

Special Issue Reprint

Monitoring, Process Control, Simulation, and Optimization in Coal Mining

Edited by
Yangyang Guo, Bo Li and Wei Zhao

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Monitoring, Process Control, Simulation, and Optimization in Coal Mining

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This is a reprint of the Special Issue, published open access by the journal *Processes* (ISSN 2227-9717), freely accessible at: https://www.mdpi.com/journal/processes/special_issues/952D80IPK1.

For citation purposes, cite each article independently as indicated on the article page online and as indicated below:

Lastname, A.A.; Lastname, B.B. Article Title. <i>Journal Name</i> Year , Volume Number, Page Range.
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ISBN 978-3-7258-8777-4 (Hbk)

ISBN 978-3-7258-8778-1 (PDF)

<https://doi.org/10.3390/books978-3-7258-8778-1>

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Monitoring, Process Control, Simulation, and Optimization in Coal Mining

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1. Introduction

Coal remains an essential energy resource supporting global industry and economic development, and the continuous depletion of shallow coal reserves has driven mining activities toward greater depths [1–3]. Compared with shallow mining, deep coal extraction is usually carried out under more severe geological and engineering conditions, including elevated temperature, high in situ stress, high gas pressure, and poor permeability [4,5]. These coupled factors can disturb the stability of coal–rock masses and intensify the risk of dynamic and gas-related disasters, such as coal and gas outbursts, rock bursts, and gas–dust explosions [6,7]. Once such accidents occur, they may cause serious casualties, interrupt normal production, and bring considerable economic losses. Therefore, revealing the occurrence mechanisms of deep coal mine disasters and developing effective prevention and control methods are of great importance for ensuring the safe, efficient, and sustainable exploitation of coal resources.

2. Overview of Contributions

This Special Issue’s contributions can be categorized into four key themes.

2.1. Mechanisms and Prevention of Dynamic Disasters

Li et al. (contribution 4) investigated the failure characteristics of coal–rock structures with different bursting liabilities and analyzed the influence of primary fracture structures on coal–rock deformation and failure before peak stress. Tang et al. (contribution 6) studied the mechanical and permeability characteristics of gas-bearing coal under different bedding angles and showed that bedding angle has an important influence on peak stress, permeability evolution, and failure tendency. Chen et al. (contribution 7) examined borehole failure mechanics in gas-bearing soft coal seams under complex geological conditions and identified two main types of borehole failure: fracture-dominated failure and stress-dominated failure. Shen et al. (contribution 9) studied the initial gas desorption characteristics of tectonic coal under vibration and explained how vibration can accelerate coal fragmentation, increase gas expansion energy, and promote coal and gas outbursts. Shen et al. (contribution 11) further analyzed the pore structure multifractal evolution of vibration-affected tectonic coal and revealed the relationship between pore structure changes and gas desorption and diffusion responses. Li et al. (contribution 13) constructed

a damage analysis model of coal under hydraulic load and provided guidance for the prevention and control of gas outbursts and rockbursts in deep coal mines.

2.2. Coal Bed Gas Adsorption, Desorption, Diffusion, and Emission Characteristics

Xu et al. (contribution 5) conducted gas desorption tests on different coal ranks and found that coal metamorphism has a strong influence on the gas desorption rate and effective diffusion coefficient. Xiao et al. (contribution 2) studied the automatic gas emission width of coal bodies in the goaf near outbursting coal seams using theoretical analysis, strain-softening simulation, and field testing, providing a basis for adjacent roadway design and gas prevention measures. Chen et al. (contribution 3) quantitatively identified emission sources and emission dynamics of pressure-relieved methane under different mining intensities by combining geostatistical inversion, FLAC3D-COMSOL coupled simulation, and isotope tracing. The results showed that adjacent seams were the dominant source of pressure-relieved methane and that targeted long directional drainage boreholes could effectively reduce methane emissions. Shen et al. (contribution 9) revealed that vibration can increase the initial desorption capacity of tectonic coal. Shen et al. (contribution 11) showed that vibration improves pore connectivity and enhances the gas desorption and diffusion capacity of tectonic coal. These studies provide important support for understanding gas migration, gas over-limit, and the development of gas-related disasters in coal mining.

2.3. Gas Extraction Technology and Permeability Enhancement of Low-Permeability Coal Seams

Chen et al. (contribution 10) investigated the combined effect of multiple hydraulic slotting spatial parameters on gas extraction efficiency through numerical analysis and field verification. Their results showed that hydraulic slotting can significantly improve pressure relief, fracture development, and permeability enhancement, thereby reducing borehole gas pressure and improving gas extraction efficiency. Hao et al. (contribution 12) studied the effect of unloading paths on coal damage and permeability evolution and showed that unloading damage can reduce coal strength, improve permeability, and enhance gas diffusion and extraction. Li et al. (contribution 13) analyzed the damage mechanism of coal under hydraulic load and proposed a theoretical basis for the design and evaluation of small-scale hydraulic fracturing in underground coal mines. Bi et al. (contribution 14) reviewed the theories, techniques, and materials for sealing coalbed methane extraction boreholes in underground mines and summarized future development directions, including intelligent sealing technology, leakage detection, non-destructive monitoring, and improved sealing materials. These studies are closely related to gas extraction in low-permeability coal seams and provide useful technical support for improving gas drainage efficiency and reducing gas disaster risks.

2.4. Process Safety, Mining Control, and Engineering Optimization

Duan et al. (contribution 1) developed a new testing method and rheometer for measuring the rheological properties of large-particle gangue filling slurry, addressing the difficulty of accurately measuring slurry containing large gangue particles and supporting the development of coal mine filling technology. Shen et al. (contribution 8) proposed non-pillar coal mining by driving a roadway during the mining period in a high-gas top-coal-caving working face. Their study optimized coal pillar width and roadway layout, reduced surrounding rock deformation, controlled gas concentration in the return airway, and improved the coal extraction rate. Xiao et al. (contribution 2) also provided data support for roadway location design by determining automatic gas discharge width in the neighboring goaf. Chen et al. (contribution 3) offered an adaptive gas control method for rapidly advancing working faces through stress–permeability coupling analysis and

targeted methane drainage. Together, these studies reflect the role of process control, simulation, and optimization in improving coal mine safety, reducing gas accumulation, and supporting efficient coal production.

3. Conclusions

The contributions in this Special Issue highlight advances in understanding dynamic disasters, coal mine gas behavior, gas extraction in low-permeability seams, gas explosions, and adsorption and desorption processes. The contributions provide comprehensive insights into the mechanisms of dynamic disasters, including coal–rock failure, rockbursts, and outburst-prone structures, which are critical for improving safety in deep and complex mining environments. Research on coal bed gas adsorption, desorption, diffusion, and emission has clarified the factors influencing gas migration and over-limit events, supporting more accurate risk assessment and early warning strategies. Studies on gas extraction technologies and permeability enhancement methods offer practical solutions for low-permeability coal seams, including hydraulic slotting, borehole sealing, and unloading techniques, improving gas drainage efficiency and mitigating disaster risks. Furthermore, investigations into mining process control, roadway layout optimization, and filling operations demonstrate the effectiveness of engineering strategies and numerical simulations in enhancing operational safety and extraction efficiency. Together, these contributions provide valuable guidance for future research and practical applications in coal mine safety and process optimization.

Funding: This research received no external funding.

Acknowledgments: Thanks to all the authors and peer reviewers for their valuable contributions to this Special Issue ‘Monitoring, Process Control, Simulation, and Optimization in Coal Mining’. I would also like to express my gratitude to all the staff and people involved in this Special Issue.

Conflicts of Interest: Yangyang Guo was employed by the company China Coal Research Institute Co., Ltd., Beijing 100013, China. The remaining authors declare that the research was conducted in the absence of any commercial or financial relationships that could be construed as a potential conflict of interest.

List of Contributions:

1. Duan, X.; Huang, Y.; Hao, Y.; Zhang, L. Research on the Testing Method for the Rheological Properties of Large-Particle Gangue Filling Slurry. *Processes* **2025**, *13*, 789. <https://doi.org/10.3390/pr13030789>.
2. Xiao, J.; Bi, R.; Chen, X.; Li, S.; Chen, Z.; Chen, J. Automatic Gas Emission Width of Coal Bodies in the Goaf near Outbursting Coal Seams. *Processes* **2025**, *13*, 715. <https://doi.org/10.3390/pr13030715>.
3. Chen, X.; Chen, X.; Hu, J.; Xiao, J.; Sun, J.; Yan, Z. Quantitative Identification of Emission Sources and Emission Dynamics of Pressure-Relieved Methane Under Variable Mining Intensities. *Processes* **2025**, *13*, 704. <https://doi.org/10.3390/pr13030704>.
4. Li, H.; He, S.; Mo, Y.; Sun, Z.; Li, L. A Study on the Failure Characteristics of Coal–Rock Structures with Different Bursting Liabilities. *Processes* **2025**, *13*, 652. <https://doi.org/10.3390/pr13030652>.
5. Xu, H.; Qi, X.; Wang, H.; Liang, Z.; Yang, T.; Zou, Y.; Jiang, Q.; Jin, L. Gas Desorption Characteristics of Different Coal Ranks. *Processes* **2025**, *13*, 570. <https://doi.org/10.3390/pr13020570>.
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Article

Research on the Testing Method for the Rheological Properties of Large-Particle Gangue Filling Slurry

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Abstract: Coal mine gangue cementation filling technology has increasingly become an effective and major means of dealing with “coal mining under buildings, railways, and bodies of water” and other complex hard-to-mine coal seams; but also, an important part of a large number of treatments of coal gangue stockpiled on the ground is to realize the green mining of coal mines. Coal mine cement filling often contains gangue particles with particle sizes larger than 15 mm; however, the viscometer and rheometer currently used at home and abroad are unable to accurately measure the rheological parameters of the slurry containing large-particle-sized gangue. In order to accurately measure the rheological parameters of slurry containing large-sized gangue particles combined with the site filling materials, the torque values obtained on the mixing blades at different speeds were generated by the combined action of the slurry between the blade side edge and the mixing drum wall, as well as the slurry between the blade lower edge and the mixing drum bottom. A new type of gangue slurry rheometer was developed. The new type of gangue slurry rheometer mainly included components such as the power system, sensing system, mechanical system, and other auxiliary units. Finally, using Fluent software ANSYS2023 to numerically simulate the fluidity of the slurry under the same conditions, the results obtained after the calculation and the test results showed that the error was within a reasonable range, indicating the correctness of the test principles of the new gangue slurry rheometer and the effectiveness of the instrument. This research offers new insights for accurately measuring the rheological parameters of particles with large sizes.

Keywords: large-particle gangue; filling slurry; rheological properties; rheometer

1. Introduction

The development and utilization of mineral resources have greatly promoted the rapid development of human economy, society, and the progress of social civilization, and will continue to be a source of human development in the future [1–4]. With the continuous development of mineral resources, the number, scale, and depth of mining operations continue to increase [5–9].

Long-term large-scale and continuous mining has led to the depletion of mineral resources with good storage conditions [10]. The recovery and utilization of complex and difficult to mine coal seams, “coal mining under buildings, railways, and bodies of water”, roadway coal pillars, and corner coal, as well as the re-mining of residual coal under other conditions, have gradually entered people’s vision [11]. On the other hand, as the largest

industrial solid waste currently stored in China, the massive storage and discharge of coal gangue not only occupies a large amount of land, but its unique physical and chemical properties can also pollute soil and water bodies. At the same time, the spontaneous combustion of gangue mountains can produce a large amount of toxic and harmful gases, pollute the atmosphere, and have a great impact on the ecosystem around the mine [12–14]. However, the single utilization method of gangue, its low product added value, and small disposal and utilization capacity and scale cannot meet China's practical requirements for ecological environmental protection, pollution reduction, and carbon reduction under the "dual carbon" goal. There is an urgent need for effective integrated and systematic action plans to solve these fundamental problems. From the perspective of coal resource recovery and environmental protection, using gangue filling coal mining technology to solve the above special coal mining problems is an effective method [15].

Compared with traditional low-concentration water sand filling, high-concentration cemented filling coal mining technology (such as paste and paste-like filling) has the advantages of a high slurry concentration [16–20], no need for dehydration (or only a small amount of dehydration), the good integrity of the filling body, high strength [21], a significant roof control effect, and a good underground working environment. It has gradually been applied in coal mine goaf treatment in the past decade [22–24]. Currently, research on high-concentration cemented filling coal mining technology in coal mines mainly focuses on the development and application of filling coal mining technology and filling systems, as well as the mechanical properties of filling materials and their interaction mechanisms with the surrounding rocks [25–31]. There is relatively little research on the flow performance and pipeline transportation characteristics of gangue filling slurry [32].

Unlike the tailing filling aggregates used in metal mines, the gangue particles used in coal mine cemented filling have larger particle sizes, some of which can reach 15 mm or even larger. Traditional methods such as NXS-11A rotational viscometer and RheolabQC rheometer cannot accurately measure gangue, as due to their small measurement space, large-sized gangue particles cannot be placed inside. On the other hand, when measuring high concentration slurries containing large-sized gangue particles, the mixing element is prone to slipping due to slip effects, making it difficult to accurately test the rheological parameters of this type of slurry. Based on the above reasons, the author has developed a new type of gangue slurry rheometer suitable for measuring large-sized gangue particles from the perspective of the testing process and the principle of the viscometer for slurry. The improvement in the testing accuracy lays the foundation for the smooth pipeline transportation of coal mine gangue filling slurry.

2. Structure and Working Principle of a New Type of Gangue Slurry Rheometer

2.1. Structure of a New Type of Gangue Slurry Rheometer

In order to adapt to the smooth flow of large-sized gangue particles in the mixing drum, generate accurate shear torque, and improve the automation level and data processing accuracy of slurry testing, a new type of gangue slurry rheometer has been developed. Its structural schematic diagram is shown in Figure 1.

After assembling the various components, the new type of gangue slurry rheometer obtained is shown in Figure 2.

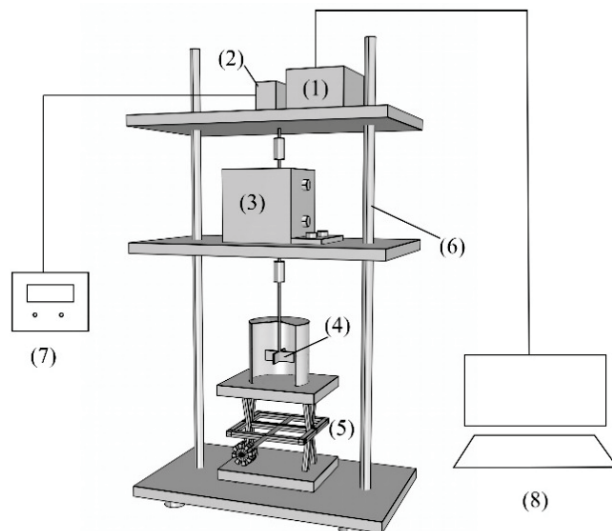


Figure 1. Structural diagram of the new type of gangue slurry rheometer. (1) Pulse generator; (2) torque servo motor; (3) torque sensor; (4) mixing blade; (5) rotating lifting test bench; (6) platform bracket; (7) stable voltage and constant current power supply; and (8) electromechanical control computer.

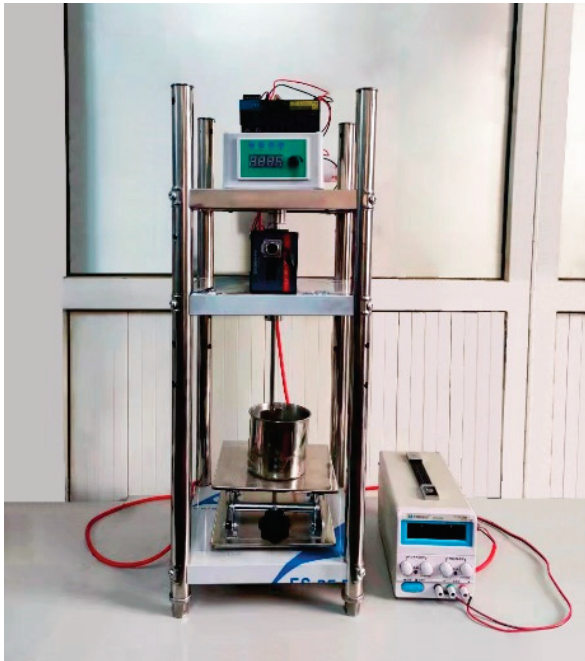


Figure 2. New type of gangue slurry rheometer.

2.2. Working Principle of a New Type of Gangue Slurry Rheometer

The new type of gangue slurry rheometer adopts a cross-blade type of mixing element. When the rheometer works, the mixing drum is fixed, and the blades are vertically placed into the slurry along the center of the mixing drum. The rotating blades drive the slurry in the drum to make a circular motion, and the slurry is layered due to the combined action of shear stress and friction force [33–35]. During this process, the slurry obstructs the stirring blades and generates torque. The received electrical signal is converted into the corresponding torque value by the torque sensor on the stirring rod, and different torque values can be obtained at different speeds. A functional relationship is established between the obtained different torque values and rotational speeds, and finally, the yield stress and viscosity of the tested slurry are calculated. The mixing system and its related dimensions are shown in Figure 3.

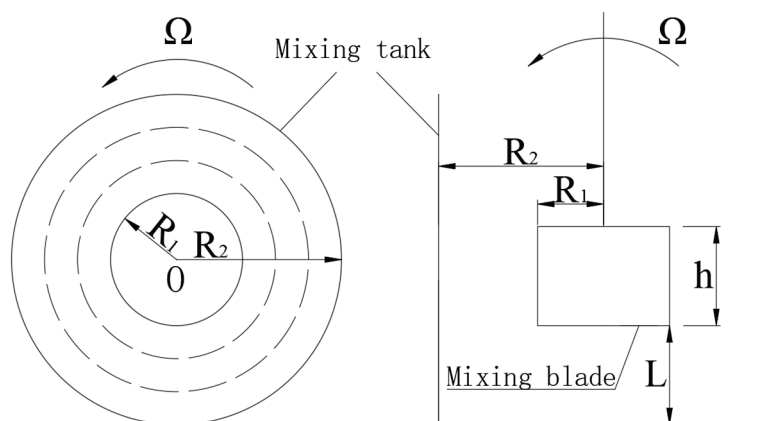


Figure 3. Working principle diagram of new type of gangue slurry rheometer (O is mixing center; R_1 is mixing blade radius; R_2 is mixing drum radius, m; Ω is rotation angular velocity of stirring blades, rad/s; h is height of stirring blades, m; and L is distance between mixing blade and bottom of cylinder, m).

When the rheometer is working, the shear deformation rate of the slurry at any point between the stirring blade and the stirring drum wall can be expressed as

$$\dot{\gamma} = \frac{d\gamma}{dt} = r \frac{d\omega}{dr} \quad (1)$$

In Formula (1), $\dot{\gamma}$ is shear rate; r is the distance between any point of the slurry between the blade and the mixing drum wall and the mixing center, m; $d\omega$ is the rotational angular velocity of the slurry at any point between the blade and the mixing drum wall; and dr is the width of the slurry at any point between the blade and the mixing drum wall, m.

Numerous experiments and studies have shown that as a high concentration fluid, the rheological properties of gangue filling slurry can be expressed using the Bingham model [12]. Therefore, the relationship between shear stress and shear rate can be expressed as:

$$\tau = \tau_0 + \eta \dot{\gamma} \quad (2)$$

In this formula, τ is shear stress, Pa; τ_0 is the yield stress; and η is viscosity. Substituting Equation (1) into Equation (2) yields the following:

$$\tau = \tau_0 + \eta r \frac{d\omega}{dr} \quad (3)$$

At this point, the micro torque generated by the slurry at any point between the mixing blade and the inner wall of the mixing drum can be expressed as:

$$M_1 = 2\pi r h \tau r = 2\pi r^2 h \left(\tau_0 + \eta r \frac{d\omega}{dr} \right) \quad (4)$$

Integral to the above equation:

$$\int_0^\Omega d\omega = \int_{R_1}^{R_2} \left(\frac{M_1}{2\pi h \eta r^3} - \frac{\tau_0}{\eta r} \right) dr \quad (5)$$

$$\Omega = \frac{M_1 (R_2^2 - R_1^2)}{4\pi h \eta R_1^2 R_2^2} - \frac{\tau_0}{\eta} \ln \frac{R_2}{R_1} \quad (6)$$

From Equation (6), it can be concluded that:

$$M_1 = \frac{4\pi h R_1^2 R_2^2}{R_2^2 - R_1^2} \eta \Omega + \frac{4\pi h R_1^2 R_2^2}{R_2^2 - R_1^2} \ln \frac{R_2}{R_1} \tau_0 \quad (7)$$

Two parameters are defined:

$$\begin{cases} k_1 = \frac{4\pi h R_1^2 R_2^2}{R_2^2 - R_1^2} \\ k_2 = \frac{4\pi h R_1^2 R_2^2}{R_2^2 - R_1^2} \ln \frac{R_2}{R_1} \end{cases} \quad (8)$$

According to Equation (8):

$$M_1 = k_1 \eta \Omega + k_2 \tau_0 \quad (9)$$

From the above equation, it can be seen that the two parameters k_1 and k_2 are only related to the size of the rheometer stirring blades and stirring drum. For a given testing instrument, these two parameters are constant values.

When measuring the rheological parameters of fine-grained mixture slurry using a coaxial cylindrical viscometer, the concave and convex settings at the upper and lower ends of the measuring inner cylinder allow a small amount of the measured slurry to overflow, while the convex setting at the lower end seals some air in the space when the measured slurry is placed in the measuring inner cylinder, forming an air cushion. At this time, both the upper and lower ends have no shear effect on the measured slurry, thereby reducing and eliminating the hindering effect of the slurry at the upper and lower ends on the measuring inner cylinder, so that the measured torque value is entirely generated by the slurry in the side slit [36].

However, for measuring the new type of gangue slurry rheometer containing the large-particle-sized gangue filling slurry, the stirring element is a rotatable blade. When it is vertically placed into the measured slurry, the influence of the end of the slurry on the torque value obtained by the blade cannot be eliminated. To achieve this, the first step is to design the blade to be immersed in the measured slurry, with its upper edge tangent to the slurry level, in order to minimize and eliminate the effect of the upper end of the stirring blade. At this point, we only need to consider and calculate the torque generated by the slurry between the lower edge of the blade and the bottom of the slurry storage cylinder on the stirring blade.

The flow behavior of the slurry at the lower end is also caused by the rotation of the mixing blade. The distance between any point on the lower edge of the mixing blade and the mixing center is r , and there is also a velocity gradient due to the frictional force. The movement speed of the slurry decreases layer-by-layer from top-to-bottom until the bottom of the mixing drum reaches 0. Therefore, it can be regarded as a flow model between flat plates, as shown in Figure 4.

Assuming that the distance between the lower part of the mixing blade and the mixing center is r and the width is d_r , a section of slurry moves uniformly in a circular motion around the mixing center at a certain speed, and the bottom of the mixing drum is considered as a stationary plate. The shear rate of the end slurry is expressed as:

$$\dot{\gamma} = \frac{\Omega r}{L} \quad (10)$$

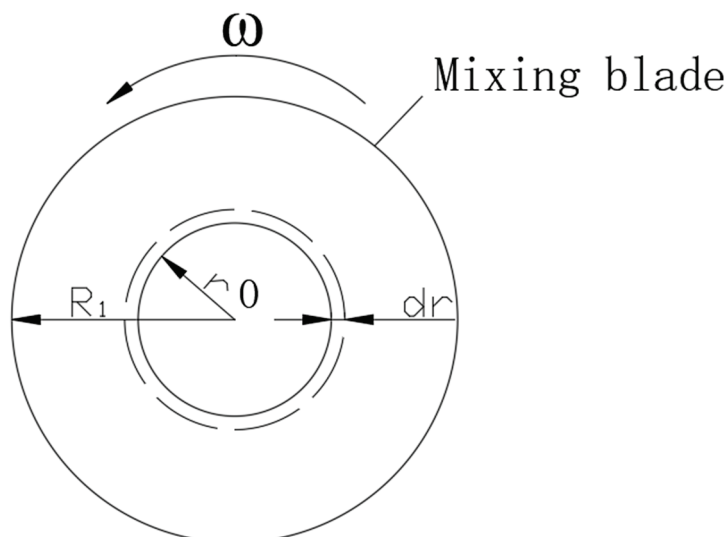


Figure 4. Schematic diagram of the end slurry movement.

The corresponding shear stress is:

$$\tau = \tau_0 + \eta \frac{\Omega r}{L} \quad (11)$$

The micro torque generated by the slurry at this point is:

$$dM_2 = \left(\tau_0 + \eta \frac{\Omega r}{L} \right) 2\pi r^2 dr \quad (12)$$

The integral to Equation (12) is:

$$\int_0^M dM_2 = \int_0^{R_1} \left(\tau_0 + \eta \frac{\Omega r}{L} \right) 2\pi r^2 dr \quad (13)$$

The additional torque of the end slurry is:

$$M_2 = \frac{\pi R_1^4}{2L} \eta \Omega + \frac{2\pi R_1^3}{3} \tau_0 \quad (14)$$

Two parameters are defined equally:

$$\begin{cases} k_3 = \frac{\pi R_1^4}{2L} \\ k_4 = \frac{2\pi R_1^3}{3} \end{cases} \quad (15)$$

$$M_2 = k_3 \eta \Omega + k_4 \tau_0 \quad (16)$$

As can be seen from the above equation, K_1 and k_2 are still two parameters that are only related to the size of the rheometer stirring blades and stirring drum, and are also constant values for a given testing instrument.

Therefore, the shear torque obtained when testing the rheological parameters of the new gangue slurry rheometer should be:

$$M = M_1 + M_2 = (k_1 + k_3) \eta \Omega + (k_2 + k_4) \tau_0 \quad (17)$$

By adjusting the rotational speed of the stirring blades to obtain their corresponding different torque values, multiple sets of recorded data are analyzed through regression to obtain a linear function. The ratio of the intercept of the curve on the shear torque axis to $(k_2 + k_4)$ is the yield stress of the tested slurry, and the ratio of the curve slope to $(k_1 + k_3)$ is the viscosity of the tested slurry:

$$\begin{cases} \tau_0 = \frac{M_0}{k_2 + k_4} \\ \eta = \frac{k}{k_1 + k_3} \end{cases} \quad (18)$$

In the formula, M_0 is the intercept of the curve on the torque axis, and $N \cdot m$; k is the slope of the curve.

According to the size of the mixing system, it can be calculated that:

$$\begin{cases} k_1 + k_3 = 0.000189477 \\ k_2 + k_4 = 0.000181156 \end{cases}$$

3. New Type of Gangue Slurry Rheometer for Testing Rheological Parameters of Slurry

3.1. Preparation of Gangue Filling Slurry

This article selected the actual filling gangue used in the Linxi mine of Kailuan Mining, with a maximum particle size of 15 mm, and the nearby power plant produced fly ash, local PO42.5 ordinary Portland cement, and urban tap water. The specific content is shown in Table 1.

Table 1. Test ratio of gangue slurry.

Content	Cement	Fly Ash	Gangue	Water
76%	76 g	228 g	456 g	240 g

First, all the dry ingredients were poured into a large beaker according to the ratio, mixed well with a spatula, then stirred with water, and finally poured into the test mixing cylinder for testing.

3.2. Test Data Analysis of Gangue Slurry at Different Rotational Speeds

The torques of the gangue slurry at different speeds obtained through testing are shown in Table 2.

Table 2. Test results of slurry with a concentration of 76%.

n (r/min)	200	250	300	350	400
M (N·m)	0.0255	0.0262	0.0289	0.0299	0.033
Ω (rad/s)	20.935	26.173	31.409	36.628	41.888

Figure 5 shows the expression of the curve obtained by fitting the torque values obtained at different rotational angular speeds of the blade, thus obtaining the slope of the curve and its intercept on the torque axis. The yield stress and viscosity values of the tested slurry can be obtained from Formula (18), as follows:

$$\begin{cases} \tau_0 = \frac{0.01748}{0.000181156} = 96.49 (Pa) \\ \eta = \frac{0.000357188}{0.000189477} = 1.89 (Pa \cdot s) \end{cases}$$

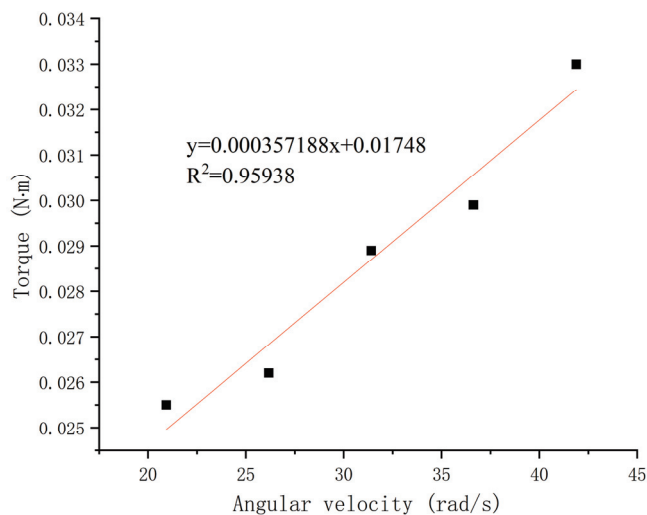


Figure 5. Fitting curve of different angular velocities and blade edge torques.

4. Numerical Simulation of the Testing Process of a New Type of Gangue Slurry Rheometer

Using Fluent numerical simulation software ANSYS2023, a three-dimensional model of the mixing system for the gangue slurry is established to simulate the velocity and pressure distribution of the cross-shaped mixing blades at different gangue slurry speeds, and the corresponding measured torques are obtained.

4.1. Model Establishment and Grid Division

The mixing element of the new type of gangue slurry rheometer consists of two rectangular thin sheets of identical size that pass through the axis and are perpendicular to each other, as shown in Figure 6. The blade size is 40 mm long, 30 mm high, and 2 mm thick, and the transmission connecting rod is 60 mm long. The mixing drum is a standard cylinder with a diameter and height of 100 mm. The three-dimensional model of the mixing blades and drum is shown in Figure 7.

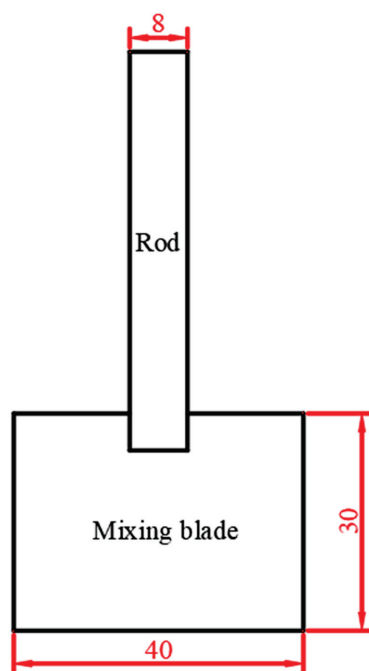


Figure 6. The size of the propeller.

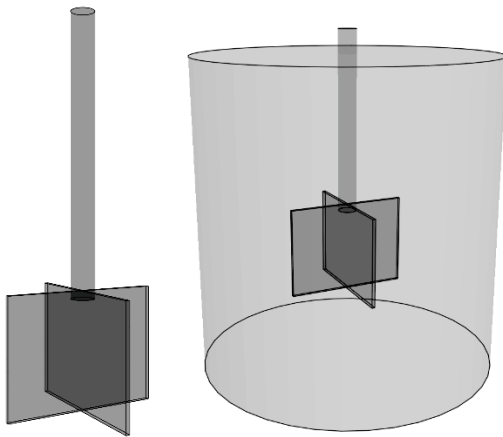


Figure 7. Three-dimensional model of blade and mixing drum.

A three-dimensional model of the mixing blade and mixing drum are established, and sliding mesh technology is used to divide the model calculation domain into two nested concentric cylinders. The data continuously transitioned and interacted through the Match Interface between the two cylinders.

In order to describe the flow behavior of the slurry at the blade rotation in detail, using a tetrahedral mesh partitioning model, as shown in Figure 8, the mesh refinement at the blade is set to 0.02 mm, and the rest is set to 0.08 mm. Therefore, a total of 275,584 tetrahedral mesh elements are partitioned.

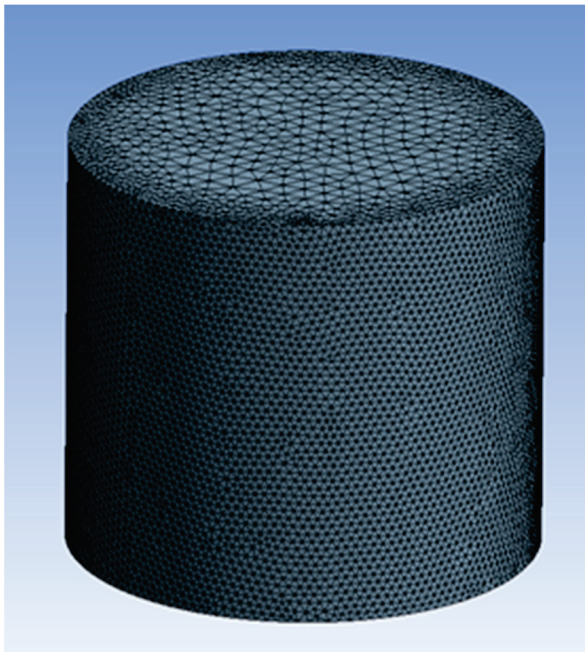


Figure 8. Mesh of the model.

4.2. Parameter Setting of Slurry Model

The numerical simulation of the flow behavior of gangue slurry requires the physical parameters of the measured slurry, including density, viscosity, and yield stress, as shown in Table 3.

The constitutive equation of the Herschel–Bulkley model is used to define the gangue slurry. During initialization, the upper edge of the blade is set to coincide with the slurry level, and the lower edge of the blade is 2.5 cm away from the bottom of the mixing drum. When the blade speeds are 200 r/min, 250 r/min, 300 r/min, 350 r/min, and 400 r/min, the

flow behavior of the slurry at a cross-section 1.5 cm below the slurry level, where the lower edge of the mixing blade is located, is observed, and the torque obtained by the mixing blade is detected.

Table 3. Physical parameters of slurry.

Physical Parameters		
Density	1800	kg/m ³
Viscosity	1.89	Pa · s
Yield stress	96.49	Pa

4.3. Observation of Velocity Distribution of Cross-Sectional Slurry at Different Rotational Speeds

Figure 9 shows the velocity distribution cloud map of the slurry observed at different speeds. It can be seen from the figure that the flow velocity of the slurry at the edge of the stirring blade is the fastest on the observed section, gradually decreases as it moves away from the blade edge, and reaches zero at a certain distance from the blade edge. This is because the slurry is subjected to the thrust generated by the rotation of the blades, causing the slurry near the edge of the blades to have the same rotational speed as the stirring blades. However, due to the frictional resistance inside the slurry, the speed gradually decreases to zero as it moves away from the blades, showing a gradient change pattern. As the rotational speed of the blade increases, the velocity of the slurry near the edge of the blade also increases. At the same time, the range of the maximum velocity gradually increases and extends away from the edge of the blade. Therefore, the area of minimum velocity gradually decreases.

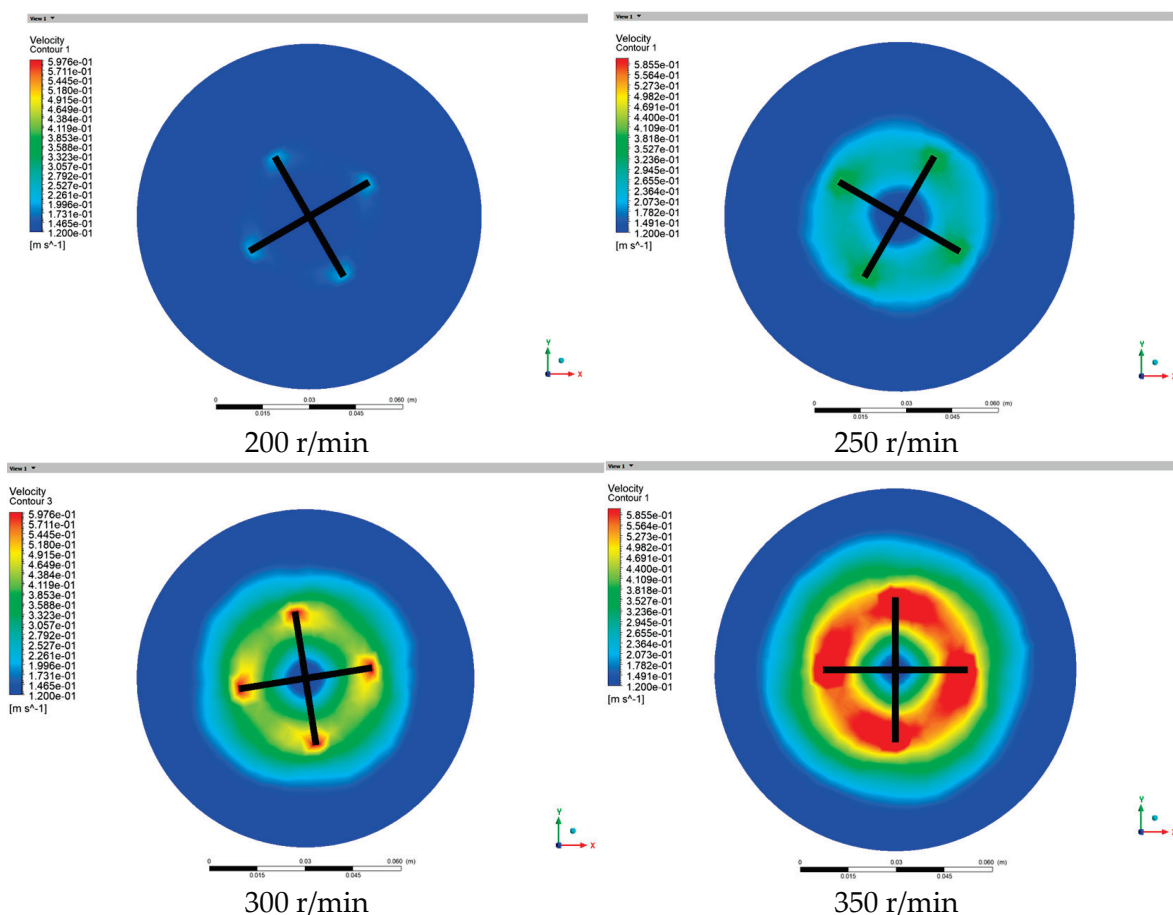


Figure 9. Cont.

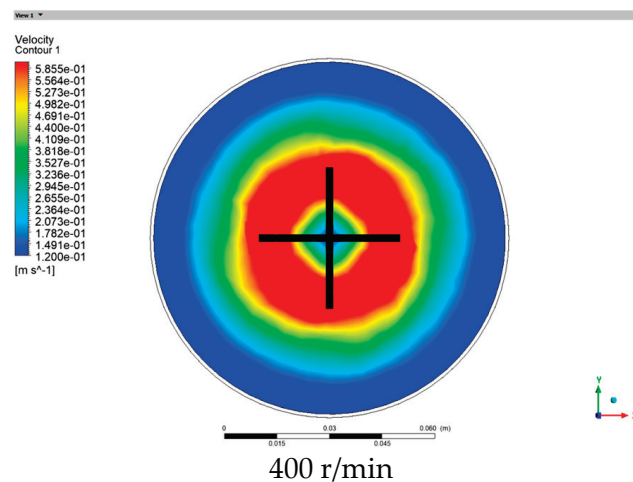


Figure 9. Cloud map of velocity distribution on the observation surface at different rotational speeds.

4.4. Pressure Distribution of Observed Cross-Sections at Different Rotational Speeds

Figure 10 shows the pressure distribution cloud map of the observed cross-section of slurry at different speeds, which reflects the force situation of the slurry during the mixing process on the observed cross-section. From the figure, it can be seen that the stirring blades rotate counterclockwise, and the slurry located in front of the blade movement on the observation section is subjected to compressive stress, as shown in the red area in the figure. The slurry located behind the blade movement is subjected to tensile stress, as shown in the blue area in the figure. The maximum stress occurs a little further in from the edge of the mixing blades. As the blade speed increases, the range and magnitude of the compressive stress on the front of the blade movement gradually increase, and stress minima occur at the blade crossings immediately adjacent to the blades. At the same time, the range and magnitude of tensile stress on the back of the blade movement also gradually increase and eventually become integrated, closely adhering to the back of the blade movement direction. The difference between the two stresses gradually increases with the rotational speed and eventually forms a clear boundary.

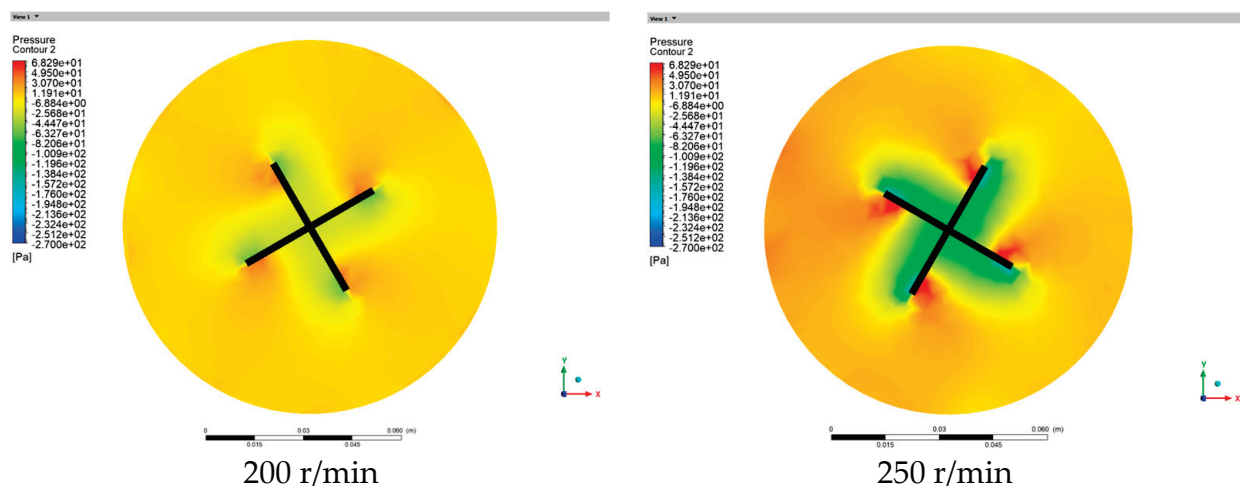


Figure 10. Cont.

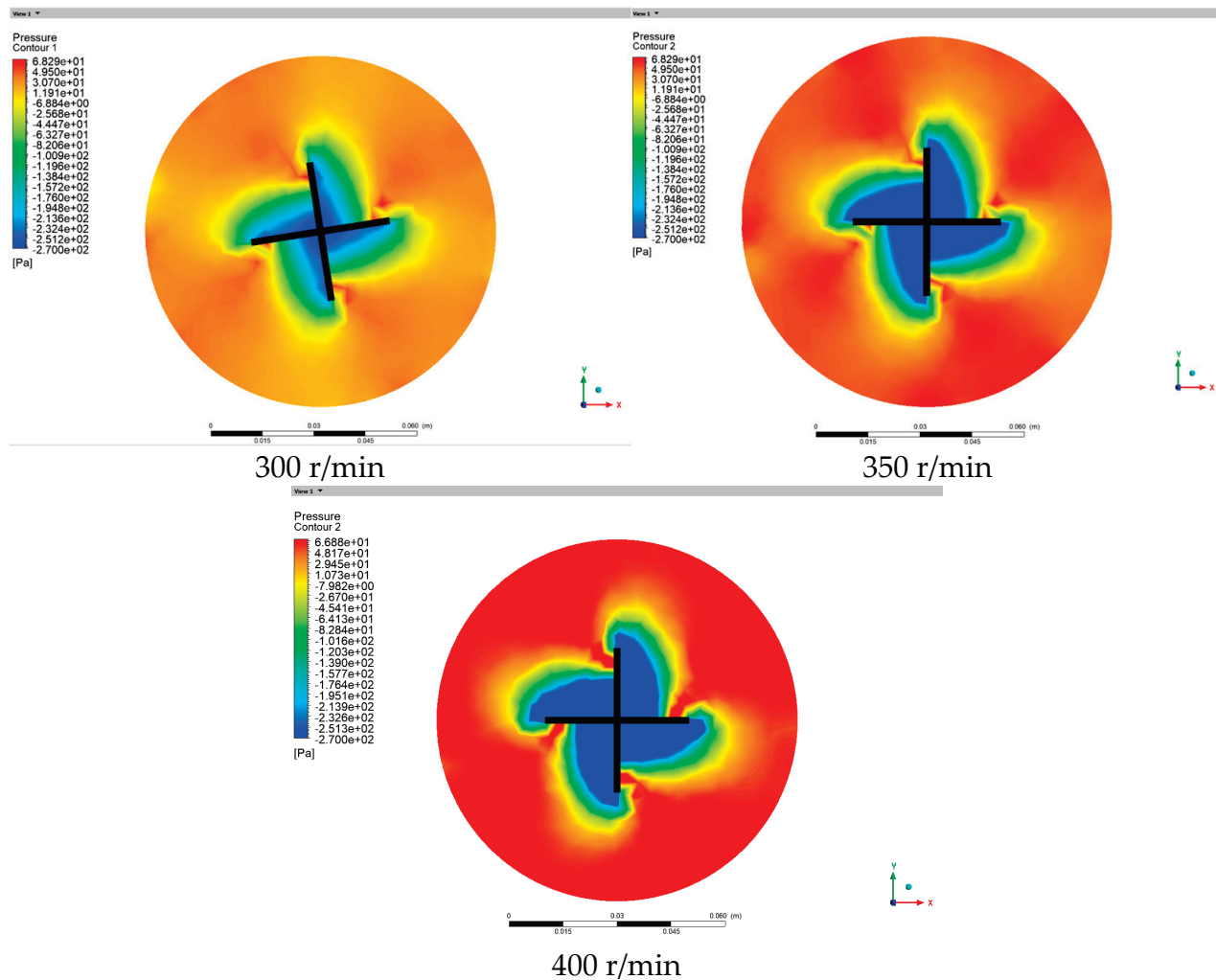


Figure 10. Cloud map of pressure distribution on the observation surface at different rotational speeds.

4.5. Torque of Stirring Blades at Different Speeds

During the testing process of the new type of gangue slurry rheometer using numerical simulation, the torque obtained by the stirring blades over time at different speeds can be obtained through a monitor, as shown in Figure 11. The torque on the stirring blades gradually increases within a short time range and eventually stabilizes before no further changes occur.

The torque values on the stirring blades at different speeds are shown in Table 4.

Table 4. Simulated torque values on the blade at different speeds.

