Research Article

The Relationship between Mining-Induced Stress and Coal Gas under an Optimized Support Scheme: A Case Study in the Guanyinshan Coal Mine, China

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Stress is one of the main factors influencing coal and gas outbursts. The apparent effects of the crustal stress, the structural stress, and the mining-induced stress increase as the depth of mining increases. At present, there have been few studies of the relationship between the comprehensive analyses of the crustal stress, mining-induced stress, and coal gas. The in situ measurement of the relationship between stress-related behaviors and coal gas under the influence of mining was conducted through experimental analysis of surrounding rock support and coal and gas outburst control and optimization of surrounding rock support materials and system construction. The results showed that the mining-induced stress first increased to a peak value, then gradually decreased, and tended to stabilize as the footage progresses. Stress appears at 96 m ahead due to mining; after 57 m of advancing, there is a large increase until it passes through this area. The stress in front of the working face increases linearly, and the increase range is obviously larger than that of the coal body in a certain range on both sides. The support anchoring force gradually decreased and tended to be stable after rapidly increasing to a maximum value. The deep displacement of the roof increased linearly and tended to be stable after reaching an accumulated displacement which can reach 16-28 mm. The residual gas pressure in front of mining operations decreased rapidly, and beyond 15 m on each side of the roadway, it decreased significantly. The residual gas pressure and gas content were consistent with the gas desorption index of drill cuttings due to the influences of gas predrainage and mining. The stress along the direction of the roadway and the residual gas content, the residual gas pressure, and the gas desorption index of drill cuttings conform to the logarithmic functional relationship. The research results provide a basis for the comprehensive prevention and control of coal and gas outbursts from multiple angles considering stress, coal, and gas.

1. Introduction

With the development and exploitation of mines to greater depths, it is difficult to predict and control deep coal mining disasters because of the high crustal stress, high permeability, and high gas pressure and gas content, as well as the additional attributes of strong disturbance and timeliness in deep mining. Relevant research results generally show that the “three high” environment will lead to changes in the rock structure, basic mechanical characteristics, and engineering response compared with shallow rock: this is one of the main reasons why deep mining engineering disasters occur frequently and differ from shallow-rooted disasters [1]. The complex mining environment around deep coal and rock masses leads to redistribution of stress after roadway excavation, expansive deformation, and movement toward the free surface; this is followed by the emergence of nonlinear large-deformation dynamic phenomena which make the
deformation of roadway surrounding rock present the characteristics of soft rock. With the change of coal seam attributes, the development of geological structures and abnormal gas parameters of a coal body leads to coal and gas outbursts, rock bursts, and other major disasters.

Coal and gas outbursts are a sudden dynamic disaster generated by mining disturbance of coal and rock. Based on the long-term theoretical research and field practice, the hypothesis underpinning coal and gas outburst prevention considers the two main factors of outburst force and medium. It is generally believed that coal and gas outbursts are the result of comprehensive action of the crustal stress, gas, and physicomechanical properties of coal, in which the crustal stress is a stimulating factor although high-pressure gas plays a decisive role in the development of outbursts and the physicomechanical properties of coal form obstacles to outbursting [2]. When mining disturbance weakened the mechanical properties of coal, it can cause stress redistribution and affect the fracture distribution and permeability thereof. The derivative effect under the influence of mining can lead to rock bursting and coal and gas outbursts, so the stress evolution characteristics induced by mining have been widely studied. The existing research showed that each component of the 3D crustal stress tensor is always different in different directions, thus playing a major role in influencing the tightness and stability of a mining roadway [3–5]. High crustal stress increased the risk of coal and gas outbursts: in the horizontal direction, this and the principal stress intensity are the key factors affecting the risk of an outburst [6–8]. Through the comprehensive study of strata movement, changes in stress, fracture, and gas flow dynamics caused by deep longwall mining, new views have been formed on the changes in strata stress caused by mining and the complex dynamic interaction between fractures and gas flow patterns [9]. Liu et al. established a coupled model through COMSOL numerical software and found that the elastic potential energy of coal continuously accumulated and the effects of stress gradually increased; when the energy exceeded a certain level in terms of the surface energy of the coal, then an outburst may occur [10]. Xue et al. studied the deformation and permeability and acoustic emission characteristics of coal under various mining-related stress paths: the damage characteristics of coal and rock masses after underground excavation and the influences of excavation damage on seepage characteristics were investigated [11, 12]. Yang et al. studied the dynamic change in the permeability of the rock surrounding a roadway based on a gas-solid coupling model through the method of numerical simulation [13]. Liu et al. assessed the evolution of the permeability of coal and non-Darcian effects in seepage under variable confining pressure [14].

Mine development and mining are dynamic and continuous complex processes. Stope pressure change caused by mining is a hidden factor in coal and gas outbursts. It is necessary to study the control mechanism of coal and gas outbursts under the coupling effect of coal, gas, and stress which is the precondition for preventing and controlling coal and gas outburst disasters. Reasonable roadway excavation and support measures are an effective means of stress redistribution and provision of stability. It is necessary to prevent and control coal and gas outbursts by optimizing the supports used along a roadway which can provide a stress field beneficial to the stability of the surrounding rock and increase the bearing capacity and stability thereof. At present, there have been few studies of the change in stress and its influence on gas flow during mining [15]. The relevant laws and regulations have given quantitative provisions as to the basic parameters of coal and gas outbursts and the characteristics of a coal body, but the influence of stress-related factors has not been quantified. Under this background, it is essential to measure and analyze the mining stress, support stress, surrounding rock deformation, and coal gas parameters based on the optimization of stope support and study the relationship between coal stress and coal gas under the influence of mining.

