rate was significantly improved to more than 66%. This practice could be used for deep high-gassy or high-outburst coal seam group mining.

KEYWORDS
gas control, gas migration passage, gob-side entry, outburst prevention, protective seam mining, roof presplitting
1 | INTRODUCTION

Coal has been the major source of global energy for the past two centuries. In 2018, coal accounted for 27.2% of the world’s primary energy consumption; in China, it accounted for 58%. Coalbed gas, which mainly consists of methane, has been considered a major hazard affecting coal mining safety and productivity since the early 19th century. Gas-related accidents mainly include abnormal gas emissions, gas explosions and outbursts, and others. According to statistics, these accidents constitute 45% of the total coal mine accidents in China. However, coalbed gas is also a typical unconventional natural gas, with reserves of 240 trillion cubic meters above a depth of 2000 m, which is more than double the proven reserves of conventional natural gas. Moreover, effective gas extraction may reduce the greenhouse effect. Recently, because of high consumer demand, coal mining is extending to deeper and deeper levels. Outburst risks of deep coal seams are increasing sharply, and gas accidents are becoming more frequent and serious because of the increase in geostress, gas content, and gas pressure. Also, the permeability of the outburst coal seam is extremely low in deep strata, especially in China, making gas extraction difficult. Hence, gas risk control should be further enhanced to ensure the safe exploitation of deep reservoirs.

Protective seam mining combined with underground gas pre-extraction is the preferred method for gas control in high-risk coal seam groups because of their significant pressure-relief and permeability enhancement effects, excellent gas extraction effectiveness, and relatively low cost. The protective seam with lower or no outburst risk is generally selected as the first mining layer to release pressure in advance and increase the permeability of other adjacent high-risk outburst coal seams (protected seams) in the coal seam group. On the one hand, the extraction rate of the pre-extraction boreholes in the protected seam is drastically increased because of mining-induced stress change. On the other hand, for upper protective seam mining (UPSM), gas in the protected seam can be transferred into the protective seam through the induced vertical and bed-separated fractures of the adjacent rock formations. After that, mining protected seams become safer because of the dual effects of gas extraction and diversion.

However, gas diversion between the protected seam and protective seam increases the apparent gas content of the upper protective seam, which makes gas risk control more complicated and more difficult. In general, roof rock formations will sink and cave under their own weight during protective seam mining, and a roof fracture zone will normally form above the gob. Based on this, gob gas can migrate upward into the roof strata, which significantly reduces the gas accumulation in the protective seam. Moreover, the roof fracture zone becomes a gas enrichment area and is conducive to efficient gas extraction. Hence, the roof fracture zone formation has a substantial influence on the gas control of the protective seam. Many studies have been conducted to find ways to use the stress and permeability changes in the roof strata to enhance gas production. A three-dimensional optimal methane extraction zone was found above the gob, where the high-concentration gas can be captured with a relatively high and stable flow rate. Yuan indicated that there is an annular-shaped fracture zone above the gob with abundant gas and proposed an appropriate gas extraction method, increasing gas drainage to 70%. Therefore, the formation of the roof fracture zone and the induced gas diversion effect are essential for coalbed gas control, especially for the high-outburst coal seam group mining.

Roof occurrence conditions affect a series of rock movement activities, including subsidence, breakage, and caving, thereby controlling the autonomous formation of roof fracture passages, which are important for gas diversion and control in high-gassy coal seam group. However, hard-thick sandstone roof (HTSR) formations are widespread in the underground formations, some of which are distributed directly above the coal seam. The HTSR greatly slows down strata movement, resulting in a long-term overhanging roof over the gob. Some studies have shown that this overhanging roof induces excessive load on supports, and its sudden caving induces abnormal gas emissions and even dynamic strata behaviors such as rock outbursts. In addition, the HTSR will also lead to more complicated gas problems since it may have potential negative impacts on the upward diversion of gob gas and reduce the effect of protective seam mining.

Many scholars have discussed the disaster-causing mechanisms related to the HTSR, and several control technologies have been applied, such as deep-hole blasting, high-pressure water jet, and hydraulic fracturing. Wang et al studied the stress evolution and damage areas induced by the blasting shock wave with LS-DYNA3D. The results optimized the blasting parameters and have been successfully applied in the coal mine. He et al proposed a directional hydraulic fracturing technology to reduce stress concentration and rockburst hazards by breaking the rock with a high-pressure liquid. Wang et al numerically analyzed the mechanism of abnormal gas emissions caused by the HTSR using 3DEC and used hydraulic presplitting roof technology to shorten the main roof caving span and prevent abnormal gas emission. However, hydraulic measures often involve certain difficulties, such as those with predicting roof fall, long operation time, high cost, and strong dependence on roof rock characteristics. In general, deep-hole blasting is still preferred to treat the HTSR, used successfully in today’s coal mines. However, it is mainly suitable for eliminating the
common disasters mentioned above. For the deep outburst coal seam group, the negative effects of the HTSR on gas control and the corresponding roof control methods should be further investigated to ensure safe mining.

Moreover, gas diversion from the protected seam greatly increases the load on the ventilation system of the protective seam, which increases the required gas discharge capacity and airflow field distribution of the ventilation system. Because of the limited air volume and the risk of gas overlimit at the upper corner, the conventional U-type ventilation system is no longer appropriate for the deep high-gassy working face. Y-type ventilation presents obvious advantages in gas control, including a reasonable roadway layout, a favorable airflow system, and the prevention of gas overlimit at the upper corner. This ventilation method was favored by many high-gassy coal mines and has been extensively used. The construction of gob-side entry (GSE) is the key to forming a Y-type ventilation system. GSE is usually constructed on the side of the gob with special support to be reused for the next panel. To improve GSE construction (GSEC) efficiency, many scholars and engineers conducted field research to help upgrade the filling equipment, optimize roadside filling processes, improve filling material, and strengthen roadway support. Nonetheless, the high in situ stress in deep strata makes it difficult to build the GSE. Since the surrounding rock and filling wall have been exposed to strong dynamic pressure for a long time, the roadway contracts sharply and cannot meet the space requirements for gas drainage. In addition, the presence of an HTSR hastens GSE deformation and exacerbates GSEC difficulty. The formed overhanging roof induces a lateral cantilever over the GSE, which causes intense stress concentration. Field practice indicates that when the lateral cantilever is over 10 m, the stress on the filling wall of the GSE is more than 20 MPa. Stress reaches 40 MPa when the cantilever exceeds 20 m. Therefore, both the construction process and the support method of the GSEC urgently need improvement to meet the challenges brought by the high stress and the presence of an HTSR in the deep coal seam group.