No.	Blade Speed (rpm/min)	Blade Angular Velocity (rad/s)	Blade Torque (N·m)
1	200	20.9333	0.02531
2	250	26.1666	0.02721
3	300	31.4	0.02912
4	350	36.6333	0.03102
5	400	41.8667	0.03292

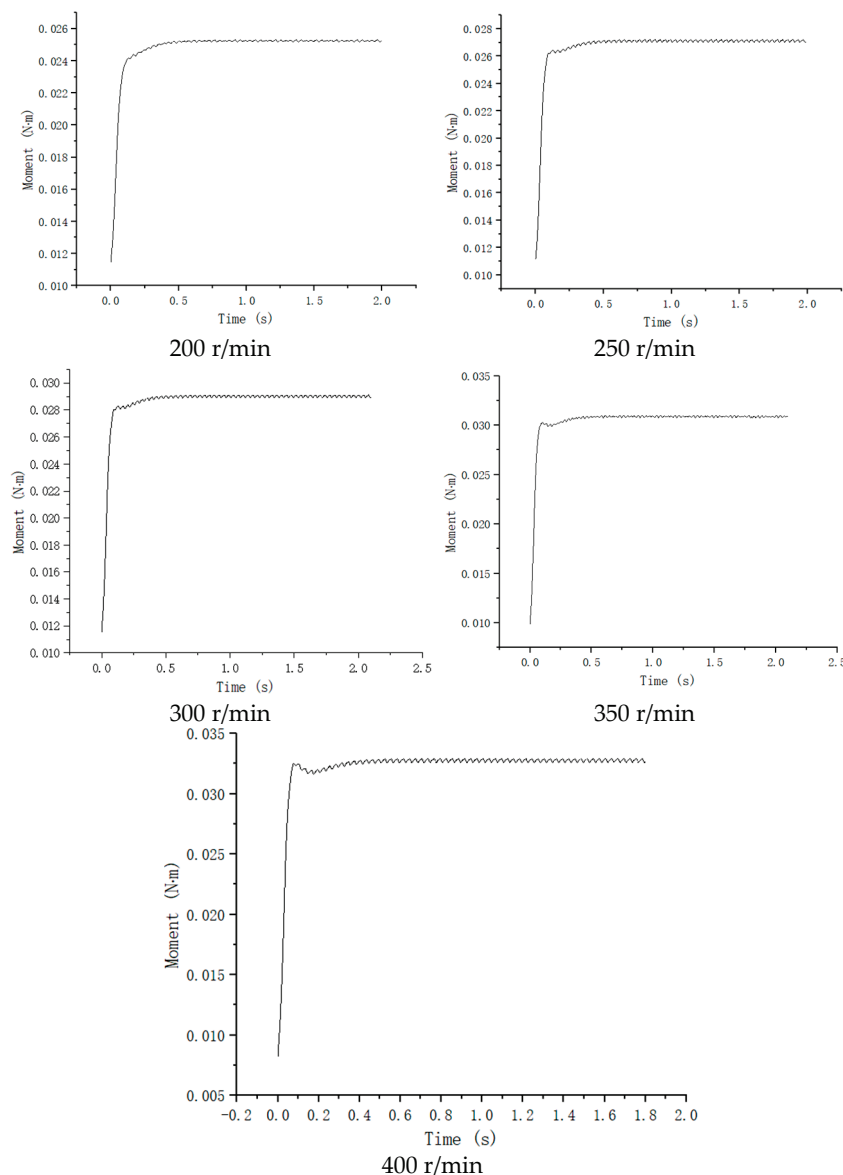


Figure 11. Torque variation curves on the blade at different speeds.

After conducting a regression analysis on the blade angular velocity and torque in the table above, the linear functions of the two were obtained, as shown in Figure 12.

The yield stress and viscosity of the simulated slurry can be obtained from Figure 12 and Formula (18), as follows:

$$\begin{cases} \tau_0 = \frac{0.01783}{0.000181156} = 98.42(Pa) \\ \eta = \frac{0.000364012}{0.000189477} = 1.92(Pa \cdot s) \end{cases}$$

From Table 5, it can be seen that the yield stress and viscosity of the slurry obtained from numerical simulation are 98.42 Pa and 1.92 Pa·s, respectively, while the measured yield stress and viscosity of the slurry under the same conditions are 96.49 Pa and 1.89 Pa·s, respectively; the differences are 1.93 and 0.03, respectively, with errors of 2% and 1.59%, respectively. It can be seen that the errors are within a reasonable range, indicating the effectiveness of the testing principle of the new gangue slurry rheometer.

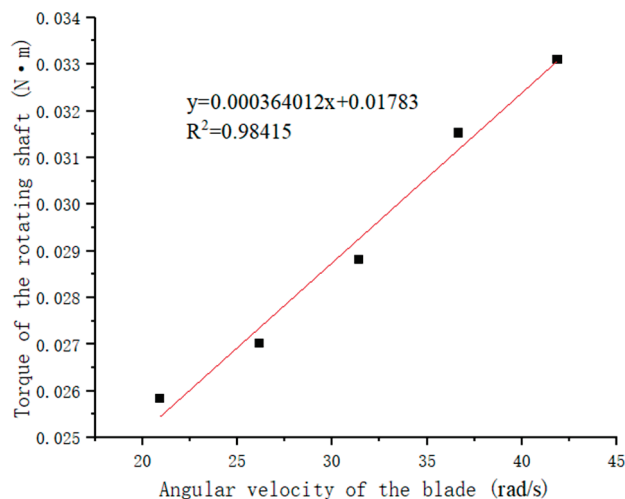


Figure 12. Fitting curve of different angular velocities and rotational axis torques.

Table 5. Comparison of rheological parameters between setup (test) and simulated values.

Physical Parameters	Set (Test) Value	Simulated Value	Relative Error
Viscosity (Pa·s)	1.89	1.92	1.59%
Yield stress (Pa)	96.49	98.42	2%

5. Conclusions

This paper analyzes the issue of being unable to obtain accurate rheological parameter test values in current coal mine gangue filling slurry due to the presence of large-sized particles. The specific research findings are as follows:

- (1) Due to the current inability to accurately measure the rheological parameters of slurry containing large-sized gangue particles, a new type of gangue slurry rheometer has been developed. The working principle of the newly developed gangue slurry rheometer is to obtain the torque of the stirring blades at different speeds and calculate the rheological parameters of the slurry. The torque is generated by the combined action of the slurry between the blade side edge and the cylinder wall, and the slurry between the blade lower edge and the cylinder bottom.
- (2) Testing is conducted on a slurry containing a 76% concentration of large-sized gangue material, and the torque on the stirring blades is obtained at a speed of 200–400 r/min. As the speed increases, the torque values obtained on the blades gradually increase. The yield stress and viscosity are calculated to be 96.49 and 1.89, respectively.
- (3) Fluent is used to simulate the flow behavior of gangue slurry under the same conditions, and it is found that the maximum velocity of the slurry appears at the edge of the mixing blade and decreases towards the direction away from the blade edge, forming a velocity gradient. The front of the blade's direction displays compressive stress, while the rear displays tensile stress. As the rotational speed increases, the difference between the two stresses gradually increases and a clear boundary appears. Based on the torque values obtained on the blades at different speeds, through regression analysis and calculation, the yield stress and viscosity of the slurry are found to be 98.42 and 1.92, respectively, with errors of 2% and 1.59%, respectively. This demonstrates the effectiveness of the new gangue slurry rheometer test.
- (4) This study provides a basis for the accurate measurement of the rheological parameters of large particle sizes, ensuring the smooth transportation of high-concentration

gangue slurry in pipelines and providing a reference for the design of filling processes and filling systems.

Author Contributions: Conceptualization, X.D. and Y.H. (Yucheng Huang); methodology, Y.H. (Yuxin Hao); software, L.Z.; validation, X.D. and Y.H. (Yucheng Huang); formal analysis, L.Z.; investigation, X.D.; resources, X.D.; data curation, X.D.; writing—original draft preparation, X.D.; writing—review and editing, X.D. and Y.H. (Yuxin Hao); visualization, X.D.; supervision, Y.H. (Yucheng Huang); project administration, Y.H. (Yucheng Huang); funding acquisition, X.D. All authors have read and agreed to the published version of the manuscript.

Funding: This research received no external funding.

Data Availability Statement: The data are contained within the article.

Conflicts of Interest: The authors declare no conflicts of interest.

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Article

Automatic Gas Emission Width of Coal Bodies in the Goaf near Outbursting Coal Seams

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Abstract: The influence of coal and gas outbursts from a coal seam adjacent to the working face is crucial for determining its automatic gas discharge width, which is an important basis for the roadway position design of the adjacent working face. This study focuses on determining the automatic gas discharge width of the coal body in the neighboring goaf, especially examining the working face of the E₁₀-32040 air mining area and the E₁₀-32060 wind tunnel of the No. 1 Mine operated by Pingmei Company. Theoretical analysis, strain-softening simulation, and field testing were adopted to study the automatic gas discharge width under the current mining conditions, and the results are as follows: (1) Back mining at the working face has a greater impact on the coal body of the neighboring goaf than roadway excavation, and the compression deformation at 50 m from the goaf after back mining is 6.18 times that during roadway excavation. (2) The gas content of the coal body of the neighboring goaf is linearly distributed, and the coefficient of determination (R^2) is 0.98024. (3) The extent of compression and deformation of the neighboring coal body follows an exponential distribution, and the coefficient of determination (R^2) is 0.99482. (4) Under the current mining conditions, the risk of protrusion can be considered eliminated when the residual gas content is below 4.45 m³/t. The compression deformation is 0.96‰ when the automatic gas discharge width is 30.11 m. The research results can provide theoretical reference and data support for adjacent roadway location design and the selection of gas prevention and control measures in coal seams.

Keywords: deep coal seam; strain softening; goaf; automatic gas emission; residual gas content

1. Introduction

Coal remains the main energy source in China, contributing immensely to its social and industrial development and serving as the cornerstone of energy security [1,2]. Coal formation is accompanied by gas formation, and the amount of gas retained in coal seams increases with depth [3–5]. As the gas content reaches a certain level, the risk of coal and gas outbursts arises [6–9]. Mining deep coal seams often contributes to the risk of coal and gas outbursts [10,11]. Stress field changes due to mining activities can alter the stress state of the coal body, causing coal body deformation and changing coal seam permeability [12–14]. The overall coal seam permeability during stress loading exhibits a V-shaped variation curve, reaching the minimum under peak stress [15]. Following the stress peak, gas permeability increases with the coal body damage, with a certain hysteresis characteristic, and the two basically show a pattern of exponential growth [16].

The increases in coal permeability under different stress paths are consistent with coal damage and failure, and the permeability increment is more pronounced if the stress borne by the coal body is smaller after the stress peak [17]. When mining an underground coal seam, the surrounding coal body is subjected to stress concentration and perimeter rock damage due to mining activity, thus improving gas seepage and creating gas flow channels for automatic gas emission. According to the difference between the mining objects, the automatic emission width can be divided into that of the coal tunnel and that of the working face goaf [18,19]. The accurate determination of the automatic gas discharge width can provide an effective basis for the location design of adjacent roadways and the selection of the sealing depth of gas extraction drill holes [20,21].

Experts and scholars have extensively researched the factors affecting the automatic gas emission width and the effect of automatic gas emission. The bearing stress of the coal body, gas pressure, and the physical state of coal are the main factors affecting the effectiveness of automatic coal seam gas emission [22–24]. The gas pressure in the coal wall of the neighboring goaf is basically zero. As mining proceeds toward the solid coal region, the gas pressure increases, and the desorbed gas is continuously transported to the mining space due to the gas pressure [3]. Tian et al. [25] proposed that the continuous flow rate of the drilled gas can be used to determine the width of the pressure relief zone in front of the coal mine working face, and the drilled gas flow rate increases first and then decreases, with the peak flow rate point marking the boundary of the pressure relief zone. Fu [26] determined the width of the unloading zone of the roadway sidewall based on the borehole leakage method. Wang et al. [23] predicted the automatic gas emission width of coal seam roadways based on the slip effect and found that the emission width increased with an increase in exposure time. Li et al. [27] simulated the ultimate emission width of a roadway under different exposure times based on COMOSOL Multiphysics and concluded that the ultimate emission width continuously increased with exposure time. Feng [28] determined the vertical stress distribution of the area surrounding the roadway based on the foundation beam theory and obtained the widths of the stress relief zone and the gas emission zone using theoretical and numerical calculations. Xi et al. [29] studied the overburden rock damage pattern and found that the width of the overpass tipping zone in front of the working face subjected to back mining was 28 to 30 m, and a distance range of 20 to 40 m from the working face during back mining is advantageous for side mining and pumping. Chen et al. [30] simulated the variation patterns of displacement and stress over time for the coal body adjacent to the 1311 working face using FLAC3D and obtained the on-site gas parameter measurement results, concluding that the area within 30 m was the effective decompression zone, and the range of 30 to 33 m was the transition zone. Zhou et al. [18] performed multi-occasion coupled numerical calculations and on-site measurements under longwall mining conditions, concluding that the gas pressure and content in the 10–23 m range decreased by 26.55% to 73.58% and 34% to 71%, respectively. Wang et al. [31] found that the coal body stress on the side of the mining zone was positively correlated with the formation time in the mining zone and that the permeability increase range was larger with a broader limit equilibrium zone.

In summary, research on the stress distribution pattern of the coal tunnel surrounding rock and the automatic gas emission width has yielded fruitful results; however, relatively few research studies have focused on the automatic gas emission width of the working face in back mining. Moreover, the automatic gas emission width is affected by the depth, coal seam thickness, overburden conditions, and coal seam permeability coefficient, with significant differences in the automatic gas emission width. Meanwhile, many researchers confuse the width of the stress relief zone with the width of the gas relief zone and simply take the peak stress point as the gas relief width, leading to inaccuracies. Even after the peak

point, the gas in the stress concentration area is in a dynamic equilibrium, and the free gas gradually migrates toward the extraction space, causing the automatic gas emission width to be generally larger than the width of the stress relief zone [32]. Thus, we studied the E₁₀-32040 and E₁₀-32060 working faces in a mine operated by Pingmei Company through theoretical analysis, strain-softening simulation, and on-site gas parameter testing. This research aimed to study the stress distribution pattern of the coal body of the neighboring goaf and the vertical compression deformation of the coal seam while testing the gas content of the coal body of the neighboring goaf. The automatic gas emission width was determined and analyzed. The findings provide data support and a theoretical basis for selecting adjacent roadways in coal seams.

2. Theory of Damage and Infiltration of Coal Bodies in the Neighboring Goaf

2.1. Stress Distribution

The redistribution of perimeter rock stress is a complex loading and unloading process. Shaft mining adjacent to a goaf's coal body involves the gravity load of the overlying rock layer, and unloading damage occurs when exceeding the limit load. Due to the large difference in area between the roadway and the working face, the scope and degree of the influence of mining stress in the roadway are relatively small. The working face's mining height and inclined length are the main factors influencing the width of the lateral stress relief zone in the goaf, which increases with the increase in these two factors, with the former having a more obvious effect [33,34]. The adjacent goaf's coal body is mainly affected by the mining stress caused by back mining at the working face, and the original stress state cannot be restored after the secondary stress equilibrium. Considering the relationship between the coal body's bearing vertical stress and the original stress, it can be divided into the unloading area (I), stress concentration area (II), and the original stress area (III) [18]. The stress distribution pattern of the neighboring goaf's coal body is presented in Figure 1.

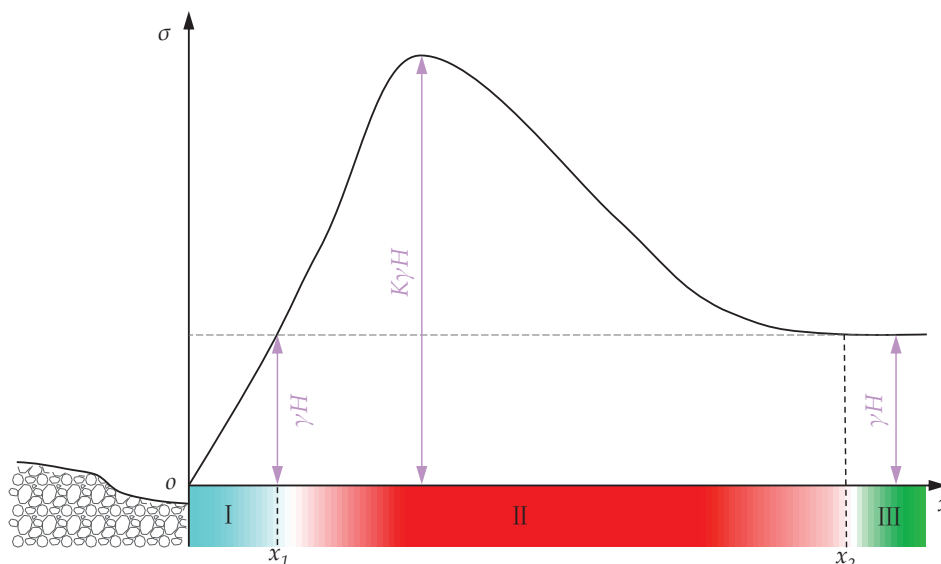


Figure 1. Stress distribution pattern of neighboring goaf's coal body.

2.2. Damage-Enhanced Seepage Pattern

During the mining stress loading, gas-containing coal seams are subjected to interactions between the deformation of the coal body skeleton and matrix and gas desorption and diffusion. The concentrated stress of the coal body increases under the mining stress, and

the neighboring goaf's coal body deforms first during the loading process. Part of the coal body starts to sustain damage until it reaches the yield limit and undergoes destruction. The coal body damage results from the accumulation and penetration of cracks initiated in the damaged part, and the final fissure formation throughout the coal body is the main cause of destabilization in the coal body [34,35]. As the loaded coal body reaches the yield strain, the cracks in the coal body gradually accumulate and expand with the increase in axial strain. At this time, the effects of the expansion materialize, and the permeability of the loaded coal body significantly increases, facilitating the flow of coal gas in the coal seam [36]. The coal body in the rupture area and plasticity area is subjected to peak stress, which has exceeded the yield strain, and therefore a large number of cracks appear in the coal body. The permeability of the coal seam increases, and the gas flow capacity is enhanced. Due to the gas pressure gradient from the inside of the coal body to the coal wall, the desorbed gas in the solid coal region flows to the air-mining area. The elastic-plastic state of the neighboring goaf's coal body and the permeability pattern are shown in Figure 2.

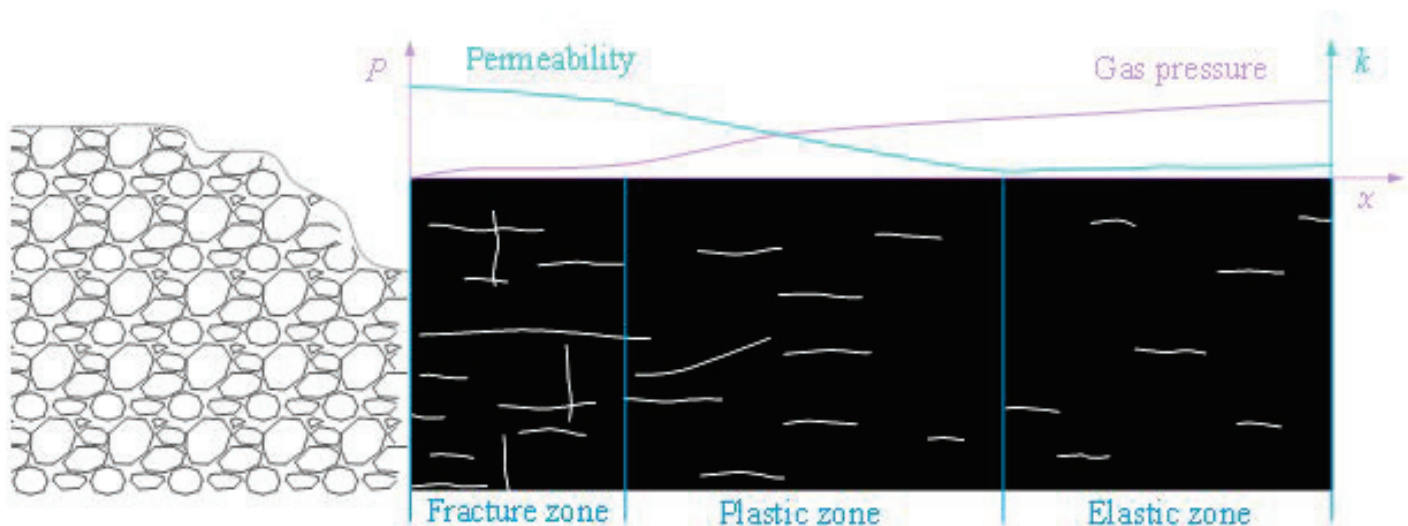


Figure 2. Elastic–plastic state and permeability of neighboring goaf's coal bodies.

3. Numerical Simulation

3.1. Geological Background Overview

Located 3 km north of the center of Pingdingshan City, Henan Province, Pingmei Company's No. 1 Mine began to be built in 1957 and started operation in 1959. After numerous upgrades, its current production capacity reaches 3.2 Mt/a, making it a prominent coal and gas mine. The total area of the mine is 29.3 km², and the mine operates in Groups C, D, E, and G. Currently, it mainly operates Groups D and E, encompassing coal seams D₅, D₆, E₈, and E₁₀. The E₁₀-32040 working face is in the west wing of the three-level E₂ mining area, starting from the downhill of the three-level E₂ belt in the east and ending at the border of the first and fourth mines in the west, adjacent to the E₁₀-32020 mining area in the south, and connected to the E₁₀-32060 working face in the north. The E₁₀-32040 working face is designed to have a strike length of 1120 m and a tendency length of 160 m, with ground elevation ranging from 160 to 290 m and underground elevation ranging from −474 to −539 m. The E₁₀-32060 working face is designed to have a strike length of 1200 m, an inclined length of 200 m, a ground elevation of 157 to 278 m, and a downhole elevation of −486 to −563 m.

The E₁₀-32040 working face is designed as a comprehensive mining face, roughly combined with coal seams E₉ and E₁₀, with the coal seam thickness ranging from 3.0 to

3.7 m and an average thickness of 3.5 m. The original gas content of the coal seams in the working face ranges from 5.98 to 6.69 m³/t, with an average original gas content of 6.31 m³/t. The inclination angle of the coal seam is 5° to 8°, and the average inclination angle is 6.5°. The top plate is mudstone, and the bottom plate is sandy mudstone. The comprehensive column shape of the strata of the working face is shown in Figure 3.

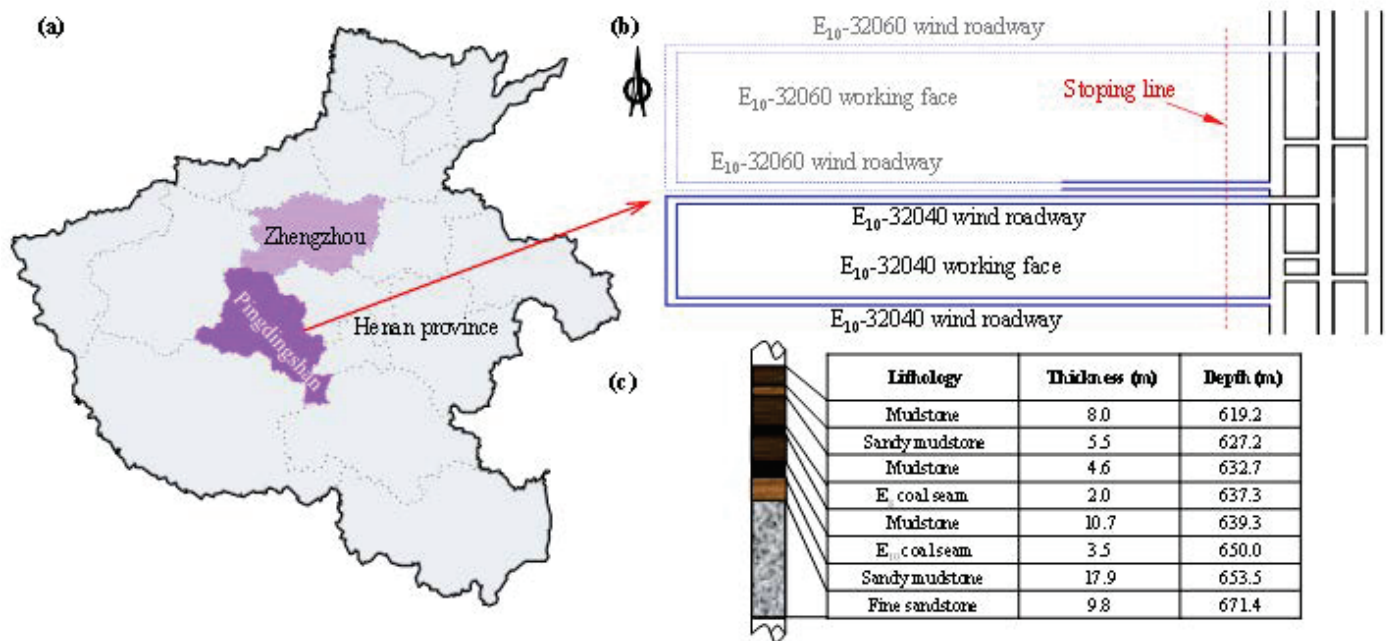


Figure 3. Schematic diagram of the working face location and coal seam column: (a) location of the test site; (b) plain view of the working face; (c) column view of the seam.

3.2. Modeling

FLAC3D is a three-dimensional numerical analysis software based on the finite difference method, featuring built-in models such as Moore–Coulomb and strain softening. It can be used to study mining stress distribution, top and bottom plates' displacement in the mining area, and the distribution of the plastic zone of the surrounding rock during underground mining [37,38]. Taking into account the basic geological conditions of the research site, a three-dimensional numerical simulation model was established based on the coal seam histogram. In order to improve the calculation efficiency and reduce the computational load, the coal and rock seams farther from the mining layer, which were less affected during mining and not the focus of this study, were combined, and the same parameters were used for the same rock seams. With the above conditions, the coordinates of each coal rock layer in the model were calculated in Excel and imported into FLAC3D to establish a 300 × 100 × 180 m 3D model (Figure 4). The numerical simulation parameters are shown in Tables 1 and 2. Coal is a non-homogeneous geotechnical material with pre-peak hardening and post-peak softening characteristics [39–41]. When simulating deep coal seam mining, the coal body around the mining space generally exceeds the strength limit, and the coal rock layer exhibits post-peak softening characteristics. The cohesion variations with strain were used in the numerical simulation model to simulate the effect of post-peak softening [42,43]. The strain-softening parameters are shown in Table 2.

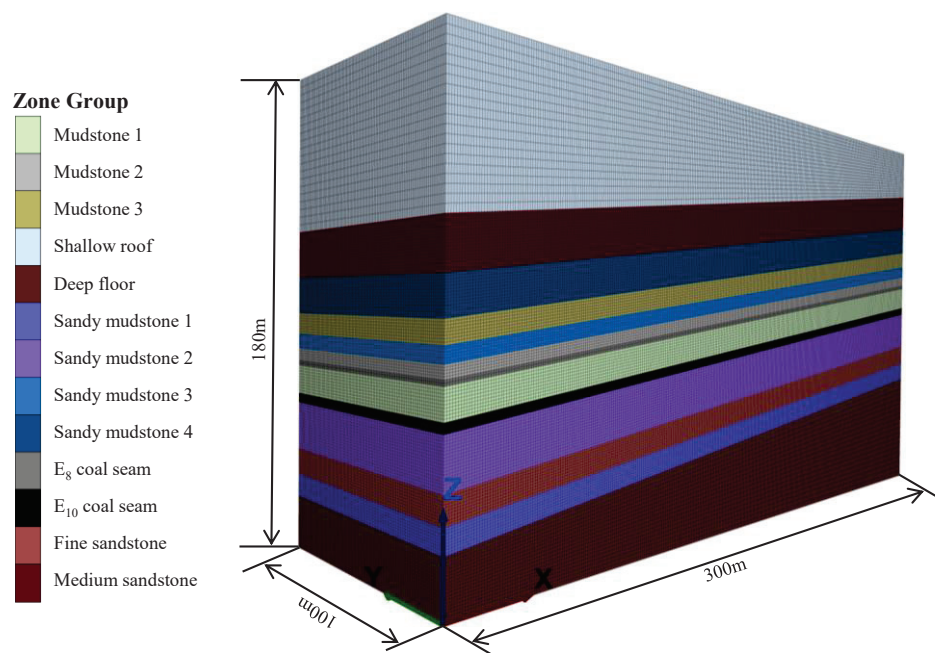


Figure 4. Model diagram.

Table 1. The modeled coal rock parameters.

Geology	Density /(kg/m ³)	Bulk Modulus/(Pa)	Shear Modulus/(Pa)	Tensile Strength/(Pa)	Cohesion /(Pa)	Friction Angle/(°)
Shallow roof	2600	9.88×10^9	6.98×10^9	5.74×10^6	5.63×10^6	45
Sandy mudstone	2390	6.22×10^9	5.14×10^9	2.12×10^6	2.64×10^6	36
Coal seam E ₈	1360	1.86×10^9	1.63×10^9	1.75×10^6	2.46×10^6	29
Fine sandstone	2450	8.66×10^9	6.35×10^9	3.68×10^6	3.51×10^6	40
Mudstone	2350	4.63×10^9	3.11×10^9	2.14×10^6	2.33×10^6	32
Coal seam E ₁₀	1420	1.49×10^9	1.52×10^9	1.44×10^6	2.37×10^6	29
Medium sandstone	2550	8.36×10^9	7.07×10^9	3.69×10^6	3.86×10^6	41
Deep floor	2600	9.88×10^9	6.98×10^9	5.74×10^6	5.63×10^6	45

Table 2. Coal strain-softening parameters.

Coal seam E ₁₀	Strain/(m/m)	0	0.025	0.033	0.049	0.055	1
	Cohesion/(MPa)	2.37	2.37	1.18	0.83	0.60	0.60

3.3. Simulation Process and Monitoring

FLAC3D was operated using a command flow code, and the simulation was performed according to the flow in Figure 5.

The numerical simulation model in Figure 5 is described in Section 3.2. As shown in the model in Figure 4, the model surface was divided into east, south, west, north, top, and bottom using the surface grouping command. The model of this structure was a strain-softened one, and the material parameters are shown in Tables 1 and 2. The east, south, west, north, and bottom surfaces of the model were constrained using a normal acceleration of zero. The top surface was subjected to the gravity of the overlying coal rock layer, which was calculated with Equation (1). The initial equilibrium results of the model are shown in Figure 6.

$$\sigma = \gamma H \quad (1)$$

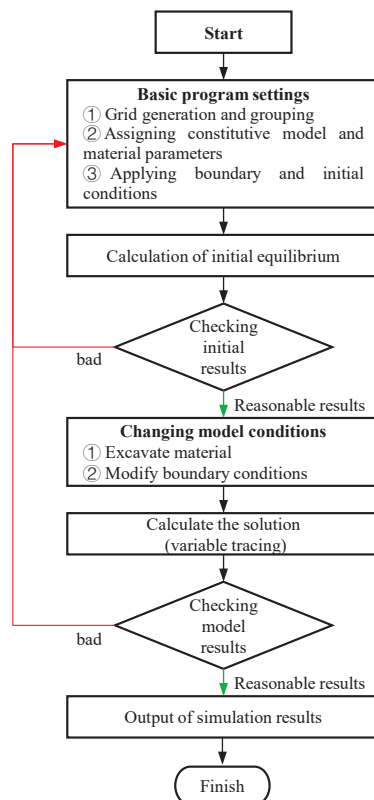


Figure 5. Flowchart of numerical simulation.

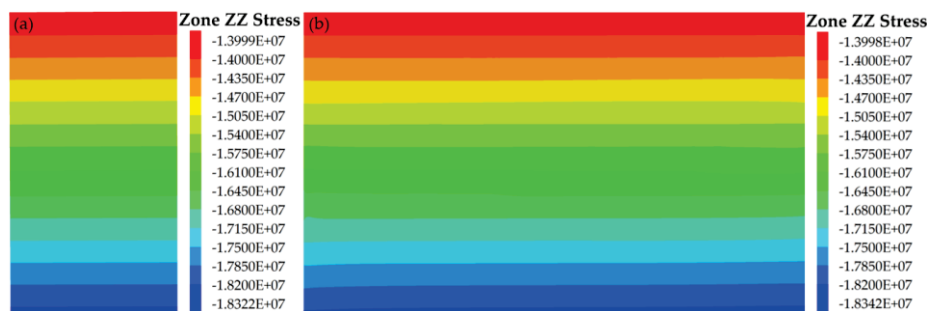


Figure 6. Initial equilibrium stress distribution: (a) strike profile; (b) inclination profile.

Here, σ denotes raw rock stresses in the overlying rock layers (MPa), γ represents the average volumetric weight of the overlying coal seam (22.5 kN/m^3), and H represents the average thickness of the overlying coal seam (650 m).

As shown in Figure 6, the modeled stress shows a pattern of gradual increase from top to bottom both in terms of the strike and inclination, and the stress at the same depth is basically the same. Since the gradual stress increase with the increase in depth aligns with the theory, the simulation results were used for subsequent calculations.

The excavation was conducted according to the actual mining sequence: (1) excavating the E₁₀-32040 machine lane and wind tunnel; (2) excavating the E₁₀-32040 working face. During excavation, 15 monitoring points were set up in the vertical section ($y = 50$) in the E₁₀ coal seam to monitor the coal seam's top-surface displacement, the coal body stress, and the coal seam's bottom-surface displacement. The monitoring point arrangement is shown in Figure 7.

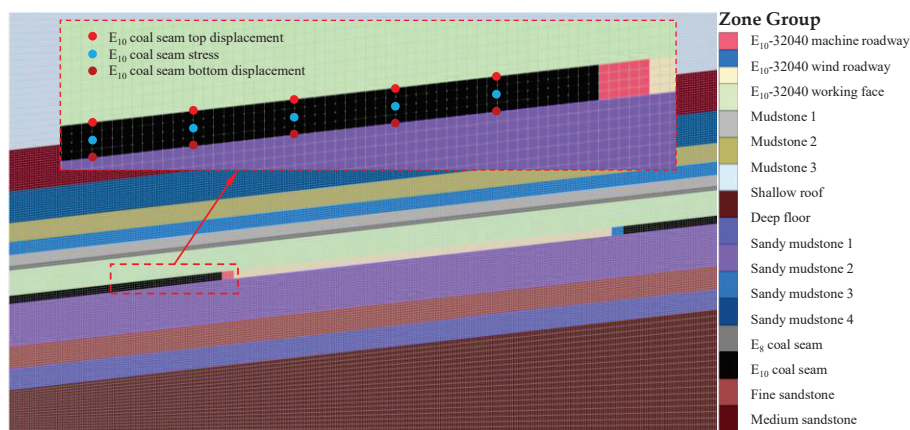


Figure 7. Arrangement of monitoring points.

The coal seam deformation during the simulation was calculated using Equation (2).

$$\varepsilon = (m_r - m_f) / m \quad (2)$$

Here, ε denotes the vertical deformation of the coal seam (%), m_t denotes the displacement in the Z-axis direction at the top of the seam (m), m_b denotes the displacement in the Z-axis direction at the bottom of the seam (m), and m denotes the thickness of the seam (m).

The Z-axis direction displacements at the top and bottom of the coal seam are positive if following the same direction as the coordinate axis and negative otherwise. When $m_r - m_f$ is positive, the coal seam expands. When $m_r - m_f$ is negative, the coal seam is compressed.

3.4. Simulation Results and Analysis

The deformation and stress changes in the coal seam at the monitoring position during the excavation of the E₁₀-32040 machine lane, the wind tunnel, and the E₁₀-32040 working face are depicted in Figure 8. The vertical stress and displacement distribution after equilibrium at the end of the excavation can be observed in Figures 9 and 10, respectively.

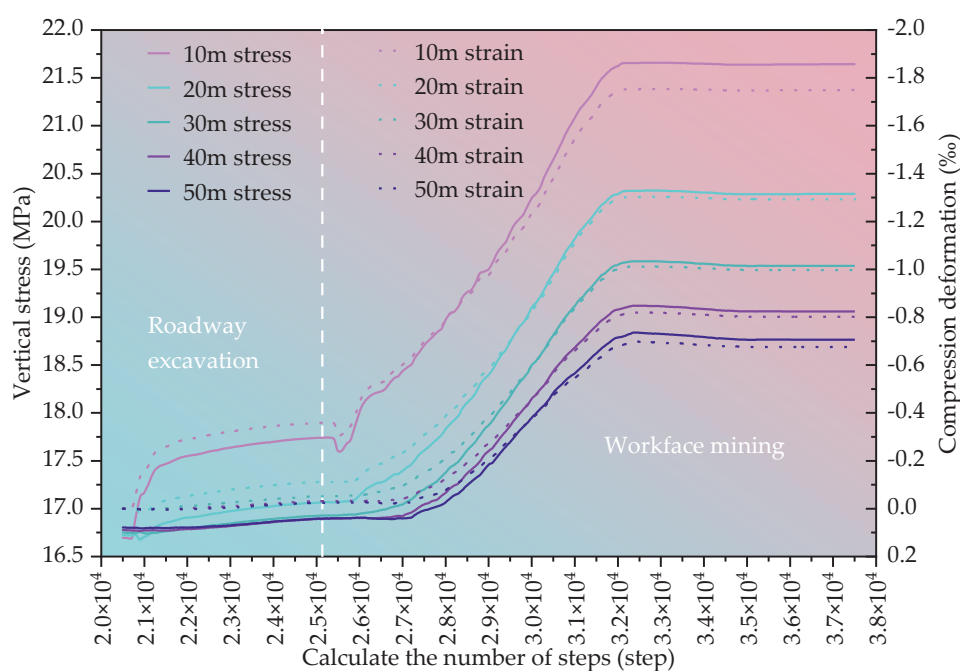


Figure 8. Coal bed deformation and vertical stress change law in the excavation process.

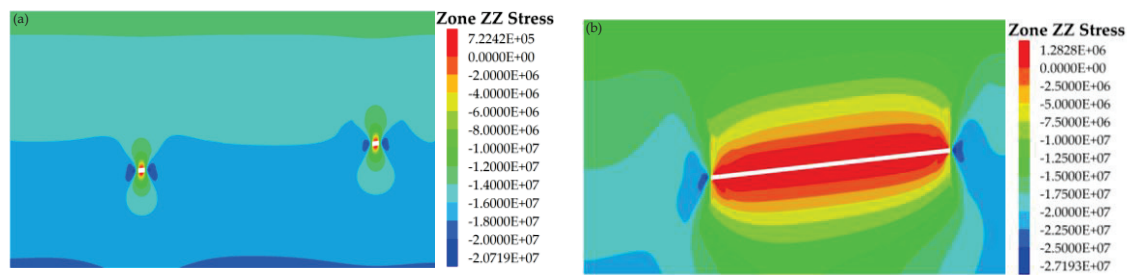


Figure 9. Vertical stress distribution after excavation: (a) roadway excavation; (b) working face recovery.

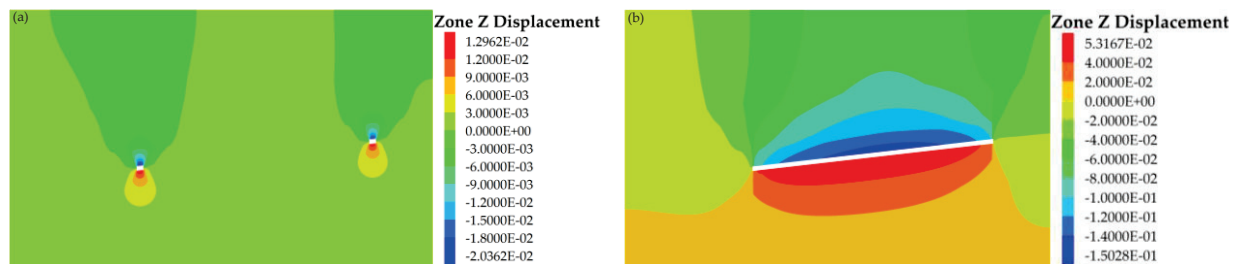


Figure 10. Vertical displacement distribution after excavation: (a) roadway excavation; (b) working face mining.

As shown in Figure 8, the compressed state of the E_{10} coal seam can be observed throughout the whole process of simulated road cutting and back mining. As the excavation of the E_{10} coal seam roadway progresses, and the working face is mined back, both the vertical stress on the coal seam and its compression level increase. However, the magnitude of change during the mining period surpasses that during the excavation period, indicating a more significant impact from working face mining on the neighboring coal body. Only at 10 m and 20 m during the roadway excavation were more obvious vertical stress changes and coal seam deformation observed, and the compression and deformation at 10 m and 20 m were 0.36‰ and 0.11‰, respectively. The rest of the locations remained basically unchanged. As the mining distance from the extraction area increased, the vertical stress and compression deformation of the coal seam gradually decreased as back mining proceeded toward the working face. The compression deformation exceeded 1.0‰ before reaching the 30 m point and reduced to 0.68‰ at 50 m. However, both the amount and magnitude of compression deformation at 50 m remained higher than those observed at 10 m during roadway excavation. These results indicate that the range and degree of the influence of the working face subjected to back mining are significantly greater than roadway excavation.

As shown in Figure 9, the vertical stress of the top and bottom plates of the roadway decreased after the excavation of the E_{10} -32040 machine lane and wind tunnel, transitioning from compressive stress to expansion stress. The roadway was subjected to stress concentration on both sides, with the stress initially increasing and then decreasing toward the coal seam direction. The maximum concentration pressure reached 20.719 MPa. After mining back the E_{10} -32040 working face, the top and bottom plates' stresses decreased, while the stress concentration on both sides increased, resulting in a maximum concentrated stress of 27.193 MPa at 3 m from the roadway. The two sides of the working face showed a trend of an initial increase, followed by a decrease, and then basically becoming stable.

As shown in Figure 10, the top plate of the roadway sank after the excavation of the E_{10} -32040 machine lane and wind lane, with a maximum sinking depth of 0.0203 m. The bottom plate of the roadway had a bottom drum, with a maximum amount of 0.0130 m, and the trend of displacement change on both sides of the roadway was not significant. The top plate of the E_{10} -32040 working face sank after mining, and the sinking depth

gradually decreased from the middle of the working face to both sides, basically showing a symmetrical distribution. The maximum sinking depth was 0.1503 m, which is 7.5 times that of the roadway when excavated. The bottom plate of the working face exhibited a wide range of undermining, and the distribution of undermining amount is consistent with the top plate's sinking amount, reaching a maximum of 0.0532 m, 4.1 times greater than that during roadway excavation.

The simulation results showed that the vertical stress concentration degree and vertical stress change amplitude were greater during working face mining. In addition, the compression deformation of the coal seam was obviously higher than that of the roadway excavation during working face mining. The vertical deformation was more than 1‰ at the depth of 30 m. In addition, the displacements of the top and bottom slabs were greater during working face mining. Specifically, the amount of top-plate sinking was 7.5 times that during roadway excavation, and the bottom plate's bottom drum was 4.1 times that during roadway excavation.

4. Field Testing of Gas Parameters

4.1. Drill Hole Layout

The mining of the E₁₀-32040 working face changed the stress environment of part of the adjacent goaf's coal body, thereby providing favorable conditions for gas emission from the gas-containing coal seam. The effect of gas release primarily manifested as a reduction in both gas pressure and content. The adjacent goaf's coal body exhibited complex fissures and unstable stress, and the gas pressure could only be measured in a single hole, posing challenges for conducting multi-data collection in a small range. Therefore, the gas content was selected as the primary index for this field test. Gas content measurement is convenient and allows for variability within a single hole, thus meeting the requirements of the test work. The study area encompassed the coal body within 47.5 m below the 600–660 m position along the E₁₀-32040 machine lane, with drilling operations conducted ahead of the E₁₀-32060 wind tunnel. A total of five holes were drilled to test the gas content of the E₁₀ coal seam in front of the tunneling work within the study area. Under normal conditions, each gas content sample was collected at intervals of 10 m during drilling. Additionally, a sample was obtained at the final location of each drill hole. When the distance between sampling points of different drill holes or the same drill hole was less than 5 m, only one sample was collected. The arrangement of drill holes and sampling points is shown in Figure 11, and the sampling parameters are listed in Table 3.

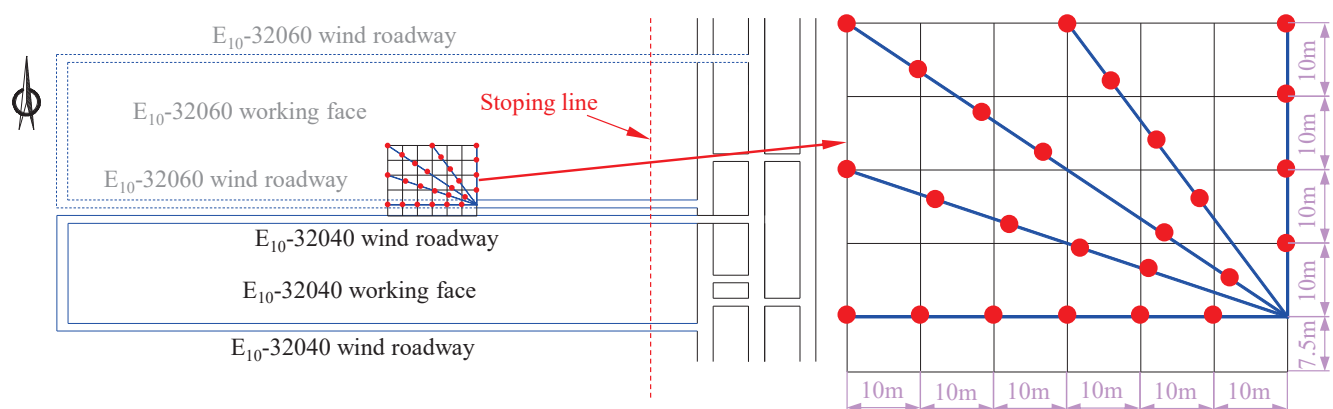


Figure 11. Sampling borehole layout.

Table 3. Parameters of gas content sampling.

Group Number	Drill Hole Number	Sampling Depth/(m)	Distance from the Goaf/(m)	Group Number	Drill Hole Number	Sampling Depth/(m)	Distance from the Goaf/(m)
1	1-1	10.0	7.5	3	3-3	40.0	29.7
	1-2	20.0	7.5		3-4	50.0	35.2
	1-3	30.0	7.5		3-5	60.0	40.8
	1-4	40.0	7.5		3-6	72.1	47.5
	1-5	50.0	7.5		4-1	20.0	23.5
	1-6	60.0	7.5	4	4-2	30.0	31.5
2	2-1	20.0	13.8		4-3	40.0	39.5
	2-2	30.0	17.0		4-4	50.0	47.5
	2-3	40.0	20.2	5	5-1	10.0	17.5
	2-4	50.0	23.3		5-2	20.0	27.5
	2-5	63.3	27.5		5-3	30.0	37.5
3	3-1	10.0	13.1	5	5-4	40.0	47.5
	3-2	30.0	24.1				

4.2. On-Site Construction and Testing

The gas content of the coal seam comprises the amount of loss, the amount of desorption in the well, and the amount of desorption tested in the laboratory. The loss amount was determined based on the obtained downhole desorption pattern. The ZDY4500LX (HENAN TFL MECHINERY AND MANUFACTURING CO., LTD., Pingdingshan, China) fully tracked hydraulic pit drilling rig was used for this site, as shown in Figure 12a. Since multiple samplings of a borehole were necessary, core drilling was chosen for sampling, and frequent backing out of the drill pipe and changing the drill bit were required. This may lead to collapsing, plugging, and deviation in drilling within the unstable neighboring goaf's coal body. Therefore, during gas content sampling, all drill holes were air-drained for pulverized coal, and samples were directly collected from the hole mouth. The sample was loaded to a distance of 1 cm from the mouth of the coal rock canister before the completion of sampling. Subsequently, downhole desorption operations were conducted, as shown in Figure 12b. Desorption can be terminated when the desorption time reaches 60–120 min or the gas desorption rate is less than 2 cm³/min. Upon completion of downhole desorption, the coal sample canister should be sealed and placed into the water for careful observation over 10 min. If any bubbles emerge during this time, the sample is invalid and must be re-sampled for testing. Additionally, no sample with air leakage can be sent to the laboratory for testing. The laboratory employs a DGC-type gas content meter for testing. Before the test, it is essential to perform a gas tightness test on the sample after desorption downhole, following which, if no air leakage is detected, it can be subjected to laboratory testing. The laboratory gas desorption amount determined by the DGC-type gas content meter was divided into two parts: the crushed desorption amount and the non-desorbable gas amount under standard atmospheric pressure. The amount of non-desorbable gas under standard atmospheric pressure was calculated by Equation (3) according to the coal body's base parameters in the laboratory test:

$$W_{CC} = \frac{0.1ab}{1 + 0.1b} \times \frac{100 - A_d - M_{ad}}{100} \times \frac{1}{1 + 0.31M_{ad}} + \frac{\pi}{\gamma} \quad (3)$$

where W_{CC} denotes non-desorbable gas content under standard atmospheric pressure (m³/t); a represents adsorption constant (m³/t), which is the limit of adsorption at the experimental temperature; b represents the adsorption constant (MPa^{−1}); A_d represents ash content of coal (mass fraction; %); M_{ad} denotes the moisture of coal (mass fraction; %); π

represents the porosity of coal (m^3/m^3); and γ denotes the capacitance of coal (apparent density; t/m^3).