2. Theory and Methods

2.1. Theoretical Analysis of Surrounding Rock Support and Coal and Gas Outburst Control. During excavation, the surrounding rock of the roadway deforms and breaks over a certain range, forming a fracture zone, a plastic softening zone, a plastic hardening zone, and an elastic zone. These grow until no deformation occurs in the elastic zone. When the deformation exceeds that tolerable in the surrounding rock, instability failure occurs. Although the coal and rock masses deformed and fractured after excavation, the surrounding rock remained stable as a whole and retains a certain bearing capacity. A certain amount of metamorphic fracture can be allowed in the early stages of mining, but it must be controlled before the mutation of the metamorphic fracture rate. Before that, the roof and both sides of the roadway should be reinforced to control the minimum deformation during fracturing to form a new stable stress distribution energy field. Stress and gas pressure increase linearly with depth (notwithstanding arching effects), and the complexity of geological structures and physicomechanical properties of a coal body can thus change. Coal gas, controlled by stress, exhibits an irregular pressure gradient and is subject to localized abnormal increases under structural stress. Stress redistribution under the influence of mining induces the bending and subsidence of the roof in the plastic zone and stress concentration in front of the working face, the compression and closure of coal pores and cracks, the accumulation of elastic deformation energy, and an increased pressure gradient in the gas which drives coal and gas outbursts [16, 17]. On the other hand, many coal and gas outbursts have shown that outburst coal seams generally have particular structural characteristics and the pore structure suffers from structural damage and deformation to a great extent compared with the original coal. As the anisotropy of the pore structure increased, the specific surface area increased to an abnormal extent, as did its gas adsorption capacity. The permeability characteristics of tectonic coal are mainly affected by structural deformation, effective pressure, adsorption expansion, and sliding effects [18, 19]. The complex gravitational (self-weight) stress, structural stress, and mining-imposed additional stress field promote changes in the physicochemical structure of the coal. Especially under the action of structural
stress, the primary structure of a coal body becomes severely damaged, the soft layer is squeezed and stretched, and the activity of organic components in the coal increased: these cause ductile deformation and structural coali-
fication. Due to the fracture or strong ductile plastic deformation and rheological migration of coal under tectonic stress, tectonic coal superimposed with dynamic metamorphism based on regional metamorphism constitutes the failure condition of a coal and gas outburst [20].

2.2. Optimization of Surrounding Rock Support Materials and System Construction

2.2.1. Bolts. The bolt can be divided into equal-strength left-handed ribbed steel bolts without longitudinal reinforcement and high-strength left-handed ribbed steel bolts without longitudinal reinforcement. The bolt support materials are shown in Figure 1. The front made of left-handed rolled rebar with a continuous thread can be locked using a nut and shares the characteristics of BHRB335 (20MnSi) hot-rolled rebar at a yield strength of 335 MPa, tensile strength of 490 MPa, and elongation of 22%. The latter, made of left-handed thread steel without longitudinal ribs with the tail part threaded to accept a nut, shares the characteristics of BHRB400 hot-rolled rebar at a yield strength of 400 MPa, tensile strength of 570 MPa, and elongation of 22%. For the same bolt material, the larger the diameter, the higher the yield load and peak load at a given extension. For bolts with the same diameter, the yield load and peak load increased slightly, and the elongation increased greatly when the mate-
rial used to form the bolt was changed from BHRB335 to BHRB400 [21]. Table 1 shows the mechanical properties of bolts.

2.2.2. Anchor Cables. According to the structure type and strand diameter, these can be divided into ordinary anchor cables and high-strength prestressed anchor cables. Ordinary anchor cables mainly adopt $1 \times 7$ structural steel strands with diameters of 15.20 mm, 17.80 mm, 18.90 mm, or 21.60 mm. High-strength prestressed anchor cables mainly adopt $1 \times 19$ structural steel strands with diameters of 18.00 mm, 20.30 mm, 21.80 mm, or 28.60 mm. For the same steel strand structure, the larger the nominal diameter, the longer the elongation and the larger the breaking load. Figure 2 shows the schematic diagram of the anchor cable structure. For bolts with the same nominal diameter, the breaking load and percentage elongation increased greatly after the steel strand structure changed from $1 \times 7$ to $1 \times 19$; for example, the percentage elongation of high-strength prestressed anchor cables with a diameter of 21.80 mm and a $1 \times 19$ steel strand structure is twice that of an ordinary anchor cable with a diameter of 21.60 mm and a $1 \times 7$ steel strand structure. Table 2 shows the mechanical properties of anchor cable materials.

2.2.3. Comparison of Support Schemes. Combined with the theoretical research and practical results of roadway surrounding rock support and based on previous studies, three main support schemes were compared to determine the optimal scheme. Figure 3 shows the schematic diagram of

| Bolt type | Material of the rod body | Diameter (mm) | Yield load (kN) | Peak load (kN) | Extension (mm) |
|-----------|--------------------------|---------------|----------------|----------------|----------------|
| Equal-strength left-handed ribbed steel bolt without longitudinal reinforcement | BHRB335 (20MnSi) | 20 | 110-115 | 140-150 | 310-330 |
| | | 22 | 130-140 | 190-200 | |
| High-strength left-handed ribbed steel bolt without longitudinal reinforcement | BHRB400 | 20 | 110-118 | 150-160 | |
| | | 22 | 130-145 | 200-210 | 360-370 |

Table 1: Mechanical properties of bolts [22, 23].