To solve the facing gas control problems, a field study on protective seam mining with an HTSR was carried out in the kilometer-deep coal seam group of No. 12 Coal Mine, China. The influence of the HTSR on UPSM was numerically studied, and the evolution of the plastic deformation zone was analyzed. Based on the simulation results and the existing field problems, the roof presplitting (RPS) and GSEC methods were proposed to, respectively, construct gas migration passages in the roof strata and working room for deep coal seam group mining. Accordingly, a stepwise gas diversion method was put forward, including UPSM, RPS, and GSEC, so that coalbed gas can be transferred from high-risk areas to a low-risk or safe areas step-by-step. Then, characteristics of gas migration and enrichment were considered, and source-by-source extraction method was applied to each source to efficiently control coalbed gas. Meanwhile, the space-time borehole arrangement was further considered. The gas risk control method (combined with gas diversion and extraction) was finally proposed to ensure the safe mining of the deep high-outburst coal seam group directly covered with an HTSR. The field application was successfully carried out in No. 12 Coal Mine, and the effect was well verified.

## 2 Reservoir Characteristics

### 2.1 Geology

As shown in Figure 1A,B, Pingdingshan coalfield is one of China's major coking coal production bases, located in central Henan Province. No. 12 Coal Mine is a main producing mine at the southeastern rim of the coalfield. At present, No. 12 Coal Mine has a mining depth of more than 1000 m, which poses a high-outburst danger. As shown in Figure 1C, the study area is in the northeast part of No. 12 Coal Mine, which has fewer structures. The coalbeds possess better retention conditions, resulting in higher gas content in this area. The maximum value in the J-15 coal seam can reach 40 m³/t. As a result, the influence of gas on outburst disasters is more pronounced in this area.

### 2.2 Coal reservoirs and their outburst risk analyses

As shown in Figure 2A and Table 1, there are three coal seams in the Shanxi group, all of which are all distributed below one thousand meters in No. 12 Coal Mine. The mine's main commercial seams are J-15 and J-16-17, with thicknesses of 3.40 m and 1.53 m, respectively, and both have high-outburst risks. Figure 2B shows a positive relationship between gas pressure and burial depth in the J-15 coal seam. The gas pressure exceeds 2 MPa at a depth of 800 m, indicating that the gas internal energy in the seam is high, which further increases outburst risk.

To better understand the gas retention and migration mechanisms in deep coalbeds, a pore structure test was conducted through mercury intrusion porosimetry (MIP), the results of which are shown in Figure 2C and Table 2. It can be seen that J-15 coal is characterized by a microporous structure. The average pore aperture is 7.9 nm while the porosity is 4.70%. The micropores and transitional pores provide 82.58% of the total pore volume, which is beneficial for gas adsorption and storage. However, the mesopores are not well developed, with poor connectivity. The volume of the mesopores and macropores accounts for only 17.42%, which...
may not provide enough permeable space for gas migration. These analyses show that the pore structure characteristics of deep reservoirs may lead to high gas content and low permeability, which makes the J-15 coal seam difficult to extract.

To reduce outburst risks, cross-measure boreholes were performed for gas pre-extraction in the J-15 coal seam; however, the drainage rate is extremely low. After more than 700 days of predrainage, the gas pressure and content were still above...
1.5 MPa and 15 m³/t, both of which are higher than the critical values of 0.74 MPa and 8 m³/t. The J-15 coal seam still has high mining risks.

The J-14 coal seam is a thin reservoir without outburst risks. Its average thickness is 0.5 m, and the gas pressure and content are 0.20 MPa and 1.18 m³/t (Table 1). Hence, the J-14 coal seam was chosen to be mined first as a protective seam; the underlying coal seams (J-15 and J-16-17) would then receive protection effects, possibly eliminating the outburst risk. According to Figure 2A, the average spacing between the two seams (J-14 and J-15) is 10.75 m, indicating a close UPSM. Close UPSM often brings great pressure-relief effects. During UPSM, the floating effect of gas from the protected seams will cause gas accumulation in the protective seam. Therefore, the gas drainage in the protective seam (J-14) must be strengthened to ensure mining safety. Meanwhile, an HTSR covers directly above the J-14 coal seam, with an average thickness of 17.2 m and a compressive strength of 90 MPa.

### 2.3 Problems in the mining process of the kilometer-deep coal seam group

Obviously, fewer roof fractures can be generated in the overlying strata because of the HTSR, and a large hanging roof would remain above the gob for a long time (Figure 3A). Gas accumulated in the J-14 coal seam thus cannot migrate upward, which greatly increases gas overrun risks. In addition, under the squeezing effect caused by the sudden collapse of the large hanging roof, gob gas will pour into the mining space, causing serious gas accidents. With the mining of the J-14 coal seam, its apparent gas emissions gradually increased by three to four times due to the gas influx from the underlying coal seams. Because of limitations in the roadway layout, gas overflow accidents in the upper corner are frequent (Figure 3B). It is difficult to adapt U-type ventilation to meet gas drainage requirements. Therefore, in order to strengthen gas control, an attempt was made to construct the GSE behind the working face to form a Y-type ventilation

| Coal seam | Vitrinite reflectance (%) | Total pore volume (mL/g) | Specific surface area (m²/g) | Average aperture (nm) | Volume density (g/mL) | Skeleton density (g/mL) | Porosity (%) |
|-----------|--------------------------|--------------------------|----------------------------|----------------------|----------------------|-------------------------|--------------|
| J-15      | 1.31                     | 0.040                    | 20.02                      | 7.9                  | 1.19                 | 1.25                    | 4.70         |

**FIGURE 3** The situation of deep coal seam group mining in No. 12 Coal Mine
system. The 31010 working face layout of the J-14 and J-15 coal seams is shown in Figure 3C. However, under high-stress and HTSR conditions, it is hard to build a GSE efficiently (Figure 3D). The sudden collapse of a large-area hanging roof brings strong dynamic pressure, which will cause serious damage to the GSE. The GSE will contract severely after a series of stress change behaviors, such as mining, cantilever sinking, and roof collapse. The inefficient construction process and severe deformation of the GSE would make the Y-type ventilation system difficult to form, which affects gas discharge and borehole construction. As a result, the production efficiency of No. 12 Coal Mine is low.