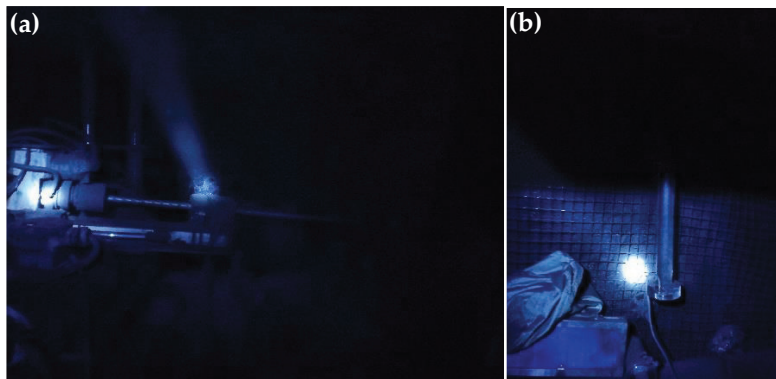


Figure 12. Field measurement process: (a) borehole construction; (b) downhole desorption.

4.3. Test Results and Analysis

The mining of the E₁₀-32040 working face changed the stress environment in a portion of the adjacent area. A total of 25 samples were tested within the study area, while 1 coal sample was selected to determine the basic parameters for calculating the non-desorbable volume at the standard atmospheric pressure. The results of the testing and calculation of the parameters necessary for determining the non-desorbable quantity under standard atmospheric pressure are shown in Table 4. The relationship between the residual gas content of the coal seam and its distance from the mining area is shown in Figure 13.

Table 4. The parameters and results of non-desorbable gas content calculation at standard atmospheric pressure.

Coal Seam	$a/(\text{m}^3/\text{t})$	$b/(\text{MPa}^{-1})$	$A_d/(\%)$	$M_{ad}/(\%)$	$\pi/(\text{m}^3/\text{m}^3)$	$\gamma/(\text{t}/\text{m}^3)$	$W_{CC}/(\text{m}^3/\text{t})$
E ₁₀	20.17	0.57	10.67	1.06	0.037	1.37	0.75

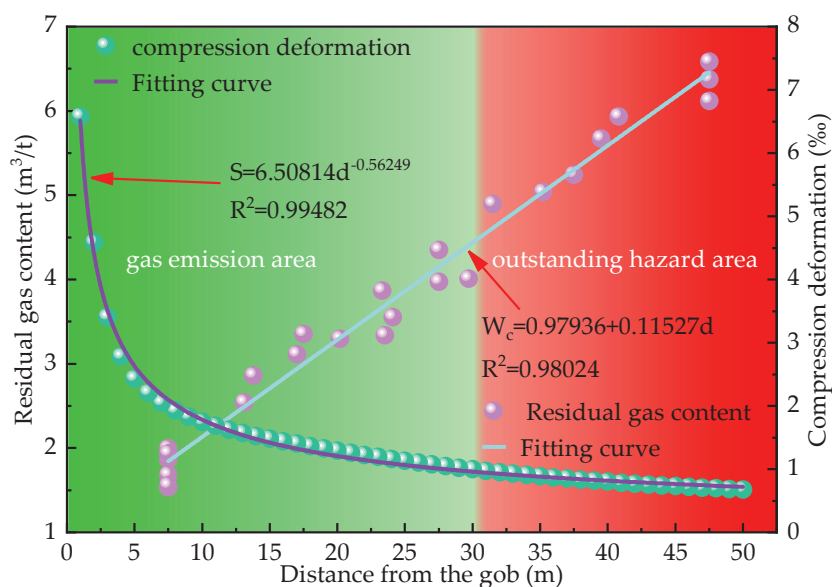


Figure 13. Variation pattern of residual gas content and compression deformation of coal seam.

Figure 13 illustrates a linear distribution with a trend of increase for the residual gas content, with lower values observed as the sampling point moves farther away from the mining area, and the fitting formula is shown in Equation (4). Additionally, an exponential

relationship is observed between the distance from the mining area and the compression and deformation of the coal seam, as expressed by Equation (5).

$$W_C = 0.11527d + 0.97936 \quad (4)$$

Here, W_C represents the residual gas content of the coal seam (m^3/t), and d represents the distance from the air-mining area (m).

$$S = 6.50814d^{-0.56249} \quad (5)$$

Here, S represents the vertical strain of the coal seam ($\%$), and d represents the distance from the mining area (m).

The coefficient of determination (R^2) obtained from the fitting of Equation (4) was 0.98024, demonstrating a strong correlation between the two parameters, and the value of the dependent variable was determined by the independent variable. Therefore, under the current mining conditions, the distance between the coal body and the goaf could be used to evaluate the residual gas content of the coal body. According to Equation (3), the residual gas content of the coal body adjacent to the goaf coal wall was $0.97936 \text{ m}^3/\text{t}$, and the residual gas content of the coal seam was $0.75 \text{ m}^3/\text{t}$, demonstrating a slight difference between the two. Therefore, the formula can accurately reflect the distribution of residual gas content in the range of 0–47.5 m. The annual average value of gas content measured at 47.5 m was $6.36 \text{ m}^3/\text{t}$, which is basically the same as the average raw gas content of $6.31 \text{ m}^3/\text{t}$ in the adjacent area. On this basis, it can be assumed that the gas content at 47.5 m is not affected by mining, and the tested gas content is the original gas content. The coefficient of determination (R^2) of Equation (5) obtained by fitting was 0.99482, demonstrating a strong correlation between the two parameters, and the independent variable could be used to determine the value of the dependent variable. Therefore, under the current mining conditions, the distance between the coal body and the mining area determined the amount of vertical deformation of the coal seam.

The E_{10} coal seam is a prominent coal seam. Under the current mining conditions, a 30% reduction in the original gas content can be considered sufficient to eliminate the risk of coal and gas protrusion, thereby establishing an automatic gas emission. Therefore, when the residual gas content is less than $4.45 \text{ m}^3/\text{t}$, the risk of protrusion can be eliminated, and safe excavation can be realized. According to Equation (4), when the residual gas content is $4.45 \text{ m}^3/\text{t}$, d equals 30.11 m, suggesting that the automatic gas emission width is 30.11 m under the current mining conditions. After substituting this value into Equation (5), S equals 0.96‰. This indicates that the effective automatic emission width was achieved when the vertical compression was 0.96‰ in the simulation.

5. Conclusions

In this study, the stress distribution and damage law of the coal body in the neighboring mining area were analyzed. Subsequently, a numerical simulation model was established according to the geological conditions of the site to examine the variation pattern of the shaded area's displacement during roadway excavation and the back mining of the working face. Finally, the effective width of automatic gas discharge was determined through experimental analysis of coal seam gas content.

- (1) The E_{10} coal seam adjacent to the goaf's coal body underwent compression, with a greater impact on the extent of deformation compared to roadway excavation during back mining. During the back mining of the goaf, a compression and deformation amount of 0.68‰ over a distance of 50 m was observed, whereas during roadway

excavation, there was a compression and deformation amount that was 1.89 times higher at a rise distance of 10 m and 6.18 times higher at a distance of 50 m.

- (2) The stress of the coal body adjacent to the E₁₀ coal seam decreased, and the concentrated stress was transmitted to the coal body on both sides. The maximum stress concentration point occurred 3 m near the auxiliary coal wall of the roadway. The concentrated stress was 20.719 MPa during roadway excavation and 27.193 MPa during the mining of the working face.
- (3) The results of the gas content test of the coal body in the neighboring goaf conform to a linear distribution, and the fitting formula is $WC = 0.11527d + 0.97936$. When the original gas content decreases by 30%, d is 30.11 m, and the width of gas automatic emission is 30.11 m. At this point, the risk of coal and gas protrusion can be eliminated.
- (4) The compression deformation of the neighboring goaf coal body follows an exponential distribution, with a fitting formula of $S = 6.50814d^{-0.56249}$. The gas automatic emission has a width of 30.11 m, and the obtained value for S is 0.96%. These findings can provide theoretical references and data support for the location of adjacent roadways and the selection of gas prevention and control measures in this coal seam.

Author Contributions: Conceptualization, J.X. and R.B.; methodology, J.X.; software, J.X.; investigation, J.X. and R.B.; resources, X.C. and S.L.; data curation, J.X., Z.C. and J.C.; writing—original draft preparation, J.X.; writing—review and editing, R.B., X.C. and J.C.; funding acquisition, R.B. and X.C. All authors have read and agreed to the published version of the manuscript.

Funding: This research was funded by the National Natural Science Foundation of China (grant numbers 52204212 and 52174181), the S&T Program of Hebei (grant number 22375401D), and the Fundamental Research Funds for the Central Universities (grant number 3142020003).

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

Acknowledgments: We would like to thank the staff of Ping Coal Company's No. 1 Mine for providing the on-site test site and for their support during the testing process.

Conflicts of Interest: We declare that we have no financial and personal relationships with other people or organizations that can inappropriately influence our work, there is no professional or other personal interest of any nature or kind in any product, service, and/or company that could be construed as influencing the position presented in, or the review of, the manuscript entitled "Automatic gas emission width of coal bodies in the goaf near outbursting coal seams".

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Article

Quantitative Identification of Emission Sources and Emission Dynamics of Pressure-Relieved Methane Under Variable Mining Intensities

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Abstract: This study addresses the abnormal emission of pressure-relieved methane under high-intensity mining conditions by integrating geostatistical inversion, FLAC^{3D}-COMSOL coupled numerical simulations, and stable carbon–hydrogen isotopic tracing. Focusing on the 12023 working face at Wangxingzhuang Coal Mine, we established a heterogeneous methane reservoir model to analyze the mechanical responses of surrounding rock, permeability evolution, and gas migration patterns under mining intensities of 2–6 m/d. Key findings include the following: (1) When the working face advanced 180 m, vertical stress in concentration zones increased significantly with mining intensity, peaking at 12.89% higher under 6 m/d compared to 2 m/d. (2) Higher mining intensities exacerbated plastic failure in floor strata, with a maximum depth of 47.9 m at 6 m/d, enhancing permeability to 223 times the original coal seam. (3) Isotopic fingerprinting and multi-method validation identified adjacent seams as the dominant gas source, contributing 77.88% of total emissions. (4) Implementing targeted long directional drainage boreholes in floor strata achieved pressure-relief gas extraction efficiencies of 34.80–40.95%, reducing ventilation air methane by $\geq 61.79\%$ and maintaining return airflow methane concentration below 0.45%. This research provides theoretical and technical foundations for adaptive gas control in rapidly advancing faces through stress–permeability coupling optimization, enabling the efficient interception and resource utilization of pressure-relieved methane. The outcomes support safe, sustainable coal mining practices and advance China's Carbon Peak and Neutrality goals.

Keywords: mining intensity; gas transport; coupled simulation; gas emission volume; quantitative source apportionment

1. Introduction

In recent years, global research on clean energy transitions and environmental sustainability has grown substantially [1–3]. Against this backdrop, coal and gas co-mining has emerged as an advanced mining philosophy, emphasizing the simultaneous and effective extraction of methane during coal production to maximize resource utilization while minimizing safety risks [4]. Gas drainage technology not only prevents gas overlimit incidents and coal–gas outbursts, thereby safeguarding miners' lives, but also plays a vital role in developing coalbed methane resources and reducing greenhouse gas emissions [5,6].

Within China's strategic push toward its "Carbon Peak and Carbon Neutrality" goals, efficient gas extraction holds profound significance for optimizing energy structures, curbing greenhouse gas emissions and achieving these dual carbon targets [7,8].

The escalating mining depths exacerbate technical challenges through low-permeability coal seams, where elevated geostresses and structural damage significantly impair drainage efficiency [9,10]. Protective seam mining has emerged as an economically viable pressure-relief permeability enhancement technology [11,12]. This technique achieves the following: (1) stress redistribution through overburden unloading, (2) fracture network development via mining-induced damage evolution, (3) permeability enhancement through porosity generation (up to 300% increase), and (4) outburst risk elimination via methane pressure reduction [13–15].

Despite its advantages, variable mining intensities significantly influence pressure-relief effectiveness. Rapid face advancement intensifies methane emission dynamics, potentially directing 60–80% of desorbed gas toward goaf areas and return airways. Without proper control measures, this may trigger methane exceedance incidents, jeopardizing face safety [16,17]. Current research emphasizes mining intensity as the dominant control parameter for emission flux in longwall configurations [18].

Although pressure-relief gas control technology via protective layer mining has been widely applied in engineering practice [19], critical gaps remain in existing research: (1) Limitations in Quantitative Gas Source Analysis: Most studies rely on single methods (e.g., statistical models or numerical simulations) for gas source contribution analysis [20,21], lacking cross-validation via multi-method integration, resulting in high uncertainty in source apportionment [22]. (2) The Neglect of Dynamic Mining Intensity Effects: Current models predominantly assume fixed mining intensities [23], failing to systematically investigate abrupt gas migration mechanisms under high-intensity mining [18]. (3) The Simplified Treatment of Geological Heterogeneity: Existing numerical models often assume homogeneous coal seams [14], inadequately capturing the impact of complex geological conditions on pressure-relief efficiency [24].

Despite significant advancements in natural gas extraction technologies, critical research gaps persist in the precise quantification of gas emissions under specific stratigraphic conditions, particularly at varying mining intensities. To address this gap, this study introduces a novel methodology combining integrated FLAC^{3D}-COMSOL coupling simulations with stable carbon–hydrogen isotopic analysis. This approach enables the following: (1) the precise reconstruction of heterogeneous methane reservoirs within the protective coal seams of the Wangxingzhuang Mine; (2) the detailed decoding of mining intensity-dependent emission mechanisms; and (3) the targeted formulation of gas drainage strategies. By bridging theoretical modeling with empirical isotopic validation, this work not only resolves limitations inherent to prior methodologies but also delivers a comprehensive solution to enhance gas recovery efficiency and operational safety in multi-seam mining. Crucially, the findings directly support China's Carbon Peak and Carbon Neutrality policy by advancing sustainable methane utilization and emission reduction frameworks [25–27].

2. Theoretical Analysis

2.1. Migration Dynamics and Transport Mechanisms of Pressure-Relieved Methane in Coal Seams

Under static conditions (non-mining state), coalbed methane maintains steady-state equilibrium through continuous adsorption–desorption cycles within the coal matrix, with the original gas distribution pattern remaining undisturbed in both coal seams and surrounding rock formations [28]. Following protective layer extraction operations, a dual stress-relief mechanism emerges: (1) substantial reduction in overburden pressure triggers coal matrix decompaction, and (2) geomechanical unloading induces structural damage

through energy release. These coupled effects produce three-dimensional fracture network propagation, significantly enhancing coal seam porosity and permeability through two fundamental mechanisms: pressure-dependent sorption swelling reversal and matrix shrinkage-induced flow path development. Such petrophysical modifications facilitate accelerated methane desorption kinetics and effective gas drainage, achieving simultaneous reduction in reservoir pressure and gas content in protected seams. Substantial empirical evidence confirms the engineering efficacy of protective layer mining for reservoir stimulation, with notable theoretical contributions including Academician Qian Minggao's pioneering "O-shaped ring" theory elucidating stress-dependent gas migration patterns during extraction processes [29]. Complementary field studies by Yang's research team [24] employing CT scanning and 3D reconstruction techniques quantitatively characterized the stratified fracture architecture facilitating gas transport in stimulated coal masses. Figure 1 systematically illustrates this multi-phase permeability enhancement mechanism.

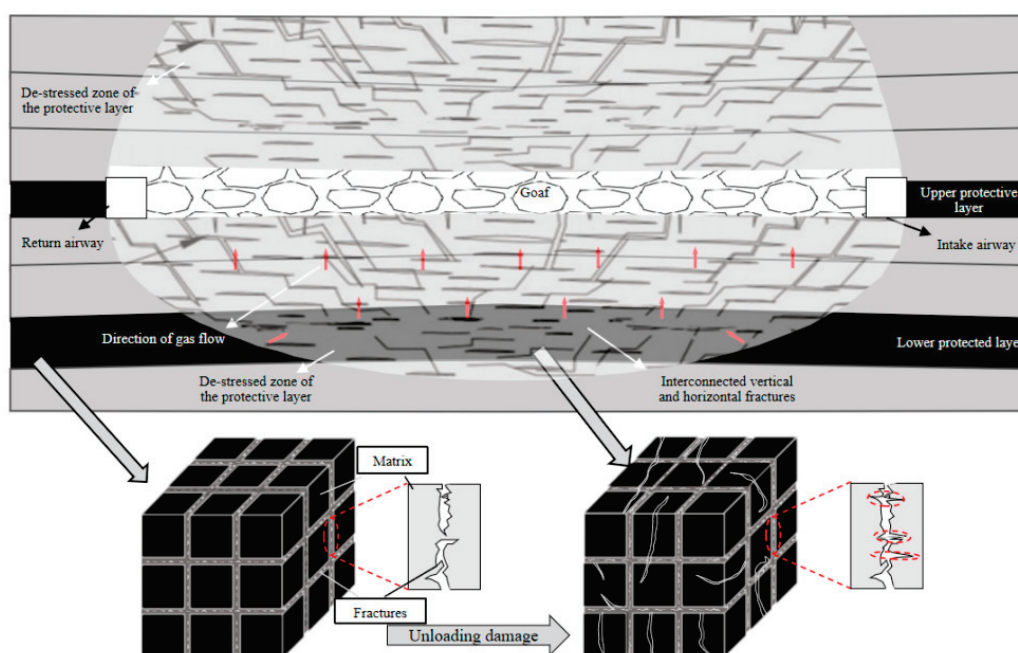


Figure 1. Diagram of stress release and permeability increase in protective layer mining.

The dynamic mining advancement disrupts initial gas-phase equilibrium, initiating methane desorption–diffusion processes along concentration gradients toward low-pressure zones. Methane emission sources exhibit spatiotemporal heterogeneity, comprising four primary components: (1) residual gas in fragmented coal, (2) coal wall emissions, (3) goaf gas accumulation, and (4) adjacent seam gas migration. Post-extraction gas liberation occurs through two dominant pathways: (a) ventilation-driven discharge via return airways, and (b) air leakage-induced gas recirculation creating hazardous accumulations near upper corners [30]. Current computational fluid dynamics models require further validation against field measurements to fully capture these complex gas transport phenomena.

2.2. Numerical Model Theoretical Equations

A dual pore medium was typically used in the present study on gas-containing coal gas–solid coupling models, meaning that the coal body is thought to be made up of fractures and a matrix that contains the coal skeleton and pores [31]. The following are the fundamental presumptions of this physical model: (1) the coal body is an ideal continuum with elastic and plastic properties; (2) isothermal circumstances satisfy the ideal

gas equation; and (3) Darcy's law of percolation and Fick's law of diffusion govern gas flow in pores and fractures, respectively [32,33].

The coal body stress field equation can be derived by taking into account the gas-containing coal body's adsorption–desorption properties:

$$Gu_{i,jj} + \frac{G}{1-2\mu}u_{j,ji} - \alpha_m p_{m,i} - \alpha_f p_{f,i} + p_i = 0 \quad (1)$$

This equation is based on the stress equilibrium of a dual pore-fracture medium model [34] where G is the shear modulus, MPa; u is the displacement, m, and μ is the Poisson's ratio; α_m is the effective stress coefficient of the pore; α_f is the effective stress coefficient of the fissure; p_i is the volumetric stress, MPa; $u_{i,jj}$ is the second-order partial derivatives of the displacement component in the i -direction in the j -direction; and $p_{m,i}$ and $p_{f,i}$ are the first-order partial derivatives of the coal body matrix, and coal body fissure gas pressure in the i -direction, respectively.

The permeability of coal is cubically related to its porosity [35]. Based on this relationship, the formula for calculating permeability is derived as follows:

$$k = k_0 \left(1 + \frac{c}{p_f}\right) \left(\frac{\varphi_f}{\varphi_{f0}}\right)^3 \quad (2)$$

$$k = \left(1 + \frac{c}{p_f}\right) \left\{1 + \frac{\alpha_m(p_m - p_{m0}) + \alpha_f(p_f - p_{f0})}{E\varphi_{f0}} + \frac{a}{\varphi_{f0}} \left(\frac{K_1}{E} - 1\right) \left(\frac{p_m}{p_m + b} - \frac{p_{m0}}{p_{m0} + b}\right)\right\}^3 \quad (3)$$

where φ_f is the porosity of coal fissure, %; φ_{f0} is the initial porosity of coal fissure, %; K_1 is the bulk modulus of coal, MPa; p_m is the coal matrix gas pressure, Pa; p_{m0} is the initial gas pressure of the coal matrix, Pa; p_f is the gas pressure of coal fissure, Pa; p_{f0} is the initial gas pressure in coal fissure, Pa; a is the Langmuir volume constant, m^3/t ; b is Langmuir pressure constant, MPa; k is the coal body's absolute permeability, mD; ρ_g is the coal body's initial permeability, mD; and c is the Klinkenberg coefficient.

The mass of gas stored in the coal matrix per unit volume includes both the gas adsorbed in the pores and the gas free in the pores [36]. The mass of gas in coal seam fractures equals the mass of gas in the free state. Gas volume is stored in each volume of coal body.

$$\rho = \rho_1 + \rho_2 = \rho_g \rho_s \frac{ap_m}{p_m + b} + \varphi_m p_m \frac{M_c}{RT} \quad (4)$$

where ρ is the amount of stored gas in unit volume of coal body, kg/m^3 ; ρ_1 is the adsorbed gas content in the unit coal matrix, kg/m^3 ; ρ_g is the density of gas, kg/m^3 ; ρ_s is the density of coal, kg/m^3 ; ρ_2 is the mass of free gas in coal matrix, kg/m^3 ; φ_m is the porosity of coal matrix, %; M_c is the molar mass of gas, kg/mol ; R is the ideal gas constant, $8.314 \text{ J}/(\text{mol} \cdot \text{K})$; and T is the internal temperature of coal, K.

The gas in the fracture flows out in the Darcy seepage mode as the negative energy source, and the mass conservation law corresponds to the unit volume of coal [37]:

$$Q_1 = \frac{\partial m}{\partial t} = -\frac{M_c}{\tau RT}(p_m - p_f) \quad (5)$$

$$\frac{\partial \rho_g \varphi_f}{\partial t} = -Q_1 - (\rho_g v) \quad (6)$$

$$v = -\frac{k}{\mu} p_f \quad (7)$$

After bringing Equation (4) into Equation (5), the time dependence equation of gas pressure within the coal body matrix is obtained:

$$\frac{\partial p_m}{\partial t} = -\frac{V_m(p_m - p_f)(p_m + b)^2}{\tau abRT\rho_s + \tau\varphi_m V_m(p_m + b)^2} \quad (8)$$

where Q_1 is the amount of gas change (mass source) per unit of coal body matrix, $\text{kg}/\text{m}^3 \cdot \text{s}$; v is the speed of gas seepage in the cracks of the coal body, m/s ; μ is the dynamic viscous damping coefficient of gas, $\text{Pa} \cdot \text{s}$; and τ is the time constant associated with the gas flow process, s .

The preceding equations describe the migration mechanism and law of gas in coal. After face mining, gas pressure in the coal seam varies, affecting coal structure, porosity, permeability, and other coal seam characteristics, as well as gas movement in the coal body. The fluid–structure coupling model is defined as the combination of Equations (1), (5) and (7). Diffusion and percolation are the mechanisms by which gas moves through the coal body. Drilling destroys the stress equilibrium of the raw coal body, causing the adsorbed gas in the coal body matrix to desorb and diffuse into the cracks according to Fick’s law, thus engaging in percolation. The gas–solid coupling model of coal and gas is created by the interaction and coupling of gas migration in coal and the coal stress field.

3. Engineering Background

3.1. Overview of the Test Area

Wangxingzhuang Coal Mine is located in the southwestern part of Xinzheng City, within Zhengzhou City, Henan Province, China, and is shown in Figure 2. The mine uses vertical shafts to extract seams No. 2₃ and 2₁, which are about 20 m apart. The seam No. 2₁ is prone to the risk of coal and gas outbursts. The thickness of seam No. 2₃ ranges from 0 to 6.54 m, with an average of 1.54 m, and is largely minable. In contrast, the seam No. 2₁ exhibits thickness varying from 0 to 21 m, with an average thickness of 6.8 m, and is characterized as a ‘three soft’ seam, indicating a soft roof, floor, and coal body, which exhibits typical features of deep coal seams in China. This seam is notably impermeable, making gas extraction challenging.

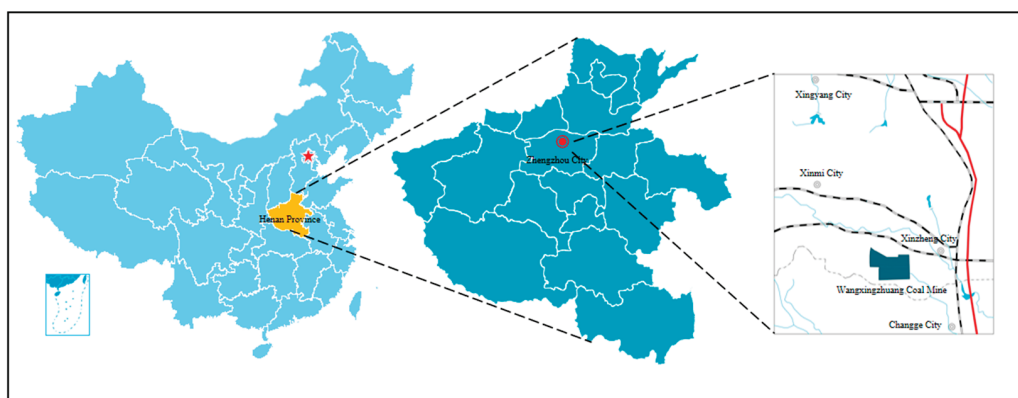


Figure 2. Location map of Wangxingzhuang Coal Mine.

For the purposes of this study, the 12021 and 12023 working faces have been designated as the experimental areas. The 12023 working face, which consists of fine-grained sandstone and mudstone above and sandy mudstone below, exhibits a broad and gentle monoclinical structure. The seam’s dip angle varies from 6 to 14 degrees, averaging at 8 degrees, with an average seam thickness of 1.7 m, and is considered to have a relatively simple structure. The 12021 working face, situated beneath, presents the following geological features: an

average coal seam thickness of 7.0 m, an average inclination angle of 8 degrees, a seam depth ranging from 385 to 427 m, an inclined width of 170 m, and a strike length of 640 m.

3.2. Gas Base Parameters

Tables 1 and 2 show the results from evaluating the gas parameters of the 12021 and 12023 working faces. The 12023 working face has a maximum initial gas pressure of 0.36 MPa and peak gas content of 6.50 m³/t. In comparison, the 12021 working face has a higher initial gas pressure of 0.78 MPa and a higher gas content of 10.52 m³/t. The seam's firmness coefficient is 0.14, with permeability coefficients of 0.0145 m²/(MPa) for both permeability and gas permeability, along with an additional coefficient of 0.0145 m²/(MPa²·d). The coal seam features a loose structure and low permeability, complicating gas extraction efforts. This is further evidenced by the borehole's gas flow attenuation coefficient of 0.07 d^{−1}. These parameters are critical for designing tailored gas control strategies, as the seam No. 2₁ poses substantial safety and operational challenges during mining.

Table 1. Basic gas parameters of 12023 working face.

Parameter Category	Measurement Result	Parameter Category	Measurement Result
Gas pressure/MPa	0.20~0.36	Initial Gas Desorption Rate/(mL/s)	10~12
Gas content/(m ³ /t)	4.75~6.50	Apparent Density/(t/m ³)	1.28
Porosity/%	5.0	True Density/(t/m ³)	1.42

Table 2. Basic gas parameters of 12021 working face.

Parameter Category	Measurement Result	Parameter Category	Measurement Result
Gas pressure/MPa	0.42~0.78	Initial Gas Desorption Rate/(mL/s)	10~12
Gas content/(m ³ /t)	8.56~10.52	Apparent Density/(t/m ³)	1.35
Coal permeability coefficient	0.0145 m ² /(MPa ² ·d)	True Density/(t/m ³)	1.47
Type of coal destruction	IV	Porosity/%	4.82
Stabilization factor of coal	0.14~0.16	Gas flow decay coefficient/d ^{−1}	0.07~0.10

Given the geological context and a thorough review of the gas parameters for each seam at Wangxingzhuang Mine, the decision was made to designate the seam No. 2₃, which presents comparatively lower gas pressure, as the upper protective layer for extraction. The actual initial gas pressure data measured at the 12021 working face were visualized on a gas pressure contour map, revealing localized gas pressures of approximately 0.80 MPa in areas prone to gas aggregation, indicating a heightened risk of gas outbursts and the potential for coal and gas outburst incidents, as depicted in Figure 3. The blue line in Figure 3 delineates the partial scope of the numerical simulation study.

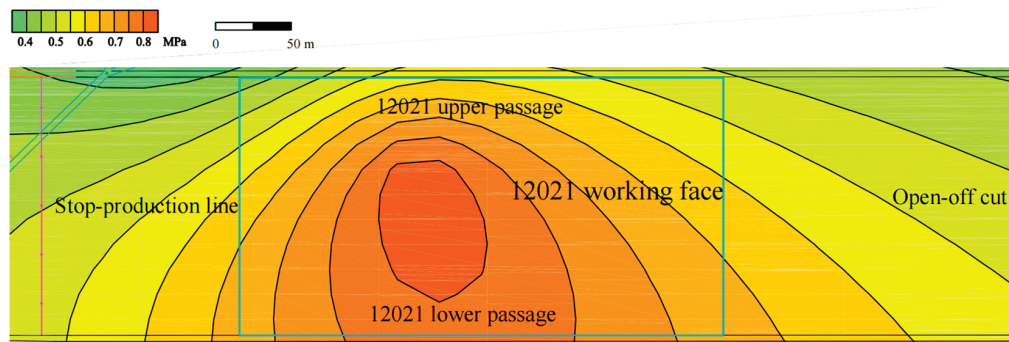


Figure 3. Gas pressure contour map at 12021 working face.

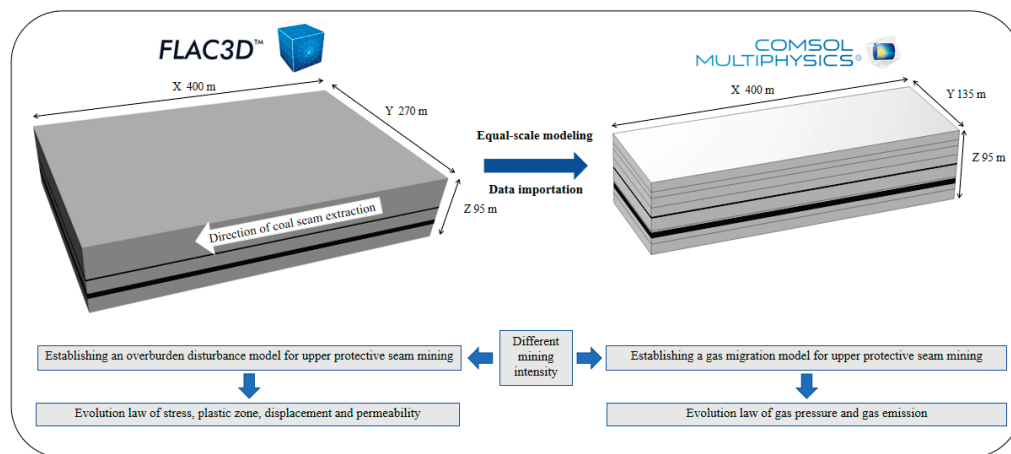
4. Numerical Simulation

4.1. Model Construction

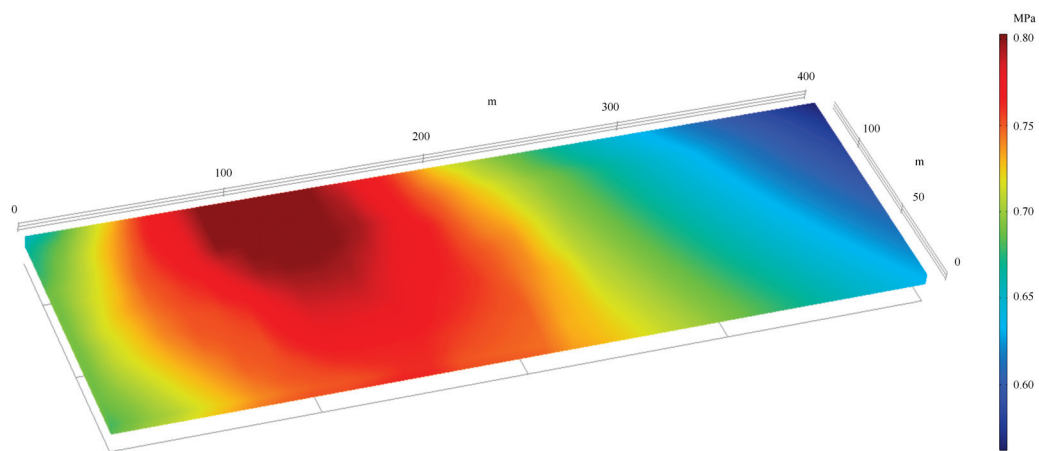
This work uses the FLAC^{3D}-COMSOL linked numerical simulation approach, as shown in Figure 4. The study object is Wangxingzhuang Coal Mine's 12023 protective layer working face. FLAC^{3D} (v6.0) software was used to create a proportionate horizontal coal seam model ($X \times Y \times Z = 400 \text{ m} \times 270 \text{ m} \times 95 \text{ m}$). Although the stress distribution exhibits symmetry at the $Y = 135 \text{ m}$ plane, the full model was retained in FLAC^{3D} to fully capture the effects of actual geological conditions, such as non-uniform stress fields and localized lithological variations. While the full model increases computational demands, it avoids the potential underestimation of plastic failure depth and stress release rate caused by symmetric boundary constraints. Subsequent COMSOL (v6.1) simulations (based on the symmetric $Y = 135 \text{ m}$ plane) were employed to simplify gas migration analysis, focusing on dominant mechanisms under symmetric conditions (Figure 4). This two-stage modeling approach balances computational efficiency with geological authenticity. The linked simulation was accomplished by importing stress–strain data from FLAC^{3D} into COMSOL using the technique described in Figure 4a.

The grid convergence study assessed the sensitivity of the simulation results to grid resolution by testing three grid configurations: coarse grid (FLAC^{3D} element size = 2 m, COMSOL element size = conventional), baseline grid (FLAC^{3D} = 1 m, COMSOL element size = refined), and fine grid (FLAC^{3D} = 0.5 m, COMSOL element size = highly refined). Figure 5 shows the data change in one monitoring point with different grid densities. The results showed that the difference in maximum vertical stress in the stress concentration zone between the baseline and fine grids was less than 2%, confirming grid independence.

To accurately portray gas geological conditions, the model's beginning condition was set to the real gas pressure of the coal seam. Given the comparatively low gas pressure of the seam No. 2₃, the starting pressure for the whole coal seam was set consistently at 0.36 MPa. The geological conditions of the seam No. 2₁ were carefully reversed using geological statistical numerical modeling. In the COMSOL simulation, the isobar map of the observed gas pressure was identified and integrated into the model as the beginning condition for the seam No. 2₁ to replicate its non-uniform occurrence features and increase simulation accuracy.



(a) The establishment of a numerical simulation model.



(b) Introduction of gas pressure.

Figure 4. Numerical simulation flowchart.

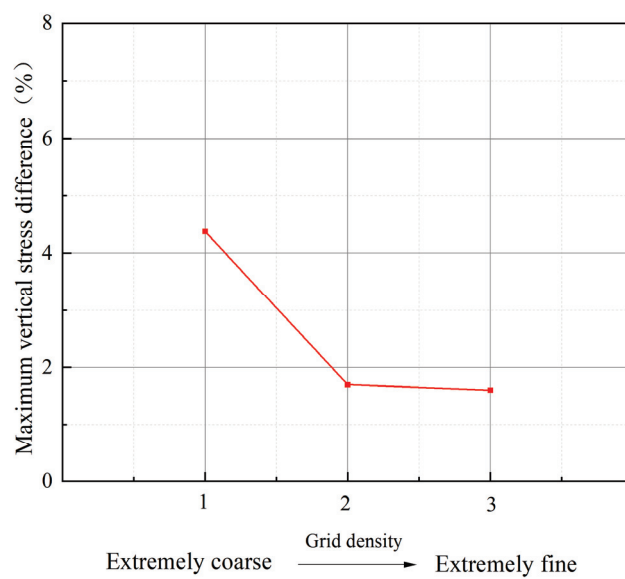


Figure 5. Grid independent result.

Furthermore, the structural stress was simulated by increasing the lateral pressure coefficient ($K = 1.15$) to better model and anticipate gas migration behavior throughout the mining operation of the protected coal seam. To prevent boundary impacts during mining, a 50 m coal pillar was left at each end of the coal seam. The mining intensity was expressed by digging various distances. Combining the real production situation on site with experimental needs, mining intensities of 2, 4, and 6 m/d were represented by digging 2, 4, and 6 m, respectively. Tables 3 and 4 indicate the particular values for coal and rock mechanical characteristics, as well as the key simulation parameters.

Table 3. Table of formation mechanic parameters.

Lithology	Thickness/m	Density/ ($\text{g}\cdot\text{cm}^{-3}$)	Bulk Modulus/ GPa	Shear Modulus/ GPa	Cohesion/ MPa	Tensile Strength/ MPa	Internal Friction Angle/ $^{\circ}$
Dazhan sandstone 1	12.5	2.6	4.5	2.3	3.0	2.2	27.0
Sandstone	8.0	2.65	4.5	2.3	3.0	2.2	27.0
Medium-grained sandstone	12.5	2.5	4.9	2.4	2.4	1.8	24.0
Sandy mudstone	12.0	2.65	2.7	1.4	2.0	1.1	26.0
No. 2 ₃ coal seam	1.7	1.5	2.8	0.5	2.0	1.1	26.0
Dazhan sandstone 1	17.0	2.6	4.5	2.3	3.0	2.2	27.0
Roof mudstone	2.3	2.65	2.7	1.4	2.0	1.1	26.0
No. 2 ₁ coal seam	7.0	1.5	2.5	0.5	2.0	1.1	26.0
Mudstone	8.0	2.6	2.5	1.3	2.0	1.1	26.0
L ₇₋₈ limestone	14.0	2.6	4.9	2.4	2.4	1.8	24.0

Table 4. Table of the values taken by the main simulation parameters.

Parameter	Value	Parameter	Value
True density of coal/($\text{kg}\cdot\text{m}^{-3}$)	1470	Molar mass of air/($\text{kg}\cdot\text{mol}^{-1}$)	0.03
Apparent density of coal/($\text{kg}\cdot\text{m}^{-3}$)	1350	Molar mass of methane/($\text{kg}\cdot\text{mol}^{-1}$)	0.016
Gas density at standard conditions/($\text{kg}\cdot\text{m}^{-3}$)	0.72	Effective stress coefficient for fractures	0.651
Elastic modulus of coal/gpa	1.51	Effective stress coefficient for pores	0.078
Elastic modulus of coal matrix/gpa	8.13	Langmuir pressure constant/mpa	6.019
Poisson's ratio of coal	0.31	Dynamic viscosity of gas/(pa·s)	1.08×10^{-5}
Initial matrix porosity/%	7.35	Molar volume of gas/($\text{m}^3\cdot\text{mol}^{-1}$)	0.026
Initial fracture porosity/%	2.55	Gas constant/($\text{J}\cdot\text{mol}^{-1}\cdot\text{k}^{-1}$)	906
Initial permeability/ m^2	1.125×10^{-18}	Temperature/k	303.15

4.2. Analysis of Numerical Simulation Results

4.2.1. Mechanical Failure Characteristics Under Different Mining Intensities

(1) Analysis of vertical stress

Figure 6 depicts the vertical stress distribution throughout the advancing process of the protective layer working surface at 2, 4, and 6 m/d. As shown in Figure 6, following mining, the overlying rock stress experiences a process of destruction and rebalancing, during which the coal wall is pressed by the overlaying rock above the goaf, and stress concentrations occur within 50 m of the goaf. When the working face is progressed to 200 m, the stress distribution at various mining intensities is a roughly symmetrical trapezoid. When the advancing speed is 2 m/d, the maximum vertical stress in the stress concentration area in the advancing direction of the working face is 28.30 MPa; when the advancing speed is 4 m/d, this value increases by 0.83%, with a slight increase; when the advancing speed is 6 m/d, this value increases by 12.89%, with a significant increase. The vertical stress is higher in the stress concentration region.

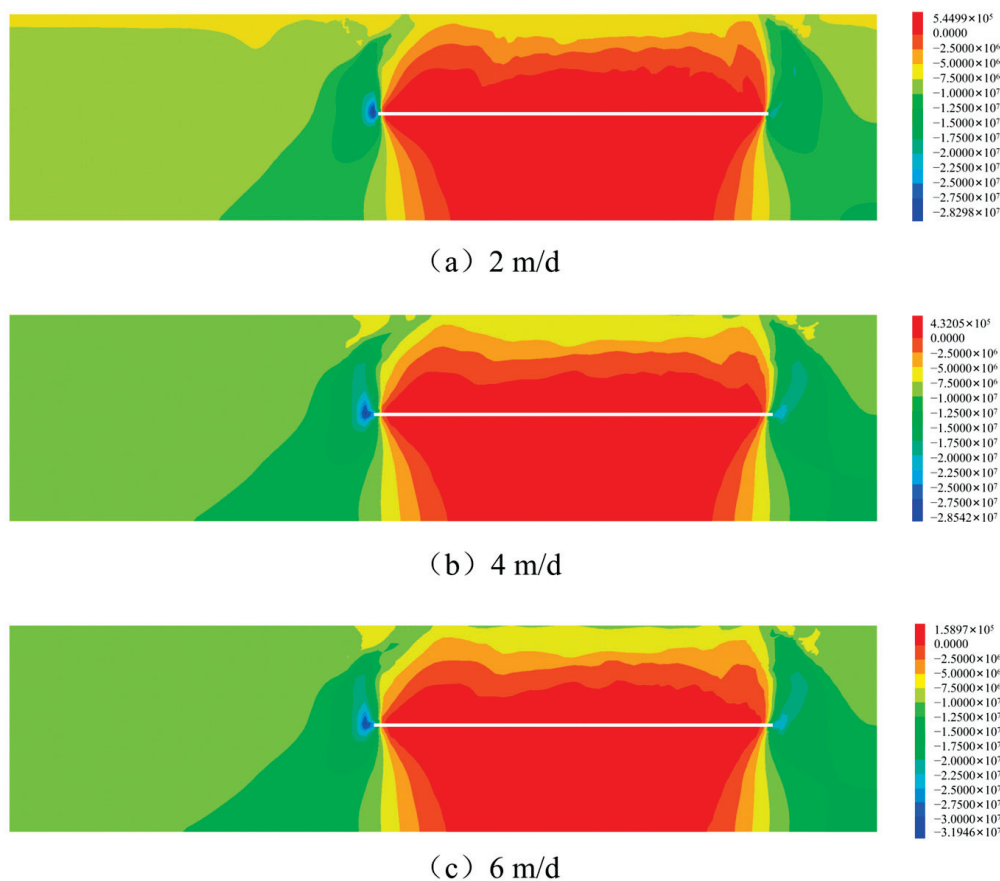


Figure 6. Variation in vertical stress under different mining intensities.

Figure 7 depicts the matching stress release rate under the bottom plate of the protective layer's working face throughout the advancing process at 2, 4, and 6 m/d. As shown in Figure 7a, when the working face of the protective layer is advanced to 180 m, the 5 m monitoring line below the floor shows obvious pressure release and pressure concentration. However, a stress release zone of coal and rock mass is formed under the goaf, but the performance varies depending on the mining intensity. When the propulsion speed is 2 m/d, the stress release rate remains steady at around 100%. As mining intensity increases, the oscillation amplitude of stress release rate approaching 100% grows greater. Figure 7b shows that the stress release rate at 10 m below the floor's bottom decreases in compari-

son to the same period, the peak value decreases from 100% to 98%, and the oscillation degree near the peak value of 6 m/d decreases, indicating that the influence on the gob's downward extension is gradually weakened. Figure 7c shows that, in the coal body of the protected layer 20 m below the mining hole, the maximum stress release rate remains steady at 97%. At the same time, it is clear that, as the mining intensity of the working face increases, the stress concentration phenomenon in the advancing direction worsens, and, similar to the change in stress release rate under the goaf, the stress concentration degree decreases with the downward movement of the monitoring line, from -97% to -33% .

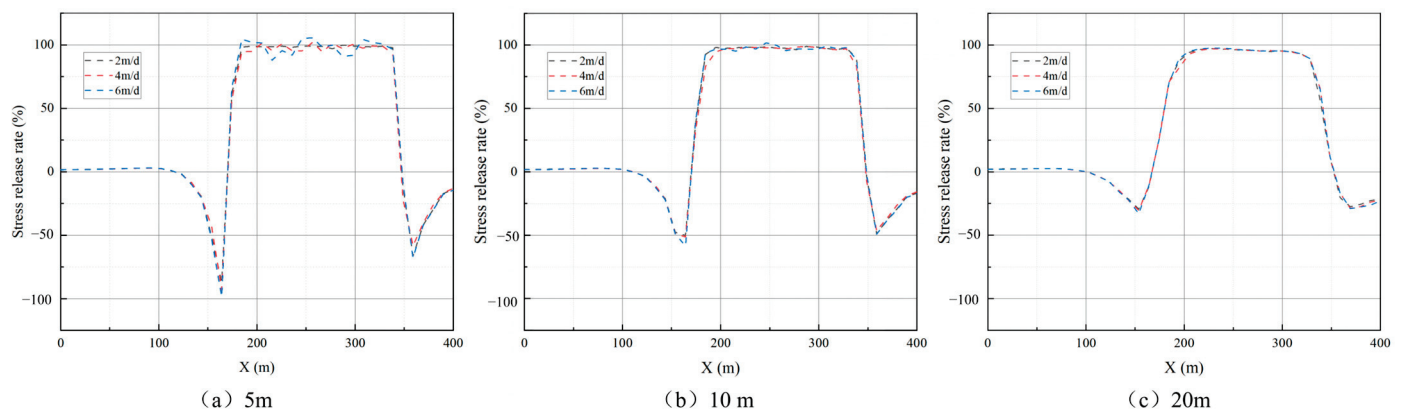


Figure 7. Stress release rate of 5 m, 10 m and 20 m below the working face of the protective layer.