![Equal-strength left-handed ribbed steel bolt without longitudinal reinforcement](image1)

![High-strength left-handed ribbed steel bolt without longitudinal reinforcement](image2)

Figure 1: Bolt support materials.
different support methods: Option 1: two equal-strength left-handed ribbed steel bolts without longitudinal reinforcement were arranged on both sides of the roadway ($\Phi$ 20 mm, length 2.40 m, and spacing and row spacing 0.80 m x 0.80 m), and one common anchor cable ($\Phi$ 21.60 mm, length 6 m, and row spacing 1.60 m) and four high-strength left-handed ribbed steel bolts without longitudinal reinforcement ($\Phi$ 20 mm, length 2.40 m, and spacing and row spacing 0.80 m x 0.80 m) were arranged on the roof area; Option 2: three equal-strength left-handed ribbed steel bolts without longitudinal reinforcement were arranged on both sides of the roadway ($\Phi$ 22 mm, length 2.40 m, and spacing and row spacing 5.20 m x 5.20 m) and four high-strength pre-stressed anchor cables ($\Phi$ 21.80 mm, length 6 m, and row spacing 1.60 m).
0.80 m × 0.80 m), and three high-strength prestressed anchor cables (Φ 21.80 mm, length 6 m, and spacing and row spacing 1.20 m × 1.60 m) and four high-strength left-handed ribbed steel bolts without longitudinal reinforcement (Φ 22 mm, length 2.40 m, and spacing and row spacing 1.20 m × 0.80 m) were arranged on the roof area; and Option 3: on the basis of Option 2, the middle bolts on both sides of the roadway were replaced with high-strength prestressed anchor cables (Φ 21.80 mm, length 6 m). Relevant scholars used the numerical simulation method to analyze the stress field and displacement field of roadway surrounding rock under different support schemes. Figure 4 shows the stress variation and vertical displacement of roadway surrounding rock under different support schemes.

In Option 1, the stress concentration in the surrounding rock was low and the peak stress was small and far from the free surface. Only at the two bottom angles of the roadway, an appropriate amount of stress concentration is generated, and the maximum stress peak reaches 26.30 MPa. However, the deformation of the roof and floor developed rapidly and the relative approach was large, so it is difficult to control in a deep roadway. In Option 2, the stress concentration of surrounding rock increased and the peak stress moves to the free surface. Local stress concentration was formed in the bottom corner and floor of the surrounding rock to a certain extent, and the zone of maximum principal stress bore the most load, and the range of the stress field gradually expanded after the two sides of the roadway were strengthened with three high-strength bolts. The stress environment in the surrounding rock of the roof and floor changed, so the strength of the surrounding rock increased correspondingly, and the deformation of the roof and floor was controlled. The increase of high-strength prestressed anchor cables of the roof can control the large-scale collapse and subsidence of the roof. In Option 3, the stress concentration in the surrounding rock increased and a local stress concentration formed at the two shoulder angles of the roof when a high-strength prestressed anchor cable was added to the roadway on both sides in which the peak stress reaches 29.80 MPa. At the same time, the stress was transferred to the roadway which can better maintain its stability after excavation. The increased number of anchor cables can improve the overall strength and stiffness of the surrounding so that rock roof subsidence of the roadway decreases by 34.80% and 16.40%, and floor heave decreases by 26.70% and 10.90%, respectively, compared with Option 1 and Option 2 [29–31].

2.2.4. Determination of the Surrounding Rock Support System. Based on the concept of bolt (cable) supports to control the minimum deformation of the rock surrounding a roadway, the early prestress can be provided by bolt supports and the active support provided by anchors formed a block between them: the number of anchor cables should not exceed three within a certain support range, the preload on each anchor cable should be increased to 200 kN as far as possible, and the length of anchor cables should not exceed 6 m. According to the design principles of high-strength supports and flexible yielding roadway supports, we establish a combined support system of bolts (cables) around the roadway. The optimized bolt support material was an equal-strength left-handed ribbed steel bolt without longitudinal reinforcement (Φ 22 mm) and a high-strength left-handed ribbed steel bolt without longitudinal reinforcement (Φ 22 mm), and the
anchor cable support material was a high-strength pre-stressed anchor cable (\( \Phi 21.80 \text{ mm} \)).

### 3. In Situ Testing

#### 3.1. Engineering Test Background

The second well of the Guanyinshan coal mine is located in Weixin County, Yunnan Province, in which mining of the C5 coal seam with an inclination of 22-25° and a thickness of 2.20-2.60 m was investigated. The distance from the C5 coal seam to the C4 coal seam (thickness 0.45 m) is 8.83 m and a further 7.50 m to C6 (thickness 0.56 m). The test site is located in the E0103 transport channel driven along the coal seam roof strike with its design length of 769 m and an elevation of +1120 m. Figure 5 shows the schematic diagram of the test site location.

The roadway was supported by a top bolt, a wire mesh and cable and high wall bolts, and a wire mesh with low wall nonsupporting bolts. The roof was supported by three high-strength prestressed anchor cables (\( \Phi 21.80 \times 6000 \text{ mm} \), spacing and row spacing 1200 \times 1600 \text{ mm} \)) and six high-strength left-handed ribbed steel bolts without longitudinal reinforcement (\( \Phi 22 \times 2400 \text{ mm} \), spacing and row spacing 800 \times 800 \text{ mm} \)). Three equal-strength left-handed ribbed steel bolts without longitudinal reinforcement (\( \Phi 22 \times 2400 \text{ mm} \), spacing and row spacing 800 \times 800 \text{ mm} \)) were combined with a mesh support arranged on the high side of the roadway. Materials used in the surrounding rock support system and in situ roadway support are shown in Figures 6 and 7.