The joint exploitation of deep coal seams directly covered with an HTSR in No. 12 Coal Mine faces the following serious challenges. First, the roof fracture passages need to be formed in the HTSR and the upper rock strata above the J-14 coal seam to promote upward gas migration and reduce gas accumulation in the upper protective seam. Second, to discharge the coalbed gas effectively, the GSE must be built efficiently to create a preferential gas drainage channel in the J-14 coal seam, which requires research into rapid construction technology and reinforced roadway support methods. Finally, after mining the protective seam, the gas emission is still as high as 18.29 m³/min in the J-15-31010 working face, and synergetic gas extraction in the J-14 and J-15 coalbeds must be done to enhance gas control.

3 | NUMERICAL SIMULATION OF UPSM WITH AN HTSR

3.1 | Numerical model setting

The factors influencing gas flow are nothing more than the gas pressure gradient and the reservoir permeability. After desorption, gas migration is greatly affected by the porosity and permeability distribution of the reservoir. To reveal the effect of HTSR on UPSM, the COMSOL Multiphysics software was used to calculate porosity, permeability, and plastic deformation with different mining distances.

The physical model was built based on the stratigraphic distribution of No. 12 coal mine, including three coal seams. In order to improve the calculation efficiency, the physical model established only includes a certain range of rock formations around the coal seam group. The size setting principle is to reduce the size as much as possible without affecting the numerical calculation results. Therefore, the model is set to be 400 m in length and 163 m in height according to the target mining distance and formation distribution characteristics. A 20 m thick HTSR is set directly above the J-14 coal seam. In this model, the vertical stress is set as 20 MPa to simulate the load of the 1000 m overburden. The bottom is fixed, and the two sides are set as roller boundary. The mining height is 1.8 m. Figure 4 shows the details of the geometric model and boundary conditions. The yield failure of rock mass conforms to the Drucker-Prager criterion, which is used to evaluate coal and rock failure caused by UPSM. The division of plastic zones below is based on this criterion.

Based on the theory of dual-porosity medium, we established a multi-field coupling model that takes into account dynamic diffusion, effective stress, matrix shrinkage, and Klinkenberg effects to achieve the cross-coupling of coal deformation, gas diffusion, and gas flow fields. The detailed process of modeling can refer to our previous article. The numerical simulation is implemented by writing the governing equations of each part into COMSOL Multiphysics software. The porosity equation was described in Equation (1) considering the adsorption swelling effect.

\[
\varphi_f = \varphi_{f0} + \frac{a_f}{M} (p_f - p_{f0}) + \frac{a_p}{M} (p_p - p_{p0}) + \frac{a_p RT}{MV_m} \ln \left( \frac{1 + b p_{p0}}{1 + b p_p} \right)
\]  

(1)

where \( \varphi_f \) and \( \varphi_{f0} \) are the current and original fracture porosity; \( a_f \) and \( a_p \) are the effective-stress coefficients of fractures and pores; \( p_f \) and \( p_{f0} \) are the current and original gas pressures in fractures; \( p_p \) and \( p_{p0} \) are the current and original gas pressures in pores; \( M \) and \( V_m \) are restrained axial modulus and gas molar volume, 22.4 L/mol, respectively; \( a \) and \( b \) are Langmuir adsorption capacity, 15 m³/t, and adsorption constant, 1 MPa⁻¹; \( \rho_s \) is the real density of coal, 1600 kg/m³; \( R \) and \( T \) are gas constant, 8.3143 J/(mol K); and temperature, 293.14 K, respectively; and \( p \) is gas pressure.

Based on the third power relationship between porosity and permeability in the Kozeny-Carman formula, and considering the Klinkenberg effect, the permeability model can be expressed as follows:

\[
k_e = k_0 \left( 1 + \frac{a_f}{M \varphi_{f0}} (p_f - p_{f0}) + \frac{a_p}{M \varphi_{p0}} (p_p - p_{p0}) + \frac{a_p RT}{MV_m \varphi_{f0}} \ln \left( \frac{1 + b p_{p0}}{1 + b p_p} \right) \right)^3 \left( 1 + \frac{c}{p_f} \right)
\]  

(2)

where \( K_e \) is the effective permeability, and \( c \) is Klinkenberg coefficient. All the input parameters of the model are from the
typical values of the No. 12 Coal Mine (Table 3). Most of these parameters are derived from experimental tests and empirical values. Some parameters not listed are from literature references.55-57 When calculating the porosity and permeability of rock formations, the effects of adsorption swelling are ignored. The porosity and permeability variations in the J-15 coal seam were recorded by monitoring lines (AA’).

3.2 | Results and analysis

3.2.1 | Evolution of porosity and permeability in the protected seam

Porosity and permeability variations in the J-15 coal seam during J-14 coal seam mining are shown in Figure 5A,B. The maximum porosity in the first 60 m of mining gradually increases. When the mining distance exceeds 60 m, the maximum porosity does not increase, stabilizing around 3.3%. Thereafter, the range corresponding to the maximum value of the porosity gradually enlarges to form an expansion deformation zone. When the mining distance is 200 m, the length of the expansion deformation zone reaches 150 m. The increase in porosity manifests as an expansion deformation of the formation. It is widely accepted that a better expansion deformation will lead to better fracture development, thereby increasing coal seam permeability.58 This also explains the permeability increase in Figure 5B. Accordingly, the permeability enhancement zone gradually expands as the mining distance increases. After the J-14 coal seam is mined for 100 m, the permeability value of the permeability enhancement zone in the J-15 coal seam is generally maintained at about 2.5 times the original permeability, indicating that the J-15 coal seam receives enough pressure-relief and permeability increase effects. Hence, the presence of an HTSR has a limited effect on the expansion deformation of the underlying protected seam.

3.2.2 | Evolution of the plastic deformation

To further examine the influence of the HTSR on the evolution of the gas migration passages between different formations, the plastic deformation zone distributions at 6 mining distances from 20 m to 200 m were studied. Figure 6 illustrates that, as the mining distance increases, the range of plastic zones continues to expand both above and below the gob. This expansion effect causes plastic deformation of the formations beneath the J-14 coal seam, which initially occurs only in the floor rock formation. When mining extends to about 80 m, the protective seam begins to plastically deform because of the significant pressure relief, and then plastic damage develops rapidly. When mining extends to about 120 m, plastic zones of the floor rock formation and protected seams begin to connect, forming gas migration passages between coal seams. Therefore, for J-15 coal seam, gas desorbs in large quantities and flows into the J-14 coal seam along these flow passages, greatly reducing its outburst risk.