Overall, mining the protective layer reduces strain on the coal and rock mass underneath it. As the mining intensity of the working face increases, so does the range and degree of this impact, and the mining influence of the seam No. 2₁ steadily improves.

(2) Analysis of plastic zone

The cloud map illustrated in Figure 8 is used to analyze the distribution of the plastic zone of coal and rock mass while advancing the protective layer working face. Figure 8 shows that, following coal seam mining, shear and tensile failure occurs in the coal and rock mass of nearby coal seams, with shear failure being the most common failure type. Mining causes very little damage to the bottom coal rock bulk, but the top coal rock layers in the goaf suffer considerable damage and caving. In comparison to the higher layer, the bottom layer of coal rock is less impacted by mining, but the upper layer displays more visible damage and caving. At this point, the growth of the fracture network in the coal seam becomes exceedingly intricate; the number of fractures increases, as does the distribution pattern. Rapid propulsion causes the coal body to be subjected to high stress in a short period of time, reducing its strength and increasing the likelihood of shear and tensile failure. The porosity and fracture rate of the coal body rise as the degree of coal crushing increases, providing more routes for the rapid release of gas, and the increased number of these channels allows the gas to move from the deep of the coal seam to the gob and working face faster.

As illustrated in Figure 8a, when the working face is progressed from 2 m/d to 180 m, the deepest plastic failure depth directly below the working face can be approximately 26.3 m, and such failure extends to both sides in the seam No. 2₁. Figure 8b indicates that the plastic zone of the goaf floor has a maximum failure depth of 42.0 m. Figure 8c demonstrates that the greatest depth of plastic failure has increased marginally to 47.9 m. It demonstrates that quicker propulsion causes a bigger region of plastic deformation because it accelerates the rate of stress accumulation and release in the ground. In contrast, the slower the progress, the smaller the region of plastic deformation, since the ground stress has more opportunity to gradually dissipate. At higher propulsion rates (such as 6 m/d),

the bottom plate may be subjected to more stress, resulting in a deeper depth of failure. At slower propulsion rates, such as 2 m/d, the floor failure depth may be reduced because the stress distribution is more uniform, and the formation has more time to react.

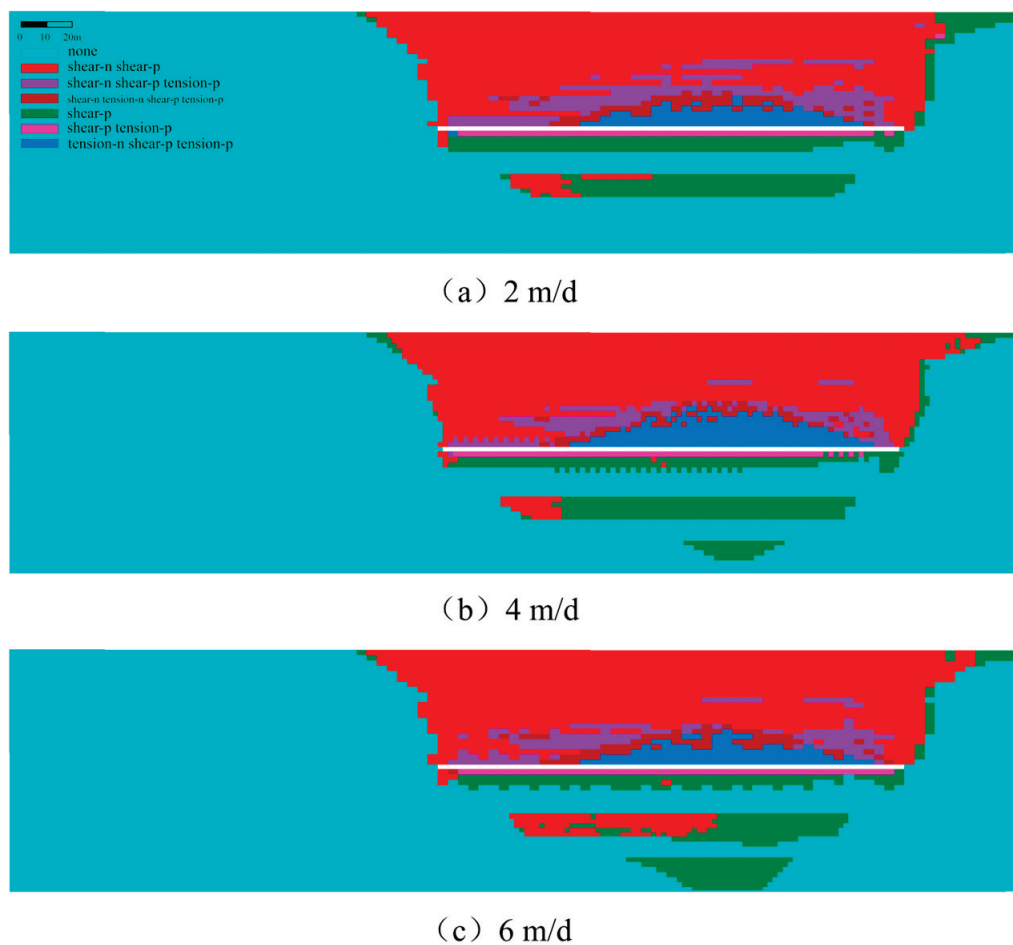


Figure 8. Variation in plastic zone under different mining intensities.

(3) Analysis of pressure-relief range of bottom plate

Figure 9 depicts the expansion and deformation rate of the seam No. 2₁ in relation to the trend and dip of the top and bottom plates during the advancing process of the protective layer working face at 2, 4, and 6 m/d. Figure 9 shows that the protected layer's expansion rate progressively increases from both sides to the middle and remains steady at more than 3‰, meeting Chinese rules on protective layer mining growth and deformation [38]. As a result, it may be concluded that mining the higher protective layer has the desired impact. Based on a critical value of 3‰, the working face has an effective discharge angle of 62°, while the dip has an effective discharge angle of 58°. The pressure-relief range on both sides is symmetric. Based on this pressure-relief range, a 3D pressure-relief model of protective layer mining is created in COMSOL for future modeling.

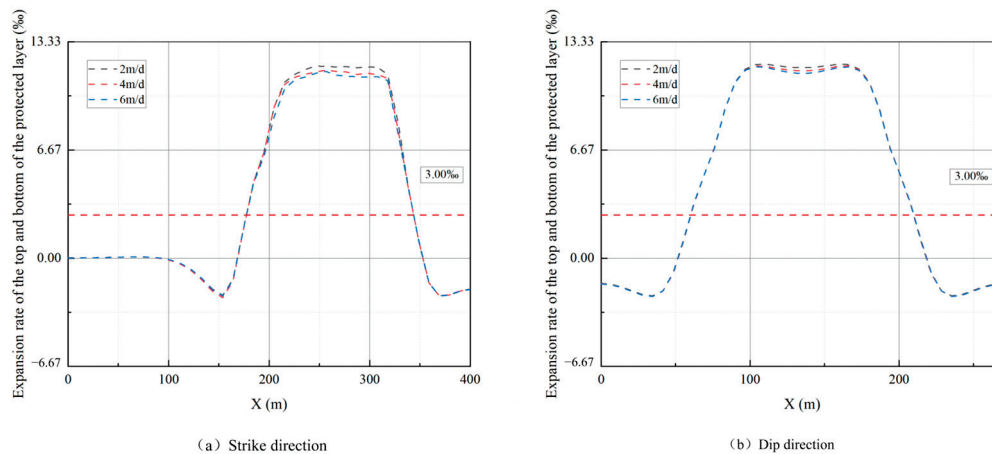


Figure 9. Curve of expansion deformation rate of the protected layer.

4.2.2. Gas Transportation Patterns

(1) Analysis of permeability

Figure 10 illustrates the permeability distribution of seam No. 2₁ during the advancement process under three mining intensities: 2, 4, and 6 m/d on the protective layer working face. As mining space expands, the depressurization of floor strata intensifies, leading to a decrease in effective stress and an extension of floor failure depth towards seam No. 2₁. Permeability rises rapidly in the plastic failure zone, peaking at the center of the goaf, indicating the strongest permeability enhancement in this area. Near the goaf, the permeability of the protected coal seam drops below pre-pressure-relief levels due to rock mass compression from stress concentration, gradually increasing towards the periphery until it matches the original coal body's permeability. As the working face progresses, the permeability spectrum of coal and rock broadens, making the enhancement effect more pronounced. At a mining intensity of 2 m/d and a distance of 180 m, the permeability of seam No. 2₁ reaches $2.45 \times 10^{-16} \text{ m}^2$, 218 times that of the original seam. At 6 m/d, the permeability values are 2.47×10^{-16} and $2.51 \times 10^{-16} \text{ m}^2$, 220 and 223 times the original, respectively, showing increases of 0.82% and 2.45%. This study demonstrates that increased mining activity elevates the coal seam's peak permeability.

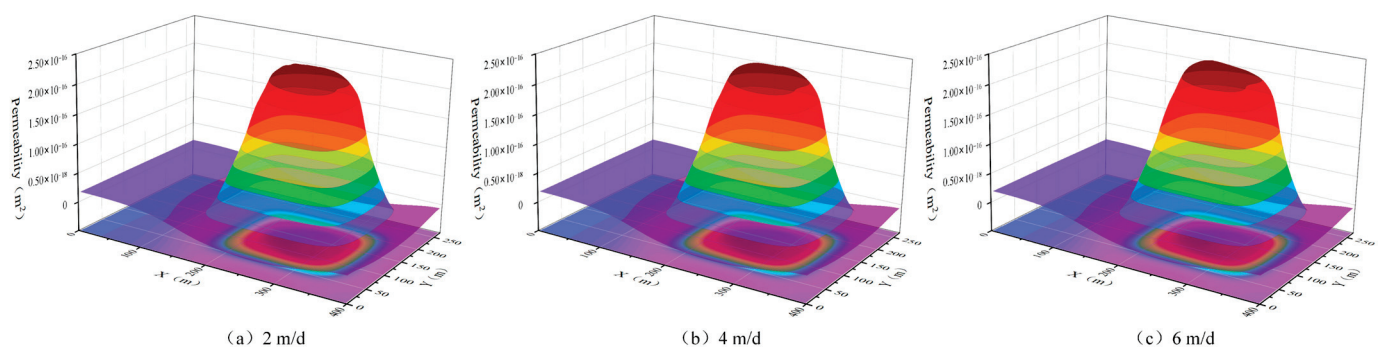


Figure 10. Variation in permeability under different mining intensities.

Figures 9 and 10 illustrate that mining operations induce significant expansion deformation in the coal body of the protected layer within the high-strength pressure-relief and reflection enhancement zone, leading to a substantial rise in permeability. This pressure relief triggers extensive gas desorption, which moves from interlayer cracks to floor cracks. The formation of a network fracture system in the floor strata enhances connectivity between the protected layer and the coal face, establishing an effective pathway for gas flow. This process not only facilitates gas release but also creates a migration route essential

for coal mine safety and gas management. The permeability of coal is closely linked to its porosity and fracture rate. Rapid advancements increase both the porosity and fracture rate of coal, thereby boosting its permeability. As permeability rises, the resistance to gas flow within the coal body decreases, while the speed and volume of gas emissions increase significantly.

(2) Transportation of gas from this coal seam

Figure 11 illustrates gas migration within the main coal seam as the protective layer working face advances at mining intensities of 2, 4, and 6 m/d. The figure presents simulated experimental results analyzing dynamic changes in gas pressure distribution across different mining intensities. The data are divided into three time intervals: 30, 60, and 90 days. Figure 11(a1–a3) reveals a significant decline in gas pressure around the goaf over time, indicating that gas pressure loss increases and the affected area expands as time progresses, assuming a constant mining distance. Figure 11(b1–b3, c1–c3) demonstrate that higher mining intensities of 4 and 6 m/d produce similar gas pressure trends but with a greater magnitude and scope of pressure reduction. This indicates that increased mining intensity enhances the unloading effect, accelerating gas migration and release. This study highlights that, as the working face advances, the coal seam experiences substantial unloading, leading to a notable decrease in gas pressure. This pressure drop creates a gradient between the coal seam and the goaf and return airway, driving gas migration and release through preferred pathways from the coal seam to these areas.

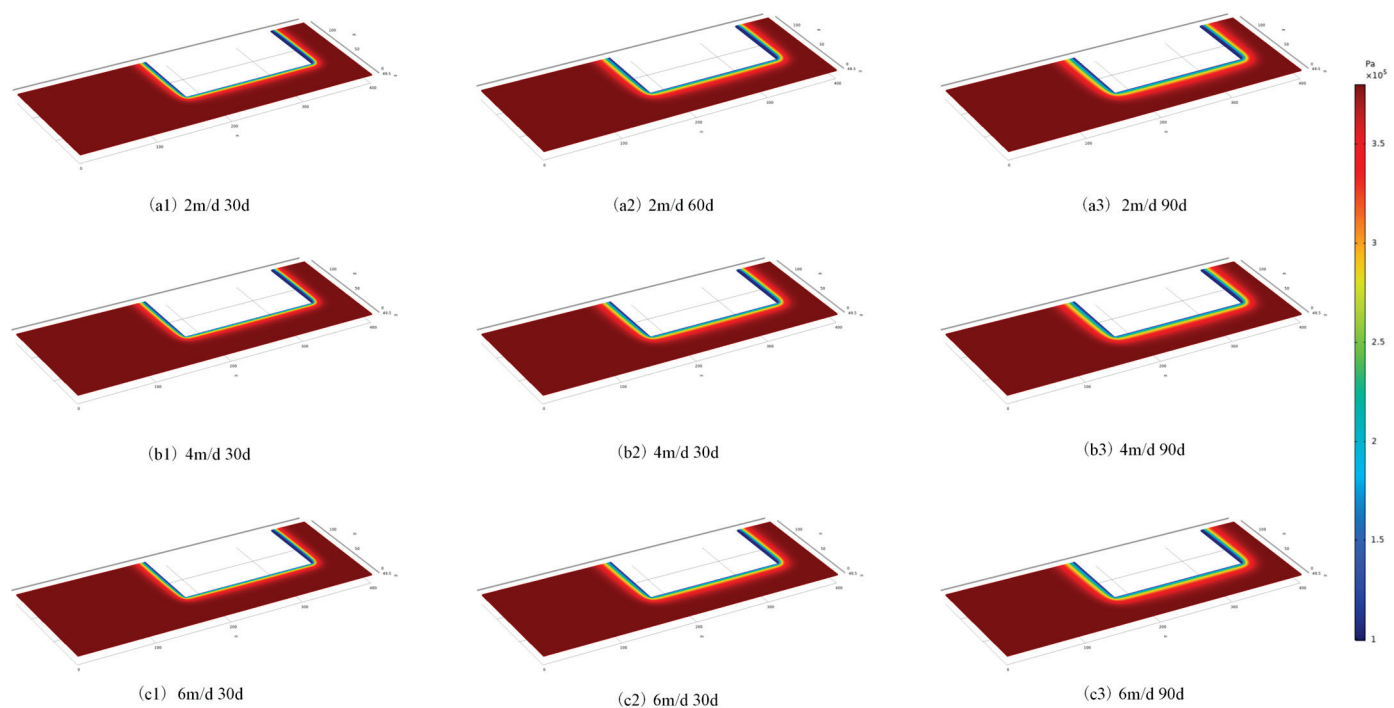


Figure 11. Characteristics of gas migration in the coal seam.

(3) Characteristics of gas transportation in neighboring seam

Figure 12 displays gas migration in the neighboring coal seam as the protective layer working face advances at three mining intensities: 2, 4, and 6 m/d. The image clearly depicts the spread of gas pressure beneath the goaf floor. The fall in gas pressure below the goaf floor is considerably negatively associated with distance from the goaf floor; the drop in gas pressure is more apparent near the goaf's core. This phenomena is primarily driven by the mining-induced unloading effect, which causes gas in the protected coal

seam to migrate upward to the goaf, resulting in a steady drop in the gas pressure of the protected layer. Furthermore, as mining activities continue, the extent of the unloading impact increasingly widens.

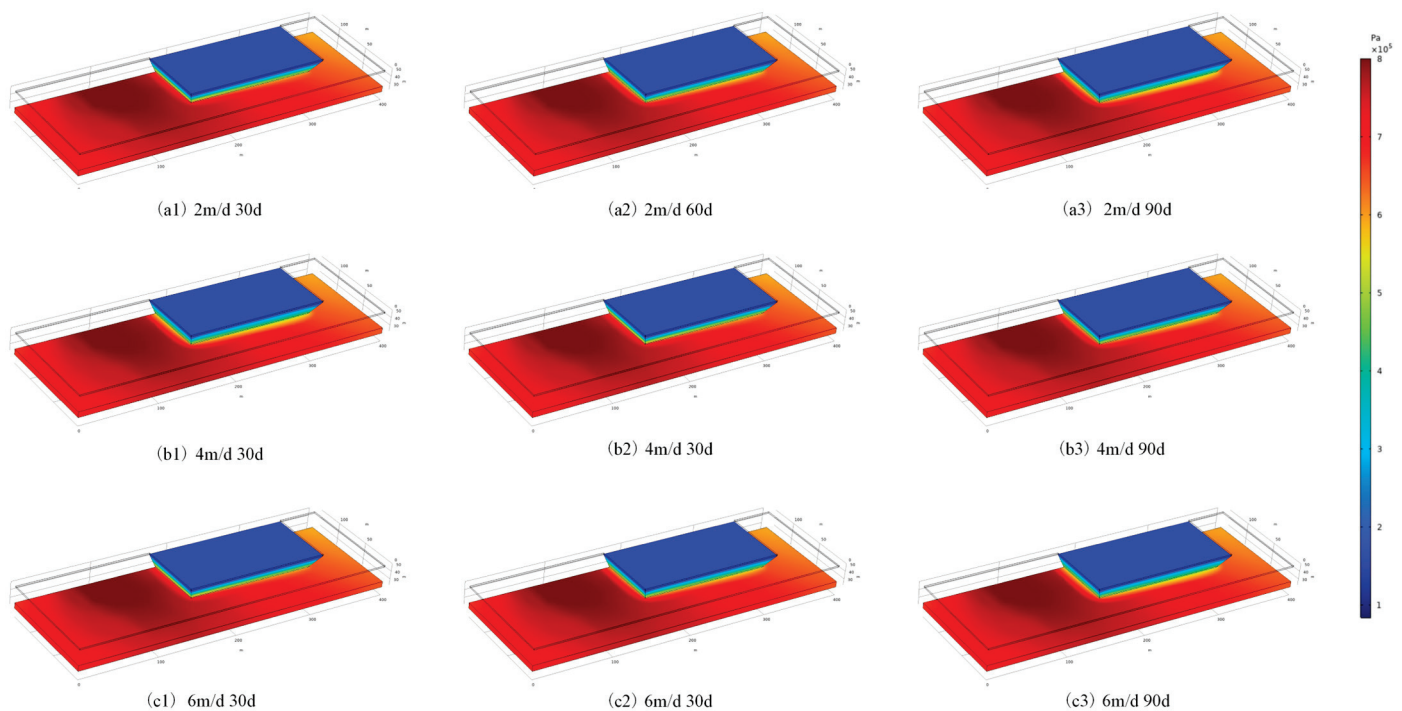


Figure 12. Characteristics of gas migration in adjacent layer.

The modeling findings reveal that the gas pressure in the gas collection area in the center of the shielded coal seam reduces dramatically as mining time increases. The findings also show that, when the mining intensity at the working face increases, the tendency of gas migration from the protected coal seam along the unloading area to the upper coal seam goaf diminishes. The increased mining intensity has a detrimental effect on gas migration and release, making it harder to relieve gas pressure in the protected layer. These findings have significant theoretical and practical implications for optimizing coal mine gas management systems and increasing coal mine safety production.

(4) Analysis of gas outflow

Figure 13 illustrates the impact of different mining intensities on gas migration. Figure 13a,b indicate that adjusting mining intensity consistently influences gas migration. Specifically, higher mining intensities enhance coal seam permeability, leading to increased gas migration rates and reduced gas pressure in both the primary coal seam and adjacent layers. This suggests that mining intensity significantly regulates gas pressure distribution within the coal seam. Figure 13c presents visual data on gas outflow, derived from a systematic analysis of gas pressure changes following the advancement of the protective layer working face at mining intensities of 2, 4, and 6 m/d. The simulation data reveal that increasing mining intensity significantly boosts gas outflow. At a mining intensity of 6 m/d, the absolute gas outflow from the neighboring layer constitutes 82.43% of the total gas output. As mining intensifies, both the total gas outflow and the proportion from the surrounding stratum increase.

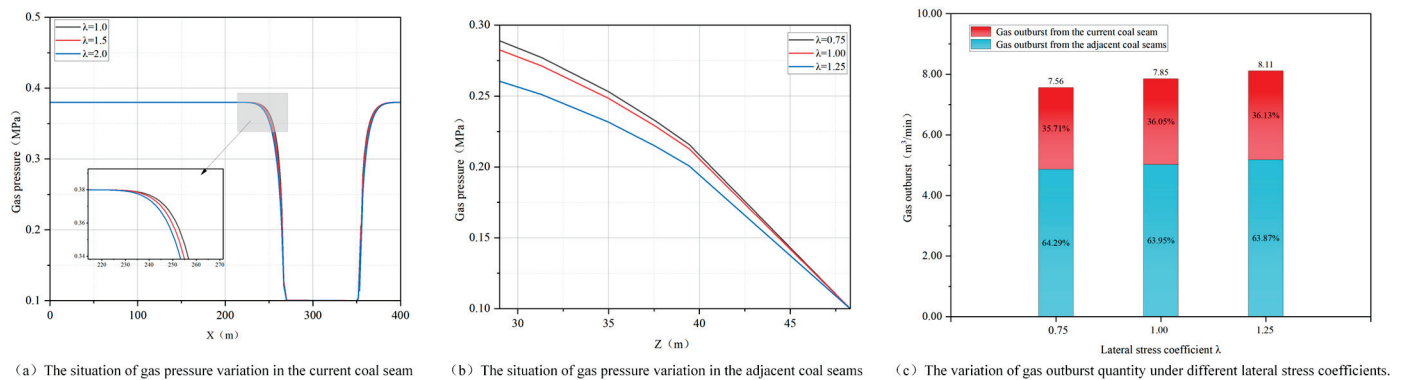


Figure 13. Gas emission quantity under different mining intensities of working face.

Coal mining accelerates the release of gas pressure within the coal body. This rapid release causes gas to desorb and flow into the goaf and working face. The rate of gas pressure decrease is directly related to mining speed; faster mining speeds result in quicker gas pressure release. Consequently, the gas pressure gradient within the coal seam intensifies, with higher pressures in deeper regions compared to the goaf and working face. This increasing pressure gradient drives gas to migrate swiftly from high-pressure to low-pressure areas, enhancing the rate and volume of gas outflow.

5. Field Analysis

5.1. Quantitative Validation Analysis of Gas Outflow Sources

In the course of employing the protective layer mining strategy, the depressurization effect leads to a substantial migration of gas into the mining airspace. As a result, real gas emissions from the working face exceed predicted values, causing gas concentration in the working face and return airflow to exceed safe limits, greatly limiting the efficient mining capacity of the upper protective layer working face [39]. Similar observations have been made during protective layer mining operations at Wangxingzhuang Mine, where pinpointing the specific sources of gas outflow has proven challenging. To address this, we used field research, carbon isotope analysis, and numerical simulations to quantify and identify the sources of gas during retreat mining of the protective layer, allowing us to deploy tailored gas management measures.

A thorough analysis identified the primary sources of gas emission at the protective seam working face in Wangxingzhuang Mine as two coal seams: the currently mined seam (2_3) and the adjacent seam (2_1). Utilizing stable carbon and hydrogen isotope techniques, we accurately and quantitatively analyzed the gas sources. Based on the principles of carbon and hydrogen isotope tracing, mixed gas samples were collected from the extraction open pipe at intervals of 140 m, 180 m, 220 m, 260 m, and 300 m along the retreating open area of the working face, yielding a total of five sets of samples. The composition and carbon and hydrogen isotopes of these samples were measured, and the proportion of gas emission sources from the borehole was determined using a two-end element calculation and analysis model. Table 5 presents the isotope measurement outcomes for end-element gases from each coal seam within the mining area and at various sampling points.

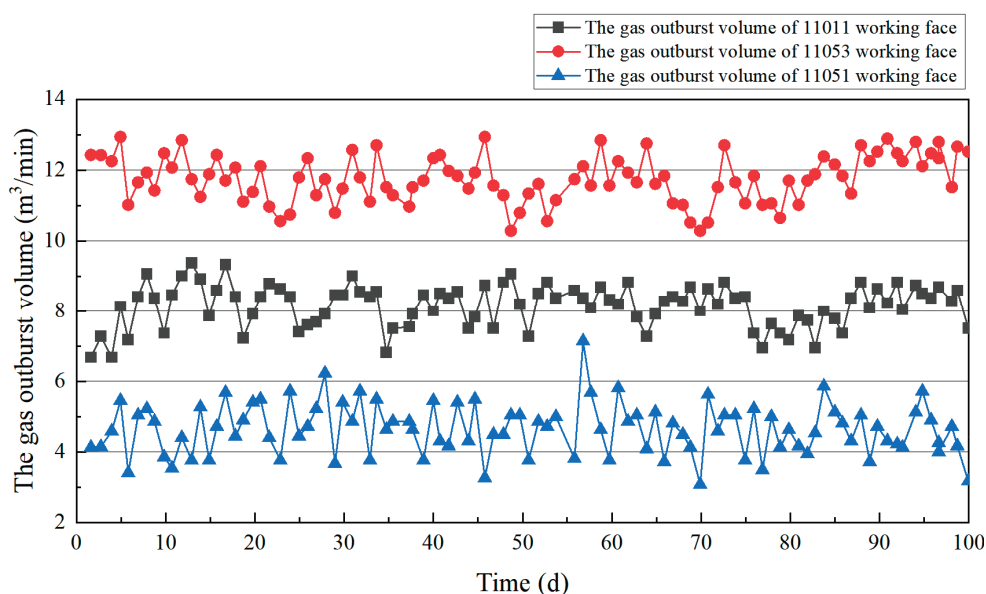
Table 5. Gas isotope determination results of each end element.

Location of Working Face	Advance Distance/m	Proportion of End Elements/%	
		The Seam No. 2 ₃	The Seam No. 2 ₁
Goaf	140	29.42	70.58
	180	28.81	71.19
	220	25.61	74.39
	260	23.32	76.68
	300	22.12	77.88

The data in Table 5 show that the proportion of gas emanating from the adjoining layer stays rather consistent as the working face advances, with the greatest proportion reaching 77.88%. This is due to the fact that the seam No. 2₃ is originally mined as a protective layer, while the seam No. 2₁ is located at the bottom plate fracture zone of the seam No. 2₃. The mining decompression effect of the seam No. 2₃ causes a considerable release of gas from the 2₁ coal seam, making it a focus point for gas management in the airspace zone during the extraction of the 2₃ protective layer.

5.2. Analysis of Past Gas Outflow Data

The temporal variation in gas outflow can be charted using data from the 11053 working face within the seam No. 2₃, the 11051 working face within the seam No. 2₁ of the protective layer, and the 11011 working face within the seam No. 2₁—which shares similar geologic conditions with the 11051 and is unaffected by the protective layer’s mining activities at Wangxingzhuang Coal Mine—as shown in Figure 14.

**Figure 14.** The measured actual gas emission volume and the proportion of each gas source.

The data analysis demonstrates that, during the retreat mining of the 11053 working face in protected seam 2₃, the stress alleviation of the surrounding rock of the bottom plate resulted in the formation of fissures and subsequent stress redistribution within the protected seam. Following stress relaxation, there was a significant desorption and release of gas from the coal seam. This gas, which flowed down the fissure channels, enhanced gas outflow from the 11053 working face while decreasing it from the 11051 working face of protected seam 2₁. The gas outflow from the 11053 working face of the 2₃ protected seam is around three times that of the 11051 working face of the 2₁ protected seam, implying that

approximately half of the gas within the protected seam has migrated into the working face. Furthermore, the gas outflow from the 11011 working face, which has a comparable burial depth and geological circumstances as the protected 11051 working face, is almost twice that of the 11051 working face. This confirms the release of a considerable amount of gas into the shielded layer from a different angle.

Figures 13 and 14 show that, when using numerical simulation techniques to assess total gas outflow, the simulated values are generally lower than the actual observed gas outflow due to the inherent limitations of the simulation methods and the simulation models' relatively small scope. Furthermore, the simulation results demonstrate that the fraction of gas outflow from the surrounding layer in the overall outflow exceeds the threshold and is near to the measured average value, indicating that the numerical model utilized is highly reliable. Meanwhile, field data reveal that, when the working face advancement distance increases, the gas outflow from the neighboring layer and its proportion in total outflow exhibit a consistent rising trend, supporting the adjacent layer's leading role in gas flow.

Based on the above analysis, it is evident that the primary goal of gas control operations should be to effectively control gas in the next layer. Under low mining intensity and the lack of unloading gas drainage mechanisms, possibly desorbable gas in the shielded layer will leak into the working face via interlayer mining-induced fissures. This technique not only increases gas outflow but also dramatically raises the safety concerns of gas explosions and outbursts. Unloading gas drainage methods can greatly limit the quantity of gas moving from the protected layer to the protective layer's operating face. Furthermore, the drainage procedure can significantly lower the gas content in the protected layer.

5.3. Measures for Extracting Decompressed Gas

This study utilizes directional long boreholes arranged longitudinally in the middle strata between the mining layer and the protected layer to precisely intercept and drain the unloading gas from the adjacent layer. This measure has advantages such as the high precision of drilling construction, a short construction period, low gas control cost, and a good drainage effect [40].

5.3.1. Calculation of Extraction Mixing Volume and Negative Pressure for Directional Long Drilling Holes

Based on the statistical evaluation of ventilation volume and gas concentration during the retreat mining of the protective layer's working face, the absolute gas outflow from the coal seam is calculated to be $3.74 \text{ m}^3/\text{min}$. This outflow significantly increases with the progression of retreat mining, reaching $9.68 \text{ m}^3/\text{min}$ in the later stages. Assuming a 75% gas interception rate for the directional long drilling holes designed for bottom plate unloading and interception, the required extraction volume is calculated to be $4.31 \text{ m}^3/\text{min}$. Selecting a 120 mm diameter for these directional long drilling holes, we apply the following Equation (9):

$$D = 0.1457\sqrt{Q/V} \quad (9)$$

where D is the inner diameter of the drill hole, 120 mm; Q is the mixed gas flow rate in the drill hole, m^3/min ; and V is the economic flow rate of gas in the gas extraction drill hole, 10 m/s.

$$Q = KQ_c / X \quad (10)$$

where K is the gas flow surplus coefficient, after the actual extraction verification take $= 1.6$; Q_c is the maximum flow rate of pure gas extraction, m^3/min ; and X is the gas concentration, %.

The calculated mixed extraction volume for a single directional long drilling hole is $6.78 \text{ m}^3/\text{min}$. Accounting for a gas concentration of 15%, the actual pure gas extraction volume per hole is determined to be $1.02 \text{ m}^3/\text{min}$. To achieve effective gas control at the mining face, a minimum of five such holes, each with a 120 mm diameter, must be constructed. For drill holes extending 300 m, the extraction branch line's negative pressure must not fall below 15 KPa to maintain gas control efficacy. Given the downward inclination of the drill holes and potential water accumulation, the negative pressure should be no less than 25 KPa to ensure effective gas extraction.

5.3.2. Calculation of Bottom Plate Destructive Depth

The destructive depth of the bottom plate is influenced by factors including mining height, inclination angle, burial depth, working face length, and the compressive strength of the bottom plate's rock layer. For the 12023 working face, which covers 170 m with a mining height of 7.0 m and an inclination angle of the coal seam of 8° , the following factors are critical:

$$h = 0.013H + 6.25 \ln(L/40) + 2.52 \ln(M/1.48) \quad (11)$$

This empirical formula is derived from statistical regression models of floor damage depth in Chinese coal mines [39], where h is the maximum destruction depth of the coal seam bottom plate (m); H is the burial depth of the coal seam (380 m); L is the length of the working face strike (170 m); and M is the mining height of the coal seam (7.0 m).

Theoretical calculations estimate the bottom plate destruction depth of seam No. 2₃ post-mining to be 16.91 m. With an average interlayer spacing of 19.3 m between the seams No. 2₃ and 2₁, the bottom plate's rock layer comprises Dazhan sandstone and mudstone. Considering lithological characteristics, drilling should optimally be arranged within the more stable Dazhan sandstone layer. Furthermore, the drilling location should fall within the damage depth and fissure zone of the upper protective layer's bottom plate. Consequently, the designed drill hole's final location is set 3 m above the top plate of the seam No. 2₁. Schematic diagrams of the directional drilling equipment, construction site, and drill hole completion are presented in Figure 15.

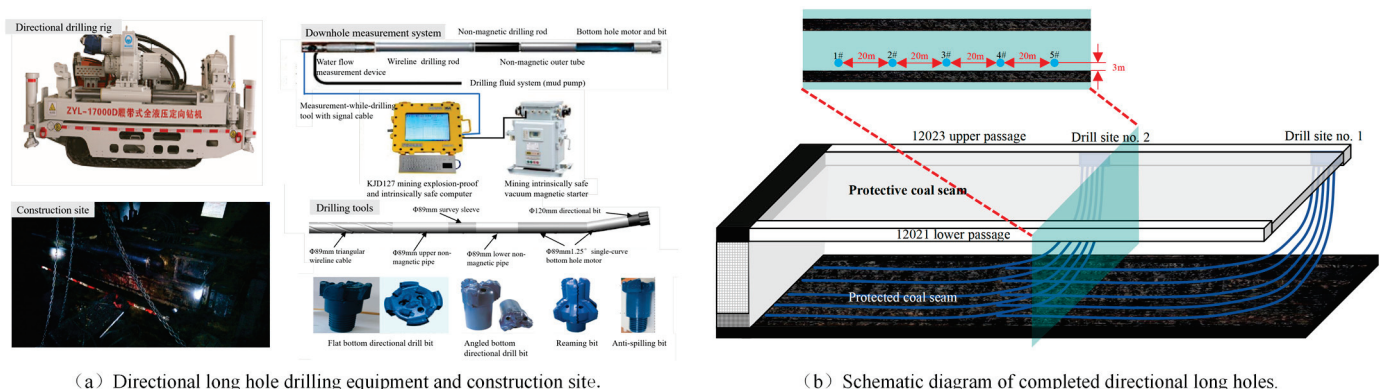


Figure 15. Directional drilling equipment, construction site, and completion plan of drilling.

5.4. Effect of Unloading Pressure Gas Extraction

The geological reserve of the seam No. 2₁ within the drilling control area of drill site 1 is 44.98 ten thousand tons, and the gas reserve is 389.09 ten thousand m³. According to the statistics of the drill site, the cumulative gas extraction volume of drill site No. 1 is 1.3540 million m³, which corresponds to the unloading gas extraction rate of 34.80%. For drill site No. 2, the geological reserve of the seam No. 2₁ within the control area of the drill hole is 43.38 ten thousand tons, and the gas reserve is 427.75 ten thousand m³. The cumulative gas extraction volume of drill site No. 2 is 175.28 ten thousand m³, and the unloading gas extraction rate is 40.95%. Further calculations reveal that the residual gas content inside the control region of drill site No. 1 is 5.64 m³/t, while the residual gas content within drill site No. 2 is 5.82 m³/t. Table 6 presents the data in full.

Table 6. Residual gas content of the seam No. 2₁.

Drill Site Number	Average Raw Gas Content (m ³ /t)	Geological Reserves (Ten Thousand Tons)	Cumulative Gas Extraction Volume (Ten Thousand m ³)	Cumulative Gas Extraction Volume (Ten Thousand m ³)	Residual Gas Content (m ³ /t)
1#	8.65	44.98	389.09	135.40	5.64
2#	9.86	43.38	427.75	175.28	5.82

Using the Wangxingzhuang Mine as a case study, the construction cost per bore-hole (including equipment and labor) is approximately CNY 174,000. For a 10-borehole system (Section 5.3.1), the total initial investment amounts to CNY 1.74 million. According to Table 6, the cumulative gas drainage volume across two drilling sites reached 3.1068 million m³. Based on China's coal mine gas utilization subsidy standard (0.3 CNY/m³), the direct revenue is estimated at CNY 932,040. Additionally, the 15% reduction in ventilation energy consumption due to lowered gas concentrations yields annual electricity savings of approximately CNY 500,000. Compared to traditional high-position drilling, the new method is less costly in terms of gas control. Although the initial investment is higher, its high extraction efficiency and long-term safety benefits significantly enhance its economic feasibility.

5.5. Changing Law of Gas Outflow Volume

Upon transitioning into the operational zones of the directional long drilling holes at drill sites No. 1 and No. 2, there is a notable decrease in both the wind-ejected gas volume and the methane concentration within the return air stream of the 12023 working face. More specifically, upon reaching a depth of 60 m into the first drilling site, the expelled gas volume was recorded to have plummeted to 2.01 m³/min, with the methane concentration concurrently reduced to 0.21%. After entering 100 m of drilling site No. 2, the air exhaust gas volume further decreased to 1.77 m³/min, and the methane concentration decreased to 0.18%. The data indicate a significant reduction in the volume of gas vented to the atmosphere as the mining face moves into the area serviced by drilling operations. Specifically, there is a minimum of a 61.79% reduction in exhaust gas volume and at least a 59.09% reduction in methane concentration. These reductions effectively address the issue of gas levels surpassing safety thresholds, which is often attributable to emissions from adjacent strata. The graphical representation of these gas outflow dynamics is depicted in Figure 16.

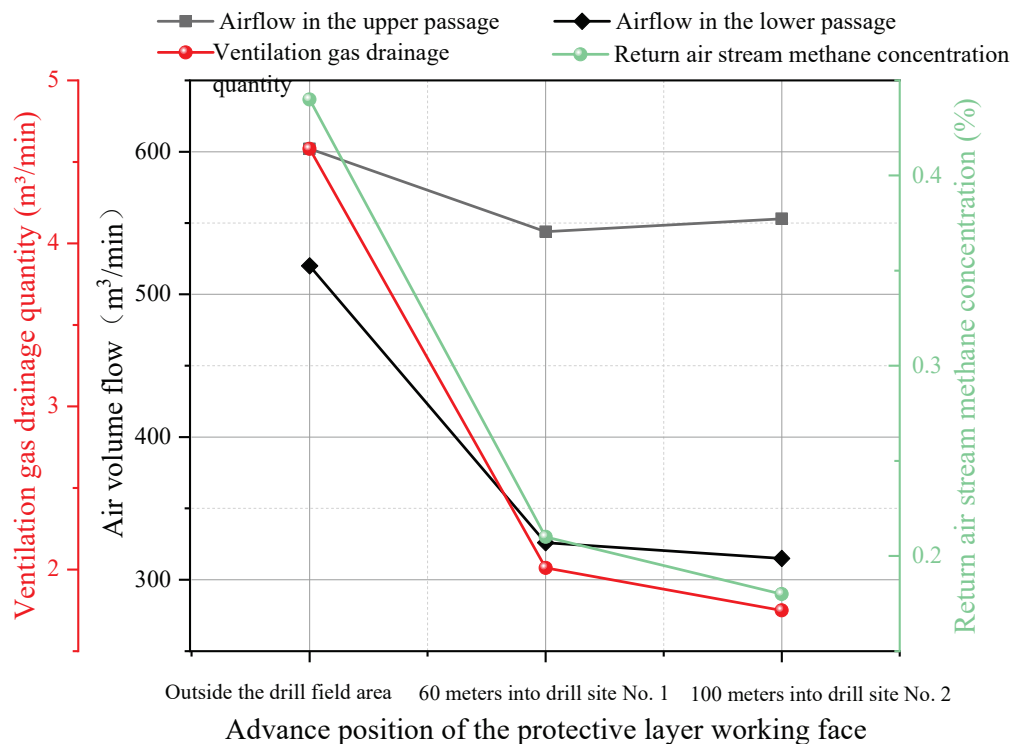


Figure 16. Changing law of gas outflow volume.

6. Conclusions

The primary reasons for the rapid gas outflow and excessive outflow volume caused by the rapid advancement of the coal seam working face are stress concentration and coal body damage due to mining, increased coal seam permeability, changes in gas pressure, the disruption of gas adsorption–desorption equilibrium, and gas accumulation and over-limit risks. These variables combine and cause abnormal increases in gas output. This paper develops a gas migration model for unloading and permeability enhancement in protective layer mining at various mining intensities, investigates the mechanism of abnormal gas outflow caused by unloading in protective layer mining, and quantitatively characterizes gas outflow to achieve precise gas control. The key findings may be summarized as follows:

(1) As the protective layer working face progresses to a position of 180 m, the vertical stress in the stress concentration area exhibits distinct changing tendencies depending on the mining intensity. When the mining intensity is 2 m/d, the maximum vertical stress in the stress concentration area is 28.30 MPa; when the mining intensity is increased to 4 m/d, the maximum value only increases by 0.83%; and when the mining intensity is increased to 6 m/d, the maximum value increases significantly by 12.89%. This finding suggests that the higher the advancing speed, the larger the vertical tension in the stress concentration area. Furthermore, on the monitoring line 5 m below the floor, when the advancement speed is 2 m/d, the stress release rate is stable at approximately 100%; but, when the mining intensity grows, so does the magnitude of the fluctuation around 100%. At 10 m below the floor, the peak value of the stress release rate drops from 100% to 98%, and, with a mining intensity of 6 m/d, the fluctuation amplitude around the peak value diminishes, showing that the downward impact from the goaf steadily reduces.

(2) Following coal seam mining, the coal and rock mass of the neighboring coal seam suffer shear and tensile damage. The damage to the lower coal and rock mass caused by mining is generally minor, but the damage and collapse of the coal and rock layers above the goaf are more severe. As mining intensity grows, so does the plastic deformation area because quick progress accelerates geostress accumulation and release. In contrast, the

slower the advancing pace, the smaller the plastic deformation area, as geostress has more time to progressively release. The greatest depth of plastic damage right below the working face can be around 47.9 m. The top and bottom plates of the protected layer experience steady deformation expansion from the edges to the center, eventually stabilizing at greater than 3%. The effective unloading angle in the striking direction of the working face is roughly determined to be 62° , while the effective unloading angle in the dip direction is 58° , with the unloading range on both sides exhibiting fundamental symmetry.

(3) As the backfill area widens, the degree of unloading of the floor rock mass rises, resulting in a lower effective stress and an expansion of the floor damage depth to the seam No. 2₁. Permeability grows dramatically in the plastic damage region, peaking in the center of the goaf. As the working face moves forward, the range of increased permeability in the coal and rock mass steadily widens, and the effect of improved permeability becomes more noticeable. When mining reaches 180 m, the maximum permeability of the seam No. 2₁ at a mining intensity of 6 m/d is $2.51 \times 10^{-16} \text{ m}^2$, which is 223 times the initial coal seam permeability. The peak value of coal seam permeability increases with increasing mining intensity.

(4) As the working face moves forward, the distribution of gas pressure near the goaf shifts downward, with the affected range steadily extending. When the working face mining intensity is increased to 4 m/d and 6 m/d, the drop in gas pressure and range marginally increase, showing that greater mining intensity has a stronger unloading impact and intensifies gas migration and release. The fall in gas pressure below the goaf floor is inversely associated with distance from the goaf floor, with a greater drop toward the center. As the working face mining intensity grows, so does the trend of gas migration along the unloading area to the upper coal seam goaf, suggesting that increased mining intensity improves gas migration and release, making it simpler to discharge gas pressure. An increase in mining intensity increases coal seam permeability and the gas migration rate and reduces gas pressure. Mining intensity adjustments have a substantial influence on gas pressure distribution. Simulations demonstrate that gas outflow rises with mining intensity. At a rate of 6 m/d, the gas outflow from the next layer accounts for 82.43%; as mining intensity increases, so does the overall outflow and fraction of the adjacent layer.

(5) A comparative analysis using a combination of on-site statistical data, carbon isotope testing, and numerical simulation methods reveals that the gas outflow and proportion from the adjacent layer increase as the advancement distance increases, conclusively indicating that the adjacent layer is the primary source of gas in the working face. The highest proportion of gas from the neighboring stratum in the goaf is 77.88%.

(6) Following the deployment of gas drainage methods utilizing directed long boreholes in the floor for interception, the drainage rates of unloaded gas reached 34.80% and 40.95%, respectively. The implementation of this measure significantly reduced gas outflow from the lower adjacent layer to the working face goaf, with the wind-discharged gas volume decreasing by at least 61.79% and the methane concentration in the working face return airflow decreasing by at least 59.09%, achieving the goal of precise gas control in multiple coal seams.

(7) Compared to existing research, this study employs multi-method validation (isotopic analysis, numerical simulation, and field statistics to conclusively identify adjacent layers as the dominant gas source. Their contribution (77.88%) exceeds the 60–70% reported by Zhou et al. [11], suggesting enhanced gas migration from adjacent layers under this mine's geological conditions. Furthermore, the efficiency of directional drilling for pressure-relief gas drainage (34.80–40.95%) aligns with field data from Guo et al. [40] (30–45%), verifying the applicability of this technology across different mining regions.

7. Discussion

In this study, we looked in depth at the anomalous gas outflow phenomena that occurred during the protective layer mining process of the 12023 working face at Wangxingzhuang Coal Mine. We discovered mechanical evolution laws, as well as unloading and permeability enhancement processes, under various mining intensities and non-uniform gas occurrence situations, using FLAC^{3D}-COMSOL linked simulations.

The findings of this study exhibit both significant connections and distinctions compared to prior empirical research on coal mine methane emissions. For instance, Yuan [19] proposed the pressure-relief gas drainage theory, suggesting that protective layer mining increases permeability by approximately 100–200 times. Our study reveals that, at a mining intensity of 6 m/d, the permeability of the seam No. 2₁ increases to 223 times its initial value, which aligns with Yuan's theoretical range. However, we further demonstrate the direct influence of mining intensity on the magnitude of permeability enhancement. Additionally, Yang et al. [24] reported through field tests that adjacent layers may contribute 70–80% of total gas emissions after protective layer mining. This finding is highly consistent with our isotopic analysis results, where adjacent layers account for up to 77.88% of gas emissions, confirming the prevalence of adjacent layers as primary gas sources. In contrast, Wang et al. [37] simulated that rapid mining advancements (>5 m/d) might limit floor failure depth to 35–40 m, whereas our study observed a plastic failure depth of 47.9 m at 6 m/d. This discrepancy may stem from the soft rock geology and high in situ stress conditions at Wangxingzhuang Mine. Such variations highlight the impact of geological heterogeneity on pressure-relief efficacy, necessitating site-specific optimization in engineering practices.

Regarding the novel contributions of this research, it pioneers the application of the integrated FLAC^{3D}-COMSOL simulation approach to investigate the mechanisms underlying gas emissions in protected seam mining operations. This groundbreaking work facilitates precise replication and a detailed quantitative assessment of gas migration dynamics. In addition, we adopted the stable carbon and hydrogen isotope method for gas traceability analysis, which improved the accuracy of gas source identification and provided a new technical means for gas management.