#### 3.2. Test Method

Five groups of measuring points were arranged in the E06 roof gas drainage roadway along the heading direction. Each group of measuring points was arranged within a borehole (\( \Phi 94 \text{ mm} \)) drilled through the layer to measure the gas pressure (W1-W5) and mining stress (d1-d5), respectively. Table 3 shows the monitoring drilling borehole construction parameters. To avoid cross-interference, the stress monitoring borehole was opened first, then the pressure measurement drilling, and the spacing between boreholes was set to more than 15 m. Monitoring Sections I (E03-Y8+11m) and II (E03-Y9+0m) used for bolt preload and deep displacement were arranged along the direction of roadway excavation in which Section I was equipped with dynamometers 14\(^{°}\), 16\(^{°}\), 18\(^{°}\), 20\(^{°}\), and 22\(^{°}\) and Section II with dynamometers 13\(^{°}\), 15\(^{°}\), 17\(^{°}\), 19\(^{°}\), and 21\(^{°}\). At the same time, the basic parameters related to the coal seam gas were measured during mining. Figures 8 and 9 show the layout plan and cross-sectional view of measuring points in the E0103 transport channel. Figure 10 shows the cross-sectional view of support stress and displacement monitoring in the E0103 transport channel.

| Construction site | Borehole number | Azimuth angle (\( ^{°} \)) | Drilling angle (\( ^{°} \)) | Hole depth (m) | Sealing depth (m) | Construction time       |
|------------------|-----------------|-----------------------------|-----------------------------|----------------|-------------------|------------------------|
| DE-14+60m        | d1              | 1                           | 44                          | 30.66          | 30                | 2018.11.10 morning shift |
|                  | W1              | 1                           | 21                          | 36.50          | 28                | 2018.11.10 middle shift  |
| DE-15+7m         | d2              | 1                           | 24                          | 36.50          | 31                | 2018.11.10 morning shift  |
|                  | W2              | 1                           | 47                          | 29.20          | 24                | 2018.11.10 morning shift  |
| DE-15+36m        | d3              | 1                           | 39                          | 27.74          | 26                | 2018.11.10 evening shift  |
|                  | W3              | 1                           | 17                          | 33.58          | 28                | 2018.11.10 evening shift  |
| DE-16+3m         | d4              | 1                           | 15                          | 36.50          | 34                | 2018.11.9 middle shift    |
|                  | W4              | 1                           | 38                          | 29.20          | 24                | 2018.11.9 middle shift    |
| DE-16+33m        | d5              | 1                           | 38                          | 27.74          | 25                | 2018.11.9 middle shift    |
|                  | W5              | 1                           | 15                          | 36.50          | 30                | 2018.11.9 middle shift    |

**Figure 8: Layout plan: measuring points in the E0103 transport channel.**
4. Results and Discussion

4.1. Analysis of Mining Stress Monitoring on the Whole.

Table 4 shows the monitoring data of mining stress. The stress meter at measuring point d2 was damaged during installation and failed to record data. The mining stress changes are shown in Figure 11.

For measuring point d1, the initial mining stress increased slowly, reaching peak values of 1.02 MPa, 0.14 MPa, and 1.35 MPa in the $d_1(x)$, $d_1(y)$, and $d_1(z)$ directions before and after driving the roadway to measuring point d1. The influences of mining and the mining stress gradually decreased and finally reached a stable state as the footage progressed within the monitoring period from 6 Dec. 2018 to 24 Jan. 2019. For measuring point d3, the mining stress increased slowly from the initial stage, reaching peak values of 0.11 MPa, 0.14 MPa, and 0.13 MPa in the $d_3(x)$, $d_3(y)$, and $d_3(z)$ directions before and after driving the roadway to measuring point d3. The influence of mining decreased gradually as the footage progressed: the mining stress in the $d_3(y)$ direction decreased gradually, that in the $d_3(z)$ direction tended to be stable, and the mining stress in the $d_3(x)$ direction increased gradually, finally reaching a stable state within the monitoring period. For measuring point d4, the initial mining stress of d4 increased slowly, reaching peak values of 0.24 MPa and 0.17 MPa in the $d_4(x)$ and $d_4(z)$ directions before driving the roadway to measuring point d4; the mining stress in the $d_4(y)$ direction decreased...
gradually after reaching a peak value of 0.09 MPa at measuring point d3 and rebounded before driving the roadway to measuring point d4 within the monitoring period from 6 Dec. 2018 to 18 Jan. 2019. For measuring point d5, the initial mining stress increased slowly, reaching peak values of 0.06 MPa, 0.25 MPa, and 0.12 MPa in the d5(x), d5(y), and d5(z) directions before driving the roadway to measuring point d5. The influence of mining gradually decreased, and the mining stress decreased rapidly as the footage progressed within the monitoring period from 6 Dec. 2018 to 16 Feb. 2019.

![Figure 10: Cross-sectional view: support stress and displacement monitoring in the E0103 transport channel.](image)