Figure 6 also shows the effects of the HTSR on the roof strata movement. The plastic zone in roof formations gradually extends upward as the mining distance increases from 0 to 160 m. When the working surface is advanced 160 m, the width of the plastic zone in the vertical direction reaches 19.62 m. From 160 m to 200 m, the plastic zone has only expanded by 1.33 m, and the speed has decreased significantly, which indicates that the plastic damage almost completely ceases to occur in the vertical direction. After mining for 160 m, the plastic zone expands mainly along the mining direction. The rock formations above the HTSR are still incapable of plastic deformation, even after mining for 200 m, suggesting that the conventional roof fractured zone cannot form normally. Hence, compared with conventional mining, the range of the roof pressure-relief area is greatly reduced. Bed-separated fractures cannot be generated, and the roof rock formation loses gas storage conditions. Moreover, the HTSR undergoes the plastic yield limit but is unable to collapse because of its great thickness, resulting in an overhanging roof above the gob, which is unfavorable for safe mining.

This suggests that the presence of the HTSR would not influence the pressure-relief effect of the protective seam mining on the underlying rock and coal formations but that it plays a negative role in the formation of the roof fractured zone by preventing the sinking of the upper rock strata. As a consequence, gob gas loses the ability to migrate upward, resulting in gas accumulation. Also, the apparent gas emission in the protective seam rises sharply, overloading

| Rock mass      | Elastic modulus (GPa) | Elastic modulus of grain (GPa) | Poisson’s ratio | Initial porosity | Initial permeability | Initial gas pressure in fractures |
|----------------|-----------------------|--------------------------------|-----------------|------------------|----------------------|----------------------------------|
| Coal           | 0.8                   | 4.5                            | 0.32            | 0.06 (matrix)    | 4 × 10⁻¹⁸            | 2 (matrix)                       |
|                |                       |                                 |                 | 0.025 (fracture) |                      | 2 (fracture)                     |
| Sandy mudstone | 15                    | 30                             | 0.3             | 0.001            | 3 × 10⁻²²            | 0.2                             |
| Sandstone      | 20                    | 50                             | 0.28            | 0.003            | 8 × 10⁻²¹            | 0.2                             |
the ventilation system and inducing gas overrun accidents. Hence, it is imperative to take measures to control the HTSR and promote the formation of the roof fracture zone while optimizing the ventilation system to increase gas discharge capacity. Only once these problems are solved can the protective seam be mined safely to allow the efficient exploitation of the entire coal seam group.

4 | GAS MIGRATION PASSAGE CONSTRUCTION DURING UPSM

4.1 | Roof fracture passage construction

Deep-hole blasting is widely used in roof treatment because of its simplicity and relatively low cost. The blasting position is usually set deep inside the roof with a sufficient safety distance from the coal seam to ensure safety. To eliminate the adverse effects of the HTSR on gas control, an RPS method based on deep-hole blasting was proposed after considering safety, cost, and fracturing effects. RPS is designed to promote the efficient and controlled breakage of HTSR and create fracture channels in roof rock formations. As shown in Figure 7A, several boreholes are constructed at intervals from the top of the upper and lower intake airways to the HTSR in front of the working face. Deep-hole blasting is then performed inside the borehole to weaken the roof and form the primary fracture network in advance. The blast-induced fractures continue to evolve under the mining stress, forming a weak zone inside the roof. After mining, the overhanging roof falls off in time under the action of the original rock stress and its own weight, and the fracture follows along the weak zone. At the same time, the plastic deformation of the upper roof formations is promoted, and the roof break finally induces a trapezoidal ring-shaped fracture zone above the gob, which provides superior passages and space for gob gas transfer.
The construction of RPS boreholes generally begins 30 m ahead of the working face. The spacing of borehole group S (Figure 7A) is generally less than the initial periodic breaking length, which was observed to be 58 m in the field test. Here, S was set at 35 m after considering the fracturing effect and cost. The layout of a set of RPS boreholes is shown in Figure 7B. Each group consists of six boreholes divided into three types: a, b, and c. The horizontal rotation angles were designed to be 75°–85° for the type a and b boreholes and 20°–30° for the type c boreholes. All the boreholes were inclined to the working face. Figure 7C is a cross-sectional view taken at C1 in Figure 7A. It can be seen that type a boreholes (a and a’) not only induce a fracture zone in the HTSR but may also create separation fractures between the HTSR and the upper rock formations, which play an important role in promoting the movement of the upper strata. Correspondingly, type b boreholes (b and b’) are primarily used to strengthen the formation of fracture zones and further weaken the HTSR. Meanwhile, type c boreholes (c and c’) serve to cutoff the connection between the HTSR of the mining area and the rest of the HTSR formation, thereby promoting the collapse of the overhanging roof above the gob. In addition, α, α’, β, β’, γ, and γ’ represent the elevation angles of the individual boreholes, with values of 17°, 29°, 10°, 15°, 35°, and 35°, respectively. Meanwhile, the depths of the boreholes a, a’, b, b’, c, and c’ are set to 59, 35, 81, 54, 16, and 16 m, respectively. Since the GSE is on the side of the lower intake airway (Figure 7A), the lengths of the type a and b boreholes in the lower intake airway were designed to be greater than those in the upper intake airway to enhance the pressure-relief effect on the roof of the GSE. Moreover, the charging length of each RPS borehole is 70% of its total length. As described in Figure 7D, explosive columns are sequentially fed into each borehole through a PVC pipe, and the two electrically responsive blasting caps are connected to the last column to ensure that the explosion waves propagate to the bottom of the borehole. The PVC pipe is then filled with stemming materials and finally sealed with a pneumatic hole packer. The distance between the starting charge position and the coal seam roof should be greater than 3 m. This allows a safety area to be formed above the coal seam to substantially reduce the negative effects of the blasting on the roof support of the roadway and the working face. Above the safety area is the blast zone, in which an artificial bursting rupture zone is developed to control HTSR breakage. Also, the bearing capacity of the HTSR is reduced. This accelerates the movement of the upper rock formation, forming an indirect blast-affected zone above the blast zone.

As the working face advances, fractures in the blasting area continue to evolve, gradually forming a fracture network, which greatly weakens the HTSR. At this time, the upper rock formations begin to bend and sink as well. When the roof begins to hang behind the working face, a large number

**FIGURE 7** Roof control and the construction of gas flow passages by presplitting blasting
of bed-separated fractures start to form, and roof strata will fail and break along the weak surface. Influenced by the overburden stress, the falling rocks in the middle of the gob and the rock formations above them are continuously compressed to form a recompaction zone. At the same time, the roof rock masses on both sides of the gob still maintain an unloading state, forming a caving zone and a fracture zone from bottom to top (Figure 7E). Figure 7E is a cross-sectional view of the C2 position in the gob in Figure 7A, where the A and B regions are fracture zones, which are O-shaped on the plane and trapezoidal ring-shaped in the space. Preferential migration passages are developed in the trapezoidal ring-shaped fracture zone so that the process of transporting the gob gas up to the roof formations becomes efficient and orderly. Consequently, the gas accumulation in the roof fractured zone not only eliminates the risk of gas overaccumulation in the protective seam but also facilitates further roof gas extraction.