Compared to existing studies, the proposed method demonstrates significant advantages in multi-scale validation and dynamic modeling: Compared to Yuan [19]'s pressure-relief theory, this study reconstructs heterogeneous gas pressure fields through geostatistical inversion, avoiding errors caused by uniform assumptions. Compared to Zhou et al. [11]'s permeability enhancement model, this study, for the first time, correlates carbon isotope results with stress release rates, revealing the spatial alignment between pressure-relief ranges and gas source contributions. Compared to traditional borehole designs, the extraction efficiency of directional long boreholes is 20–30% higher than that of high-level boreholes, validating the effectiveness of precise interception strategies.

This study acknowledges potential uncertainties in numerical simulations, including simplified assumptions, parameter variability, and boundary condition limitations. To address these uncertainties, we adopted a multi-method validation approach that integrates numerical simulations, isotope tracing, and field measurements. This comprehensive validation framework enhances model reliability and provides a robust foundation for optimizing gas control strategies.

Future research should focus on reducing uncertainties in numerical simulations by incorporating more detailed geological data (e.g., fault distributions and coal seam irregularities). Advanced techniques, such as machine learning algorithms and real-time monitoring systems, could be integrated with numerical models to improve their predictive accuracy. Furthermore, more comprehensive sensitivity analyses, including variations in mining intensity and gas adsorption–desorption dynamics, could further enhance model

robustness. These improvements will deliver more accurate representations of gas migration dynamics and support the development of adaptive gas control strategies in complex mining environments. Through these efforts, we aim to provide more scientific and efficient solutions for coal mine gas hazard prevention, thereby promoting the sustainable development of the coal mining industry. Concurrently, we will strengthen the collection and analysis of field data to validate and refine our theories and models through practical applications, further improving the precision and effectiveness of gas management.

Author Contributions: X.C. (Xuexi Chen): Conceptualization; Funding acquisition; Investigation; Methodology. X.C. (Xingyu Chen): Methodology; Software; Writing-original draft. J.H.: Methodology; Software; Supervision; Investigation. J.X.: Investigation; Supervision. J.S.: Methodology; Resources. Z.Y.: Validation. All authors have read and agreed to the published version of the manuscript.

Funding: This research was funded by the National Natural Science Foundation of China (No. 52174181), the S&T Program of Hebei (No. 22375401D), the Natural Science Foundation for Youths of Hebei Province (E2024508014), and Fundamental Research Funds for the Central Universities (No. 3142024011, 3142023065, 3142023066).

Data Availability Statement: Data will be made available upon request.

Conflicts of Interest: We declare that we have no financial and personal relationships with other people or organizations that can inappropriately influence our work, there is no professional or other personal interest of any nature or kind in any product, service, and/or company that could be construed as influencing the position presented in, or the review of, the manuscript entitled, “Mechanism and quantitative traceability of anomalous outflow of unpressurized gas from protective layer mining under tectonic action”.

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Article

A Study on the Failure Characteristics of Coal–Rock Structures with Different Bursting Liabilities

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Abstract: Research on the deformation and failure behavior of coal is a key scientific issue in the study of coal–rock dynamic disaster prevention technology. It is a critical means to grasp the structural effect of coal–rock deformation and failure behavior to explore the effects of fracture structure on coal–rock deformation and failure behavior. Our experiment on the failure characteristics of coal–rock and the evolution of deformation–fracture structures before the peak stress of coal–rock primarily investigates the influence of fracture structures on its deformation and failure behavior under loading, with a focus on analyzing the size of the primary fractures. The results indicate that the influence of the primary fracture structure on the physical and mechanical properties of coal–rock varies, and the sensitivity of different properties to these structures also differs. Compared to coal–rock without outburst proneness, the fracture structure evolution of coal–rock with strong outburst proneness before failure is more intense and exhibits significant geometric nonlinearity. The size of the fracture that plays the main role in the pre-peak deformation of coal–rock with strong outburst proneness is about one-third of the size of the specimen, and it is about one-fifth of the size of the specimen for coal–rock without outburst proneness. The fracture structure affects the whole deformation process before the failure of coal–rock with strong outburst proneness, but its influence on coal–rock without outburst proneness is gradually reduced with the loading.

Keywords: fracture; structural effect; the size of main fracture; CT scanning; experimental study

1. Introduction

The occurrence of rock burst accidents is primarily due to the loss of the original equilibrium state of the coal–rock mass under the influence of mining disturbances, leading to subsequent deformation and failure [1–4]. Therefore, to reveal the mechanism of rock burst accidents, it is essential to study the controlling mechanisms of three factors: the inherent properties of coal–rock, the structure of coal–rock, and the environmental stress during the process of disaster initiation, occurrence, and development.

Coal bursting liability is the likelihood of coal to accumulate strain energy and cause impact damage. The damage and failure of coal–rock media under external load is a nonlinear evolution process far from equilibrium [5,6]. It is also the result of the heterogeneity and collective effect of the media at macro, micro and micro levels and different scales [7–9]. The failure of coal–rock media is characterized by a sudden catastrophe, which

presents uncertainty. The basic reason causing this complex feature lies in the multi-scale coupling effect [10–12]. It is generally a trans-scale evolution process, i.e., a large number of micro-damages accumulate and induce the macro catastrophe through trans-scale non-linear cascades and development. In the whole process, some disordered structure effects at the micro-scale may be strongly amplified and give rise to significant large-scale effects, which will have an important impact on the catastrophic behavior of the system. Since it is impossible to describe the disordered structure and its sensitive effects at various scales in detail, the catastrophic behavior presents uncertainty [13,14].

The structural control of coal–rock catastrophic damage means that coal–rock damage is mostly controlled by the structure of coal–rock mass. During this process, the geometric distribution and material properties of coal–rock structural planes will directly affect the failure strength of the coal–rock mass. Researchers have carried out in-depth research on the structural control of coal–rock catastrophic failure behavior [15–18]. Moreover, Zheng et al. [19,20] analyzed the relationship between the shear strength of the structural plane and that of the rock block and discussed the failure mode and strength theory of the mass. Yang et al. [21] conducted uniaxial compression tests on through joint mass and found it has three main forms of failure, i.e., split, slippage along the joint surface and the composite form of split and slippage; in addition, the variation in joint plane inclination is the main reason for the difference in the strength, deformation and failure mode of the specimen. Nasser et al. [22] carried out triaxial test research on shale and slate, respectively. The result indicates that the failure of the rock mass can be divided into three forms: slippage, shear and composite failure along the structural plane. Sun [23] proposed that the rock mass is a discontinuous structure controlled by the structural plane, and the rock mass structure controls its deformation, failure and mechanical properties. Meanwhile, the control effect of rock material is far less obvious than that of the rock mass structure from the aspect of control effect. However, with the continuous increase in mining depth and the emergence of complex working conditions during the mining engineering process, the control effect of the rock mass structural plane will weaken or disappear along with the high ground stress conditions at its depth. In this case, shear failure becomes the main form of rock mass failure. The stress control and the synergistic control effect of structure and stress will be more significant, while the mechanical behavior of the rock mass will shift from structural control to stress control [24].

In addition, regarding the fracture structure effect of coal–rock mechanical behavior, the effect of coal–rock fracture structure on the deformation and failure of materials during loading is studied, which is the core content of the fracture structure effect of coal–rock mechanical behavior. Xiao et al. [25] found that the failure strength and deformation characteristics of deep rock masses with a single fissure are strongly influenced by the structural surface under triaxial conditions. Du et al. [26] developed a combined test system to study the size effect on the shear strength of rock joints and explored its initial applications, finding that the shear strength of rock joints exhibits a significant size effect. Zhang et al. [27] concluded that the permeability of both intact and fractured coal changes significantly under cyclic loading and unloading, with fractures playing a key role in permeability evolution. Sun [28] presents an image-based multi-grid and adaptive multi-scale modeling method to simulate the trans-scale process of reinforced concrete, from evolving random meso-damage to macro-scopic cracks and eventually to catastrophic rupture, and verified the feasibility of the method. The influence of coal–rock fracture structure on coal–rock mechanical behavior is a basic scientific issue in studying the occurrence mechanism of coal–rock dynamic disaster, especially the effect of coal–rock fracture structure on the deformation and failure behavior of coal–rock media, which is the basis to explore the occurrence and development of coal–rock dynamic disaster.

To reveal the impact of fracture structures on the deformation and failure behavior of coal–rock with different impact tendencies, this paper conducted pre-peak deformation fracture structure evolution experiments. It analyzed the evolution characteristics of fracture structures during the pre-peak deformation process of coal–rock with varying impact tendencies and identified the dominant fracture scale in the pre-peak deformation process at the laboratory scale. Meanwhile, the evolution behavior of the elastic and plastic deformation of coal–rock with different fracture structures under the same loading path and the effect of fracture development on the failure characteristics of coal–rock were analyzed.

2. Experimental Procedure

2.1. Experimental Setup

The test includes two parts: (1) Uniaxial cyclic loading/unloading test. The main equipment is the TAW-2000 electro-hydraulic servo testing machine and LVDT displacement sensor by U.S.A (as shown in Figure 1). (2) The in situ CT (computed tomography) scan test of coal–rock media after unloading. The equipment is the advanced industrial CT machine produced by U.S. BIR (as shown in Figure 2). The experiment employs a high-precision 16-bit micro-focus X-ray for full-height scanning, with a total of 1000 layers, each 0.1 mm thick. Due to equipment limitations, the criteria for different loading stages remain consistent, resulting in an image resolution of 100 μm .



Figure 1. Main test equipment: (a) TAW-2000 electro-hydraulic servo testing machine; (b) LVDT displacement sensor.



Figure 2. Computer tomography scan imaging system.

Coal samples were selected from two coal mines with different impact tendencies: Bayan Gaole Coal Mine (with strong bursting liability) and Shennan'ao Coal Mine (without bursting liability). The mechanical properties of the test coal samples are as shown in Table 1. If the uniaxial compressive strength is less than 7 MPa and the elastic energy index is less than 2, the coal mine is considered to have no outburst proneness. If the uniaxial compressive strength is greater than or equal to 14 MPa and the elastic energy index is greater than or equal to 5, the coal mine is considered to have strong outburst proneness. In the laboratory, cylindrical standard specimens with dimensions of $\varnothing 50 \text{ mm} \times 100 \text{ mm}$ were prepared. Among them, 3 specimens were prepared from Bayan Gaole Coal Mine, labeled B1~B3, and 2 specimens were prepared from Shennan'ao Coal Mine, labeled S1 and S2.

Table 1. The mechanical properties of test coal samples.

Coal Mine Name	Elastic Modulus/(GPa)	Poisson's Ratio	Tensile Strength/(MPa)	Uniaxial Compressive Strength/(MPa)	Elastic Energy Index— W_{ET}	Bursting Liability
Bayan Gaole	2.216	0.28	1.158	28.568	17.603	Strong
Shennan'ao	0.802	0.38	0.56	5.929	1.82	No

This group of tests is aimed to observe the evolution behavior of the fracture structure characteristics of coal–rock media at different loading stages. The test is completed in four stages: (1) load 5 coal–rock specimens on the testing machine, and then carry out the in situ CT scan test to acquire the initial structures; (2) load the coal–rock specimens to the elastic stage and unload them, and then carry out the in situ CT scan test; (3) load the coal–rock specimens to the plastic stage and unload them, and then carry out the in situ CT scan test; and (4) load the coal–rock specimens to the near-failure stage and unload them, and then carry out the in situ CT scan test. The loading/unloading rate in the test is 100 N/s. Due to the great difference in coal–rock samples, even the coal–rock samples with the same bursting liability have different stress values at different stages during loading. Therefore, the judgment process is mainly based on the slope of the stress–strain curve, i.e., the curve is approximately a concave arc during the compaction stage, and the slope will increase rapidly after it passes through the bottom of the arc. The slope of the primary curve is read at the interface of the testing machine every 1 kN. When the slope has no obvious change more than 3 consecutive times and is close to the elastic modulus of the sample, it can be judged that the coal–rock deformation has entered the linear elastic stage; when the slope shows a downward trend more than 3 consecutive times, it can be judged that the deformation enters the plastic stage; and when the slope decreases rapidly and greatly compared to the previous stage, it can be judged that the deformation enters the near-failure stage. See Table 2 for the specific loading/unloading scheme.

Table 2. The specific plan of test loading.

No. of Specimen	Bursting Liability	Loading Rate/(N/s)	Loading Stage	Actual Load/kN
B1	Strong	100	Elastic deformation stage	10.00
			Plastic deformation stage	28.00
			Near-failure stage	34.00
B2			Elastic deformation stage	18.00
			Plastic deformation stage	39.00
B3			Near-failure stage	54.00
			Elastic deformation stage	21.00
			Plastic deformation stage	29.00
			Near-failure stage	33.00

Table 2. Cont.

No. of Specimen	Bursting Liability	Loading Rate/(N/s)	Loading Stage	Actual Load/kN
S1	No	100	Elastic deformation stage	3.00
			Plastic deformation stage	7.00
			Near-failure stage	8.00
S2			Elastic deformation stage	5.00
			Plastic deformation stage	8.00
			Near-failure stage	9.00

2.2. Data Image Processing Method

Due to the uneven density of coal and rock media and the fact that CT scan imaging is a very complex process, it is easy for the overall high gray value of some CT scan sections and the overall low gray value of some CT scan sections to appear, which directly affects the subsequent threshold segmentation results. Therefore, the CT scan image is processed by gray standardization, so that the pixels of the image are distributed in the whole gray range. At the same time, after standardization, the difference between the gray value of the target and the background is amplified, which is convenient for subsequent recognition and segmentation. Figure 3 shows the CT scan slice effect before and after processing. On this basis, the Gaussian filtering algorithm is applied to remove some special noise and artifacts, and the difference algorithm is applied to smooth the image as a whole. The image obtained by subtracting the image obtained by the former from the image obtained by the latter is the final processed result, as shown in Figure 4. It can be seen that this processing method can highlight the edge of the fracture image on the premise of removing some special noise and artifacts, and it greatly improve the image effect after processing so as to prepare for the subsequent three-dimensional reconstruction and statistical analysis of the spatial geometric characteristics of the original fracture structure.

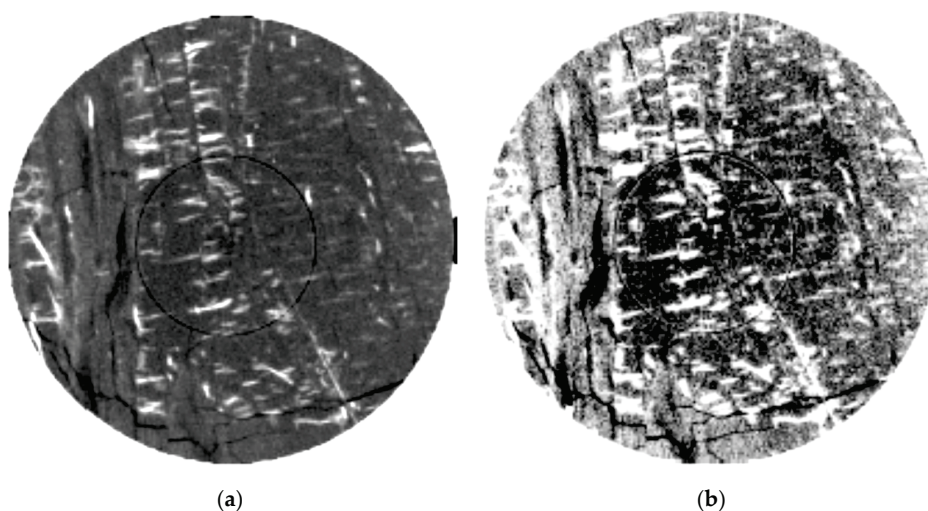


Figure 3. Effect diagram before and after the slice normalizing process: (a) original slice; (b) processed slice.

The primary fracture structures were obtained by three-dimensional reconstruction and threshold segmentation. Figures 5 and 6 show the reconstruction effect of primary fracture structures of 5 coal–rock mediums. The length and volume are selected to quantitatively describe the morphological characteristics of fracture space, in which the length is the maximum Feret diameter and the volume is the pixel volume. The fracture structure with the largest space volume and the best connectivity in the coal–rock medium is called the primary fracture structure.

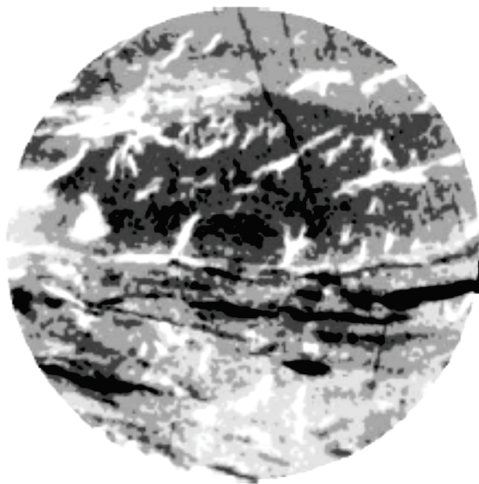


Figure 4. The effect diagram of a slice after the filtering process.

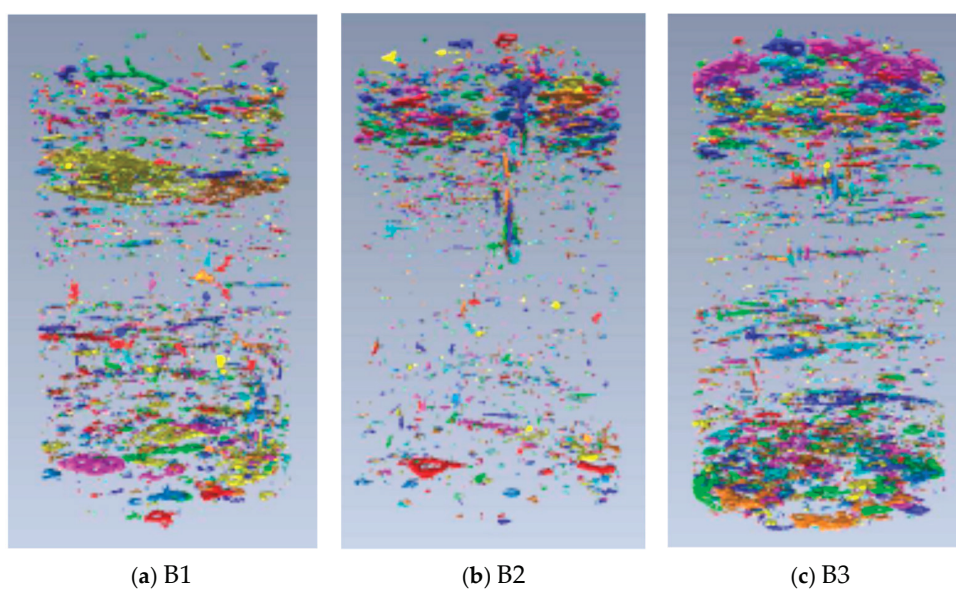


Figure 5. Three-dimensional reconstruction of primary crack structure of coal-rock with strong outburst proneness (Bayangaole Coal Mine).

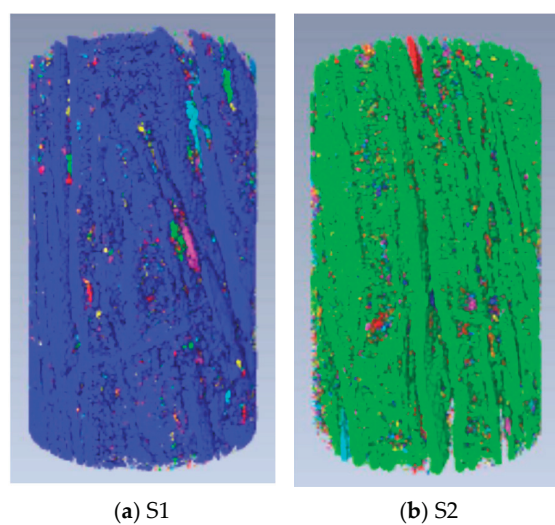


Figure 6. Three-dimensional reconstruction of primary crack structure of coal-rock without outburst proneness (Shennanao Coal Mine).

3. Results

3.1. Evolution Process of Pre-Peak Deformation Fracture Structure of Coal and Rock

Figure 7a–d respectively show the evolution behavior of two coal–rock fracture structures with different bursting liabilities. The specimen undergoes a complete loading and unloading process, forming a hysteresis loop in the stress–strain curve. The loading curve can be represented by the curve before the peak in the clockwise direction along the hysteresis loop, while the unloading process is represented by the curve after the peak. Owing to many minor damages in coal–rock, defects of a volume less than 10 mm^3 are hidden in favor of the visual observation, and the remaining large volume fractures are smoothed. For the fracture evolution behavior of coal–rock without bursting liability, it is difficult to observe the obvious changes visually; therefore, only one group is shown. But this does not mean that there is no fracture evolution behavior. Due to the high degree of the development of the primary fracture structure in its initial state, the image of subsequent crack propagation is masked, which makes it difficult to distinguish visually. However, there are obvious differences that appear later in the statistical analysis.

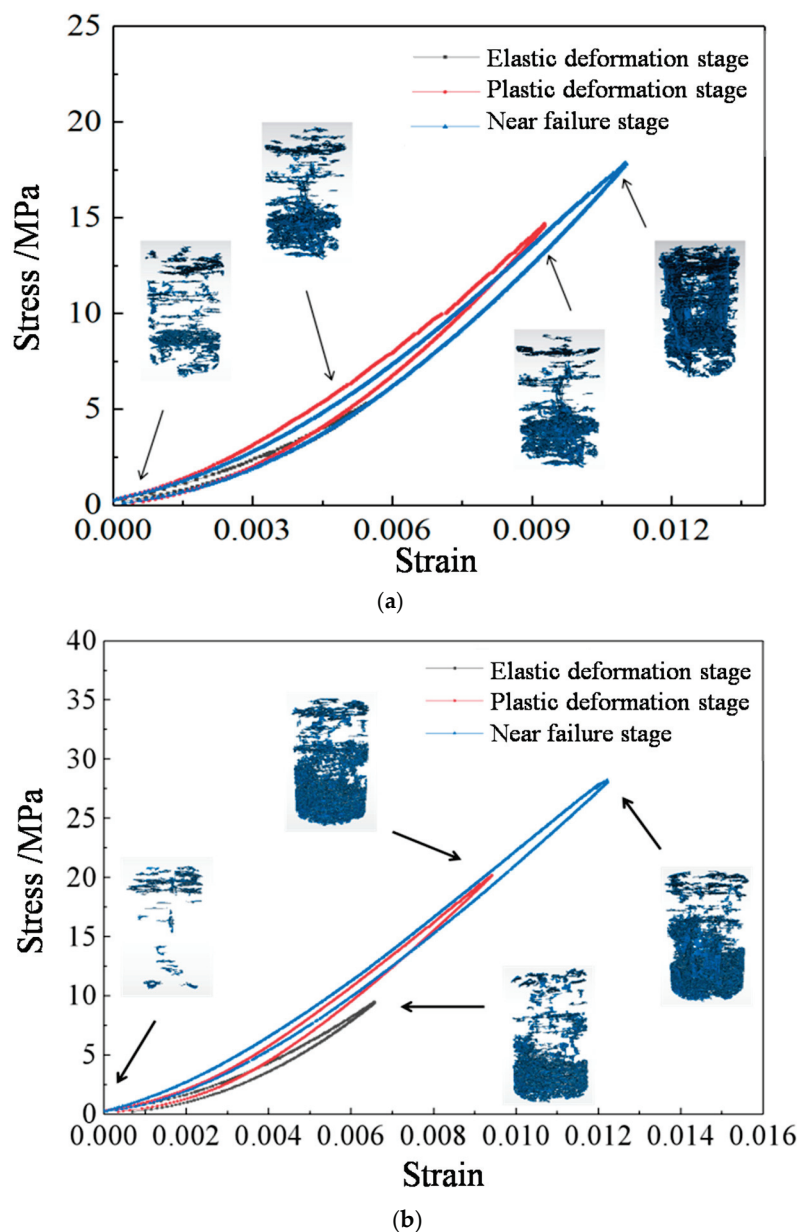


Figure 7. Cont.

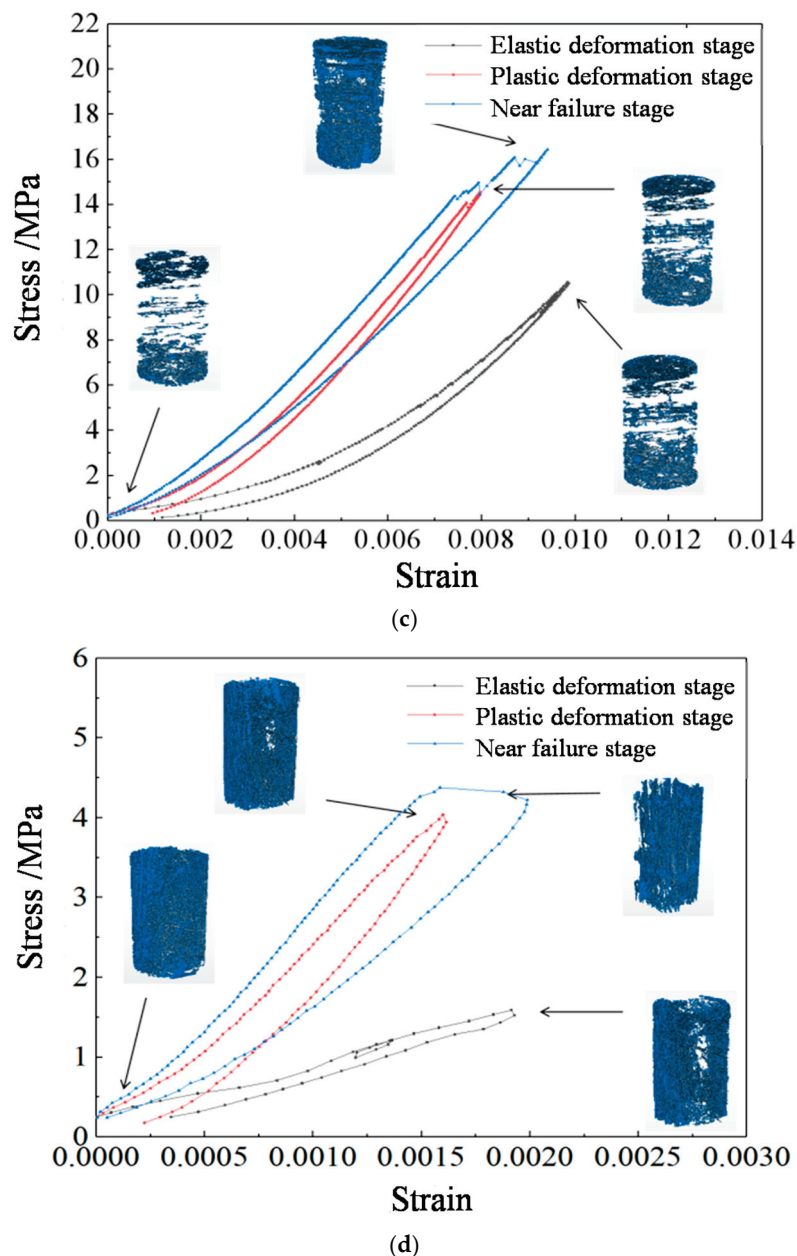


Figure 7. Fracture structure evolution diagram before peak value: (a) B1; (b) B2; (c) B3; (d) S1.

3.2. Evolution Behavior of Pre-Peak Deformation Fracture and Main Control Fracture Scale of Coal–Rock

The main control fracture refers to the fracture structure that plays a leading role in different deformation stages of the coal–rock pre-peak [29]. The coal–rock media are always accompanied by fracture structure expansion and new cracks in the process of deformation and destruction. Therefore, the dynamic evolution behavior of structure will significantly degrade the material. Some scholars [30,31] use a full stress–strain curve to classify the different stages of rock deformation and failure, crack development and propagation behavior. The reason that coal–rock exhibits different deformation and failure characteristics under loaded conditions is that the evolution of its fracture structure plays an important role.

The fracture scales at different loading stages are counted, their changes are analyzed, and the evolution coefficient η is defined to describe its scale change. In order to indicate the reason for variation, the absolute value is not taken and the sign is kept. $\eta > 0$ indicates that the number of fractures increases at this scale; $\eta < 0$ indicates that the number of

fractures decreases at this scale; $\eta = 0$ indicates that the number of fractures has no change at this scale; and $\eta = -1$ indicates that the fracture of this scale “disappears”, which may be caused by the fracture closure or by the expansion or compression of the scale fracture to other scales. The breakpoint on the curve indicates that there are new fractures at this scale.

$$\eta = \frac{p_t - p_{t-1}}{p_{t-1}} \quad (1)$$

where p_t is the number of fractures at a certain scale in the current stage, and p_{t-1} is the number of fractures at the same scale in the previous stage. Figure 8 shows the fracture scale change in coal–rock media with strong bursting liability.

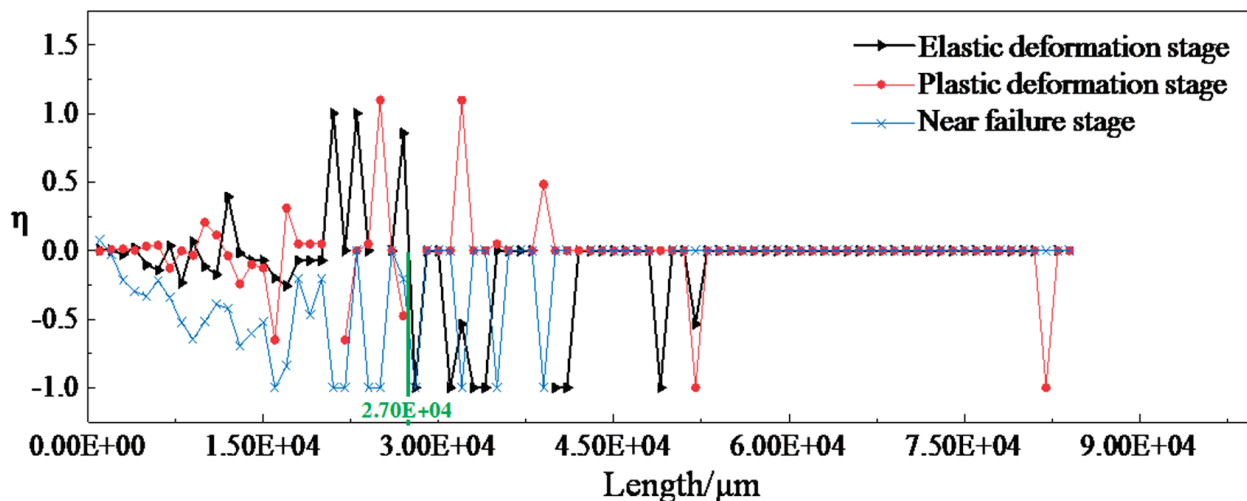


Figure 8. Fracture scale evolution of coal–rock with strong bursting liability (B1 specimen) at different stages.

It can be analyzed from the curve in Figure 8 that in the elastic deformation stage, 27,000 μm is the dividing line. The number of fractures will increase between the 0 μm and 27,000 μm scale and decrease above the 27,000 μm scale. Especially for the scale over 27,000 μm , η mostly take 0 and -1 , and there are almost no breakpoints in the curve, which indicates that the fracture evolution behavior at this scale is dominated by complete compaction. Figure 9 compares the CT scan slices of coal–rock at this stage with those in the initial state. New small fractures appear during the elastic deformation stage, so there are two reasons for the increase in fractures between the 0 μm and 27,000 μm scale: large-scale fractures are compressed to this size; and small new cracks appear. Few breakpoints occur in the whole curve, indicating that there are few new fracture scales in the elastic deformation stage, and the fracture evolution behavior is relatively continuous; therefore, the deformation characteristics of coal–rock media are relatively stable in this stage. After the deformation and failure behavior of coal–rock enters the plastic stage, the scale range of fracture evolution behavior in this stage is more concentrated, mostly between 10,000 μm and 39,000 μm . All three breakpoints in the curve appear in this scale range, i.e., the scale of new fractures is concentrated here. This is precisely the macroscopic manifestation of the self-organization characteristics of the meso organizational behavior of the coal–rock system under load, and the crack evolution will develop from disorder to order. In the elastic and plastic deformation stage, the maximum sizes of fractures are 82,000 μm and 86,000 μm , respectively. When the coal–rock enters the near-failure stage, the crack distribution scale will increase significantly compared to that in the first two stages; that is why the macro fracture begins to form. The entire curve is almost all in the negative range and tends to -1 . It indicates that almost all the fractures in the original scale “disappear” when the

coal-rock is close to failure. The three breakpoints that occurred in the curve are 42,000 μm , 49,000 μm and 104,000 μm , respectively. The 104,000 μm scale especially exceeded the full height of the specimen, which was clearly the scale of the macro fracture surface. The curve characteristic of this stage shows that in the near-failure stage of coal-rock, the continuity of deformation and failure behavior decreases, and the fracture structure begins to concentrate to several scales, especially the 104,000 μm scale. Compared to the maximum scales 82,000 μm and 86,000 μm in the elastic and plastic deformation stage, it sees a significant leap. This is strong proof of the geometrically nonlinear characteristics of the coal-rock fracture structure evolution behavior near the catastrophic moment.

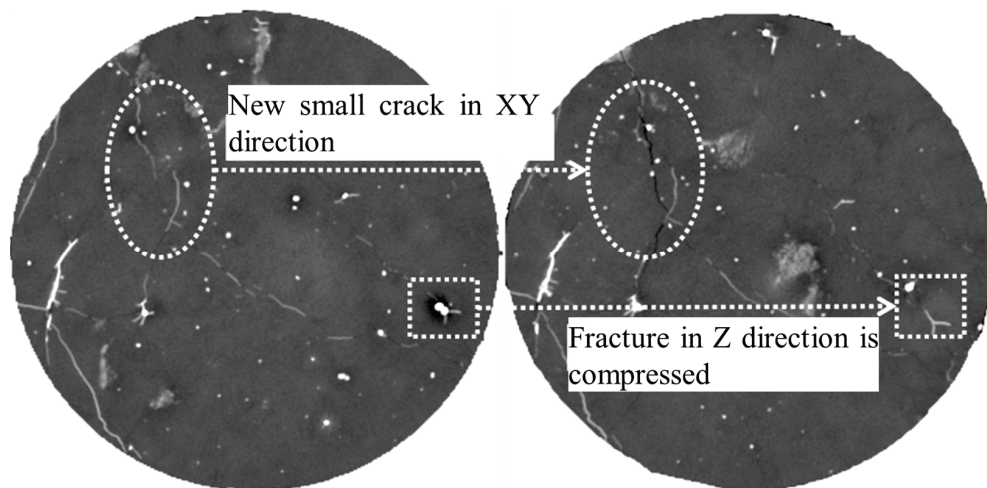


Figure 9. Comparison diagram of slices in same position of specimen (XY direction).

In the elastic deformation stage, the scale of the fracture evolution behavior of coal-rock with strong bursting liability is relatively scattered, and the overall performance is disorder. The evolution behavior of large-scale fractures is mainly characterized by compaction, and the number of them decreases, while the number of small-scale fractures increases. This may be caused by new small cracks or large-scale fractures being compressed to this scale. The dividing line of these two scales is about one-third of the specimen size. After entering the plastic stage, the self-organization characteristics of the coal-rock system come into play. The fracture evolution behavior begins to develop in order, and the fracture evolution behavior concentrates to a certain scale, i.e., approx one-third of the specimen size. In the elastic and plastic deformation stage, the fracture evolution behavior is relatively continuous, so the macro deformation behavior of coal-rock is relatively stable. In the near-failure stage, the fracture scale begins to concentrate to several points from a certain range. Here, most fractures no longer evolve, and only a few fractures play a leading role. The fracture evolution behavior exhibits geometrically nonlinear characteristics with leaps in the spatial scale, and eventually, the larger-scale fracture abruptly changes into macro breakdown. Therefore, the entire failure process of coal-rock specimens with strong bursting liability is divided into two stages: one is the catastrophe-forming stage (the elastic and plastic stage) and the other is the quasi-catastrophic stage (the near-failure stage). When it is detected that the cracks near one-third of the loaded size begin to concentrate, it is predicted that the coal-rock is at the end of the catastrophe-forming stage and is going into the quasi-catastrophic stage, which can be used as an early warning.

The coal-rock media without bursting liability and the change in its fracture scale are shown in Figure 10. It can be seen from the variation curve of S1 that the fracture distribution scale is significantly increased compared to the coal-rock with strong bursting liability. There is also a limit value (16,000 μm) for the increase and decrease in fractures

in the elastic deformation stage. η mostly takes 0 and -1 when the scale is larger than $16,000\ \mu\text{m}$. The fracture evolution behavior in the plastic deformation stage is concentrated between $8000\ \mu\text{m}$ and $22,000\ \mu\text{m}$. In the near-failure stage, the fracture evolution behavior of the coal–rock without bursting liability is different from that with strong bursting liability, and the curve shape is close to that in the plastic deformation stage. Its fracture evolution behavior is concentrated between $4000\ \mu\text{m}$ and $20,000\ \mu\text{m}$, which is slightly reduced compared to that in the plastic stage. In addition, in different deformation and failure stages, the overall fracture scale distribution range of coal–rock without bursting liability has no change. Especially when it is near the failure stage, there is no sudden change and leaps in scale. Generally, in the whole process of S1 deformation, the curve change is not as sharp as that of coal–rock with strong bursting liability, and the difference between each stage is relatively weak. From this point of view, it is more difficult to predict the loading failure of coal–rock media without bursting liability. It can be concluded from the curve that there is little evolution behavior of large-scale fractures in the whole deformation process, so it is speculated that the evolution behavior of small-scale fractures plays a leading role. The generation and expansion of small-scale cracks will make the large-scale cracks that are about to be penetrated in the initial state penetrate into the macroscopic failure surface. In the S1 specimen, the main control fracture size is less than $20,000\ \mu\text{m}$. In the process of deformation and failure, the fracture structure evolution behavior of coal–rock without bursting liability is softer than that with strong bursting liability, without sudden increases or leaps in scale or strong geometric nonlinear characteristics, and the meso structure evolution behavior is more orderly. In the whole deformation process, the fracture scale that plays a leading role in its macro failure is about $20,000\ \mu\text{m}$, i.e., one-fifth of the specimen size.

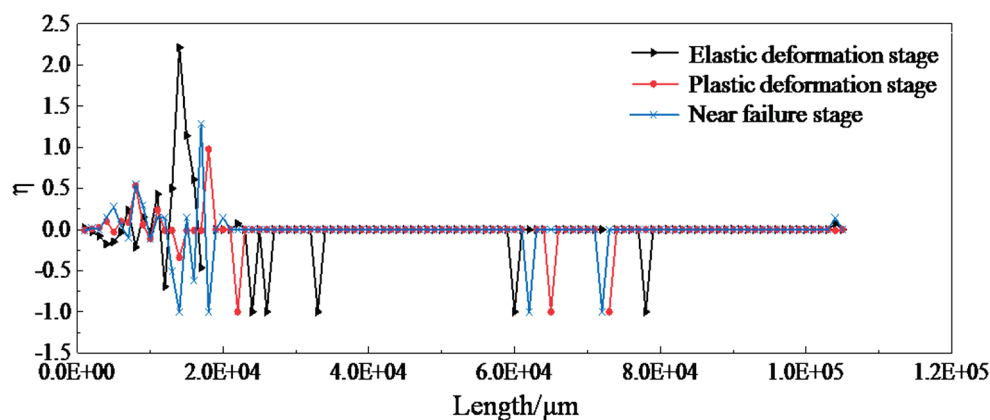


Figure 10. Fracture scale evolution of non-bursting liability coal–rock (S1 specimen) at different stages.

3.3. The Failure Characteristics Are Affected by the Fracture Development

The effect of fracture structure on the failure characteristics is also reflected in the coal–rock failure mechanism. It is known from the CT scan experiment that the integrity of coal–rock with strong bursting liability is high, and there are often fillers in the primitive fracture [32,33], so the failure is mostly from the coal matrix and fillers. Figure 11 depicts the physical graphs of coal sample failure. It can be seen from the stress–strain curve that in the initial stage of material loading, the primary fracture structure with an angle of approximately 90° to the principal stress is compacted, meaning that this part of the primitive fracture has “disappeared”. If there is no filler, the “disappeared” primitive fracture will become the primitive coal; if there is a filler, the primitive “disappeared” fracture will become the primitive rock. For the primary fracture structure with an angle of approximately 0° to the principal stress, if there is no filler, the coal matrix materials on

both sides will be affected by the tensile stress on the two surfaces of the fracture, making the fracture expand parallel to the direction of the principal stress loading, mainly forming type I cracks. If there is a filler, due to the difference in the elastic modulus between the filler and coal matrix material, the two will show inconsistent deformation behavior under load, i.e., different displacements, and the propagation fractures will mostly be type I and II cracks. Under the loading condition, the strength of coal–rock with strong bursting liability is mainly that of the media themselves, and its failure is mainly the failure of coal–rock materials.

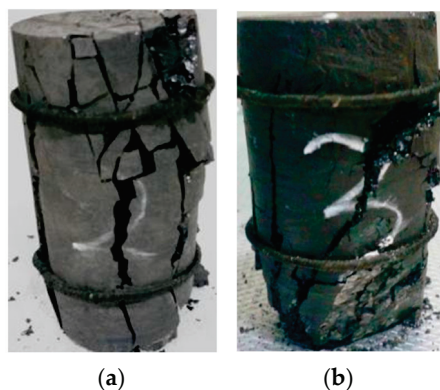


Figure 11. Physical graphs of coal sample failure: (a) B1; (b) S1.

The main fracture structure of the coal–rock without bursting liability is approx. $30\sim 60^\circ$ to the main stress, or it is exhibited, as a net structure, as a whole in the coal–rock. The failure of the material is mainly shear failure or plastic deformation. The failure mechanism of coal–rock with strong bursting liability is tension failure.

According to the principle of elasticity [34],

$$\varepsilon_3 = \frac{1}{E} [\sigma_3 - \nu(\sigma_1 + \sigma_2)] \quad (2)$$

The ultimate tensile strain of the material is assumed to be $\varepsilon_{3,0}$, and its failure criterion is

$$\sigma_3 - \nu(\sigma_1 + \sigma_2) = -E\varepsilon_{3,0} \quad (3)$$

When $\sigma_3 = \sigma_2 = 0$,

$$\varepsilon_0 = \frac{1}{E} \sigma_c \quad (4)$$

where σ_c is the uniaxial compressive strength of the material. Therefore,

$$\varepsilon_{3,0} = \nu_0 \varepsilon_0 = \nu_0 \frac{\sigma_c}{E} \quad (5)$$

$$\nu_0 = \frac{\varepsilon_{3,0}}{\varepsilon_{1,0}} \quad (6)$$

Combining with (3) and (5), then

$$\sigma_3 = \nu_0(\sigma_1 + \sigma_2 - \sigma_c) \quad (7)$$

If it is a false triaxial condition, then

$$\sigma_1 = \frac{1 - \nu_0}{\nu_0} \sigma_3 + \sigma_c \quad (8)$$

Equations (7) and (8) are the tensile fracture criteria of materials.

The failure of coal–rock media without bursting liability is mainly shear failure, the plastic deformation of rock material, or the sliding of rock material along the fracture structure. The shear failure follows the maximum shear stress theory and is brittle failure; the plastic deformation follows Mohr–Coulomb’s strength theory and is flexible failure. The former is a special case of the latter, but the two are the same in the essence. According to Mohr–Coulomb’s criterion [35], under the plane stress condition,

$$\frac{1}{2}(\sigma_1 - \sigma_3) = \frac{1}{2}(\sigma_1 + \sigma_3) \sin \varphi + c \cos \varphi \quad (9)$$

where c is the cohesion of the material and φ is the internal friction angle of the material. Equation (9) is rewritten into

$$\frac{1}{2}(\sigma_1 - \sigma_3) \frac{1}{\cos \varphi} = f \left(\frac{\sigma_1 + \sigma_3}{2} \right) + c \quad (10)$$

$$\frac{1}{2}(\sigma_1 - \sigma_3)(1 + f^2)^{1/2} = f \left(\frac{\sigma_1 + \sigma_3}{2} \right) + c \quad (11)$$

$$\sigma_1[(1 + f^2)^{1/2} - f] = \sigma_3[(1 + f^2)^{1/2} + f] + 2c \quad (12)$$

When $\sigma_3 = 0$, $\sigma_1 = \sigma_c$.

When $\sigma_1 = 0$, $\sigma_3 = -\sigma_t$.

σ_t is the unique strength of the material. When combined with Equation (12), then

$$2c = \sigma_c[(1 + f^2)^{1/2} - f] \quad (13)$$

$$2c = \sigma_t[(1 + f^2)^{1/2} + f] \quad (14)$$

When combined with Equations (13) and (14), then

$$\frac{\sigma_t}{\sigma_c} = \frac{(1 + f^2)^{1/2} - f}{(1 + f^2)^{1/2} + f} \quad (15)$$

When combined with Equations (12)–(14), then

$$\sigma_1 = \sigma_c + \frac{\sigma_c}{\sigma_t} \sigma_3 = \sigma_c + \frac{(1 + f^2)^{1/2} + f}{(1 + f^2)^{1/2} - f} \sigma_3 \quad (16)$$

When combined with Equation (8), then

$$\frac{1 - \mu_0}{\mu_0} = \frac{(1 + f^2)^{1/2} + f}{(1 + f^2)^{1/2} - f} \quad (17)$$

then

$$\mu_0 = \frac{1}{2}(1 - \sin \varphi) \quad (18)$$

The criterion of transformation from tensile failure to shear failure or plastic deformation can be obtained accordingly. When

$$\mu_0 < \frac{1}{2}(1 - \sin \varphi) \quad (19)$$

The material will be damaged in the form of tensile failure controlled by tensile strain; when

$$\mu_0 > \frac{1}{2}(1 - \sin \varphi) \quad (20)$$

In the material, shear failure or plastic deformation will occur. Because the structural characteristics of materials will affect the relationship between Poisson's ratio and the internal friction angle, it will affect the failure mechanism of coal-rock media with different bursting liabilities.

The correlation between the primary fracture structure of coal-rock and the Poisson's ratio of material is quite weak. Therefore, the effect of structure on the coal-rock failure mechanism is manifested in its effect on the internal friction angle in essence [36]. The damage degree of coal-rock media with different bursting liabilities is significantly different, so it is assumed that the complete coal-rock materials can be subjected to different degrees of damage and deterioration treatment in order to obtain the coal-rock media with different bursting liabilities. According to the nonlinear brittle damage dynamic model [37], the relationship between the internal friction angle of intact rock material and the effective internal friction angle of damaged rock material is as follows:

$$\tan \varphi^* = (1 - D^{3/2})^n \tan \varphi \quad (21)$$

where n is the material parameter. It can be seen from Figure 12 that the internal friction angle of the material will gradually decrease with the increase in damage; the greater the n , the higher the sensitivity of the internal friction angle to the damage. Therefore, when the bursting liability decreases, the failure mechanism will gradually transit from the tensile failure to the shear failure and plastic deformation.