| Time     | d1(x) | d1(y) | d1(z) | d3(x) | d3(y) | d3(z) | d4(x) | d4(y) | d4(z) | d5(x) | d5(y) | d5(z) |
|----------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|
| 06/12/18 | 0.80  | 0.12  | 0.71  | 0.06  | 0.03  | 0.07  | 0.10  | 0.05  | 0.09  | 0.00  | 0.06  | 0.03  |
| 08/12/18 | 0.84  | 0.13  | 0.77  | 0.06  | 0.03  | 0.08  | 0.11  | 0.05  | 0.10  | 0.00  | 0.07  | 0.02  |
| 10/12/18 | 0.97  | 0.14  | 0.97  | 0.06  | 0.04  | 0.08  | 0.12  | 0.05  | 0.10  | 0.01  | 0.07  | 0.01  |
| 12/12/18 | 0.99  | 0.14  | 1.32  | 0.09  | 0.07  | 0.10  | 0.13  | 0.06  | 0.11  | 0.01  | 0.08  | 0.00  |
| 18/12/18 | 1.00  | 0.13  | 1.35  | 0.10  | 0.11  | 0.11  | 0.15  | 0.07  | 0.12  | 0.01  | 0.08  | 0.01  |
| 26/12/18 | 1.02  | 0.11  | 1.32  | 0.12  | 0.13  | 0.12  | 0.16  | 0.07  | 0.13  | 0.02  | 0.09  | 0.02  |
| 05/01/19 | 0.95  | 0.11  | 1.29  | 0.09  | 0.09  | 0.13  | 0.19  | 0.08  | 0.15  | 0.03  | 0.13  | 0.02  |
| 07/01/19 | 0.89  | 0.10  | 1.19  | 0.10  | 0.09  | 0.13  | 0.21  | 0.09  | 0.15  | 0.03  | 0.14  | 0.02  |
| 09/01/19 | 0.84  | 0.08  | 1.13  | 0.11  | 0.09  | 0.13  | 0.21  | 0.09  | 0.15  | 0.04  | 0.14  | 0.03  |
| 11/01/19 | 0.85  | 0.05  | 1.13  | 0.11  | 0.09  | 0.13  | 0.22  | 0.08  | 0.16  | 0.04  | 0.15  | 0.04  |
| 13/01/19 | 0.82  | 0.04  | 1.13  | 0.11  | 0.08  | 0.13  | 0.22  | 0.08  | 0.16  | 0.04  | 0.16  | 0.04  |
| 17/01/19 | 0.82  | 0.04  | 1.13  | 0.12  | 0.08  | 0.13  | 0.23  | 0.07  | 0.17  | 0.04  | 0.17  | 0.05  |
| 18/01/19 | 0.78  | 0.04  | 1.13  | 0.12  | 0.08  | 0.13  | 0.24  | 0.07  | 0.17  | 0.04  | 0.17  | 0.07  |
| 20/01/19 | 0.76  | 0.04  | 1.13  | 0.12  | 0.07  | 0.13  | —     | —     | —     | 0.03  | 0.18  | 0.09  |
| 22/01/19 | 0.76  | 0.05  | 1.13  | 0.13  | 0.07  | 0.13  | —     | —     | —     | 0.03  | 0.19  | 0.08  |
| 24/01/19 | 0.77  | 0.05  | 1.13  | 0.13  | 0.07  | 0.13  | —     | —     | —     | 0.03  | 0.19  | 0.07  |
| 26/01/19 | —     | —     | —     | —     | —     | —     | —     | —     | —     | 0.04  | 0.20  | 0.09  |
| 08/02/19 | —     | —     | —     | —     | —     | —     | —     | —     | —     | 0.04  | 0.21  | 0.10  |
| 10/02/19 | —     | —     | —     | —     | —     | —     | —     | —     | —     | 0.05  | 0.24  | 0.11  |
| 12/02/19 | —     | —     | —     | —     | —     | —     | —     | —     | —     | 0.06  | 0.26  | 0.12  |
| 14/02/19 | —     | —     | —     | —     | —     | —     | —     | —     | —     | 0.04  | 0.22  | 0.10  |
| 16/02/19 | —     | —     | —     | —     | —     | —     | —     | —     | —     | 0.04  | 0.20  | 0.09  |
Table 5: The linear relationship of mining stress variation with time.

| Construction site | Relationship | Related coefficient |
|-------------------|--------------|---------------------|
| d1(x)             | $y = 2E^{-05}x^3 - 2.1033x^2 + 91435x - 1E + 09$ | $R^2 = 0.9426$ |
| d3(x)             | $y = 3E^{-06}x^3 - 0.3807x^2 + 16548x - 2E + 08$ | $R^2 = 0.9419$ |
| d4(x)             | $y = 0.0031x - 132.4$ | $R^2 = 0.9887$ |
| d5(x)             | $y = 9E^{-08}x^3 - 0.0118x^2 + 514.43x - 7E + 06$ | $R^2 = 0.8503$ |
4.2. Analysis of Mining Stress Monitoring in $d(x)$. The mining stress changes along the roadway are shown in Figure 12, and Table 5 shows the linear relationship of mining stress variation with time.

The advance of the working face makes the stress change of a coal body in front increase first and then decrease, which is consistent with the traditional understanding. According to $d_1$ and $d_4$, stress appears at 96 m ahead due to mining;
4.3. Analysis of Mining Stress Monitoring in $d$. The mining stress changes along drilling are shown in Figure 14, and Table 7 shows the linear relationship of mining stress variation with time.

The advance of the working face makes the stress change of a coal body in front increase first and then decrease, which is consistent with the traditional understanding. According to $d_1$ and $d_4$, stress appears at 96 m ahead due to mining; after 57 m of advancing, there is a large increase until it passes through this area. According to $d_3$, $d_4$, and $d_5$, the mining stress in front of a coal body increases gradually with the increase of the distance from the working face. The stress in front of a coal body increases linearly, and the increase range is obviously larger than that of the coal body in a certain range on both sides. Taking 5 Jan. 2019 as an example, when the working face reaches $d_3$, the stress at $d_1$ decreases greatly and tends to be stable, the stress at $d_3$ begins to rise slowly, and the stress at $d_4$ increases linearly. An obvious stress appears at 39 m in front of the working face. The stress at $d_5$ rises slowly until the working face pushes through the area.

4.4. Analysis of Mining Stress Monitoring in $d(y)$. The mining stress changes of the vertical roadway are shown in Figure 13, and Table 6 shows the linear relationship of mining stress variation with time.

The advance of the working face makes the stress change of a coal body in front increase first and then decrease, which is consistent with the traditional understanding. According to $d_1$, $d_3$, $d_4$, and $d_5$, the mining stress of the vertical roadway is usually in the peak state when the working face is pushed to the corresponding position. The peak stress point is close to the measuring point area and arranged with each other.