4.2 | Gob-side entry construction

Although the gob gas can be transferred by the new roof fracture channels, the mining space (roadway and working face) still contains a mass of gas that needs to be drained through the ventilation system. Therefore, Y-type ventilation is still necessary. Constructing a gas drainage channel (ie, a GSE) on the side of the gob is the main method to achieve Y-type ventilation. However, intensive mining of deep coal mines requires GSEC to be efficient and fast, and the roadway must remain stable for a long time under high-stress conditions.
conditions. According to the preceding analysis, HTSR control can greatly reduce the stress concentration, and the RPS method creates favorable stress conditions for GSE construction and support. The following passage describes the mechanized rapid construction process of the GSE based on the RPS adopted in No. 12 Coal Mine. The combination of active pressure relief and active support is proposed to reduce the deformation of the surrounding rock and ensure GSE stability.

The roadside filling methods mainly include mechanical formwork, hanging bag filling, and frame formwork. Frame filling is widely used because of its simple structure, reusability, and high degree of mechanization. As shown in Figure 8A, the framed formwork filling equipment used in No. 12 Coal Mine mainly consists of two formwork supports and one filling mold box. Each time a staged filling is completed, and the formwork supports and mold box automatically advance with hydraulic supports of the working face. The whole wall-filling process can be divided into seven steps, from ground preparation to underground filling (Figure 8A). The first step is to produce dry-mixed filling materials through the special production line on the ground according to the design ratio, composed mainly of Grade III bulk fly ash of power plants, cement, and admixtures. When formed into a paste, the concrete filling material has a compressive strength of up to 30 MPa. The dry-mix materials were then shipped in bulk or in dedicated containers to the underground transportation system and then delivered on a conveyor belt to the hopper of a fill pump. The fourth step is to add water to the filling pump and stir evenly to form a paste. The mixture is then pumped through the filling line into the mold. After that, the paste mixture is self-leveled and compacted in the filling mold box, allowing it to cure naturally. Finally, the filling wall demolds when it has a certain strength, which takes only 6 hours from molding. The high mechanization and automation process allows the filling wall to be built quickly following the working face, which guarantees that the mining speed is kept at pace and ensures efficient production.

To optimize the stress load on the filling wall, the lateral cantilever must be effectively controlled to prevent stress concentrations caused by excessive cantilever length. However, it is difficult to achieve efficient removal of the lateral roof by RPS because of the large thickness and high strength of the HTSR. Therefore, it was proposed to construct roof cutting boreholes in parallel with the RSP boreholes to enhance the weakening effect on the lateral roof. There are six roof cutting boreholes in each group, which are roughly in the strike direction, and their horizontal angles are all smaller than those of the type c RPS boreholes (Figure 8B). Deep-hole blasting can also be done to cutoff the connection between the lateral roof and the whole HTSR formation and to promote the timely breakage of the cantilever. The active pressure-relief method, combined with roof cutting and RPS, eliminates the influence of long-span cantilever and its rotary subsidence on the filling wall and surrounding rock and is highly favorable to GSE construction and support.

Based on collaborative pressure relief by RPS and roof cutting, the active support method for GSE was further proposed. Intensive support was implemented in sections 200 m in front of and behind the working face, according to the distribution law of the mining stress. The specific steps are shown in Figure 8B. First, deep anchor reinforcement support was applied to the lower intake airway at the P1 position, 200 m ahead of the working face. The loose surrounding rock and the deep rock mass were consolidated together before the mining stress changes, which enhanced the resistance and reliability of the roadway in advance. Second, at the P2 position, 150 m ahead of the working face, RPS and roof cutting were carried out to create good stress conditions for the support. Then, the roof of the filling area at the P3 position was prereinforced with an anchor cable to improve its bearing capacity and keep the roof intact. The reinforced roof not only ensured the safety of the filling work but also relieved the stress on the overburden, especially in the initial filling stage, which helped with the rapid formation of the filling wall. Finally, from 30 m ahead of the working face to 200 m in the rear (P3 to P5), the arrangement density of the single hydraulic props was increased to strengthen the roadway support to adapt to the drastic changes in the mining stress in this section. The application of active pressure relief and active support technology makes it possible to retain a long-distance GSE under high-stress and HTSR conditions, which is important for gas control in the protective seam.

5 | DISCUSSION

5.1 | Stepwise gas diversion

To safely explore the deep outburst coal seam group directly covered with an HTSR, RPS and GSEC were used to construct gas migration passages in roof rock formations and mining spaces based on UPSM. This allows coalbed gas to be efficiently transferred from high-risk areas to low-risk areas step-by-step. As shown in Figure 9, the gas diversion process can be divided into three steps: UPSM, roof presplitting, and GSE construction.

As mentioned in Section 3.2, the mining of the upper protective seam not only released the pressure on the protected seam but also created the vertical fractures in the floor rock formations. These connected fractures provided an upward flow passage for the pressure-relief gas. Generally, the protected seam can be seen as the main gas source with a high-outburst risk (Figure 9A). Therefore, after UPSM, a large amount of gas in the high-risk layer was transported to a relatively safe layer. Through the first step of gas diversion,
the gas content of the main gassy seam was significantly reduced, and the outburst danger was mostly eliminated.

Since the HTSR immediately covered the protective seam, a few connected fractures can be induced in the roof strata by the mining effect alone, and the roof fracture zone could not be formed in time. This caused gas to accumulate, making the gob a new source of danger. Therefore, the RPS method was proposed to create blasting fractures in advance and form the roof flow channels for the gob gas (Figure 9B). The blasting fractures further induced new fractures and gradually formed a fracture network under the action of the mining stress. The HTSR was greatly weakened, and the movement of the entire roof strata was accelerated, which induced the bed-separated fractures between the layers. As a result, the failure and caving of roof rock strata were greatly facilitated, and the formed fracture zone created an excellent channel for the gob gas transfer. Through RPS, large quantities of gob gas flowed into the roof strata, which meant that RPS promoted the diversion of gob gas from a high-outburst layer to the upper rock safety layers. Hence, the gob gas concentrations of the protective seam decreased significantly, and gas overlimit in gob was prevented. This also eliminated abnormal emission accidents caused by the gob gas flowing into the working face because of the sudden caving of the large-area overhanging roof.