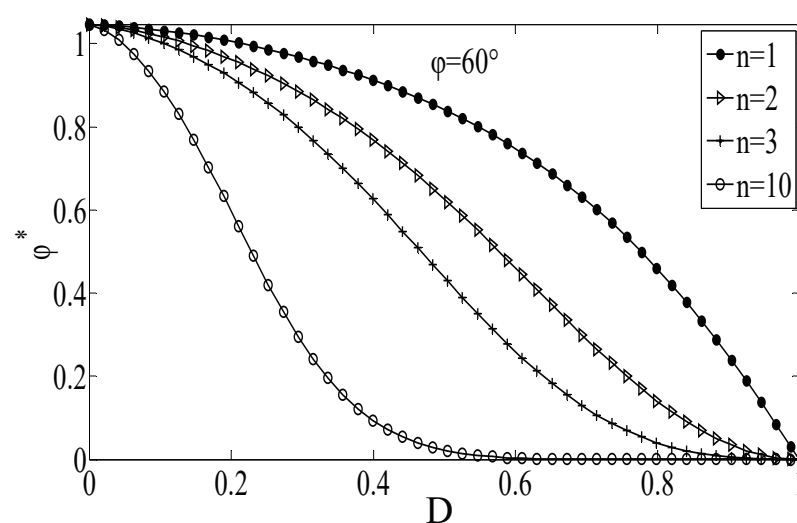


Figure 12. Effect of damage degree on internal friction angle.

4. Conclusions

Based on the test of coal-rock failure characteristics, the evolution test of coal-rock pre-peak deformation fracture structure and the deformation numerical test of coal-rock with different structural characteristics, this paper focuses on the effect of coal-rock fracture structure on its deformation and failure behavior under load and obtains the following understanding:

- (1) The fracture evolution behavior of coal-rock specimens with strong bursting liability in their elastic deformation stage exhibits disorder and the scale is relatively dispersed. The boundary is about one-third of the specimen size; after entering the plastic deformation stage, the fracture evolution behavior becomes orderly, the continuity is improved, and the scale is gradually concentrated in the range of about one-third of the specimen size. When it is observed that the fractures in this range begin to develop

concentratedly, the specimen can be considered to be in the “quasi-catastrophic” stage, and an early warning can be given.

- (2) Under the loading condition, the fracture evolution behavior of coal–rock with no bursting liability is relatively soft and orderly without a leap or sharp rise in the scale. The damage of the material is mainly controlled by the increase in small fractures, which contributes to the penetration of large fractures. The size of this small fracture is less than one-fifth of the specimen size.
- (3) The failure mechanism of coal–rock is affected by the fracture structure. Regarding the coal–rock media with different bursting liabilities, two different failure mechanisms under uniaxial loading will occur: tensile failure and shear failure/plastic deformation. The root cause lies in the relationship between Poisson’s ratio and the internal friction angle. Specifically, the property of the coal–rock fracture structure controlling the internal friction angle of the material makes the coal–rock failure mechanism transition from tensile failure to shear failure or plastic deformation as the bursting liability decreases.

Author Contributions: Conceptualization, H.L. and Y.M.; methodology, S.H.; software, Y.M.; validation, S.H. and Z.S.; formal analysis, L.L.; investigation, S.H. and Y.M.; resources, H.L.; data curation, Z.S.; writing—original draft preparation, S.H. and Y.M.; writing—review and editing, H.L.; visualization, L.L.; supervision, H.L.; project administration, H.L.; funding acquisition, H.L. All authors have read and agreed to the published version of the manuscript.

Funding: This work was supported by the Science and Technology Development Fund Project of the CCTEG China Coal Research Institute (No. 2023CX-I-05) and China Coal Research Institute Science and Technology Development Fund Project Technology Innovation Fund Class I (2023CX-I-06).

Data Availability Statement: The original contributions presented in this study are included in the article. Further inquiries can be directed to the corresponding author.

Conflicts of Interest: Author Lei Li was employed by the company China Coal Science and Technology Group Energy Technology Development Co., Ltd. The remaining authors declare that the research was conducted in the absence of any commercial or financial relationships that could be construed as a potential conflict of interest. The China Coal Science and Technology Group Energy Technology Development Co., Ltd. had no role in the design of the study; in the collection, analyses, or interpretation of data; in the writing of the manuscript, or in the decision to publish the results.

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Article

Gas Desorption Characteristics of Different Coal Ranks

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Abstract: It is crucial to understand the desorption and diffusion characteristics of coal seam gas to prevent gas disasters in coal mines. This study conducted constant-temperature gas dispersion tests on lignite, fiery coal, and anthracite to uncover the complexities of gas diffusion in coal samples under the influence of coal metamorphism. The main focus was on determining their gas desorption and diffusion characteristics and analyzing the mathematical models of gas dispersion for coal samples with varying degrees of metamorphism over different durations. The results indicated that highly metamorphosed coal samples reached the desorption limit quickly, while low-rank lignite required more time to reach the limit. The gas desorption diffusion rate was notably sensitive to the level of coal metamorphism. In the first 10 min, the gas desorption rate remained stable for lignite and fiery coal, whereas anthracite exhibited a rapid desorption rate in the initial 4 min followed by stability in the subsequent 6 min. Additionally, the effective gas diffusion coefficient showed a strong negative linear correlation with the degree of metamorphism, indicating reduced gas diffusion ability with increased metamorphism. The empirical formulas applied to anthracite showed stable correlation indices, with Formula 3 considered more suitable for describing the gas desorption processes of lignite and fiery coal.

Keywords: coalbed methane (CBM); desorption; diffusion; degree of metamorphism

1. Introduction

China's energy is characterized by its rich coal resources, poor oil resources, and little gas. Its proven coal reserves account for 33.8% of the world's coal reserves [1,2]. With the increase in mining depth, the ground stress and gas pressure of coal seams increase, and dynamic gas disasters in coal mines become more and more serious [3–5]. Studying the desorption and diffusion characteristics of gas not only has an important impact on the development of coalbed methane resources but also forms the theoretical basis for the prevention of gas disasters in coal mines. Gas desorption and adsorption are mutually reversible processes that can also be called mass transfer processes [6–8]. In the process of coal adsorption and desorption, there are some complex mechanisms of change, such as the effects of temperature and pressure, which also affect the adsorption and desorption performance of coal [9,10]. China's energy profile is defined by its abundant coal, limited oil, and scarce gas resources. The country's proven coal reserves make up 33.8% of the global total. As mining depth increases, the ground stress and gas pressure in coal seams rise, leading to more severe dynamic gas disasters in coal mines [3–5]. Research on the desorption and

diffusion characteristics of gas not only significantly impacts the development of coalbed methane resources but also forms the theoretical basis for preventing gas disasters in coal mines [6–8]. Gas desorption and adsorption are reversible mass transfer processes that play a crucial role in coal adsorption and desorption, with complex mechanisms influenced by factors such as temperature and pressure, ultimately affecting the performance of coal in adsorption and desorption processes [9,10].

Different ranks of coal possess varying structures, leading to differences in their desorption capacities. The Langmuir adsorption constant of coal, determined by analyzing pore distribution characteristics, can accurately predict gas content and pressure in coal seams [11–14]. This constant also aids in assessing the risk of coal and gas outbursts and predicting their likelihood. It serves as a vital parameter for evaluating the surface adsorption capacity of coal, influenced by internal factors such as the coal metamorphism degree, rock composition, test conditions, and parameter settings [15–17].

In coal containing gas, free and adsorbed gasses reach a dynamic equilibrium, and any changes in gas pressure, temperature, shocks, or oscillations lead to the establishment of a new equilibrium state [18,19]. According to Yang et al. [20], the initial effective diffusion coefficient during gas desorption is linked to factors like the pressure, temperature at adsorption equilibrium, and coal sample particle size. Pores play a critical role in gas adsorption, desorption, seepage, and diffusion, impacting the diffusion rate, adsorption capacity, and desorption capacity of gas, thus influencing production efficiency. Li et al. [21] discovered that as the structural damage in coal gradually increases, changes occur in pore volume and specific surface area. The ratio of large and mesoporous pores increases gradually, while the ratio of micropores decreases gradually. Upon reaching a certain level of structural damage, the volume ratios of large and mesopores undergo changes, whereas the volume ratio of micropores remains stable.

Furthermore, the structure of large and medium pores can lower the gas osmotic pressure and decrease its diffusion rate, thereby enhancing gas desorption efficiency effectively [22]. Gasparik, M. et al. [23] conducted a comprehensive study on the influence of coal rank on coalbed methane desorption, exploring changes in saturation adsorption capacity and adsorption curve during desorption. Kusuma, M.I. et al. [24] utilized molecular dynamics simulation to investigate the impact of the coal rank on coalbed methane desorption, offering theoretical insights into the interaction between adsorbent and gas and desorption rate during the process. Kang et al. [25] observed that shorter micropores in coal lead to a reduced gas migration distance within the coal body, shorter transport time, and higher initial-stage adsorption capacity. Hu et al. [26] noted that with increased coal metamorphism, the significance of large pores in gas desorption and diffusion laws diminishes, while the influence of micropores grows.

To date, researchers have compared and tested coal adsorption and desorption behaviors in various mining regions and with different coal ranks, establishing the corresponding relationships. These studies involve analyzing factors such as the coal metamorphism degree, reservoir pressure, temperature, coal rank, and water content. Among these factors, the coal rank notably impacts the adsorption and desorption capacity, with the adsorption capacity decreasing as the coal rank increases. However, limited research has focused on the gas desorption and diffusion characteristics of coals with varying metamorphism degrees, and a unified mathematical expression for gas desorption is lacking. To address this gap, desorption tests were conducted on coals with different metamorphism degrees, employing a more extensive mathematical formula to fit gas desorption curves and deriving gas release patterns for coals with varying metamorphism degrees. These findings will serve as a valuable reference for coal mine safety.

2. Methodology

2.1. Sample Preparation

The experimental samples included lignite from Anjialing Coal Mine, fiery coal from Yushuling Coal Mine, and anthracite from Xinyuan Coal Mine, as shown in Figure 1. After sampling, the coal samples to be measured were broken, crushed, and screened, respectively. Experimental coal samples with a particle size of 0.20–0.3 mm were screened and placed in the oven for 6 h, and the drying temperature was 70 °C. Then, the samples were allowed to stand and cool down to 21 °C, sealed, and stored until being used.

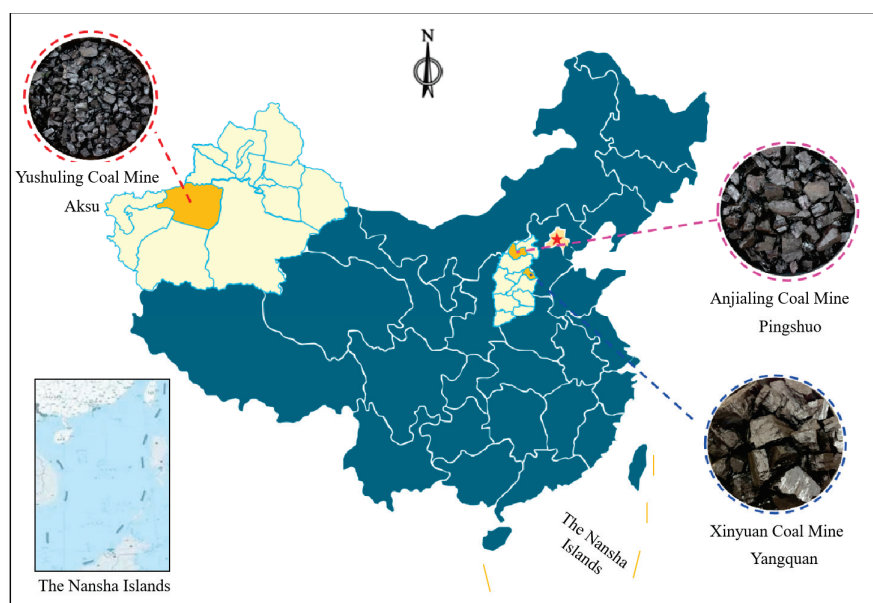


Figure 1. Geographical locations of the coal samples.

According to the parameters obtained through an industrial analysis, all experimental coal samples were reduced to a 1–3 mm particle size for use, dried in a drying oven at 70 °C for 6 h, cooled to 21 °C, and sealed until being used. The related gas parameters and maximum vitrinite reflectance of the selected coal samples are shown in Table 1 (according to GB/T212-2008, GB/T40485-2021) [27,28].

Table 1. Coal quality analysis and related gas parameters of mines.

Coal Mine	Coal Variety	$M_{ad}/\%$	$A_d/\%$	$V_{daf}/\%$	Porosity /%	$a/(m^3 \cdot t^{-1})$	b/MPa	$R_{0,max}/\%$
Anjialing	Lignitic	1.09	16.34	34.25	3.83	8.975	1.146	0.308
Yushuling	Fiery coal	1.10	5.31	41.13	3.82	13.914	1.258	0.721
Xinyuan	Anthracite	1.76	13.09	7.02	0.13	40.546	1.147	2.382

M_{ad} is moisture, A_d is ash, V_{daf} is a volatile, a and b are the Langmuir adsorption constants, and $R_{0,max}$ is the maximum vitrinite reflectance.

2.2. Test Method

2.2.1. Scanning Electron Microscopy

The pore morphology of the coal surface was determined by scanning electron microscopy (SEM), which revealed the morphology characteristics of micrometer cracks and large pores on coal surface through high-magnification optical photography. During the experiment, the appropriate amount of coal sample was uniformly glued to the special base. Due to the poor electrical conductivity of the coal, it was necessary to spray gold to prevent the accumulation of static electricity and improve the imaging contrast. After that,

an FEIQuantaTM250 scanner electron microscope was used to observe the development of pore cracks on the surface of coal samples by using a high-vacuum mode.

2.2.2. Low-Temperature Nitrogen Adsorption

Both the MIP and N₂GA methods have advantages in the characterization of coal pore structures. While the former is capable of measuring the distributions of pores of all sizes, its accuracy for minipores and micropores is low due to pressure damage. Thus, N₂GA is the better choice.

The N₂GA test was carried out with a Quantachrome NOVA-4200e specific area and pore size analyzer, which is designed for static physical adsorption–high-purity (99.999%) nitrogen adsorption–desorption in a liquid nitrogen environment (77.3 K) and is applicable to pores between 0.35 and 500 nm in diameter (according to “Pore size distribution and porosity of solid materials by mercury porosimetry and methane adsorption, Part 1: Mercury porosimetry” (GB/T 21650.1-2008) [29]. The N₂GA test was carried out with a Quantachrome NOVA-4200e specific area and pore size analyzer, which is designed for static physical adsorption–high-purity (99.999%) nitrogen adsorption–desorption in a liquid nitrogen environment (77.3 K) and is applicable to pores between 0.35 and 500 nm in diameter.

2.2.3. Gas Isothermal Adsorption and Desorption Test

This experiment was carried out using an independently developed isothermal adsorption and desorption system that consisted of six main parts: a gas supply and gas control system, adsorption and desorption system, pressure collection system, vacuum degassing system, temperature measurement system, and desorption gas collection system. The test pressure range was 0–6 MPa, as shown in Figure 2.

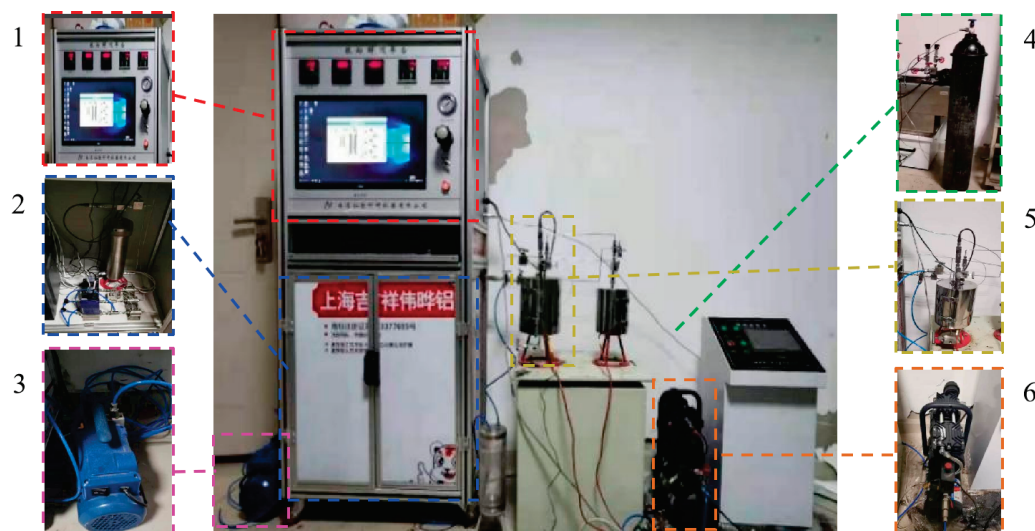


Figure 2. Isothermal adsorption desorption system. 1, computer acquisition system; 2, reference tank; 3, vacuum pump; 4, gas cylinders; 5, coal sample tank; 6, vacuum pump.

In order to study the adsorption–desorption kinetic characteristics of three coal samples under different gas pressures, 99.99% CH₄ was used to carry out the test, and three different gas pressures were set, which were 0.74 MPa, 1.06 MPa, and 1.46 MPa. In order to avoid the chance of bias in the test results, three groups of experiments were carried out in each scheme, and one group of data was selected for analysis when there was no large error in the experimental results. The specific test steps were as follows:

(1) In the preparation stage of the experiment, an air tightness test was carried out first.

(2) Then, 300 g of the coal sample was placed in an adsorption tank. The constant temperature water bath system and vacuum pump were turned on at the same time, with the degassing the sample to be tested in a constant-temperature water bath at 60 °C for 12 h and the removal the water and gas from the coal sample until the air in the coal sample reached the vacuum degree of 4 Pa.

(3) Gas was injected into the tank with the coal sample at a preset pressure, constant-pressure adsorption was maintained for 24 h, and it was ensured that the internal temperature of the adsorption tank was less than room temperature.

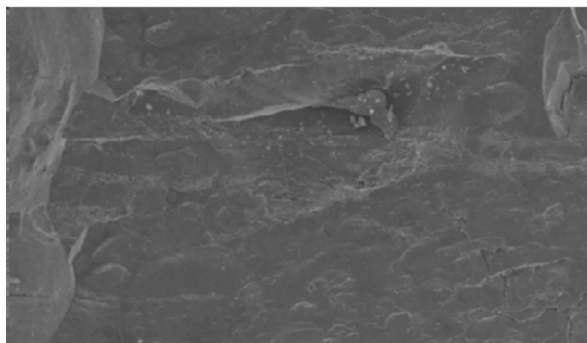
(4) Tests were conducted according to the GB/T19560-2008 standard [30], and desorption data were continuously recorded every 5 s for more than 100 min.

3. Results and Discussion

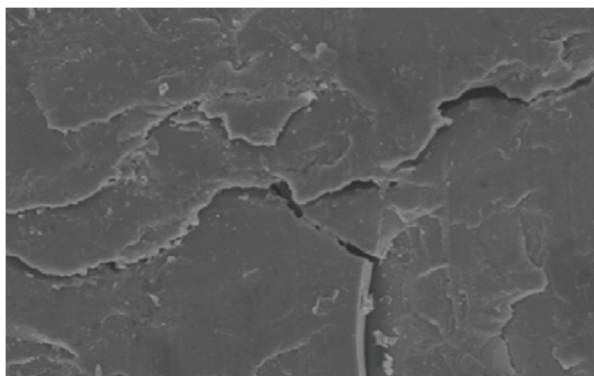
3.1. Coal Sample Pore Structure

3.1.1. SEM

It can be seen from Figure 3 that the Anjialing, Yushuling, and Xinyuan coal samples all have obvious fractures and better pore development. By comparing the scanning electron microscope images of the three coal samples, it can be found that the surface of Anjialing coal samples is the most flat, and most of the cracks are brittle tensile cracks or shear cracks in the shape of a cross. The flatness of the Yushuling coal sample is the second-highest; the fracture morphology and pore morphology are similar to that of the Shuangliu coal sample, but the mineral particle filling is more obvious. The surface of the Yushuling coal sample has a large number of scaly structures and a large number of pyrolytic pores. From the above characteristics, it can be concluded that the pore development status of experimental coal samples is basically as follows: Anjialing < Yushuling < Xinyuan. This result can be explained by the relationship between coal metamorphism and porosity.

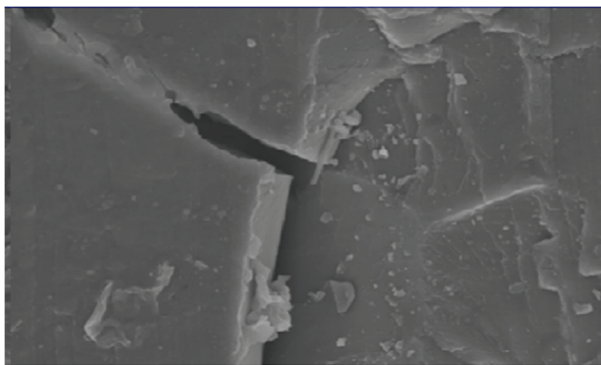


(a) Anjialing.



(b) Yushuling.

Figure 3. Cont.



(c) Xinyuan.

Figure 3. SEM results of three coal samples. (a) Anjialing; (b) Yushuling; (c) Xinyuan.

3.1.2. N₂GA

It can be seen from Figure 4 that the gas adsorption volumes of the three coal samples are 2.04 cm³/g, 3.87 cm³/g, and 5.85 cm³/g, respectively. The adsorption capacity of anthracite is higher than that of fiery coal and lignite, and the adsorption capacity of fiery coal is higher than that of lignite, indicating that the adsorption capacity of coal increases with an increase in coal metamorphism. There are hysteresis loops in all three types of coal, indicating that all coal contains open pores. Lignitic coal has the smallest hysteresis loop and is not closed, and the inflection points of adsorption and desorption curves are not obvious, and the adsorption isotherms and desorption isotherms basically coincide. This phenomenon indicates that a cylindrical shape with one end closed and a slit shape with four sides open are common in this rank of coal. Fiery coal has large hysteresis loops, many open and breathable pores, and good pore connectivity. The adsorption and desorption curves roughly coincide, and the adsorption and desorption curves basically coincide with the higher and lower sections of relative pressure, indicating that there are many semi-closed pores and open and breathable pores in its interior. The adsorption isotherm and desorption isotherm of anthracite show a separate state and do not close at the stage of relatively low relative pressure, which indicates that anthracite has a high degree of micropore development and is dominated by cylindrical micropores with one end closed, and the pore connectivity is poor.

As one of the important observation indicators of pore structure characteristics in coal, the pore size distribution can directly reflect the pore structure characteristics. The pore size distribution of coal based on BJH method analysis is shown in Table 2.

Table 2. Pore size distribution of test coal samples.

Coal Variety	Diameter/nm	Cumulative Pore Volume/(cm ³ ·g ^{−1})	Percentage/%
Lignitic	<10	0.0021	55.3%
	10–100	0.0011	28.9%
	>100	0.0006	15.8%
Fiery coal	<10	0.003875	65.3%
	10–100	0.001205	20%
	>100	0.0000860	14.7%
Anthracite	<10	0.0079	80%
	10–100	0.00131	13.3%
	>100	0.000665	6.3%

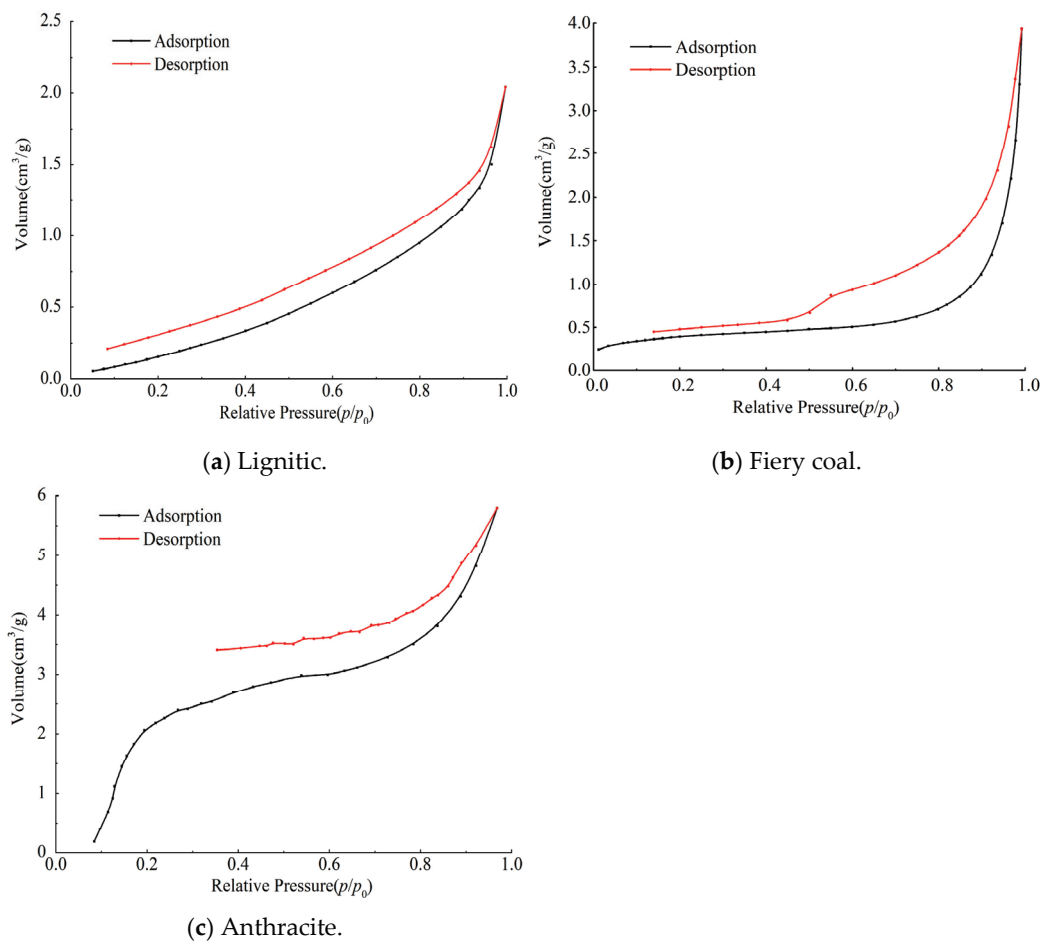


Figure 4. N₂GA adsorption and desorption isotherms of coal samples. (a) Lignitic; (b) Fiery coal; (c) Anthracite.

3.2. Influence of the Degree of Metamorphism on the Amount of Gas Desorption in Coal

Figure 5 presents the Q-t isothermal gas desorption curve of coal samples with different metamorphic degrees at various gas equilibrium pressures over a 100 min period. It is evident that, under consistent external environmental conditions, the cumulative gas desorption from coal increased gradually as the desorption time was extended. Once reaching a certain threshold, the cumulative gas desorption showed minor fluctuations around a stable value. Notably, regardless of the desorption time, the curves obtained under high pressure consistently exceeded those obtained under low pressure, indicating that a higher equilibrium pressure correlated with greater cumulative gas desorption.

During the initial phase of gas desorption, the amount of desorbed gas increased with the degree of coal metamorphism. Highly metamorphic coal samples exhibited a more pronounced increase in gas desorption. For lignite, gas desorption increased with rising pressure. The first 10 min of desorption displayed the highest gas desorption amount and rate, representing a rapid desorption stage. Subsequently, the desorption rate stabilized gradually, and gas desorption showed a slow increase, marking the onset of a stable desorption stage at around 80 min. Similarly, fiery coal showed increased gas desorption with pressure increments. The highest desorption amount occurred within the first 5 min, accompanied by a relatively high desorption rate, indicating the rapid desorption stage. Beyond 5 min, while the desorption amount continued to increase gradually, the desorption rate slightly decreased, signaling the beginning of a stable desorption stage. Anthracite's desorption capacity also varied with pressure changes, increasing as the pressure rose. The most substantial gas desorption took place within the initial 10 min, with the highest

desorption rate, representing the rapid desorption stage. After 10 min, the desorption rate stabilized gradually, and gas desorption increased slowly, entering a stable desorption stage. These observations provide valuable insights into the desorption characteristics of different coal samples under varying pressures, delineating distinct stages of rapid and stable desorption.

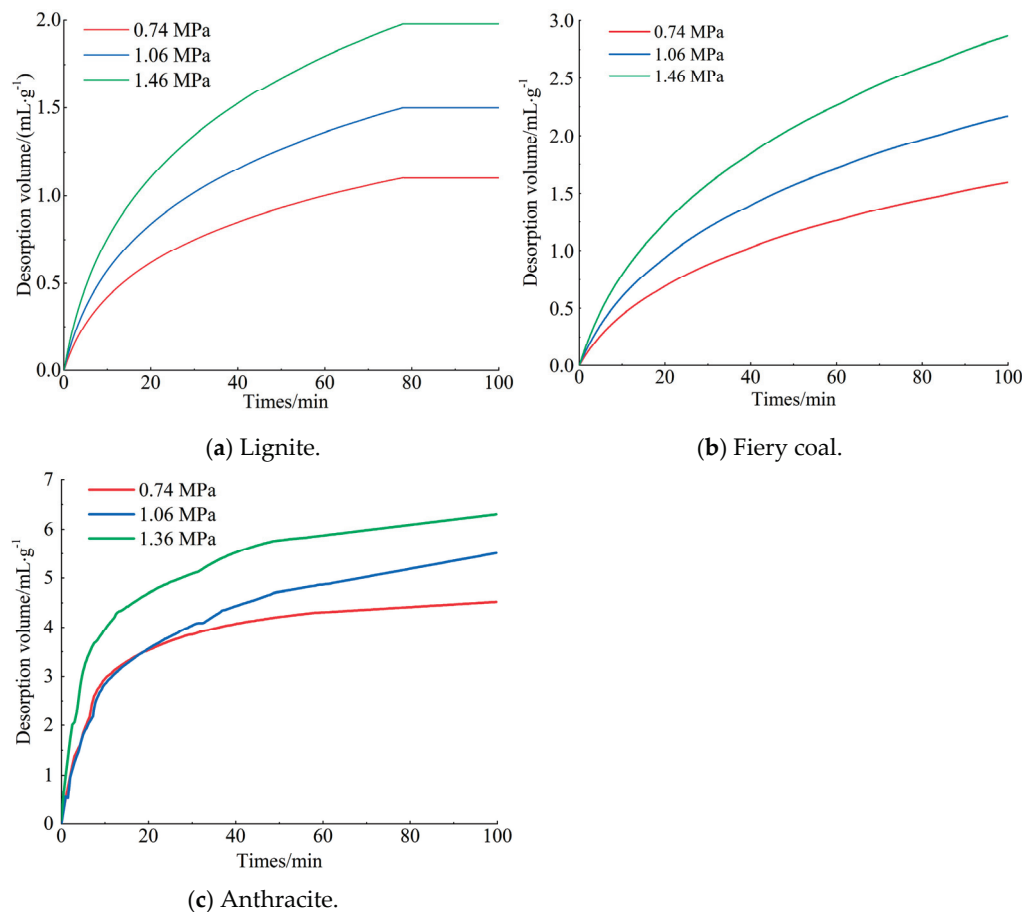


Figure 5. Coal desorption curve. (a) Lignite; (b) Fiery coal; (c) Anthracite.

The most significant gas desorption characteristics of the coal occurred within the initial 3 min. Considering the degrees of metamorphism, the cumulative desorption amounts of the three coal samples (lignite, fiery coal, and anthracite) within the first 10 min represented 76.1%, 45.2%, and 36.7% of the total desorption, respectively. This indicates that coal samples with higher metamorphic degrees reached the limiting value comparatively quickly, while low-rank lignite required a longer time to reach the limit. However, with prolonged desorption time, the gas desorption rate of highly metamorphic coal samples showed a progressively faster decline, leading to a subsequent deceleration in the desorption rate.

3.3. Effect of the Metamorphic Degree on the Gas Desorption Rate of Coal

The changes in the desorption rates of the three coals over time are depicted in Figure 6. It is evident that the trends in the changes in desorption velocities differed among the three coal ranks. However, the initial gradient of the curve under high pressure was greater than that under low pressure, indicating a higher initial gas desorption velocity under high equilibrium pressure compared to low equilibrium pressure. Additionally, the maximum gas desorption rate under different pressures and the rate of decline in the desorption rate varied with the degree of coal metamorphism. Among the three coal samples, anthracite,

which had the highest coal rank, exhibited the smallest initial gas desorption velocity when the equilibrium pressure ranged from 1.36 to 1.46 MPa. However, as the desorption process progressed, its desorption velocity decreased and became the slowest within the same time frame.

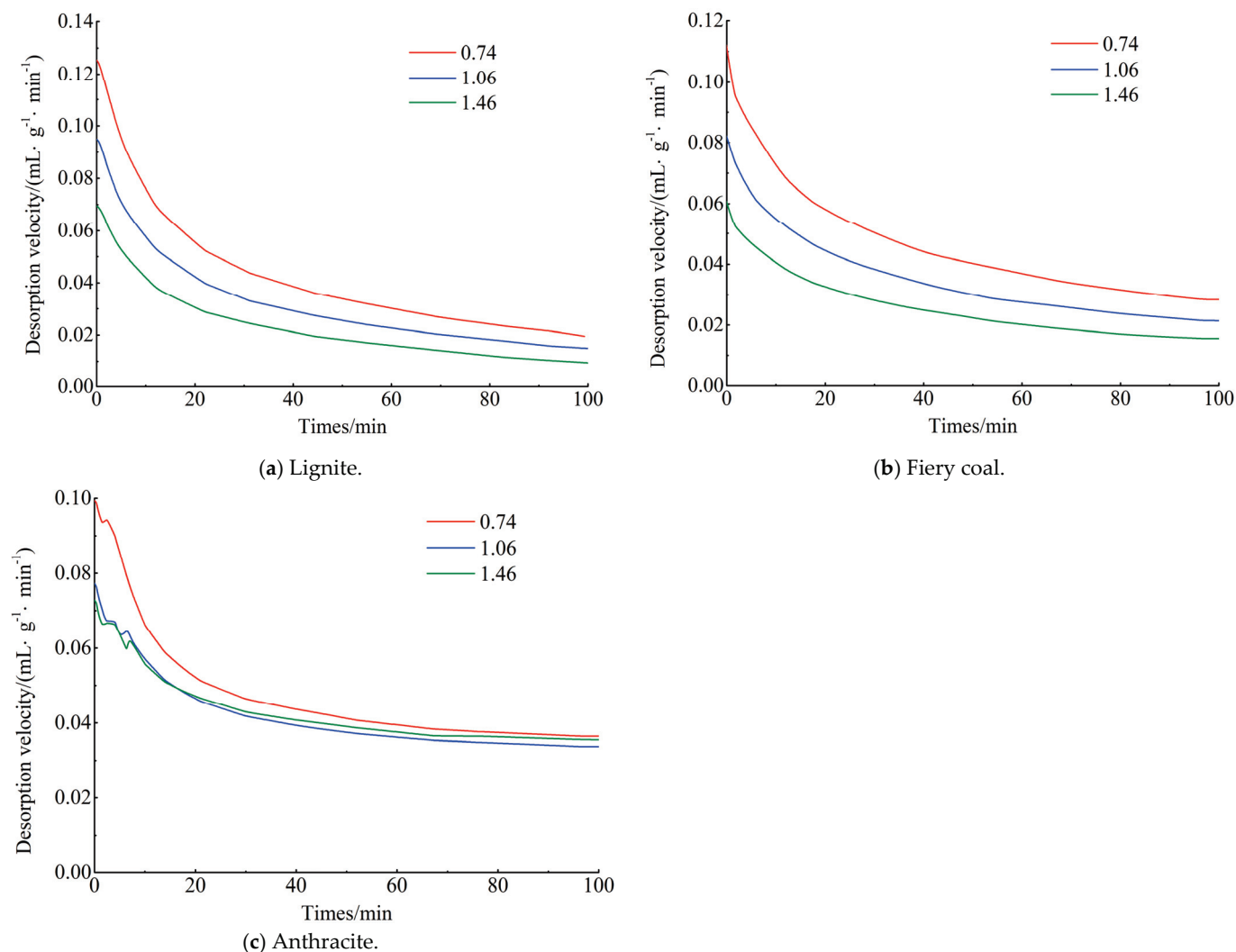


Figure 6. Changes in the desorption rate of coal under various gas equilibrium pressures are examined. (a) Lignite; (b) Fiery coal; (c) Anthracite.

The initial desorption rate of the low metamorphic fiery coal was low, and it gradually decreased as the desorption process continued. Among the three coal samples, lignite, with the lowest degree of metamorphism, exhibited the highest initial gas desorption rate, but it also experienced the fastest subsequent decline. Specifically, when the equilibrium pressure was 1.06 MPa, low-rank lignite displayed the fastest initial gas desorption rate, which swiftly decreased as the desorption progressed. On the other hand, gassy coal, with a metamorphic degree intermediate to the other coal samples, demonstrated a relatively faster initial gas desorption rate, which gradually decreased over time. Anthracite had the lowest initial gas desorption rate, and its desorption speed decreased the slowest over the same duration.

When the equilibrium pressure was 0.74 MPa, the intermediate metamorphic fiery coal exhibited the smallest initial desorption rate. As the desorption process continued, the desorption rate slowly decreased over the same duration. Remarkably, the initial desorption rate of anthracite, with a high degree of metamorphism, did not differ greatly

from that of lignite, which had the lowest degree of metamorphism. However, as the desorption process continued, anthracite experienced the smallest decrease in desorption rate over the same duration, while lignite exhibited the greatest decline.

Data were collected at half-minute intervals during the initial 10 min, and a curve was plotted based on the difference between the former and latter groups. The ratio of the former group's desorption rate to the latter group's is presented in Figures 7 and 8. It is evident from these Figures that the desorption rate curves for the three experimental coals exhibited distinct trends. The difference in desorption rate within the first 10 min under varying pressures was more pronounced for lignite compared to fiery coal and anthracite, implying a greater decline in desorption rate for lignite. Furthermore, the difference in desorption rate for lignite within the initial 10 min showed no significant variation between the first 4 min and the last 6 min, while the difference in desorption rate for fiery coal did not significantly differ from that of the last 6 min. The linear correlation between the ratio of differences in the desorption rate within the first 4 min was stronger than that of the last 6 min, and the linear correlation coefficient increased with the increase in gas pressure.

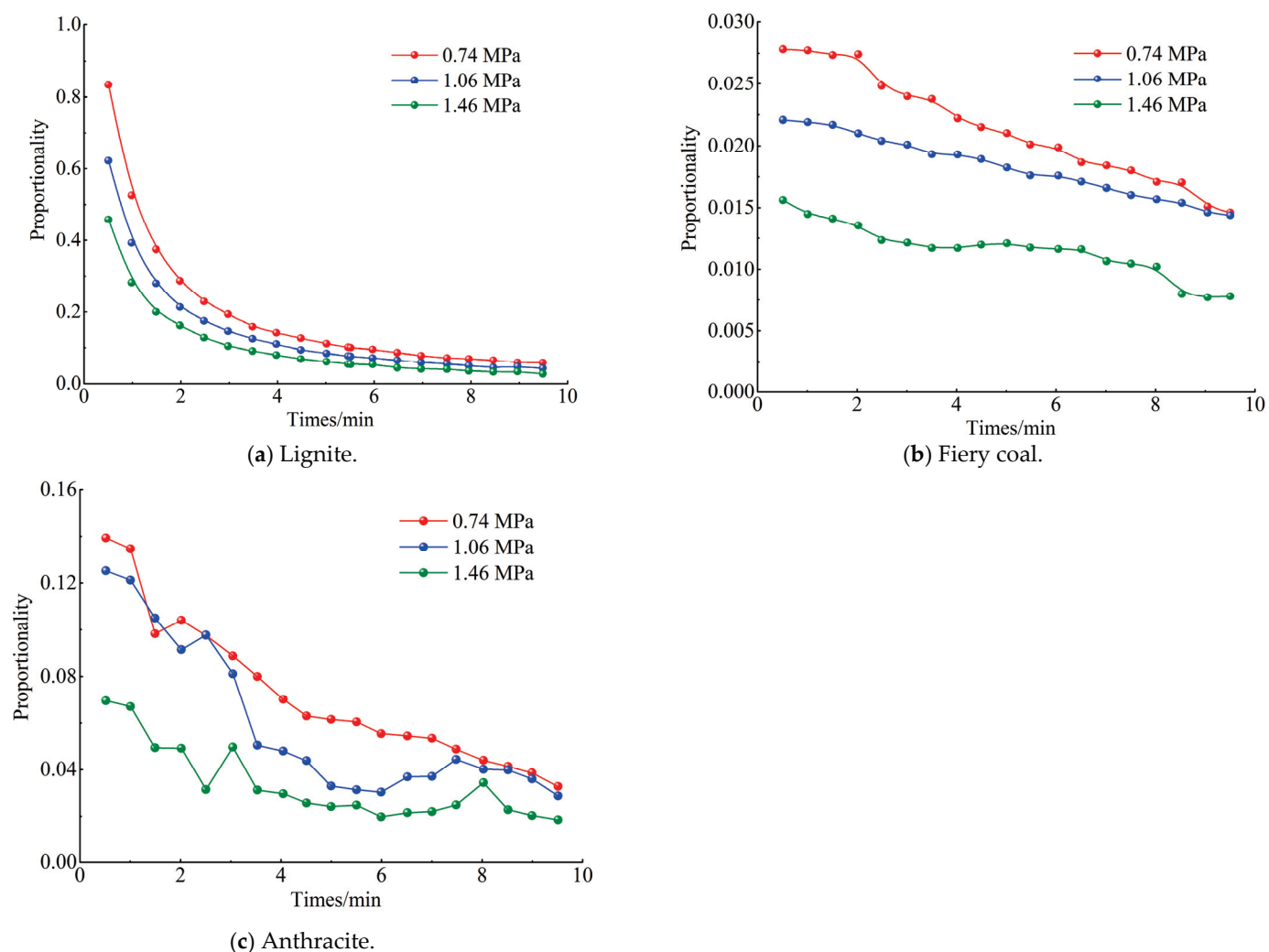


Figure 7. Proportionality of the desorption rate of coal under different gas pressures. (a) Lignite; (b) Fiery coal; (c) Anthracite.

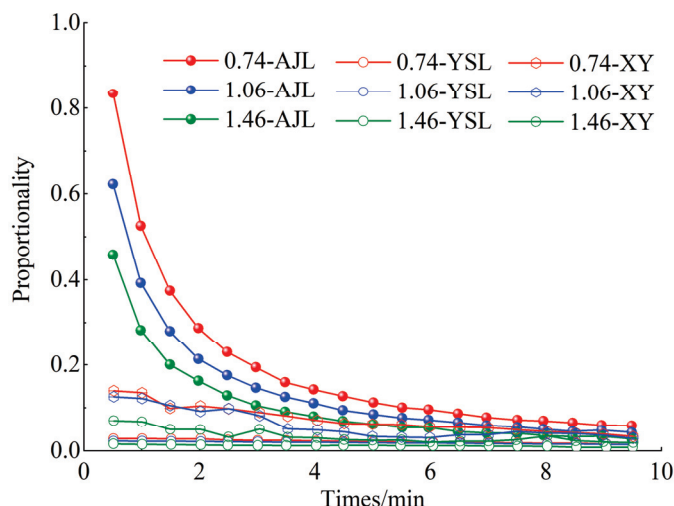


Figure 8. Summary of the desorption rates of coal under different pressures.

3.4. Effect of the Metamorphic Degree on the Gas Diffusion Characteristics of Coal

After undergoing the processes of adsorption and desorption, gas permeates the pores of the coal matrix. By utilizing desorption data and applying Fick's second law of diffusion, it is possible to calculate a physical parameter that characterizes the gas diffusion capacity within coal, known as the effective diffusion coefficient. Through linear regression analysis between the effective diffusion coefficient and the maximum reflectance of vitrinite, the impact of the degree of metamorphism on gas diffusion performance can be examined. The diffusion coefficient signifies the ability of coal to facilitate gas diffusion, and its numerical value reflects the rate at which gas migrates through coal pores. In this study, the effective diffusion coefficient was introduced as a means of quantifying the gas diffusion capacity. The relevant expressions are provided below:

$$\frac{Q_t}{Q_\infty} = 1 - \frac{6}{\pi^2} \sum_{n=1}^{\infty} \frac{1}{n^2} \exp(-n^2 \pi^2 D_e t) \quad (1)$$

$$Q_\infty = \left(\frac{abp}{1+bp} - \frac{abp_0}{1+bp_0} \right) [1 - M_{ad} - A_{ad}] \quad (2)$$

where Q_t is the amount of gas desorption from coal at time t (mL/g); Q_∞ is the limiting gas desorption capacity of coal (mL/g); t is the desorption time (s); D_e is the effective diffusion coefficient (s^{-1}); p_0 is the gas pressure during the test (MPa); and M_{ad} and A_{ad} are the moisture and ash in coal (%).

The direct determination of the effective gas diffusion coefficient using the formula mentioned above is not feasible. To address this, we set $n = 1$ in Equation (1) and performed a logarithmic transformation of both sides of the equation, thus converting it into a linear fitting formula for the calculation.

$$\ln \left(1 - \frac{Q_t}{Q_\infty} \right) = -\frac{\pi^2 r^2 D_e}{a^2} t + \ln \frac{6}{\pi^2} \quad (3)$$

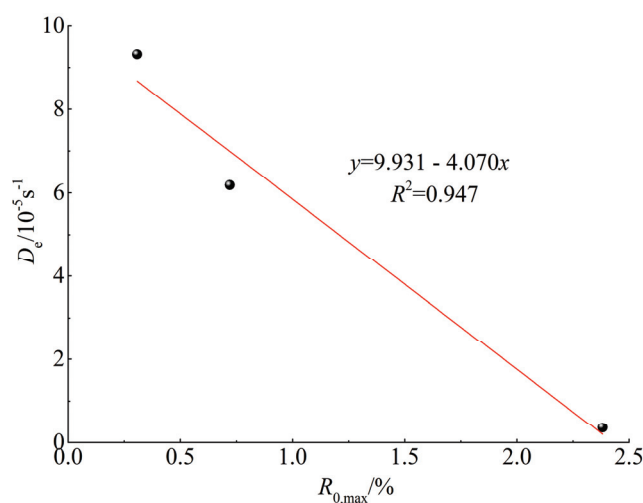
Here, $-\frac{\pi^2 r^2 D_e}{a^2}$ is $\ln \left(1 - \frac{Q_t}{Q_\infty} \right)$ in the linear fitting slope of the curve. The effective gas diffusion coefficients of the three coal samples were then calculated through inverse fitting, and the calculated results are shown in Table 3.

Table 3. Effective gas diffusion coefficients of the coal samples.

Coal Samples	Lignite	Fiery Coal	Anthracite
D_e/s^{-1}	9.32×10^{-5}	6.18×10^{-5}	0.36×10^{-5}

The effective diffusion coefficients of the coal samples, as shown in Table 3, followed a descending order: lignite, fiery coal, and anthracite. This observation markedly contrasts with the pre-desorption amount and adsorption constant of the coal samples mentioned earlier, suggesting that the impact of metamorphism on the gas diffusion capacity differs significantly from its influence on the adsorption and desorption capacity.

The relationship between the effective gas diffusion coefficient and the metamorphic degree of the coal samples was calculated based on the measured results for the maximum reflectivity of vitrinite and the effective gas diffusion coefficients. Figure 9 demonstrates this relationship. Figure 9 shows that there is a strong negative linear correlation between the effective gas diffusion coefficient and the maximum reflectance of vitrinite, indicating that the ability of gas to diffuse in coal decreases as the metamorphic degree increases. The variation in effective gas diffusion coefficients among coal samples with different degrees of metamorphism can be primarily attributed to the maceral components of the coal itself, as well as compaction and dehydration during the coalification process. Coals with a lower degree of metamorphism tend to have a higher content of inertinite and chitinite at the maceral level, resulting in abundant large pores within these macerals. These pores provide ample space for gas diffusion, facilitating the gas diffusion process.