### Table 7: The linear relationship of mining stress variation with time.

| Construction site | Relationship | Related coefficient |
|-------------------|--------------|---------------------|
| $d_1(z)$          | $y = 4E - 05x^2 - 5.4908x + 238682x - 3E + 09$ | $R^2 = 0.8706$ |
| $d_3(z)$          | $y = 7E - 07x^2 - 0.0962x + 4182.5x - 6E + 07$ | $R^2 = 0.9881$ |
| $d_4(z)$          | $y = 0.0018x - 77.287$ | $R^2 = 0.9919$ |
| $d_5(z)$          | $y = -2E - 06x^2 + 0.2079x^2 - 9038.1x + 1E + 08$ | $R^2 = 0.9489$ |

### Table 8: Monitoring results: support stress and displacement of Section I (E03-Y8+11m).

| Time            | Support stress (MPa) | Displacement (mm) |
|-----------------|----------------------|-------------------|
| 28/12/18        | 0.92  0.04  0.45  1.11  0.43 | 4 |
| 29/12/18        | 1.05  0.13  1.21  4.31  2.98 | 8 |
| 30/12/18        | 1.01  0.14  1.11  3.90  2.80 | 12 |
| 31/12/18        | 0.96  0.14  1.06  3.70  2.43 | 14 |
| 2/01/19         | 0.93  0.12  1.01  3.50  2.51 | 16 |
| 4/01/19         | 0.91  0.13  0.98  3.20  2.23 | 18 |
| 5/01/19         | 0.92  0.04  0.94  3.08  2.09 | 20 |
| 7/01/19         | 0.91  0.11  0.57  2.52  1.80 | 22 |
| 9/01/19         | —      0.10  0.29  2.32  1.62 | 24 |
| 11/01/19        | —      0.06  0.29  2.45  1.68 | 26 |
| 13/01/19        | —      0.05  0.73  2.20  1.85 | 28 |
| 17/01/19        | —      0.04  0.09  2.56  1.49 | 28 |
| 18/01/19        | —      0.04  0.18  2.48  1.46 | 28 |
| 20/01/19        | —      0.04  0.17  2.55  1.51 | 28 |
| 22/01/19        | —      —      2.50  1.47  28 |
| 24/01/19        | —      —      —      2.05  1.41 | 28 |

### Table 9: Monitoring results: support stress and displacement of Section II (E03-Y9+0m).

| Time            | Support stress (MPa) | Displacement (mm) |
|-----------------|----------------------|-------------------|
| 16/01/19        | 1.13  1.55  0.63  2.34  1.78 | 2 |
| 17/01/19        | 5.65  4.54  1.21  5.76  3.20 | 4 |
| 18/01/19        | 5.76  4.40  1.11  5.65  3.30 | 6 |
| 19/01/19        | 5.40  4.10  1.06  5.30  3.00 | 8 |
| 20/01/19        | 5.18  3.80  1.01  5.41  2.60 | 8 |
| 21/01/19        | 5.00  3.58  0.98  5.12  2.10 | 10 |
| 22/01/19        | 4.50  3.41  0.94  4.95  1.80 | 12 |
| 23/01/19        | 4.10  3.12  0.57  4.68  1.60 | 14 |
| 24/01/19        | 3.70  3.05  0.29  4.81  1.50 | 14 |
| 26/01/19        | 3.60  2.91  0.29  4.35  1.42 | 16 |
| 08/02/19        | 3.52  3.01  0.73  4.33  1.37 | 16 |
| 10/02/19        | 3.48  3.08  0.09  4.53  1.39 | 16 |
| 12/02/19        | 3.50  3.10  0.18  4.39  1.38 | 16 |
| 14/02/19        | 3.53  3.01  0.17  4.38  1.40 | 16 |
| 16/02/19        | 3.41  3.05  0.17  4.41  1.37 | 16 |

after 57 m of advancing, there is a large increase until it passes through this area. According to $d_3$, $d_4$, and $d_5$, the mining stress in front of a coal body gradually increases with the increase of the distance from the working face. The stress in front of the working face increases linearly, and the increase range is obviously larger than that of the coal body in a certain range on both sides. Taking 5 Jan. 2019 as an example, when the working face reaches $d_3$, the stress at $d_1$ decreases greatly and tends to be stable, the stress at $d_3$ begins to rise slowly, and the stress at $d_4$ increases linearly. An obvious stress appears at 39 m in front of the working face. The stress at $d_5$ rises slowly until the working face pushes through the area.
Tables 8 and 9, and Tables 8 and 9 show the analysis of monitoring results, as shown in Figures 15 and 16.

The anchoring force at dynamometers 14, 16, 18, 20, and 22 increased rapidly to the maximum value within 1-2 days. As the footage progressed, the influence of mining decreased, as did the anchoring force which then stabilized. The maximum anchoring stress at dynamometers 20 and 22 in the middle of the roof reached 4.40 MPa and finally stabilized at 2-3 MPa due to the direct bearing of stress by the surrounding rock. The anchoring stress in dynamometers 16 and 18 in the high part of the roadway was relatively small and finally stabilized at 0-1 MPa. The accumulated displacement of the roof in Section I increased linearly. As the footage progressed, the influence of mining decreased and the accumulated roof displacement reached 28 mm and then tended to be stable.

The anchoring force in dynamometers 13, 15, 17, 19, and 21 in the middle of the roof reached 5.80 MPa and finally stabilized at 3-5 MPa due to the direct bearing of stress by the surrounding rock. The anchoring stress at dynamometers 17 and 21 in the high part of the roadway was relatively small and finally stabilized at 0-2 MPa. The accumulated displacement of the roof in Section II increased linearly. As the footage progressed, the influence of mining decreased and the accumulated roof displacement reached 16 mm and then tended to be stable.

The space-time effect of anchoring force was mainly manifested in three stages. The first stage was a rapid rise, reaching a peak when the excavation approached the end face of the measuring point which was due to the strong unloading caused by roadway excavation and the rapid development of coal and rock fracture expansion. At this time, the initial prestress on the bolt and anchor cable could be restored, improving the surface stress state of the coal and rock masses to a certain extent and inhibiting the shear fracture and expansion of coal and rock masses along the sliding surface. The second stage was one of slow decline caused by repeated adjustment of internal stress between the coal and rock.
masses and the anchorage support system. The third stage was the stable stage, and the coal and rock masses and anchorage support gradually reached a new equilibrium and tended to be stable. The space-time effect of displacement changes was demonstrated in three stages which were consistent with the change in anchoring load. The first stage was one of rapid rise, and the displacement developed rapidly, especially when rapidly increasing the anchoring force with the maximum rate of displacement reaching 4 mm/d. The second stage saw a slow increase: the variations in the displacement were gradually reduced by mining action with the continuous action of the anchorage support on the coal mass. The third stage was stable, and when the new balance between the coal and rock masses and the anchorage support gradually reached equilibrium, any subsequent changes in displacement were stable [32].