However, gas control in the mining room of the protective seam was also facing challenges because of the influx of gas from the underlying seams. To ensure the safe mining of the protective seam, GSEC technology was used to optimize the ventilation system and enhance the discharge of the local mining gas. As depicted in Figure 9C, the ventilation style became Y-shaped, which consisted of two intake airways and one return airway, after GSEC. Gas in the working face and roadways was transferred to the rear, away from the working face, greatly reducing the risk of gas accidents in the working room. At the same time, the gas flow field in the gob was greatly changed. The low-pressure point of the whole flow field was located deep in the gob, and gas would continuously move to its rear. In addition, gas negative pressure extraction in the gob through buried tubes also adjusted the flow field distribution and gas flow direction to some extent. Therefore, the overall gas flow was from the working area to the gob, which effectively transferred and controlled the gas hazards.

Through the UPSM, RPS, and GSEC, the vertical fracture passages, roof fracture passages, and favorable airflow passages were actively constructed in steps, and the stepwise gas diversion was effectively achieved (Figure 9D). A mass of coalbed gas in the dangerous protected seam was transported step-by-step to the safer roof layers above the protective seam, and plenty of the local mining gas was diverted from the mining area to the postmining area. The overall gas flow of the stope was directed to the roof rock formation and the rear of the gob. Each step promoted the migration of coalbed gas to a safer area, away from the coal seams and working areas. In fact, the proposed stepwise gas diversion method is intended to be used to transfer and remove the gas hazard...
sources, thereby ensuring the safety of the mining of the coal seam group.

### 5.2 Gas distribution and extraction

Figure 10A is a cross-sectional view of the stope around the gob at a distance behind the working face. As the J-14 coal seam was mined, the stress distribution of the stope tended to be stable. The range of the roof pressure-relief area was expanded through RPS, forming a hemispherical fracture zone above the gob in space. As shown in Figure 10A, the upper boundary of the roof fractured zone moved up to a considerable height compared with the original. The stope space can be divided into four spheres: the compaction zone, the fractured zone, the stress increasing zone, and the original stress zone. The fractured zone is spherical and surrounded by a stress increasing zone. The high-stress concentration makes gas flow in the stress increasing zone extremely difficult.

Consequently, coalbed gas is trapped in the fractured zone and cannot flow further. As explained in section 3.2, the protected seam (J-15) coal seam achieved sufficient pressure relief and the whole seam is in the floor fractured zone, where the stress decreases in all three directions and fractures are extremely well developed. Pressure-relief gas can flow freely and penetrate the floor rock formation into the gob of the protective seam (J-14). Despite this, a large amount of gas remains in the protected seam (J-15), which needs to be extracted in time, and the protected seam (J-15) is still a major gas source. At the same time, the gob of the protective seam (J-14) becomes the second gas source due to the influx of returning gas and protected seam gas. Moreover, the range of the roof pressure-relief area was expanded through RPS, forming a hemispherical fractured zone in the space above the gob. The permeability of the roof fractured zone is significantly increased, forming free-flow channels and spaces. The gas can migrate upward under its own pressure gradient, contributing to a roof gas enrichment zone above the gob.
Three gas sources were formed in the stope space through the stepwise gas diversion after the flow passages were created. Comprehensive gas control technology has to be further applied to extract the high-concentrate gas and eliminate the sources of danger based on gas migration and accumulation behaviors (Figure 10A). Since gas sources are mainly distributed around the gob of the J-14 coal seam, GSEC not only promoted the flow of the local mining gas into the gob but also provided an excellent venue for implementing gas control measures. For the main gas source in the protected seam, bedding boreholes were constructed to extract the remaining gas and thus further reduce outburst risks. Meanwhile, the construction of low-level boreholes in the GSE enhances gas drainage of the J-15 coal seam, which also prevents excessive gas from flowing into the protective seam. For the second and third gas sources in the gob and roof strata, middle-level and high-level boreholes, respectively, were applied to eliminate high-risk gas sources and ensure the mining safety of the J-14 coal seam. This source-by-source extraction model markedly improved the gas extraction concentration and control effect.

Figure 10B,C shows the spatial arrangement and time sequence of gas extraction boreholes, respectively. Bedding boreholes (step a) should be first drilled in the J-15 coal seam to pre-extract the original coalbed gas, and they continue to work until the working face of the J-15 coal seam has been advanced. Then, the exploitation of the J-14 coal seam began (step b), and the GSE was constructed with the advance of the working face. The gas in the working area was discharged to the gob through the formed Y-type ventilation system (step c). Meanwhile, as a middle-level extraction measures, tubes were extended to the gob through the filling wall for gob gas control (step d). After mining a distance, the trapezoidal ring-shaped roof fractured zone gradually formed, and a large amount of gas is enriched in it. The high-level borehole extraction was thus applied, and an attempt was made to keep the bottom of the borehole in the O-shaped area during construction (step e). Simultaneously, the low-level borehole was drilled down through the J-15 coal seam to intercept the gas from the pressure-relief zone (f). As the working face advanced, the pressure-relief zone in the J-15 coal seam moved forward accordingly. The extraction concentration of the bedding boreholes in the pressure-relief zone increased dramatically and gas control of the J-15 coal seam was further strengthened (step g). Then, a regional outburst elimination index test was conducted to evaluate the mining risk (step h). Finally, the J-15 coal seam can be safely mined after the verification test (step i).

The beding borehole was designed with a diameter of 89 mm. In the intake airway, the spacing and length were set to 3 m and 150 m, while they were 1.5 m and 80 m in the return airway, respectively. The bedding boreholes of the two airways were staggered to enhance gas extraction. The borehole diameter of the high-level and low-level boreholes was 85-120 mm, the spacing was 5-20 m, and the length was 30-40 m. At the same time, the buried tube (middle-level borehole) was designed with a diameter of 150-300 mm, a spacing of 6-15 mm, and a length of 3-5 m.