**Figure 9.** Relationship between the effective diffusion coefficient and the degree of metamorphism in coal.

It can be seen from the above research that the more developed the pore structure of the coal sample, the stronger the corresponding desorption capacity. Because the pore structure of medium and large pores is more developed, it can provide a larger surface area and permeability, which can adsorb and desorb gas more effectively. At the same time, coal gas adsorption and desorption represent a reversible process, and the stronger the desorption capacity, the faster the corresponding gas adsorption speed. The analysis of the pore structure characteristics of coal samples combined with low-temperature liquid nitrogen adsorption experiments also confirmed that the gas adsorption capacity of AJL and YSL coal samples was significantly lower than that of XY coal samples. With the poor opening of micropores in coal samples, gas is not easily dispersed from micropores to large pores. With the increase in pores, once gas is released to large pores, it is very

easily dispersed to the outside world due to the good opening of large pores. In the initial stage of gas desorption, gas desorption is difficult, so it is preferred to desorption in large pores. XY coal samples have more micropores and poor connectivity, so the decrease in gas desorption rate is uneven and large, while AJL coal samples have more large pores, so the overall decrease trend in the desorption rate in the first 10 min is stable. Based on the relationship between pore structure and gas adsorption and desorption ability in coal of different coal ranks, it can be concluded that micropore structure plays a decisive role in gas adsorption ability, while the development degree of medium and large pores plays a decisive role in gas adsorption balance and rapid desorption ability, and there is a positive correlation between them.

4. Gas Desorption Dynamics Models

Numerous studies by domestic and foreign scholars have focused on the mathematical modeling of gas desorption in coal, resulting in the proposal of various formulas. However, some of these formulas have limitations. To investigate the gas desorption behavior of the three coal samples and identify suitable mathematical expressions, a commonly employed formula was utilized to fit the desorption process under different pressures. The mathematical expressions employed are presented in Table 4.

Table 4. Mathematical expression for gas desorption fitting.

NO.	-
Formula 1	$Q_t = at^i$
Formula 2	$Q_t = v_0 \left[(1+t)^{1-n} - 1 \right] / (1-n)$
Formula 3	$Q_t = Q_\infty \sqrt{1 - e^{-kbt}}$

It can be seen from Figure 10 that the fitting correlation coefficients of the three equations are all above 0.9, indicating a good fitting effect. In terms of the R^2 correlation index, when the desorption equilibrium pressure was 0.74 MPa, Formula 1 displayed a lower R^2 value compared to the other two empirical formulas, suggesting a weaker fitting correlation. Conversely, the other two formulas exhibited better fitting effects. As the equilibrium pressure increased, the correlation coefficients for all formulas also increased, albeit less noticeably for Formulas 2 and 3. When Formula 1 reached 1.46 MPa, its R^2 value increased to 0.977, significantly improving the fitting degree. However, Formula 2 had an R^2 value of 0.987, and Formula 3 had an R^2 value of 0.990, indicating even better fitting results. Overall, the correlation coefficient (R^2) increased with increasing equilibrium pressure for all empirical formulas. Formula 2 showed a correlation coefficient range of 0.971–0.989, while Formula 1 had a range of 0.949–0.977. Formula 3 exhibited the smallest range of variation for its correlation coefficient, ranging from 0.979 to 0.990, resulting in the best fitting effect among the empirical formulas. The fitting results indicated that the regression coefficients of all three formulas changed with variations in the equilibrium pressure. However, the changes were significant for Formula 1, while being relatively minor for Formulas 2 and 3.

It can be seen from Figure 11 that the fitting results for the fiery coal samples were satisfactory across the three different equilibrium pressures. At an equilibrium pressure of 0.74 MPa, Formula 3 exhibited some deviation during the middle stage of gas release, while Formula 1 displayed significant deviation in the middle and late stages. Conversely, Formula 2 demonstrated the highest degree of fitting. At an equilibrium pressure of 1.06 MPa, Formula 2 exhibited the best fitting results, although Formulas 3 and 1 showed some deviation during the early and middle stages of gas dispersion. Under an equilibrium

pressure of 1.46 MPa, both Formulas 3 and 1 deviated during the middle and late stages of gas release, while Formula 2 continued to display the highest degree of fitting. This suggests that Formula 2 provided a good fitting effect. Throughout the entire desorption process, the correlation coefficient of Formula 2 varied only between 0.993 and 0.998. Additionally, Formula 1 had a fitting correlation coefficient of 0.993–0.997, indicating its ability to accurately describe the diffusion process of coal samples. Considering the fitting regression coefficient, Formula 3 exhibited the smallest range of variation, implying that using Formula 3 data in different equilibrium pressure segments to calculate gas loss would result in minimal differences and greater stability.

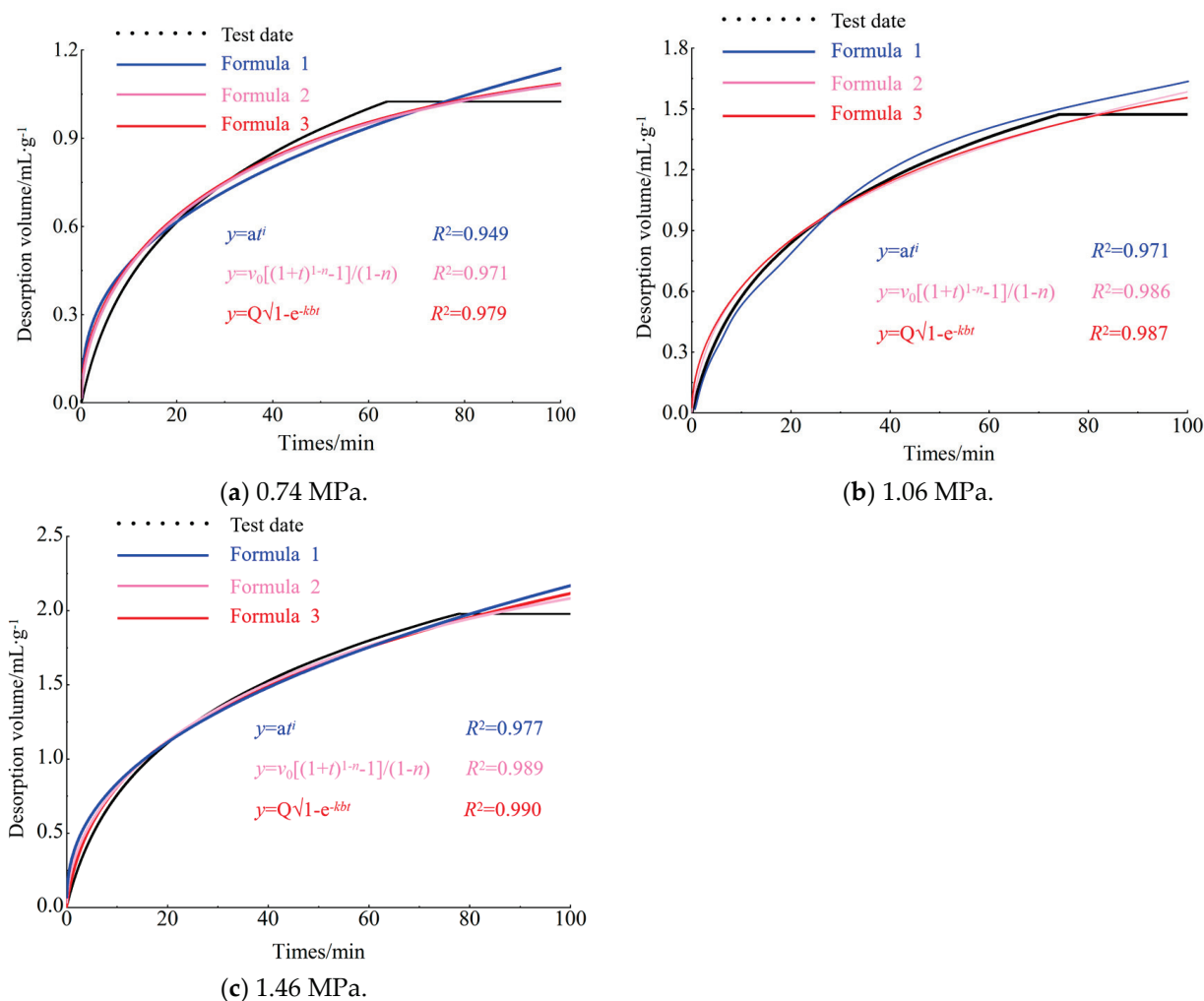


Figure 10. Fitting curve for the amount of desorption in lignite. (a) 0.74 MPa; (b) 1.06 MPa; (c) 1.46 MPa.

It can be seen from Figure 12 that the empirical formula displayed a good fitting effect in the early stage for the anthracite coal samples. The correlation coefficient (R^2) of the fitting equation gradually increased with increasing equilibrium pressure and then gradually decreased after reaching a certain point. The correlation coefficients obtained using Formulas 2, 1, and 3 were all greater than 0.95. However, Formula 2 consistently yielded correlation coefficients above 0.99 for the entire desorption process, reflecting a strong fitting effect.

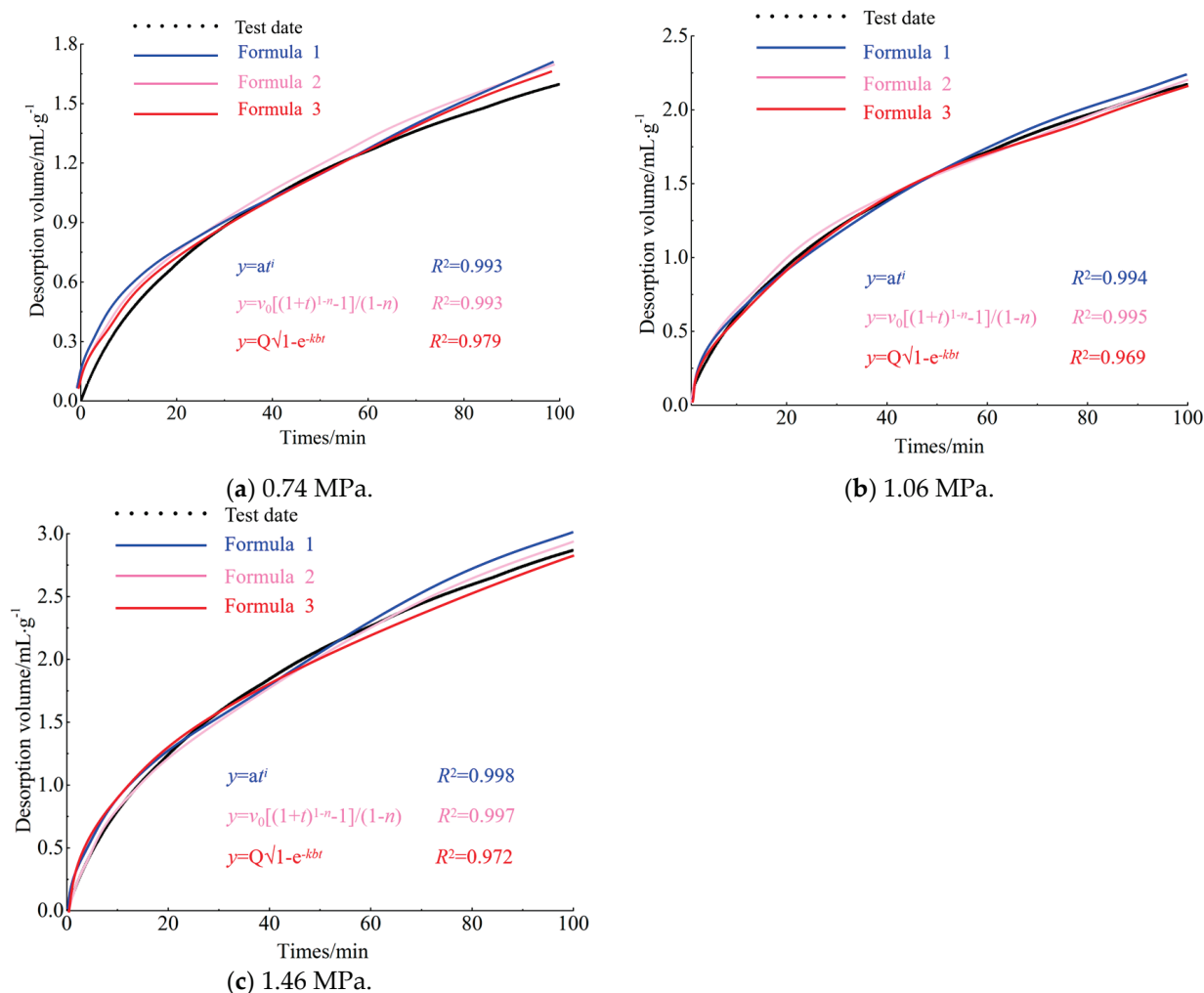


Figure 11. Desorption fitting curve for fiery coal. (a) 0.74 MPa; (b) 1.06 MPa; (c) 1.46 MPa.

At an equilibrium pressure of 0.74 MPa, Formula 2 exhibited some deviation in the early stage, while Formula 3 showed the highest degree of fitting and Formula 1 displayed deviations during the early, middle, and late stages of gas dispersion. Under an equilibrium pressure of 1.06 MPa, Formula 2 demonstrated the highest degree of fitting, while Formulas 3 and 1 exhibited certain deviations in the early and middle stages. Similarly, under an equilibrium pressure of 1.46 MPa, Formula 2 again had the highest degree of fitting, while Formula 1 deviated during the middle and late stages of gas dispersion.

The fitting results indicate that the overall applicability of the three empirical formulas was relatively good, especially at an equilibrium pressure of 1.46 MPa. Formulas 2, 1, and 3 all achieved high correlation index values, indicating a relatively good fitting effect.

In the experiment, different classical empirical formulas were adopted to conduct a regression analysis on the experimental data obtained for desorption in three groups of coal samples under different equilibrium pressures. The three empirical fitting formulas used for anthracite yielded relatively stable correlation index values, indicating a good fitting effect. The fitting results of Formulas 2, 3, and 1 under an equilibrium pressure of 1.46 MPa were relatively ideal. For lignite, Formula 3 exhibited a relatively large correlation coefficient (R^2) with a small range of variation, indicating a better fitting effect compared to the other two empirical formulas. Additionally, for fiery coal, Formula 3 accurately described the gas release process of the coal samples.

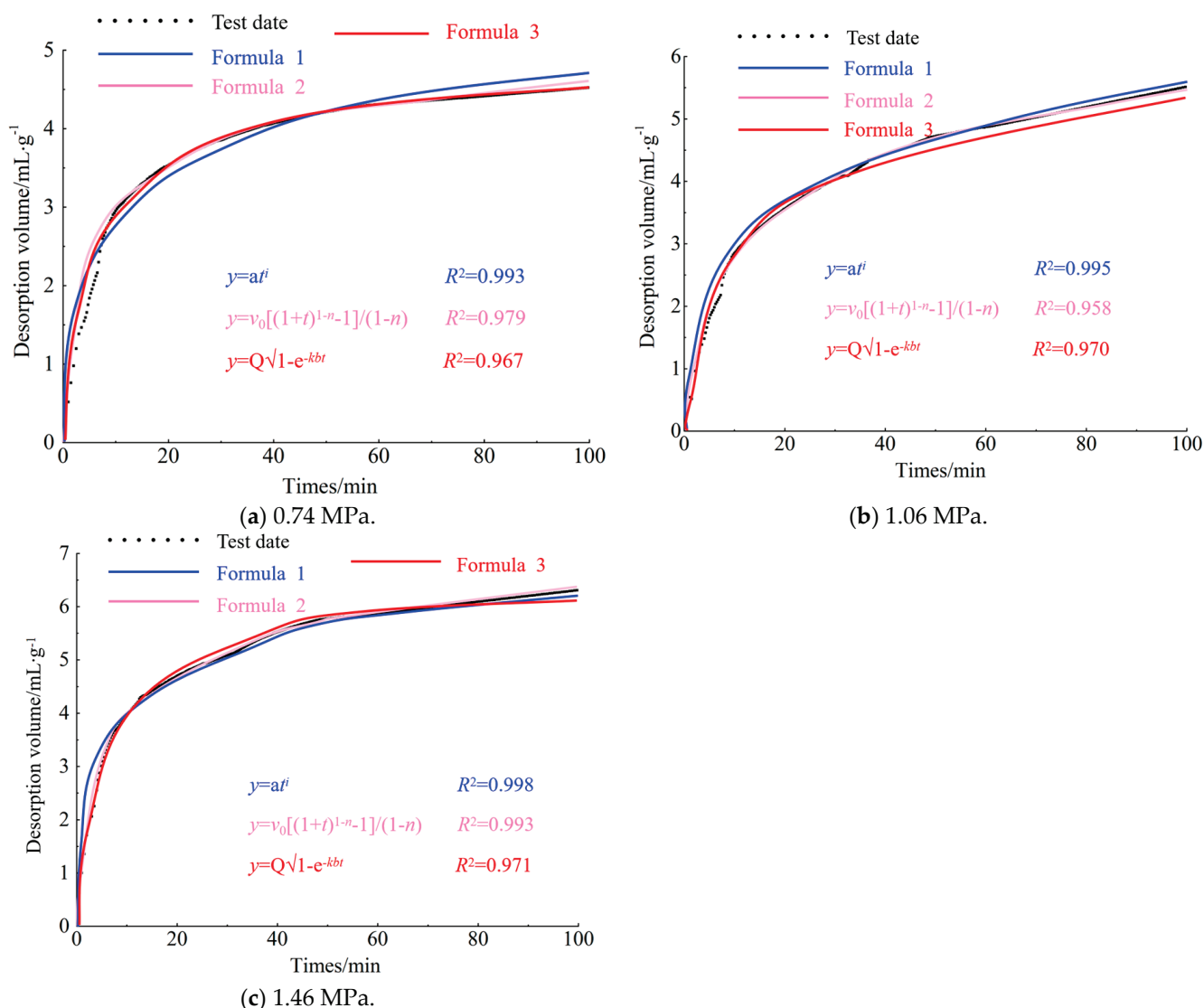


Figure 12. Desorption fitting curve for anthracite. (a) 0.74 MPa; (b) 1.06 MPa; (c) 1.46 MPa.

Building upon prior research theories, this paper scrutinizes the gas desorption and diffusion behaviors of lignite, bituminous coal, and anthracite. It employs more comprehensive mathematical formulas to fit gas desorption curves, revealing gas release patterns and applicable models for coals with varying metamorphic degrees. This analysis not only enhances comprehension of gas desorption and diffusion laws but also offers insights for predicting and preventing coal and gas outbursts. It establishes a solid foundation for accident prevention and holds practical significance for enhancing mine safety practices. Nonetheless, the study in this paper only tested three representative coal samples, which could potentially introduce deviations in the results. The author intends to conduct additional gas desorption and diffusion tests and model studies on a broader range of coal samples and ranks in future research endeavors.

5. Conclusions

- (1) Coal samples with a higher degree of metamorphism exhibited a faster attainment of the desorption limit value, whereas low-rank lignite required a longer period to reach this limit. However, as the desorption time increased, the desorption rate of highly metamorphic coal samples gradually declined, resulting in a progressively slower desorption rate.

- (2) As coal metamorphism increased, the rate of decline in the gas desorption rate slowed down, and the higher the degree of coal metamorphism, the greater the amount of gas desorption. The effective gas diffusion coefficient of the coal samples exhibited a strongly negative linear correlation with the degree of metamorphism, indicating that the ability of the coal samples to facilitate gas diffusion decreased as the degree of metamorphism increased.
- (3) Among the three empirical fitting formulas applied to anthracite, a relatively stable correlation index was obtained, and the fitting effect was satisfactory. For an equilibrium pressure of 1.46 MPa, the fitting results for Formulas 2, 3, and 1 were relatively ideal. Formula 3 yielded a relatively high correlation coefficient (R^2) and exhibited a smaller range of coefficient variation for lignite, making it a more effective fitting formula compared to the other two. For fiery coal, Formula 3 was deemed more suitable for describing the gas desorption process of the coal samples.

Author Contributions: H.X.: Conceptualization, Data curation, Writing—original draft. X.Q.: Data curation, Formal analysis. H.W.: Funding acquisition, Methodology. Z.L.: Formal analysis. T.Y.: Formal analysis, Visualization. Y.Z.: Formal analysis. Q.J.: Formal analysis. L.J.: Formal analysis, Visualization. All authors have read and agreed to the published version of the manuscript.

Funding: This research was funded by the Natural Science Foundation of Hebei Province (E2024508006), the National Natural Science Foundation of China (52274200).

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

Conflicts of Interest: Authors Zhongqiu Liang and Yongming Zou were employed by the company CCTEG Shenyang Research Institute. The remaining authors declare that the research was conducted in the absence of any commercial or financial relationships that could be construed as a potential conflict of interest.

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Article

Mechanical and Permeability Characteristics of Gas-Bearing Coal Under Various Bedding Angles

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Abstract: Coal structures are commonly found in coal rock formations. Understanding the evolutionary laws of mechanics, deformation, and permeability of gas-bearing coal rock during the failure process at different bedding angles is crucial for studying the prevention and control techniques of coal and rock gas dynamic disaster mitigation. In this study, a mechanical seepage test of gas-bearing coal under various bedding angles was conducted using the fluid–solid coupling triaxial servo test system. The results indicate the following corrections: ① Both axial peak strain (ϵ_1) and radial peak strain (ϵ_3) initially increase and then decrease, reaching their maximum values at 45° , indicating that the specimen eventually slips along the bedding plane and fails. ② As the bedding angle increases, the peak stress of the coal body shows a “V”-shaped distribution, with the peak strength of the gas-bearing coal sample being the lowest at 60° . ③ The minimum permeability of the coal sample increases with the rise in the bedding angle. The bedding direction of the coal samples at 90° and 75° aligns with the axial direction, leading to more seepage channels. ④ At a bedding angle of 60° , the minimum dissipated energy (U_d) is required for sample failure, indicating that the sample is highly prone to failure.

Keywords: gas-bearing coal; bedding angle; mechanics; permeability rate; energy

1. Introduction

Coal is an important energy source for industrial development worldwide, accounting for a significant portion of global primary energy consumption. In China, coal accounts for a staggering 56% of the energy consumption [1,2]. With the development of the economy, the demand and output of coal increase year by year, leading to the gradual depletion of shallow coal resources [3]. In eastern China, the annual growth rate of coal mining depth can reach 10–25 m. In deep mining environments, the stress conditions surrounding coal formations are significantly different from those during shallow mining periods. The mining conditions deteriorate, and the occurrence of gas-bearing coal becomes more complex. As a result, the frequency and intensity of coal and gas outburst disasters gradually increase [4,5].

Understanding the mechanical and deformation evolution patterns of gas-bearing coal formations under mining-induced stress disturbances is crucial for studying coal and gas dynamic disaster prevention and control techniques [6–8]. A large number of scholars

have used the triaxial test to analyze the damage and failure forms of coal and rock masses under various stress states. Shen et al. [9] hold that the mechanical peak strength and strain of coal samples gradually decrease as a whole with the increase in axial prestress σ_1 . Based on the damage and degradation caused by multiple impacts, according to Liu et al. [10], the dynamic peak stress and Young's modulus of the specimen decrease with an increasing number of impacts.

Du et al. [11] analyzed the changes in total input energy and elastic strain energy by establishing a three-dimensional microstructure reconstruction model of coal and rock formations, considering the variations in confining pressure and gas pressure. Yin et al. [12] used PFC software to study and find that the existence of cracks in coal will lead to a reduction in compressive strength and crack initiation stress, and promote the occurrence of AE. Similarly, the permeability evolution law is also an important means to characterize the damage degree of coal and rock masses [13,14]. Li et al. [15] conducted measurements of permeability and adsorption characteristics using the pulse transient method. The results indicate that for adsorbed gas, permeability initially decreases due to expansion with increasing gas pressure. Li et al. [16] pointed out that gas flow rates increase with increasing pore pressure, causing the coal matrix to undergo expansion deformation due to gas adsorption. Zhu et al. [17], Xu et al. [18], and Wang et al. [19] discussed the damage–permeability aging characteristics of mined coal and the influence of mechanical paths on the permeability characteristics, based on the experimental results of the deformation and gas flow state of highly gassy coal rock formations under mining-induced stress and gas pressure.

The physical and mechanical properties of a coal and rock mass are the internal causes of dynamic coal mine disasters [20,21]. Coal, as a sedimentary rock, undergoes long-term geological processes that result in the development of numerous fractures and bedding planes [22]. The CT scanning of stratified coal shows that most of the original fractures in the coal body develop along parallel stratification directions; bedding planes are one of the primary structures of coal seams and are considered to be the dominant factor influencing the strength, deformation, and failure modes of layered rocks [23,24]. In view of the anisotropy of the mechanical behavior of coal under different bedding conditions, many scholars have carried out corresponding research. Mu et al. [25] conducted experimental research on the influence of different bedding angles on the acoustic emission and mechanical characteristics of coal. Song et al. [26,27] conducted uniaxial compression tests on coal samples with different bedding angles and studied the anisotropic mechanical behavior of coal from aspects such as acoustic characteristics, strength characteristics, and deformation characteristics. They proposed an anisotropic strength model for coal. Hao et al. [28] discussed the effects of bedding angle and confining pressure on the mechanical behavior of shale on the basis of conventional triaxial compression tests. Liu et al. [29] studied the mechanical properties and fracture characteristics of fractured rock masses with bedding plane defects by establishing a pre-fracture model with different bedding plane attributes (strength degradation factor, bedding plane angle, and bedding plane spacing). Yang et al. [30] simulated the fracture mode of bedding rocks and observed that there were five kinds of bedding plane fracture rotation during the fracture process. Zuo et al. [31] analyzed the influence of bedding on shale fracture propagation by using a three-point bending test, and found that when the fracture propagation direction deviates from the bedding inclination angle by $0^\circ \sim 90^\circ$, the fracture toughness, energy, and fracture path complexity increase significantly. The development of pore and fracture networks in coal is crucial to the storage space and migration path of coalbed methane, which means that the bedding structure also affects the permeability of coal [32–34].

Bedding coal is common in nature, and the bedding plane angle is diversified due to geological movement. It is generally believed that bedding is the stress release surface

of coal seams, and its angle difference has an important effect on the strength and failure of coal. However, there are few studies on the influence of bedding plane angle on the failure process of coal under triaxial loading conditions. In view of this, this paper uses the fluid–solid coupling triaxial servo test system for gas-bearing coal to carry out mechanical seepage tests of gas-bearing coal under different bedding angles. The influence of bedding angle on the stress–strain, strength, permeability, and energy characteristics of coal is analyzed, and the research results can provide a reference for the stability analysis of gas-bearing coal in deep environments and the prediction and prevention of instability disasters.

2. Test Equipment and Scheme

2.1. Sample Preparation

The specimens used in this experiment are standard samples with dimensions of 50 mm × 100 mm, and the long-flame coal from the Baliancheng Mine (Group) Co., Ltd., Hunchun, Jilin province, China was selected as the research object. The Baliancheng mine field is located to the west of Hunchun coalfield. It has developed coal-bearing deposits from the Paleogene and Eocene to the Oligocene, with many faults and obvious rules. And it is a high-capacity gas mine; the maximum measured gas content is 6.3 m³/t, the absolute gas emission amount is 58.77 m³/min, and the relative gas emission amount is 10.62 m³/t.

In accordance with the requirements of the “Standard Test Method for Engineering Rock Masses” (GB/T50266-2013 [35]), large coal blocks with significant structural bedding were transported to the laboratory for cutting and grinding to produce standard coal samples with a diameter of 50 mm and a height of 100 mm. It was ensured that the non-parallelism between the two end faces of the coal sample was less than 0.05 mm, with the end faces perpendicular to the axis of the coal sample and a maximum deviation not exceeding 0.25°. Coal samples were taken at angles of 0°, 15°, 30°, 45°, 60°, and 90° between the bedding planes and the horizontal direction, as shown in Figure 1.

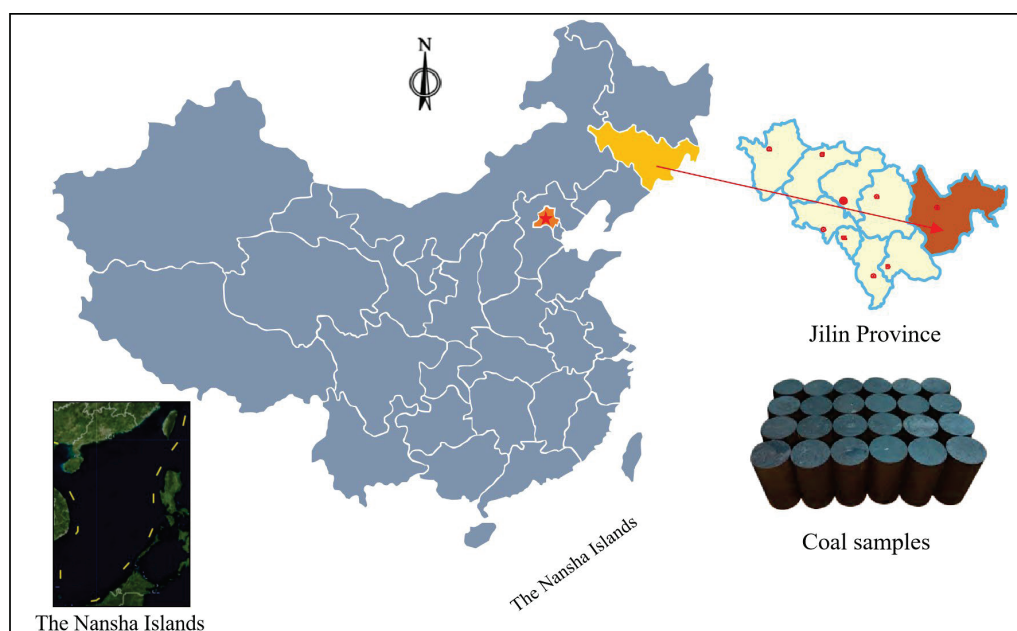


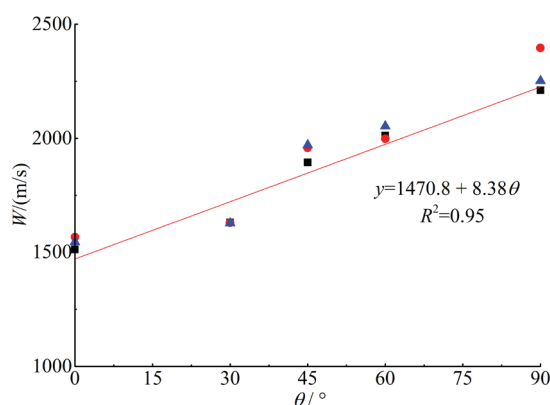
Figure 1. Sampling location of coal samples.

After the processed samples were dried, 3 coal samples were taken from each angle for P-wave velocity measurement. The coal samples without abnormal wave velocity, that were intact, and apparently without cracks and defects, were selected for experimental research. Table 1 shows the wave velocity statistics of the coal samples used in the experiment.

Table 1. Wave velocity statistics of coal samples with different bedding angles.

$\theta/(^{\circ})$		$W(m/s)$	
0	1513	1567	1545
30	1632	1630	1628
45	1894	1958	1971
60	2012	1998	2053
90	2210	2396	2252

Figure 2 shows the change in coal axial wave velocity with the increase in bedding angle. As the bedding angle increases, the wave velocity of the sandstone samples increases nearly linearly. The average wave velocity increases from 1542 m/s at $\theta = 0^{\circ}$ to 2286 m/s at $\theta = 90^{\circ}$, with an increase of 744 m/s. This is because as the bedding dip angle increases, the number of bedding planes that the transmitted wave passes through decreases, reducing the travel time and thus increasing the wave velocity. This indicates that the propagation of sound waves in layered coal bodies exhibits significant anisotropy.

**Figure 2.** Variation curve of P-wave velocity with bedding angle.

2.2. Test Equipment

The experiment was conducted on a gas-bearing coal thermal fluid–solid coupled triaxial servo test system, which consists of four main parts: an axial loading unit, a permeability control unit, a gas injection and vacuum unit, and a data acquisition and real-time monitoring unit. The structural composition of the experimental system is shown in Figure 3.

**Figure 3.** Gas-bearing coal thermal fluid–solid coupled triaxial servo test system.

2.3. Test Scheme

To avoid interference from moisture, all specimens were dried in a drying oven at a temperature of 105 °C for 6 h. To prevent delamination during the experiment and to ensure effective gas permeation, a layer of 704 silicone gel was uniformly applied to the surface of the specimens prior to the test. After allowing the gel-coated specimens to stand for 24 h, they were fixed together using heat shrink tubing.

According to previous research [5], the adsorption, desorption, and permeation behaviors of CO₂ and CH₄ in coal are similar. Therefore, CO₂ can be used as a substitute for CH₄ in physical simulation experiments. For safety reasons, this study conducted the experiments using CO₂ with a concentration of 99.9%. In the following sections, the term “gas” refers to CO₂.

The steps for conducting gas-bearing coal mechanical permeability tests under triaxial loading conditions are as follows: (1) Close the air outlet valve and open the vacuum pump to vacuum the coal sample chamber for 6 h; (2) Apply hydrostatic pressure $\sigma_1 = \sigma_3$ to 8 MPa (where σ_1 is the axial stress and σ_3 is the confining pressure), and the stress balance is 3 h; (3) 1.5 MPa CO₂ was injected into the coal sample chamber and adsorbed for 24 h. When the strain was less than 1%, the sample was considered to have reached adsorption equilibrium. (4) Open the stress–strain collection device and gas flow collection device and load the axial stress at a rate of 0.1 mm/min until it is damaged. The loading path is shown in Figure 4, and the coal sample size and experimental steps are shown in Table 2.

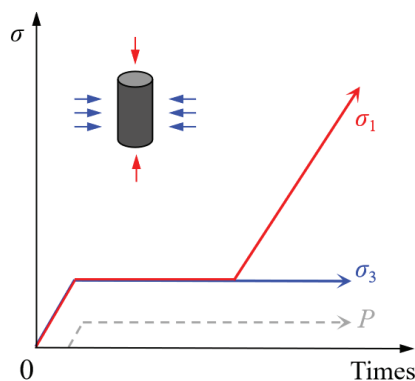


Figure 4. Load path diagram.

Table 2. Specimen size parameters and test scheme.

No.	Diameter/D (mm)	Height/H (mm)	Density/ ρ (g/cm ³)	Confining Pressure/ σ_3 (MPa)	Gas Preessure/P (MPa)	Bedding Angle/ θ (°)
1	49.90	99.67	1.36	8	1	0
2	50.12	99.98	1.33			30
3	50.01	100.10	1.30			45
4	49.96	100.03	1.28			60
5	50.01	99.96	1.32			75
6	50.05	100.08	1.31			90

3. Test Results and Analysis

3.1. Stress–Strain Characteristics

Figure 5 displays the stress–strain curve of gas-bearing coal under triaxial compression testing at 8 MPa. It can be observed that the stress–strain variations in the specimens exhibit a similar trend overall. The difference in principal stresses initially increases and then decreases with the increase in axial strain. Taking Figure 5a as an example for pattern

analysis, the stress–strain curve can be divided into four stages: the elastic deformation stage, the stable crack propagation stage, the unstable crack propagation stage, and the post-peak instability and failure stage. In the elastic stage, the low level of axial stress does not cause internal damage in the specimen, and the deformation and stress exhibit a linear increasing trend. In the stable crack propagation stage, the increase in axial stress leads to the formation of new cracks inside the specimen, causing the deformation curve to deviate from linearity. In the unstable crack propagation stage, the growth rate of internal cracks intensifies with increasing stress, resulting in a transition of overall deformation from compression-dominated to expansion-dominated. In the post-peak instability and failure stage, the internal cracks intersect, causing the stress in the specimen to drop until failure occurs.

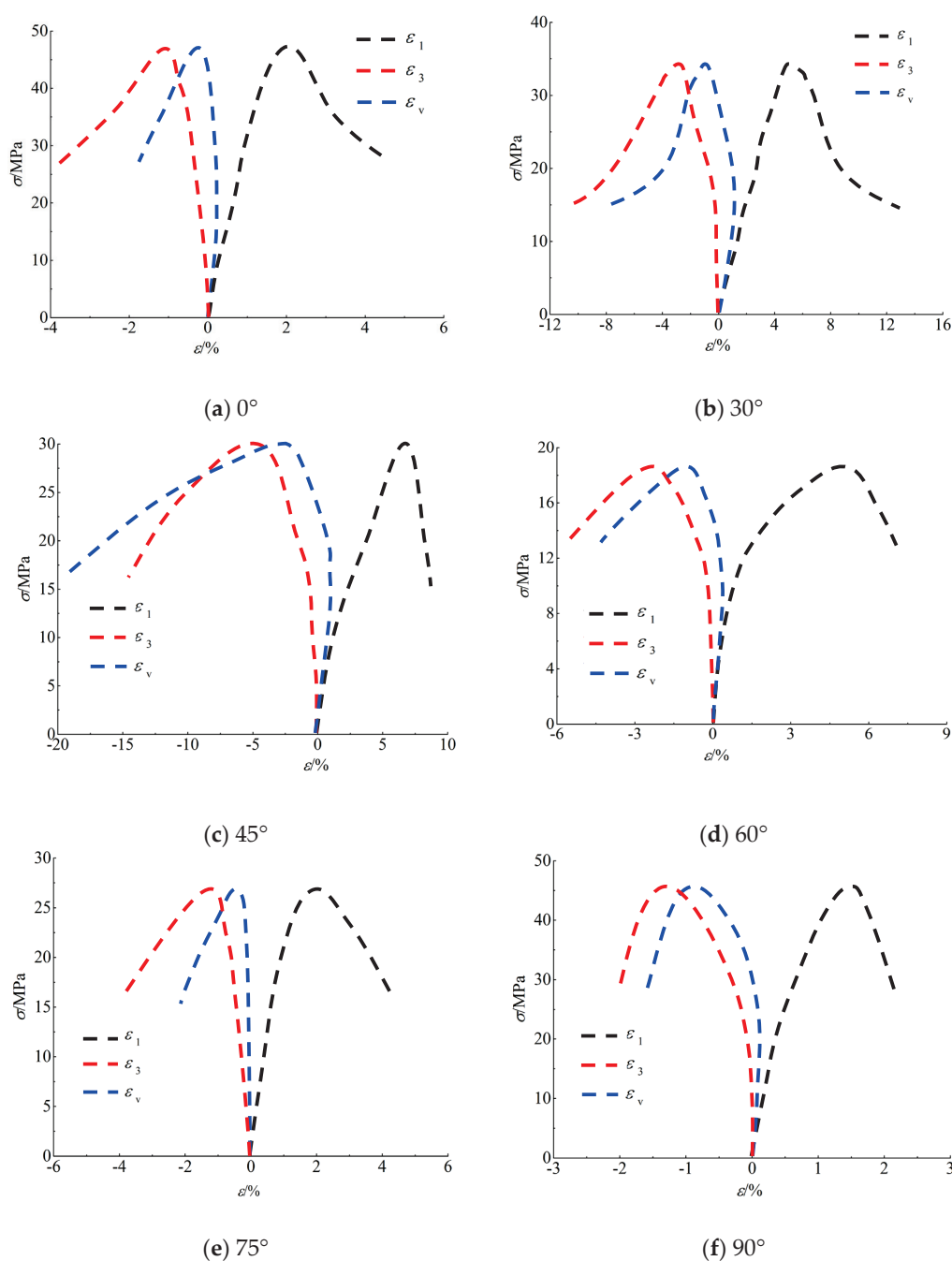


Figure 5. Stress–strain of gas-bearing coal under triaxial loading test at different bedding angles.

The deformation of gas-bearing coal is influenced by the angle of bedding, as shown in Figure 6. The variation in bedding angle affects the deformation capacity of gas-bearing coal samples. As the bedding angle increases, both the axial peak strain ε_1 and the radial peak strain ε_3 of the specimen show a trend of initially increasing and then decreasing. At an angle of 0° , when reaching the stress peak, the axial peak strain ε_1 and the radial peak strain ε_3 of the specimen are 2.03 and 1.18, respectively. At a bedding angle of 30° , ε_1 and ε_3 are 4.89 and 2.98, respectively. When the bedding angle increases to 45° , ε_1 and ε_3 are 6.90 and 4.91, respectively, reaching their maximum values. As the bedding angle gradually increases to 60° , 75° , and 90° , ε_1 decreases to 5.31, 2.03, and 1.51, respectively, while ε_3 decreases to 2.35, 1.25, and 1.35, respectively. This is because at a bedding angle of 45° , the failure of the gas-bearing coal sample is significantly influenced by the bedding angle; once the shear force along the bedding plane exceeds the cementation strength between the bedding planes, this eventually results in unstable failure along the bedding plane due to sliding.

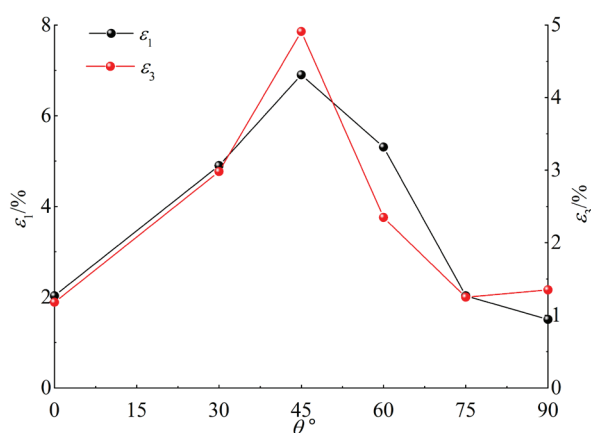


Figure 6. Variation law of peak strain of coal samples.

3.2. Mechanical Strength Characteristics

In order to gain a better understanding of the influence of bedding angle on the peak strength of gas-bearing coal samples, the experimental results were organized and analyzed, as shown in Table 3 and Figure 7, revealing the anisotropic characteristics of different bedded coal formations.

Table 3. Mechanical properties of coal samples with different bedding angles.

$\theta/(^\circ)$	$\Delta\sigma/\text{MPa}$	E/MPa	ν
0	46.21	3082	0.30
30	35.02	2867	0.28
45	29.87	2604	0.27
60	18.63	2061	0.22
75	27.22	2588	0.31
90	44.93	3379	0.42

With the increase in bedding angle, the peak deviator stress, elastic modulus, and peak strain of the coal exhibit strong anisotropy. The peak deviator stress of the coal forms a “V”-shaped distribution. The peak strength of the gas-bearing coal samples is lowest at a bedding angle of 60° and highest at a bedding angle of 90° , indicating the strongest load-bearing capacity. The peak strength of the gas-bearing coal samples initially increases and then decreases with increasing bedding angle, exhibiting a characteristic of increasing again at the end. The elastic modulus reflects the ability of coal to resist deformation

during loading. The trend of the elastic modulus of the coal samples generally decreases with the change in the bedding angle θ , reaching a minimum value at 60° and gradually increasing thereafter.

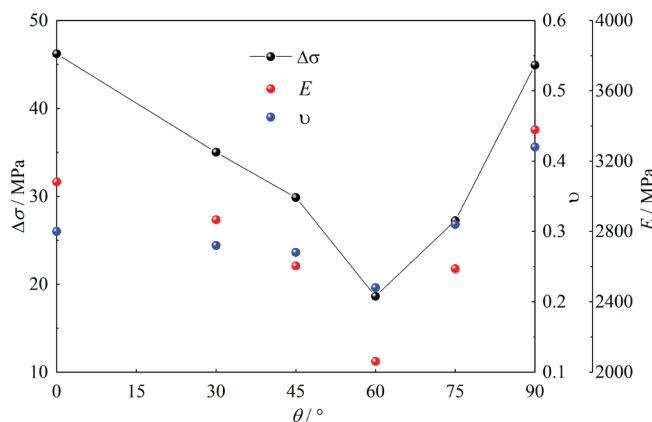


Figure 7. Influence of bedding angle on peak strength of coal samples.

According to the characteristics of coal samples at different bedding angles, the differences in the mechanical properties of coal samples can be analyzed as follows. For a coal sample with a bedding angle of 0° , the bedding planes are perpendicular to the loading direction. The load-bearing capacity of the coal sample mainly depends on the support of the coal matrix and the bond strength between the bedding planes. In the case of a coal sample with a bedding angle of 60° , shear slip failure occurs, and the compressive strength mainly depends on the bond strength between the weak bedding planes, resulting in the lowest peak deviator stress. In contrast, for a coal sample with bedding angle of 90° , the load-bearing capacity is more inclined towards the material itself, resulting in the maximum load-bearing capacity.

The coal sample with bedding angle of 0° was significantly affected by the cutting action of the cleavage planes, resulting in the most significant increase. In the case of a coal sample with a bedding angle of 60° , the compressive strength mainly depends on the bond strength between the weak bedding planes, and the radial constraint and bedding planes are at a 60° angle. As a result, the increase in peak deviator stress is less prominent, leading to a “V”-shaped distribution of the increase with increasing bedding angle.

3.3. Seepage Change Characteristics

In this study, the seepage of gas in coal is believed to obey Darcy's law, and the permeability calculation formula is

$$k = \frac{2p'_0 Q \mu L}{A(p_1^2 - p_2^2)} \quad (1)$$

where k represents permeability in m^2 , Q represents the gas flow rate in m^3/s , μ represents the absolute viscosity of the gas, L represents the length of the specimen in meters, A represents the effective area of permeability in m^2 , p'_0 represents the standard atmospheric pressure in MPa, p_1 represents the inlet pressure in MPa, and p_2 represents the outlet pressure in MPa.

The permeability of the specimen represents the strength of gas permeability. By observing the permeability variation curve shown in Figure 8, it can be observed that as stress is applied, the permeability initially decreases and then increases. This is because during the initial stage of the experiment, as external stress is applied, the inherent pores and cracks inside the specimen are compressed, leading to a narrowing of gas flow channels

and a decrease in permeability. Subsequently, cracks in the specimen start to develop, leading to damage expansion, which increases the gas flow channels and thus increases the permeability. As stress continues to increase, internal cracks start to connect, resulting in the unstable failure of the specimen, opening up the gas flow channels, and causing a rapid increase in permeability.