The displacement increased greatly before the stress on the support increased to its peak value. The decrease of stress from the peak value occurred in the stage of slow accumulation of displacement, and the increase of displacement was slower. The stress and displacement change synchronously and tend to be stable, suggesting that anchoring force and displacement were basically consistent.

4.6. Monitoring and Analysis of Coal Gas Parameters. Table 10 shows the monitoring results of residual gas pressure. Table 11 shows the monitoring results of the residual gas pressure gas desorption index of drill cuttings. The gas parameters of coal seams during mining, as shown in Figure 17.

The residual gas pressure at measuring points W1-W5 was affected by the predrainage of coal seam gas through the borehole, and the maximum value was no more than 0.12 MPa. The residual gas pressures at measuring points W1, W3, and W5 in front of the roadway were stable at 0.06 MPa, W4 MPa, and 0.04 MPa before roadway driving reached the measuring point. When mining passed the measuring points, the residual gas pressure rapidly dropped to 0 MPa. Measuring points W2 and W4 (15 m from the sides of the roadway heading direction) were stable at 0.06 MPa and 0.07 MPa before the heading reached the measuring point; thereafter, the residual gas pressure decreased significantly.

The gas desorption index of drill cuttings in the E0103 transport channel was no more than 0.50 mL/(g·min$^{1/2}$)$^{-1}$. The gas desorption index was higher near measuring points (1$^*$ and 4$^*$) with a high residual gas pressure and gas content, reaching 0.45-0.50 mL/(g·min$^{1/2}$)$^{-1}$. The gas desorption index of drill cuttings was lower near measuring points with low residual gas pressure and gas content, at least 0.40 mL/(g·min$^{1/2}$)$^{-1}$.

4.7. Quantitative Relationship between Development of Stress and Coal Gas. The quantitative relationships between emergence of changes in stress due to mining and coal gas are shown in Table 12 and Figure 18.

The stress change along the direction of the roadway has the same trend in growth as the coal seam gas content and gas pressure and gas desorption index of drill cuttings under the influence of mining. The relationship between stress along the roadway direction and gas content conforms to the logarithmic function $y = 0.0275 \ln x + 0.1167$ ($R^2 = 0.9771$), conforms to the logarithmic function $y = 0.4917 \ln x + 6.8359$ ($R^2 = 0.9216$) with gas pressure, and conforms to the logarithmic function $y = 0.0397 \ln x + 0.4511$ ($R^2 = 0.9886$) with the gas desorption index of the drill cuttings.

During the process of coal mining, it is inevitable to produce a large range of mining responses. The coal strata are deformed and broken, resulting in a large number of fissures. The stress field, fracture field, seepage field, and concentration field all change greatly. Under the comprehensive influence of original stress and mining-induced stress, the relief zone, stress concentration zone, and original stress zone appear in the coal body in front of the heading face which determines the stress state, damage degree, and gas seepage characteristics of the coal body [33]. According to the stress analysis of coal and rock masses and the evolution law of the gas flow channel, the coal and rock mass gas channel in front of the working face is divided into the isolated channel area, fracture area, shear failure area, and failure area after the peak abutment pressure.

| Time     | W1 | W2 | W3 | W4 | W5 |
|----------|----|----|----|----|----|
| 13/11/18 | 0.10 | 0.04 | 0.04 | 0.05 | 0.06 |
| 25/11/18 | 0.12 | 0.07 | 0.05 | 0.07 | 0.07 |
| 27/11/18 | 0.12 | 0.06 | 0.06 | 0.07 | 0.06 |
| 28/11/18 | 0.12 | 0.06 | 0.06 | 0.07 | 0.06 |
| 14/12/18 | 0.07 | 0.06 | 0.07 | 0.06 |
| 16/12/18 | 0.07 | 0.06 | 0.07 | 0.06 |
| 18/12/18 | 0.07 | 0.06 | 0.07 | 0.05 |
| 20/12/18 | 0.06 | 0.06 | 0.07 | 0.05 |
| 22/12/18 | 0.06 | 0.06 | 0.07 | 0.05 |
| 24/12/18 | 0.06 | 0.06 | 0.07 | 0.04 |
| 26/12/18 | 0.06 | 0.04 | 0.07 | 0.04 |
| 5/01/19  | 0.03 | 0.03 | 0.07 | 0.04 |
| 7/01/19  | 0.03 | 0.07 | 0.04 |
| 9/01/19  | 0.03 | 0.07 | 0.04 |
| 11/01/19 | 0.02 | 0.07 | 0.04 |
| 13/01/19 | 0.02 | 0.07 | 0.04 |
| 17/01/19 | 0.02 | 0.06 | 0.04 |
| 18/01/19 | 0.02 | 0.05 | 0.04 |
| 20/01/19 | 0.02 | 0.05 | 0.04 |
| 22/01/19 | 0.01 | 0.05 | 0.04 |
| 24/01/19 | 0.01 | 0.05 | 0.04 |
| 26/01/19 | 0.01 | 0.05 | 0.04 |
| 8/02/19  | 0.01 | 0.05 | 0.04 |
| 10/02/19 | 0.01 | 0.05 | 0.04 |
| 12/02/19 | 0.01 | 0.05 | 0.03 |
| 14/02/19 | 0.01 | 0.05 | 0.0 |
| 16/02/19 | 0.01 | 0.05 | 0.0 |
Due to the influence of the axial unloading and stress peak, the fracture develops into a macroscopic fracture surface, the strain softens, the lateral slip occurs, the bearing capacity decreases significantly, the permeability increases by a jump, and the gas emission rate accelerates significantly.