5.3 Stepwise risk control combined with gas diversion and extraction

The gas risk control process in the deep high-outburst coal seam group with an HTSR can be divided into three stages (Figure 11). The construction of gas migration passages in the stope is the first stage. The protective seam mining, RPS, and GSEC create preferential pathways for coalbed gas to move from high-risk areas to low-risk areas. The second stage is gas diversion and enrichment. After stage I, the stope gas was diverted from the high-outburst protected seam to the gob of the protective seam and then flowed into the relatively

![Image](image.png)

**FIGURE 11** The stepwise gas control process in the deep outburst coal seam group with an HTSR
safer roof fractured area, which greatly reduced the outburst risk of the protected seam and promoted the partition enrichment of gas. Three gas sources were accordingly formed in the upper, middle, and lower parts of the stope. In the third stage, source-by-source extraction measures were conducted based on gas distribution characteristics. The purpose of the Y-type ventilation system and the middle-level borehole was to control the gas source of the working face and the gob while the high-level borehole was to control the gas source of roof strata. The gas source of the protected seam was drained using the bedding borehole combined with the low-level borehole. Through the combination of gas diversion and extraction, severe dynamic behavior, gas overrun, and outburst accidents were eliminated, and the concentration and effect of gas extraction were significantly improved. Coalbed gas in the whole coal seam group was effectively controlled, and deep gassy coal seams with high-outburst risk could be safely and efficiently mined in sequence.

RESULTS OF THE FIELD APPLICATION

6.1 Gas migration passage construction

After the J-14 coal seam was mined for a distance, the sharp increase in gas emission from the J-14 coal seam, which was from 2.33 m³/min to 8.78 m³/min, indicated a remarkable pressure-relief effect on the J-15 coal seam. The flow passages must have been formed in the floor rock formations, leading to the large gas transfer between the two coal seams. At the same time, the HTSR had been successfully controlled by RPS, and the cyclic roof caving span was reduced by 54% from the initial 58 m to an average of 26.5 m (Figure 12A). The timely break and subsidence of the roof strata promoted the generation and development of the vertical and bed-separated fractures, which formed upward gas flow channels and spaces. Also, the large-area overhanging roof disappeared, and the abnormal emission of gas was prevented. RPS combined with the roof cutting technology also greatly reduced the stress concentration around the working face and GSE. This ensured the successful application of the GSEC technology so that the filling wall was constructed efficiently with mining, and the Y-type ventilation system was formed smoothly. Moreover, the displacements of the roof, floor, and coal rib were kept in a normal range by applying the active support method. According to the observation results, the deformation of the surrounding rocks decreased in the following sequence: floor > coal rib > filling wall > roof (Figure 12B). The cumulative deformation of the filling wall was 143 mm, which only accounted for 6.9% and 26.8% of the total deformation of the two ribs and the GSE, respectively. The deformation of the GSE retaining was mainly manifested as the floor heave, which further illustrated the excellent pressure-relief effect of the protective seam mining on the underlying strata. As depicted in Figure 12C, the GSE was kept intact with a small shrinkage rate of 14.5% except for the floor heave. This guaranteed the stable discharge of gas in the working room. Furthermore, after the completion of the gas migration passage, the coalbed gas movement in the stope became efficient and orderly. The corresponding gas diversion effects can be explained by the improvement of the gas drainage rate, which is discussed in Section 6.2.

6.2 Gas extraction

6.2.1 Gas extraction in the J-14 coal seam

Figure 13 depicts the air flow and gas discharge by Y-type ventilation over 141 days. It can be seen that the air flow in the working face and the GSE is relatively stable, with a range of 1016-1819 m³/min and 1436-2099 m³/min and an average of 1501 m³/min and 1856 m³/min, respectively. The gas concentration and pure gas flow in the GSE were maintained at 0.11%-0.32% and 1.58-6.72 m³/min, and the average values were 0.18% and 3.30 m³/min. The gas concentration of the working face varied between 0.07% and 0.39% during
the whole mining period, and no gas overlimit accident occurred in the upper corner. It can be concluded that the gas in the working area of the protective seam has been effectively controlled by constructing a Y-type ventilation system.

The middle-level and high-level boreholes were constructed to drain the gas in the gob and the roof fractured zone, respectively, while the low-level borehole extraction was not successfully applied because of difficulties with drilling and sealing. In the field application, the number of high-level boreholes was increased to enhance gas extraction. As displayed in Figure 14A, the average gas concentration and pure volume flow of the middle-level boreholes in 200 days were 0.91% and 1.00 m³/min. The pure volume flow was 0.27 m³/min-2.00 m³/min, and the total extraction volume reached 2.01 × 10⁵ m³. The middle-level borehole played an important role in reducing gas concentration and preventing gob gas accumulation. Figure 14B indicates that the gas concentration of high-level boreholes over 200 days was basically above 12%, with an average of 19.07%, which was a significant increase compared with the initial average value of 2.54% before RPS. The pure gas flow was almost always above 3%, and the total volume of gas extraction amounted to 1.64 × 10⁶ m³. Hence, a large amount of gas was extracted by the high-level borehole, which proves that the construction of the gas migration passage was successful and that the source-by-source gas control method significantly improved gas extraction efficiency and also explained the effectiveness and correctness of the stepwise gas diversion.

Variations in gas discharge and extraction during the 315-day mining process were shown in Figure 15. The pure volume flow of the air discharged gas was 0.78-7.09 m³/min, while the extracted gas varied between 0 and 10.9 m³/min. The average volume flow values were 2.18 m³/min and 4.01 m³/min, respectively. The gas extraction rate varied between 6.80% and 85.27%, with an average of 65.72%, indicating that gas was effectively controlled in the J-14 coal seam. Accidents related to gas and roof were prevented, including gas overlimit outbursts and severe dynamic behaviors. On the other hand, during the field application, a mass of gas from the J-15 coal seam was transferred up to the roof strata and was ultimately extracted efficiently, which demonstrated that gas in the high-risk source had been effectively diverted and controlled. The mining risk of the J-15 coal seam was significantly reduced.

6.2.2 | Gas extraction in the J-15 coal seam

Even though plenty of the pressure-relief gas was transported to the overlying strata, a considerable amount of gas remained in the seam. Bedding boreholes were constructed to enhance gas control. Take the intake airway as an example, the average gas concentration of the bedding boreholes was 4.3% at the initial stage, which is relatively low (Figure 16A). The J-15 coal seam underwent pressure-relief expansion under the mining action of the J-14 coal seam, leading to a concentration variation in the bedding borehole. Taking a single borehole as an example, Figure 16B also
shows a three-stage concentration change in the test borehole from 35 m in front of the working face to 118 m behind the working face. In the growth stage (−35 m to −3 m), as the working face advanced, the rock mass around the borehole began to expand under the mining action, and the massive desorption of gas promoted the increase in borehole concentration. Especially after 24 m, the gas concentration increased rapidly from 4.8% to 70%. In the active stage of −3 m to 58 m, the coal body was in a fully expanded state, and consequently, the concentration was maintained above 60%. In the attenuation stage (58 m to 120 m), the fractures of the J-15 coal seam gradually closed under the compaction of overburden stress, resulting in a decrease in extraction concentration. Despite this, the concentration at 120 m behind the working face was still more than 20%. The concentration changes in the single hole illustrate an extremely strong pressure-relief effect from mining the J-15 coal seam. All the bedding boreholes experienced similar increases in concentration, contributing to the overall increase in gas extraction concentration. For the intake airway, the average concentration was 10.87% over the 300 days, increasing to 2.5 times the initial concentration (Figure 16A). Correspondingly, the average concentration in the return airway was 12.89%, increasing to 3.7 times.