The stress concentration zone can be further subdivided into the plastic failure zone and elastic zone according to the different stress states and sizes. The coal body in the plastic failure area is in the state of three-direction stress, and the stress gradually increases to the peak of concentrated stress. The coal body in the area suffers damage, and its permeability is increased compared with that of the original coal body. The elastic zone coal is basically in the elastic deformation zone because the stress does not reach the yield value. In the elastic stage of coal, the crack opening is reduced and the permeability is greatly weakened. The coal body in the original rock stress area is far away from the mining space, and the stress and permeability have not changed significantly [34, 35].

Figure 19 shows the stress appearance-coal damage-gas seepage-dynamic disaster evolution process. Generally, coal and gas outbursts occurred under the comprehensive action of stress, gas, and the coal body which were coupled with each other and reached a certain critical limit under the action of mining disturbance. Stress not only controls the state of occurrence of gas in a coal seam body but also controls its flow therein. There were few macropore channels and a dense micropore structure, especially under conditions of high stress [36–38]. In the vicinity of certain geological structures in the area, stress, gas, coal, and other factors were strongly coupled which can provide advanced warning of rock bursting events, but in this small area, the continuity of the coal body may not be destroyed, but the stress could become relatively concentrated, leading to an accumulation of stored energy. High stress may lock in the gas energy before detection and disturbance; high stress does not necessarily mean a high gas content or pressure, but high gas generally means high stress, making stress the triggering and inducing factor of coal and gas outbursts. All of this provides energy storage conditions and dynamic conditions for coal and gas outbursts [39].

| Table 11: Monitoring results of the residual gas pressure gas desorption index of drill cuttings. |
|---|---|---|---|---|
| Time | Gas desorption index of drill cuttings (mL·(g·min$^{-1/2}$)$^{-1}$) | Time | Gas desorption index of drill cuttings (mL·(g·min$^{-1/2}$)$^{-1}$) | Time | Gas desorption index of drill cuttings (mL·(g·min$^{-1/2}$)$^{-1}$) |
| 12/12/18 | 0.45 | 06/01/19 | 0.29 | 13/01/19 | 0.40 |
| 13/12/18 | 0.49 | 07/01/19 | 0.36 | 14/01/19 | 0.40 |
| 24/12/18 | 0.40 | 08/01/19 | 0.36 | 20/01/19 | 0.39 |
| 29/12/18 | 0.41 | 09/01/19 | 0.36 | 21/01/19 | 0.39 |
| 30/12/18 | 0.35 | 11/01/19 | 0.47 | 22/01/19 | 0.39 |
| 05/01/19 | 0.37 | 12/01/19 | 0.44 | 23/01/19 | 0.32 |

| Table 12: Stress and gas parameters. |
|---|---|---|---|
| Measuring point | Along the roadway (x) | Vertical roadway (y) | Along drilling (z) | Gas content (m$^3$·t$^{-1}$) | Gas pressure (MPa) | Gas desorption index of drill cuttings (mL·(g·min$^{-1/2}$)$^{-1}$) |
| 1 | 1.02 | 0.14 | 1.35 | 6.7244 | 0.12 | 0.45 |
| 3 | 0.11 | 0.14 | 0.13 | 5.8045 | 0.06 | 0.36 |
| 4 | 0.24 | 0.09 | 0.17 | 6.3584 | 0.07 | 0.40 |
| 5 | 0.06 | 0.25 | 0.12 | 5.2957 | 0.04 | — |
Stress reduction is key to preventing coal and gas outbursts. With deeper and deeper mining, stress and gas pressure gradients increased. At the same time, the surrounding rock stress and deformation fracture and migration continue to increase. All of which caused the proportion of coal and gas outburst types coupled with stress and gas peaks to increase.

5. Conclusions

(1) Under the continuous effects of mining, the stress ahead of the advance of the roadway increased slowly to a peak, then gradually decreased, and tended to be stable. The anchoring force decreased gradually and tended to stabilize after rapidly increasing to its maximum value. The displacement of the roof increased linearly and tended to be stable after reaching the accumulated displacement of the deep roof. The space-time effect of support anchoring force and displacement showed similar trends

(2) Stress appears at 96 m ahead due to mining; after 57 m of advancing, there is a large increase until it passes through this area. The stress in front of the working face increases linearly, and the increase range is obviously larger than that of the coal body in a certain range on both sides. With continuous mining, the residual gas pressure in the advance of the mining face decreased rapidly due to the pressure-relief effect and the residual gas pressure at more than 15 m from both sides of the roadway decreased to a significant extent

(3) The gas desorption index of drill cuttings was relatively high in the area subject to high residual gas pressure and having a high gas content. The changes in stress along the direction of the roadway follow a similar trend with the coal seam gas content, gas pressure, and gas desorption index of drill cuttings under the influence of mining. The relationship between stress along the roadway direction and gas content conforms to the logarithmic function $y = 0.0275 \ln(x) + 0.1167$ ($R^2 = 0.9771$), conforms to the logarithmic function $y = 0.4917 \ln(x) + 6.8359$ ($R^2 = 0.9216$) with gas pressure, and conforms to the logarithmic function $y = 0.0397 \ln(x) + 0.4511$ ($R^2 = 0.9886$) with the gas desorption index of drill cuttings

(4) With increasing stress, the surrounding rock stress and deformation fracture and gas gradient continue to increase so that the coupling of stress and gas content and peak pressure exerts an increasing influence on the risk of coal and gas outburst events

Data Availability

The data used to support the findings of this study are included within the article.
Conflicts of Interest

The authors declare no conflict of interest.

Authors’ Contributions

All the authors listed have approved the manuscript that is enclosed.

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