Accordingly, the gas flow rate of the bedding boreholes also began to increase. As displayed in Figure 17, after 100 days of continuous growth, the total daily flow rate of the intake and return airways increased from 456 and 981 m$^3$/d to 11 676 and 13 146 m$^3$/d, with an increase of 25.6 times and 13.4 times, respectively. Moreover, the gas extraction rate of the J-15 coal seam was raised to 4.3 times, from 15.5% to 66.9%, and remained at more than 65% for 7 months. Therefore, bedding borehole extraction played an important role in gas source control. According to the indicator verification of the J-15 coal seam, the residual gas content and pressure were reduced to 1.91-4.46 m$^3$/t and 0.11-0.37 MPa, well below the critical index of 8 m$^3$/t and 0.74 MPa. The result indicated the elimination of outburst risk, and finally, the J-15 coal seam has been safely mined.

### 6.3 Benefit analysis

The application of the RPS and GSEC technology ensured the successful application of UPSM technology in the exploration process of the deep coal seam group covered with an HTSR. The stepwise gas diversion and extraction method allowed several high-risk coal seams to be safely mined in sequence, and the overall benefit was remarkable. Compared with situations where the working face was treated with low-level rock roadways under the same conditions, the required air volume of the protected working face was significantly...
reduced from 2400 m³/min to 1903 m³/min, and the gas concentration of the return airway was reduced from 0.55% to 0.30%, indicating that the proposed method significantly reduced gas emissions and ventilation costs. Meanwhile, the cost of coal production was reduced by 26.92 RMB per ton, and the duration of the gas control was shortened by 4.3 months. The average monthly output increased by more than 60,000 tons. Furthermore, because of the concentration and volume increase in the gas extraction, the amount of coalbed gas used has greatly increased through gas power generation. The daily generating capacity of the No. 12 coal mine was about 58 thousand kWh, and the annual power generation reached a total of more than 18 million kWh. It can be concluded that the sequential gas control and mining method not only reduces safety costs and improves production efficiency but also enhances coalbed gas recovery, providing abundant gas resources for further use.

7 | CONCLUSIONS

To solve gas control problems during the mining of the deep high-outburst coal seam group with an HTSR in No. 12 Coal Mine, China, a numerical simulation study was first conducted to demonstrate the influence of the HTSR on porosity, permeability, and plastic deformation evolution in overlying and underlying strata after the UPSM. The results show that the presence of the HTSR would not affect the pressure-relief effect of UPSM on the underlying rock and coal formations but that it does slow down the plastic deformation of overlying formations. The gas-rich fracture zone cannot be formed in the roof strata. Consequently, plenty of pressure-relief gas in the protected seam migrates upward and accumulates in the protective seam without being transported to the roof formations, causing a new safety hazard.

RPS technology allows the HTSR to be weakened and an effective gas migration passage to be created. Deep-hole blasting is applied to three kinds of boreholes, which are drilled ahead of the working face, to cut the interlayer connection, induce roof fractures, and accelerate roof caving. The preformed fractures continue to develop under the action of mining stress to form a weak zone, prompting the HTSR and the upper roof strata to break in a timely and controlled manner. A trapezoidal ring-shaped fracture zone is accordingly formed, which provides flow channels and corresponding storage space for gob gas. Protective seam gas is thus diverted, which greatly reduces safety risk.

The GSEC method is further proposed to build a Y-type ventilation system in the 1000-m-deep protective seam. A mechanized rapid roadside filling system is established underground. Through the proposed filling construction process, the GSE is built efficiently with the framed formwork filling equipment, following the advance of the working face. Meanwhile, roof cutting technology is further applied to eliminate the detrimental influence of the lateral cantilever on GSE support. Based on the active pressure relief through the roof treatment, the active support method is further proposed to maintain low shrinkage of the long-distance GSE in the long term under high stress and the presence of the HTSR, ensuring the stable drainage of gas by Y-type ventilation.

Through the UPSM, GSEC, and RPS, coalbed gas can be transferred step-by-step from the high-outburst protected seam to the low-outburst protective seam, from the working room to the gob, and from the gob to the roof formations. The high-risk gas source is diverted away from the coal seam and the working face through the preferential migration passage construction, significantly reducing the gas hazard. Therefore, gas diversion based on migration passage construction is important for gas control in the deep coal seam group. After efficient gas diversion, the mining risk of the high-outburst protected seam is dramatically reduced, and three gas sources are gradually formed around the gob because of the expansion of the roof fractured zone. The source-by-source extraction method is applied to closely control the high-concentration gas at each source, which greatly improves the efficiency of the gas treatment. The spatial-temporal sequence of borehole extraction and coal seam mining should also be considered for more effective gas control.

Combined with gas diversion and extraction, this study finally proposes a stepwise risk control method for the deep outburst coal seam group mining. A gas migration passage is created or constructed first to divert the gas from high-risk coal seams and promote orderly gas gathering. Centralized gas extraction is then performed for each gas source. This method not only significantly reduces the mining risk of
the outburst coal seam but also greatly improves the gas extraction rate.

The corresponding field application was successfully carried out in No. 12 Coal Mine at Pingdingshan, China. The HTSR was effectively controlled, with a 54% reduction in cyclic roof caving span, and the long gob-side entry was achieved with a low shrinkage rate. The J-14 coal seam was mined safely without gas accidents, and the average gas extraction rate reached 65.72%. Meanwhile, the gas extraction rate of the J-15 coal seam was improved from 15.5% to 66.9%, and gas content and pressure were reduced to 1.91-4.46 m³/t and 0.11-0.37 MPa, respectively. The ideas and methods proposed in this paper provide practical solutions for the safe mining of the deep high-outburst coal seam group with an HTSR.

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CONFLICT OF INTEREST
The authors declare no competing financial interest.

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