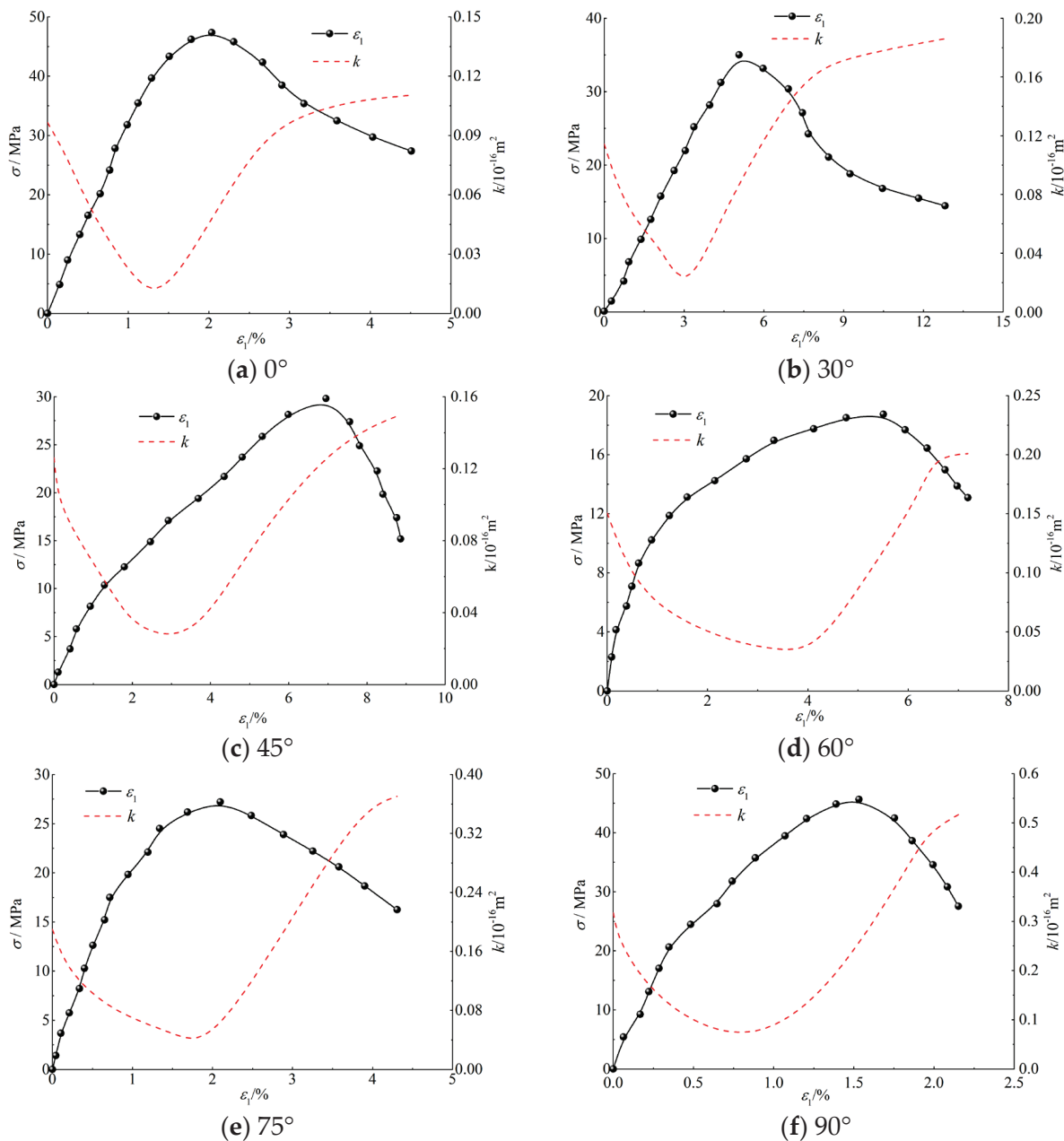


Figure 8. Variation trend of permeability.

As shown in Figure 9, overall, the minimum permeability of the coal sample increases with the increasing bedding angle during the loading and failure process of the specimen.

Coal samples with bedding angles of 90° and 75° have bedding orientations consistent with the axial direction, resulting in well-developed bedding fractures and more flow channels. As a result, these samples exhibit higher initial permeability compared to coal samples with other bedding angles.

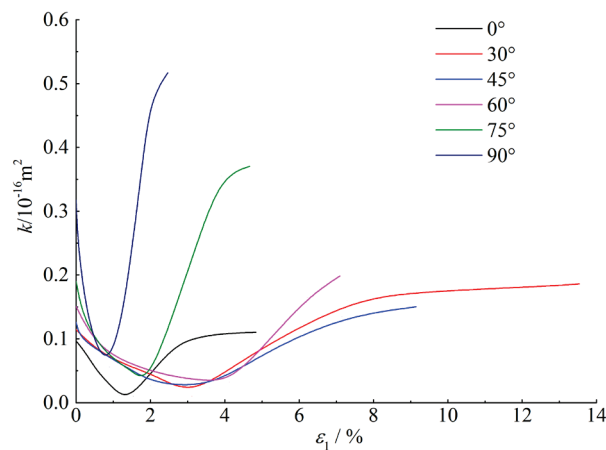


Figure 9. Influence of bedding angle on permeability of coal sample.

Coal samples with bedding angles less than 75° have a certain angle with the flow direction due to their bedding orientation, which prevents gas from flowing solely along the bedding fractures. Instead, gas flow also needs to traverse the cleavage fractures across the bedding planes. As the bedding angle increases, the area requiring cross-bedded flow increases. However, the number of cross-bedded cleavages is smaller and their development is limited. This results in fewer and more tortuous flow channels, leading to significantly increased flow resistance. Therefore, the initial permeability of these samples is much lower than that of nearly parallel bedded coal samples, and it decreases with increasing bedding angle. Coal samples with larger bedding angles have limited space for primary fractures along the axial direction, and their permeable pathways have less compressibility.

3.4. Energy Evolution Characteristics

The loading failure of gas-bearing coal involves the transfer and exchange of energy between the coal and its surroundings. During the stress loading of the specimen, external forces perform work on it, resulting in energy absorption by the specimen. The total input energy (U) absorbed during the specimen's failure process is transformed into elastic strain energy (U_e) and dissipated energy (U_d) [34,36], as shown in Figure 10, where

$$U = U_e + U_d \quad (2)$$

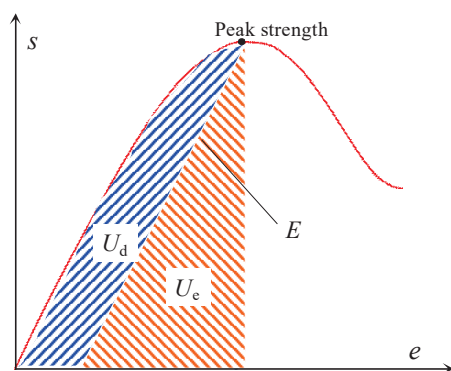


Figure 10. Energy evolution diagram of the failure process of gas-bearing coal.

In the presence of gas pressure during the permeability testing, the influence of gas pressure on the test should be taken into account when calculating the work performed on

gas-bearing coal under external forces. Specifically, the work applied to the coal should be the effective principal stress, and it can be calculated using the following formula [37]:

$$\sigma'_i = \sigma_i - \alpha P \quad (i = 1, 2, 3) \quad (3)$$

So, we have:

$$U = \int_0^{\varepsilon_1} \sigma'_1 d\varepsilon_1 + \int_0^{\varepsilon_2} \sigma'_2 d\varepsilon_2 + \int_0^{\varepsilon_3} \sigma'_3 d\varepsilon_3 + U_0 \quad (4)$$

U_e can be calculated as:

$$U_e = \frac{\sigma'_1 \varepsilon_1^e + \sigma'_2 \varepsilon_2^e + \sigma'_3 \varepsilon_3^e}{2} \quad (5)$$

According to a generalized Hook's law:

$$\varepsilon_1^e = \frac{\sigma'_1 - v(\sigma'_2 + \sigma'_3)}{E} \quad (6)$$

$$\varepsilon_2^e = \frac{\sigma'_2 - v(\sigma'_1 + \sigma'_3)}{E} \quad (7)$$

$$\varepsilon_3^e = \frac{\sigma'_3 - v(\sigma'_1 + \sigma'_2)}{E} \quad (8)$$

Then, we have:

$$U_e = \frac{1}{2E} [\sigma_1'^2 + \sigma_2'^2 + \sigma_3'^2 - 2v(\sigma'_1 \sigma'_2 + \sigma'_1 \sigma'_3 + \sigma'_2 \sigma'_3)] \quad (9)$$

For the true triaxial test:

$$E = \frac{\sigma'_1 - v(\sigma'_2 + \sigma'_3)}{\varepsilon_1} \quad (10)$$

$$v = \frac{B\sigma'_1 - \sigma'_3}{B(\sigma'_2 + \sigma'_3) - (\sigma'_1 + \sigma'_2)} \quad (11)$$

$$B = \frac{\varepsilon_3}{\varepsilon_1} \quad (12)$$

In this study:

$$\varepsilon_2 = \varepsilon_3 \quad (13)$$

where α is the Biot coefficient, which is approximately 1; U_0 is the total energy absorbed by the specimen in the hydrostatic stage, kJ/m^3 ; B is the ratio of minimum principal strain to maximum principal strain; E is the elastic modulus of the specimen, MPa; and v is the Poisson's ratio of the specimen. The elastic modulus and Poisson's ratio are generally those of the specimen in the elastic stage.

The energy indexes of samples with different bedding angles during peak stress and sample failure are shown in Figure 11. The main energy triggering coal sample failure is the stored elastic strain energy inside the specimen. When the stress reaches its peak, the coal sample reaches its energy storage limit, and a larger elastic strain energy indicates the sample is less likely to fail. The evolution of strain energy in coal samples with different bedding planes exhibits strong isotropy, as shown in Figure 11a. At the peak stress stage, with the increase in the bedding angle, the total strain energy U and elastic strain energy U_e first decrease and then increase in a V-shaped trend. Both are highly sensitive to bedding and show obvious anisotropy. When the bedding angle is 0° , the total strain energy U and elastic strain energy U_e of the coal sample reach their maximum values. When the bedding angle is increased to 60° , the total strain energy U and elastic strain energy U_e reach their

minimum values. The variation trend in dissipated energy U_d is close to the horizontal, the sensitivity to the bedding angle is low, and there is no obvious anisotropy.

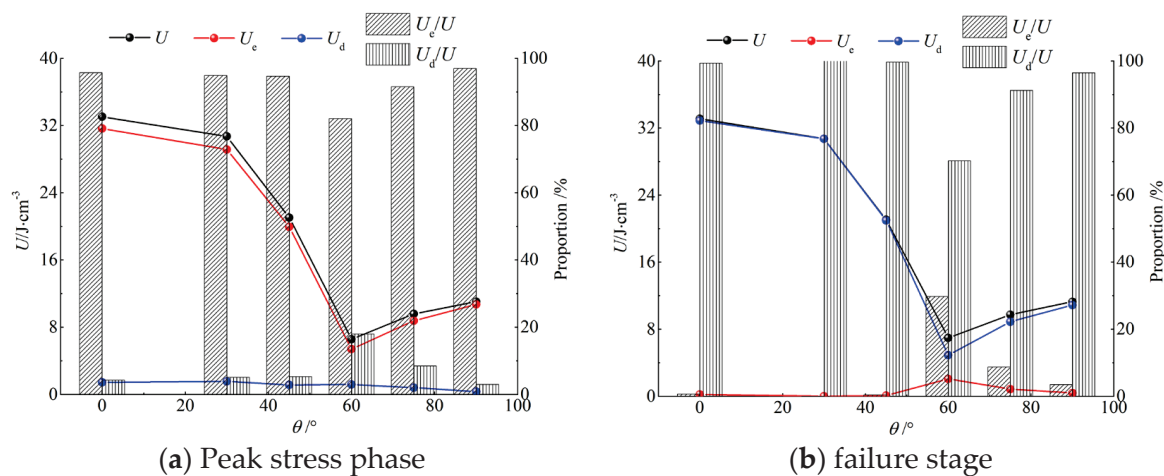


Figure 11. Variation in energy with different coal bedding angle.

During the failure phase, almost all U_e stored in the coal sample before the peak is converted to U_d for rapid crack propagation and penetration. With the increase in the bedding angle, the curves of total strain energy U and dissipated energy U_d basically coincide, showing a V-shaped trend, and the anisotropy is significant. When the bedding angle is 0° , the dissipation energy required for sample failure is at its largest. When the bedding angle is 60° , the dissipation energy required for sample failure is at its minimum, and the sample is most prone to failure. This is directly related to the failure mode of the sample. The sample at 60° showed shear slide failure along the plane, and the energy loss was minimal. On the other hand, the 0° specimen underwent composite failure, penetrating both the matrix and the bedding plane, resulting in greater energy loss. Therefore, specimen failure is the most difficult to occur.

The effects of layer -rated inclination angle on the strain, strength, penetration rate and energy evolution of gas-bearing coal. Studies have found that when the layers of inclination angle are 45° , 60° , and 75° , the intensity of the test parts is low, and the strain varies when the test parts are damaged. In addition, with the increase in the coal seam inclination angle, the energy storage limit of the sample decreases, and the coal seam is more likely to lose stability. When the inclination angle is 60° , the proportion of the strain can be converted into decentralized energy, and the risk of coal, rock, and gas disasters is high; protective measures should be taken. At the same time, the mining angle and mining process of the working surface can also be adjusted according to the coal seam inclination angle. In addition, in the process of natural gas mining, the diamond angle and natural gas coal seam inclination can be adjusted. When the angle is 60° , the penetration behavior of the gas is more convenient.

4. Discussion

In order to further understand the relationship between coal bedding angle and the form of failure, the failure mechanism of the coal body is further analyzed with the Moore–Coulomb criterion. For intact rocks, according to the M-C strength criterion [5], when shear failure occurs along a certain interface, the shear strength is shown in Equation (14):

$$\tau_d = \sigma'_n \tan \varphi + c \quad (14)$$

where τ_d is shear strength, MPa; σ_n is normal stress, MPa; c is the cohesive force inside the coal, MPa; φ is the internal friction angle, ($^\circ$).

As shown in Figure 12a, according to the M-C stress circle theory, the relationship between normal stress and shear stress on the failure shear plane is shown in formulas (15) and (16):

$$\sigma'_n = \frac{\sigma'_1 + \sigma'_3}{2} + \frac{\sigma'_1 - \sigma'_3}{2} \cos 2\alpha \quad (15)$$

$$\tau_d = \frac{\sigma'_1 - \sigma'_3}{2} \sin 2\alpha \quad (16)$$

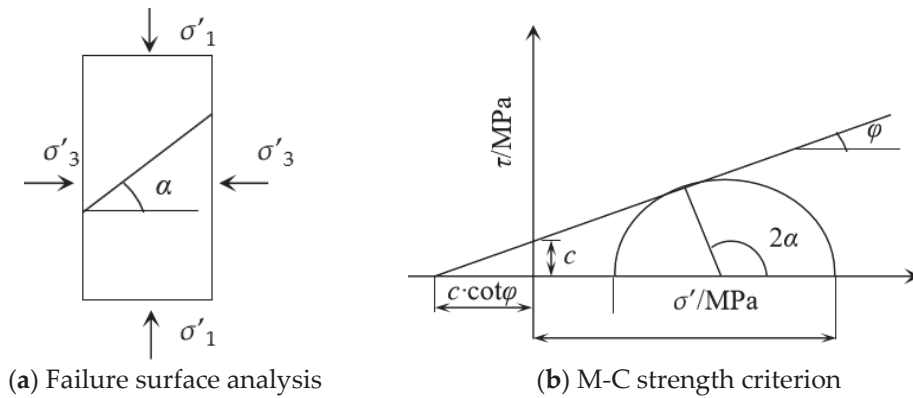


Figure 12. Rock failure surface and M-C strength criterion.

As shown in Figure 12b, if the failure of the intact rock complies with the M-C strength criterion, the conditions for rock failure along the shear plane can be obtained by substituting Equation (17), as shown in the following equation,

$$\sigma'_1 - \sigma'_3 \geq \frac{2c + (\sigma'_1 + \sigma'_3) \tan \varphi}{(1 - \tan \varphi \cot 2\alpha) \sin 2\alpha} \quad (17)$$

where α is the angle between the failure surface and the minimum principal stress, ($^\circ$).

For coal rock with a weak bedding plane, shear slip failure may occur along the weak bedding plane, on the one hand, and shear failure may occur across the bedding plane, on the other hand, with a change in bedding direction when loading the coal sample.

As shown in Figure 13, according to the Moore–Coulomb strength criterion, when the molar circle is located below the failure envelope of the structural plane, that is, when $2\alpha < 2\alpha_1$ or $2\alpha > 2\alpha_2$, the coal body does not fail along the bedding plane, but shear failure occurs across the bedding plane.

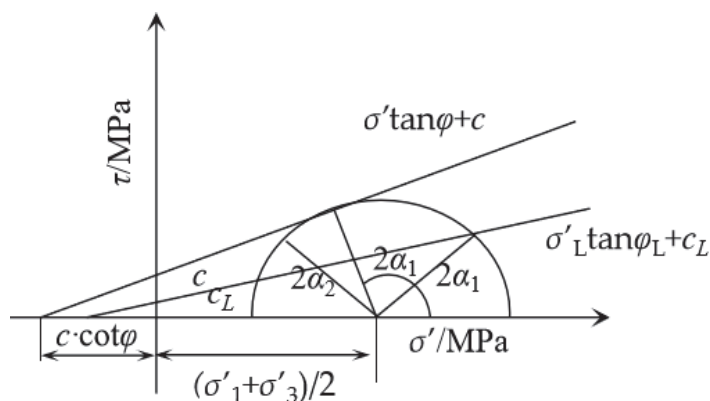


Figure 13. Shear failure criterion of coal.

When the mole circle is located above the structure line, that is, when $\alpha_1 \leq 2\alpha \leq 2\alpha_2$, the coal body will be destroyed along the weak plane of bedding, and the $\alpha = 45^\circ$ coal sample will conform to this failure form. When coal shear slip failure occurs along a weak bedding plane, the relationship between shear failure strength and normal stress on weak the bedding plane is shown in Equation (18):

$$\tau_L = \alpha'_L \tan \varphi_L + c_L \quad (18)$$

By substituting formula (18) into formula (15) and (16), it can be obtained that the failure condition of a coal body along the weak plane of bedding is shown in Formula (19),

$$\sigma'_1 - \sigma'_3 \geq \frac{2c_L + (\sigma'_1 + \sigma'_3) \tan \varphi_L}{(1 - \tan \varphi_L \cot 2\alpha) \sin 2\alpha} \quad (19)$$

where τ_L is the shear failure strength of the weak plane of bedding, MPa; σ'_L is the normal stress on the weak surface of bedding, MPa; φ_L is the internal friction angle of the weak plane of bedding, °; and c_L is the cohesion between the weak planes of the bedding, MPa.

As shown in Figure 13, the angle between the minimum principal stress and the bedding surface is α_1 . The conditions for coal sample failure along the structural plane are shown in Equation (20),

$$\frac{c_L \cot 2\varphi_L + \frac{\sigma'_1 + \sigma'_3}{2}}{\sin(2\alpha_1 - \varphi_L)} = \frac{\frac{\sigma'_1 - \sigma'_3}{2}}{\sin \varphi_L} \quad (20)$$

After simplification, the calculation formula of the critical angle α_1 can be obtained, as shown in Equation (21),

$$\alpha_1 = \frac{\varphi_L}{2} + \frac{1}{2} \arcsin \left[\frac{2\varphi_L (c_L \cot \varphi_L + \sigma'_1 + \sigma'_3)}{\sigma'_1 - \sigma'_3} \right] \quad (21)$$

Similarly, the formula for calculating the critical angle α_2 can be obtained.

As can be seen from formula (19), when $2\alpha \rightarrow 0^\circ$, $2\alpha \rightarrow 180^\circ$ or $2\alpha \rightarrow \varphi_L$, $\sigma'_1 - \sigma'_3 \rightarrow \infty$, the coal will inevitably not be damaged along the weak plane of bedding, but will tend to be damaged by the material itself, reflecting the strength characteristics of the material itself. In the process of coal rock failure from structural plane failure to complete rock failure, the coal body surface comes to be in a composite failure form, with shear failure accompanied by tensile failure [38].

Through the relationship between α and the failure mode, the change in failure mode of a coal sample with the angle of bedding can be deduced. According to the concrete test results, shear slip failure occurs when coal is loaded along a certain range of bedding angles, and the failure form of 0° , 45° , 60° coal sample conforms to this situation. At $\theta = 90^\circ$, the loading direction of the coal is parallel to the bedding direction, and the coal will be adjusted to the radial direction during the loading process, and the cutting action of the cutting makes the integrity of the coal poor, resulting in the tensile failure of the coal, resulting in the failure of the intact rock. The failure mode of the coal sample with $\theta = 0^\circ$ is shear failure across the bedding plane, and the loading direction is perpendicular to the bedding plane. Under the action of confining pressure, the coal sample is more inclined to the shear failure of intact coal.

The relationship between α and the failure mode can be used to deduce the change in failure mode of the coal sample with the angle of bedding. According to the concrete test results, the shear slip failure occurs when the coal and rock are loaded along a certain range of bedding angles, and the failure form of the 30° , 45° , and 60° coal samples conforms to this situation.

When the loading direction of the $\theta = 90^\circ$ coal sample is parallel to the bedding direction, the coal body will adjust to the radial direction during the loading process, and the cutting action of the cutting makes the integrity of the coal body poor, resulting in the tensile failure of the coal body, resulting in the failure of the intact rock. When the $\theta = 0^\circ$ coal sample undergoes shear failure across the bedding plane, and the loading direction is perpendicular to the bedding plane, under the action of confining pressure, the coal sample is more inclined to the shear failure of intact rock. When the 90° coal sample is loaded, the existence of cleavage cracks weakens the integrity of the coal body. The carrying capacity of the coal sample mainly reflects the bonding strength between the coal particles and the bonding strength between the bedding planes, and the cohesion is also smaller than that of the 90° coal.

This paper studies the mechanical–exudation–energy evolution process of gas-containing coal under different levels of rational angle at different levels of rational angles, and analyzes the theoretical analysis of the mechanism of damage to the coal body. A preliminarily mechanism for the loss of stability and destruction of gas-containing coal under different levels of rational angle was obtained. It is inspired by the prevention and control of disasters in coal mining. The mining method of coal seams at different angles and the need for pressure removal, soaring, and supporting measures requires specific means, such as the main timely decay of the mining of vertical inclination coal seams. Measures to prevent accidents and, when the coal seam dumping angle and mining angle presents as 45° , 60° , and 75° , the smoothness of the coal seam can be considered. However, due to the complex geological conditions of a coal mine, the surrounding air fields, temperature fields, and stress fields are intertwined. The test results and the actual situation at the scene can differ. According to the above conclusions, the author intends to find a typical coal seam layout stress monitoring device at the coal mine of Xunchun Mining Group to conduct on-site monitoring of the coal seam deformation to guide the safety and efficient production of working faces.

5. Conclusions

This study conducted a triaxial stress–permeability test on gas-bearing coal under different bedding angles. It analyzed the impact of bedding angle on strain, strength, permeability, and the energy evolution characteristics of gas-bearing coal. The following conclusions were drawn:

- (1) The change in bedding angle affects the deformation ability of coal samples containing gas. ε_1 and ε_3 both first increase and then decrease. When the bedding angle is 45° , the failure of the coal sample containing gas is greatly affected by the bedding angle, and instability failure occurs along the bedding plane.
- (2) With the change in bedding angle, the coal exhibits strong anisotropy in terms of peak stress, elastic modulus, peak strain, and other characteristics. The minimum permeability of the sample increases with the increase in bedding angle. The bedding direction of the 90° and 75° coal samples is consistent with the axial direction, the bedding fractures are well developed, and the seepage channels are greater.
- (3) At the peak stress stage, the total strain energy U and the elastic strain energy U_e first decrease and then increase with the increase in the bedding angle. During the failure stage of the specimen, the curve of the total strain energy U and the dissipated energy U_d almost overlap, exhibiting a V-shaped trend. At an angle of 60° , the minimum dissipated energy U_d is required for specimen failure, indicating that the sample is more prone to failure.

Author Contributions: X.T.: conceptualization, data curation, and writing—original draft. F.X.: data curation and formal analysis. Y.B.: formal analysis. R.B.: funding acquisition and methodology. Y.G.: formal analysis and visualization. N.H.: formal analysis and visualization. All authors have read and agreed to the published version of the manuscript.

Funding: This research was funded by the National Natural Science Foundation of China (52404223, 52204212), the China Postdoctoral Science Foundation (2024M751711), the S&T Program of Hebei (22375401D), and the Fundamental Research Funds for the Central Universities (3142020003).

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

Conflicts of Interest: Author Xiaojun Tang was employed by the company Hunchun Mining Group Co., Ltd. The remaining authors declare that the research was conducted in the absence of any commercial or financial relationships that could be construed as a potential conflict of interest.

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Article

Borehole Failure Mechanics and Influencing Factors in a Gas-Bearing Soft Coal Seam Under Complex Geological Conditions

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Abstract: The present research focuses on the mechanical properties and stress evolution of gas-bearing soft coal seams during drilling, which are affected by a multitude of complex factors such as high ground stress, gas pressure, and pre-existing fractures. In this study, a combination of PFC2D (Particle Flow Code in 2 Dimensions) numerical simulation and theoretical analysis is employed to investigate the borehole mechanics and fracture evolution characteristics under diverse complex conditions and to determine the factors influencing different forms of borehole failure in soft coal seams. The principal outcomes are as follows: (1) At a horizontal displacement of 0.1 m from the borehole orifice of the soft coal seam, a stress peak value of 13.9 MPa is attained; the peak value of the coal body contact force is 15.8 MPa; the peak value of the displacement is 0.008 m; and the porosity of the coal body around the borehole ranges from 0.14 to 0.35. (2) With an increase in the number of pre-existing fractures, the inclination progressively aligns with that of the pre-existing fractures. Maximum values of contact force (5.13–51.9 MPa), stress (3.19–37.2 MPa), shape dimension, and fracture angle (140–150°) are achieved under the highest lateral pressure coefficient and gas pressure (1.5 MPa). (3) The borehole energy is directly proportional to the number of pre-existing fractures, the lateral pressure coefficient, and gas pressure. The number of pre-existing fractures has the most significant impact on the damage degree, followed by the lateral pressure coefficient and then the gas pressure. (4) Two types of failure are identified: fracture-dominated failure, which is controlled by the geometric distribution of pre-existing fractures, and stress-dominated failure, wherein the failure zone gradually extends both upward and downward with an increasing lateral pressure coefficient.

Keywords: gas-bearing soft coal seams; borehole mechanics; fracture evolution; complex conditions; failure types

1. Introduction

To help achieve its “double carbon” goal, China is pursuing green and low-carbon development, including safe and efficient energy mining. However, total domestic energy consumption reached a peak in the last decade, with coal consumption representing

55.3% of the total value in 2023. Furthermore, coal production was predicted to reach 4.7–4.8 billion tons in 2024 [1,2]. As the main energy source in China, coal contains abundant associated resources, such as gas. Coal seam gas, an unconventional natural gas present in coal seam holes and fractures, is an important green energy source in China. Moreover, the efficient extraction of this gas can effectively prevent coal mine gas disasters.

Drilling coal seams allows gas to migrate into boreholes through desorption and seepage under the action of negative pressure. However, boreholes are prone to collapse and deformation during this process. Therefore, borehole quality is a major factor influencing the efficacy of gas extraction. In a previous study investigating borehole stress and failure, Wang et al. [3] established a borehole stress model to analyze failure characteristics and observed that holes in the broken zone and jet holes in the plastic zone could easily collapse during excavation and that a large stress gradient around the borehole in the radial direction is conducive to fracture expansion parallel to the principal stress direction. Dong [4] analyzed the mechanics of a coal body during drilling and concluded that the stress in the plastic zone increased linearly with the cohesive force and friction angle of the coal body. The borehole fracture strength and coal body connectivity exhibit substantial regional differences [2]. Typically, with increasing stress during borehole excavation, the borehole enters a pre-peak stress concentration stage, and new fractures appear and expand in the coal body along the direction of the borehole principal stress [5]. Once the borehole enters the post-peak stress concentration stage, local deformation occurs in the borehole, and new fractures interconnect to form a large fracture network [6]. Zhang et al. [7] conducted uniaxial tests on borehole samples with different compressive strengths and observed that boreholes with low compressive strengths exhibited more complex failure forms, faster fracture expansion, and larger displacements.

Coal seams in China have complex geological conditions. In particular, soft coal seams are widely distributed and are characterized by high gas content, high gas pressure, and low permeability. Moreover, the difficulty in drilling holes in soft coal seams increases the risk of borehole failure. In previous research investigating the failure mechanism and fracture development characteristics of boreholes in soft coal seams or complex structures, Zhao [8] constructed an SRM (three-dimensional synthetic rock mass) model and performed numerical simulations of multi-stage reaming. They observed that under a high confining pressure, an increase in the structural coal burial depth led to a more significant borehole failure and lower porosity. Moreover, an experimental study examining the stress variation characteristics around boreholes in soft coal seams demonstrated that increasing the ground stress and gas pressure continuously increased the stress around boreholes and expanded the high-stress area to a greater depth [4]. Another study [9] adopted ABAQUS-2022 (ABAQUS is a powerful finite element software for engineering simulation) technology to conduct numerical simulations under different ground stresses and gas pressures, and the results indicated that the peak borehole stress in a gas-bearing coal seam moves toward the periphery of the borehole with increasing ground stress and gas pressure. Furthermore, Sun et al. [10] established a numerical model of a gas extraction borehole using PFC2D-5.00.25 software and demonstrated that both the stress and fracture development around the borehole increased with increasing burial depth and lateral pressure coefficient. Yang and Zhao [11] used COMSOL-6.2 (COMSOL Multiphysics, abbreviated as COMSOL, is a multi-physics simulation software) numerical simulation software to explore the degree of borehole failure under different ground stresses and concluded that an increase in vertical stress increases the plastic zone around the borehole. Some scholars have studied borehole failure mechanisms and characteristics under other complex conditions. Lin et al. [12] reported that a larger aperture reduces the drilling initiation stress. However, borehole failure is more significant under high-stress and high-temperature conditions. In recent

years, a considerable number of scholars have comprehensively examined the relationships among multiple factors, including in situ stress, temperature field, formation permeability, and the level of borehole wall instability. They have also delved into the laws governing stress distribution around the borehole [13]. It has been determined that the non-uniform distribution of thermal stress within the borehole under diverse temperature fields exerts a substantial influence on the initiation and propagation of cracks in the borehole [14,15]. After exploring the influences of borehole burial depth, lateral pressure coefficients, and other factors on borehole failure during deep coal seam extraction using a three-dimensional borehole model, Li et al. [16] concluded that ground stress was the main factor affecting borehole instability. Evidently, borehole instability exhibits an exceedingly high level of complexity. Notably, within the gas-abundant regions of deep coal seams, instabilities in stress, temperature, gas, and liquid states are present [17,18]. As a result, the “solid-liquid-gas” three-phase coupling environment renders the mechanism and influence of borehole instability under complex circumstances still subject to further exploration.

However, previous research investigating coal seam drilling and excavation has focused on the fracture development and stress change characteristics around the borehole. Therefore, only the macroscopic laws of borehole failure have been determined. Additionally, previous numerical simulations of borehole fractures in soft coal seams under complex conditions have focused on fracture development and mechanical characteristics under different factors. Thus, the basis for establishing a borehole model remains unclear. Moreover, summaries and classifications of the complex factors that influence borehole failure are lacking. Finally, studies investigating the mechanical fracturing of boreholes in gas-bearing soft coal seams under different structural conditions have considered only a single borehole failure method.

Therefore, in this study, the stress distribution characteristics around boreholes in gas-bearing soft coal seams are analyzed and the failure mechanisms of coal bodies under gas pressure are explored. The mechanical properties of the coal body around the borehole, including stress, strain, force chain, and displacement, are obtained through PFC2D numerical simulations. Subsequently, the borehole mechanics and fracture evolution characteristics of soft coal seams under different complex conditions are analyzed using fractal dimension calculations. Then, the degree of influence of different factors is determined according to the number of borehole fractures, thereby revealing the main factors affecting borehole failure and instability. Finally, the major types of borehole failure in soft coal seams are determined. Thus, the results provide an important reference for optimizing borehole design for deep coal gas extraction.

2. Theoretical Analysis

2.1. Mechanical Stress Distribution Around a Borehole in a Gas-Bearing Soft Coal Seam

The original stress state of the coal body around the borehole changes during the construction process, and a new stress-concentration area is formed around the borehole after construction is completed. Accordingly, the stress state around the borehole changes successively with distance from the borehole as follows: depressurization failure zone, post-peak stress-concentration zone, pre-peak stress-concentration zone, and original rock stress zone. Along the direction of the bore diameter, there are four stages successively—broken zone, plastic zone, elastic zone, and primary rock stress zone [3]—as shown in Figure 1.

(1) Elastic deformation stage

$$\sigma_{\theta} + \alpha p_r \delta_{ij} = K_p (\sigma_r - \alpha p_r \delta_{ij}) + \sigma_c \quad (1)$$

Here, σ_{θ} is the tangential stress of the coal, α is the equivalent pore pressure coefficient, p_r and δ_{ij} are radial gas pressure and Kroneker symbols, K_p is the degree of softening of

the coal, σ_r is the radial stress of the coal, and σ_c is the uniaxial compressive strength in plastic deformation.

(2) Plastic deformation stage

For a constant internal friction angle, a change in cohesion will lead to a decrease in coal strength; thus, the conditions for plastic deformation are

$$\sigma_\theta + \alpha p_r \delta_{ij} = K_p(\sigma_r - \alpha p_r \delta_{ij}) + \sigma_c(\varepsilon_1^p) \quad (2)$$

Here, ε_1^p is the compressive strength.

(3) Residual deformation stage

The strength of coal is reduced to the residual strength, and the equilibrium condition is

$$\sigma_\theta + \alpha p_r \delta_{ij} = K_p(\sigma_r - \alpha p_r \delta_{ij}) + \sigma_c^* \quad (3)$$

Here, σ_c^* is the residual compressive strength.

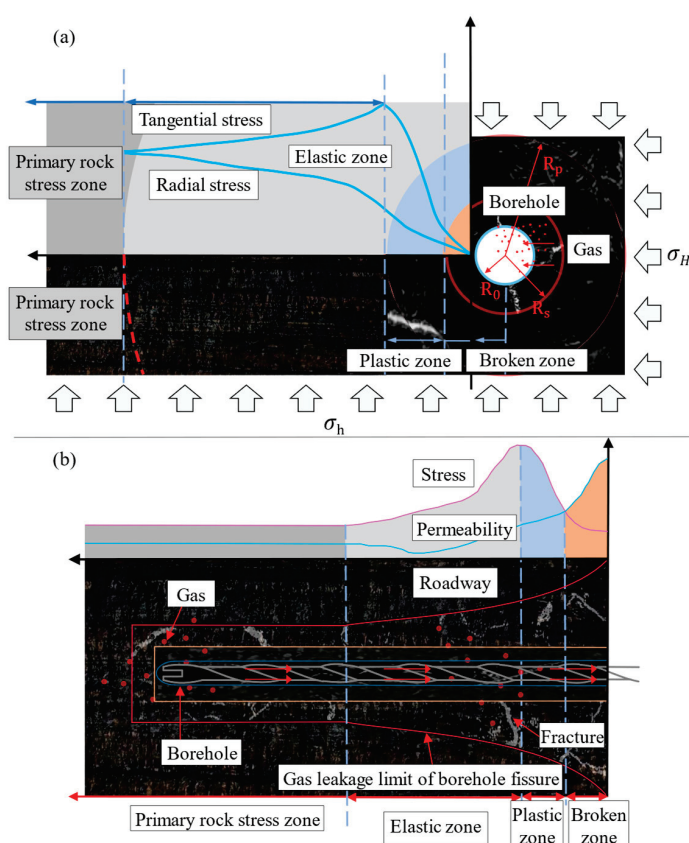


Figure 1. Distribution of (a) axial and (b) radial stresses around a borehole in a coal body.

2.2. Theoretical Analysis of Coal Weakening Mechanism Under Gas Pressure

The deformation and failure of gas-bearing coal are governed by the external stress and the mean pore gas pressure. As depicted in Figure 2, the mechanism through which gas pressure impairs the mechanical characteristics of gas-bearing coal is presented. Specifically, the Mohr circle associated with gas-bearing coal progressively migrates to the right. The strength line of gas-bearing coal exhibits a diminished τ intercept in comparison to that of ordinary coal. This implies that an augmentation in gas pressure not only diminishes the peak strength and elastic modulus of gas-bearing coal but also curtails the cohesion and internal friction angle, thereby leading to a reduction in mechanical strength. Such

a variation in the mechanical strength parameters of gas-bearing coal is manifested by the cohesion [19]:

$$\Delta C = p \tan \varphi \quad (4)$$

where p is the gas pressure, φ is the internal friction angle of coal, and ΔC is the reduced cohesion value. The cohesion strength of gas-bearing coal is then reduced and corrected as follows:

$$C_{loc} = \frac{C - p \tan \varphi}{\omega} \quad (5)$$

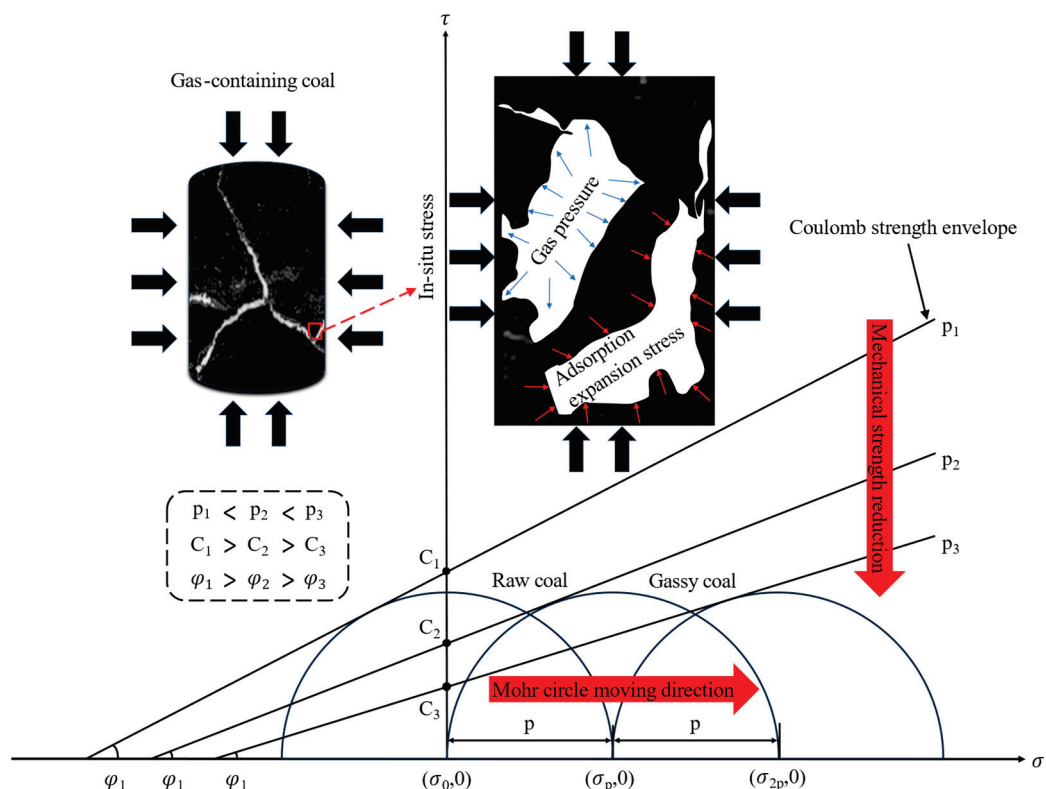


Figure 2. Schematic of the mechanical strength reduction in gas-bearing coal.

Here, C_{loc} is the reduced cohesion and ω is the reduction factor, which is defined as the safety factor.

2.3. Quantitative Characterization of Borehole Fractures

To reflect the complexity and difference in the development of microscopic fractures in coal-seam boreholes, fractal dimension analysis is employed [20]. The basic concept of the box-counting method is shown in Figure 3. Firstly, ImageJ-1.54h software was used to binarize the fracture image (Figure 3a) to obtain an area (Figure 3b) containing two gray values, in which the black area represents the coal matrix and the white area represents the fractures. Fracture identification and extraction of the image in Figure 3c were performed on the image in Figure 3b, and the fractal dimension of the image in Figure 3c was calculated using the box-counting method. In two-dimensional images, the box-counting method covers the fracture image with a square grid with side length r , as shown in Figure 3d. The number of empty meshes without fracture coverage was denoted as $N(r)$, and the number of multiple groups of empty meshes was obtained by changing the side length of the mesh, as shown in Figure 3e. Logarithms of the mesh side length r and the number of empty meshes $N(r)$ in each group were, respectively, taken, and the slope, D , was obtained by linear fitting, giving the fractal dimension (Figure 3f). The fractal dimension can be expressed as follows:

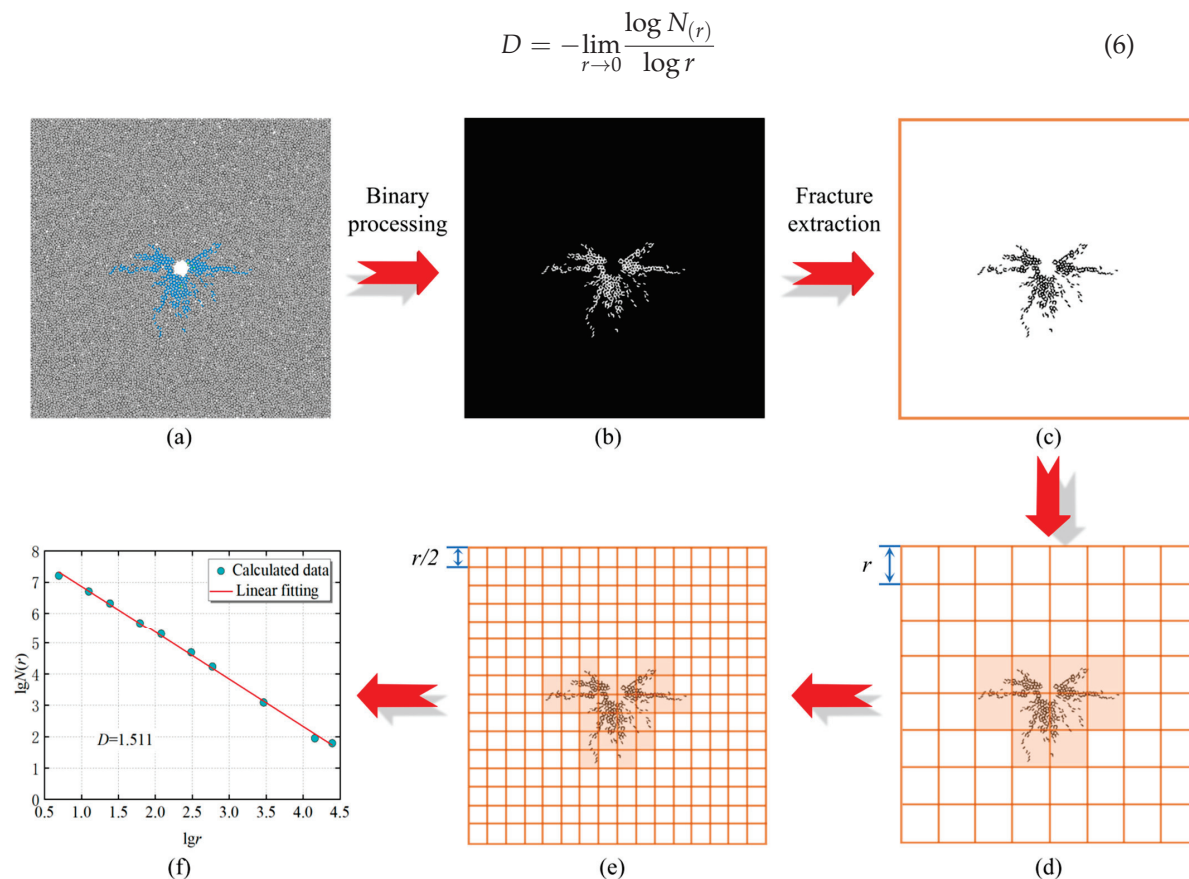


Figure 3. Principle of calculating fractal dimensions via the two-dimensional box-counting method: (a) Fracture image; (b) Image binarization; (c) Fracture extraction results; (d) Grid division with side length r ; (e) Grid division with side length $r/2$; (f) Fractal dimension calculation.

3. Discrete Element Numerical Simulation Analysis

When the peak stress exceeds the compressive strength of the coal seam, the coal body around the borehole experiences shear and tensile displacement, resulting in a fracture area and fracture field surrounding the borehole.

3.1. Parameter Calibration and Numerical Modeling

PFC2D contains several built-in contact models. The linear parallel contact model can lead to relative motions inside the material and produce forces and moments. Once the maximum normal stress and shear stress exceed the bonding strength, the parallel bond will rupture, which makes it capable of accurately simulating the properties of soft and fractured coal samples. Numerical calculations were carried out using the linear parallel contact model. Firstly, parameters are calibrated based on the mechanical parameters of raw coal. In this study, a test coal sample from a working face of the 5# coal seam in Xiangshan Mine, Hancheng Mining area, Shanxi Province, China, is utilized. This sample is a highly metamorphic bituminous coal mainly comprising pulverized and granular coal, with low hardness and a minimum firmness coefficient of 0.21. This working face has experienced relatively severe borehole failure with complex fracture development.

Figure 4 displays the results of parameter calibration for the model; the specific mechanical parameters are shown in Table 1 [21]. A borehole numerical model was constructed, having a height and width of 2 m and the minimum and maximum particle sizes being $R_{\min} = 0.008$ m and $R_{\max} = 0.012$ m, respectively (Figure 5a). In the simulation, a confining pressure of 10 MPa was set up around the model. Then, the borehole excavation

procedure was realized by deleting the circular area with a diameter of 10 cm in the middle of the numerical model. Regarding data monitoring, measurement circles were disposed to monitor the changes in stress and porosity around a borehole with a radius of 0.05 m. The measurement circles were configured to the same size to more accurately obtain the stress and porosity alterations around the borehole, taking the borehole coordinates as the origin. Ten measurement circles were arranged in both the positive and negative directions of the x -axis, and the radius of the measurement circles was fixed at 0.05 m (Figure 5b,c).

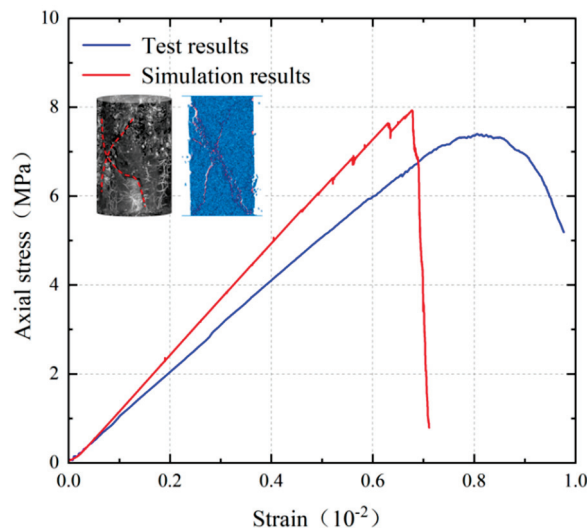


Figure 4. Calibration results of the uniaxial stress–strain curve.

Table 1. Linear parallel contact model parameters.

Serial Number	Parameter	Symbol and Units	Name	Value
1	Rmin	m	Minimum radius of particles	0.008
2	Rmax	m	Maximum radius of particles	0.012
3	Damp	1	Damping coefficient	0.1
4	ρ	Kg/m ³	Particle density	2000
5	Emod	GPa	Contact elastic modulus	16
6	Kratio	1	Contact stiffness ratio	1.5
7	fric	1	Friction coefficient	0.5
8	pb_fa	1	Parallel bonding internal friction angle	30
9	pb_ten	MPa	Parallel bond tensile strength	20
10	pb_coh	MPa	Parallel bond cohesion	60

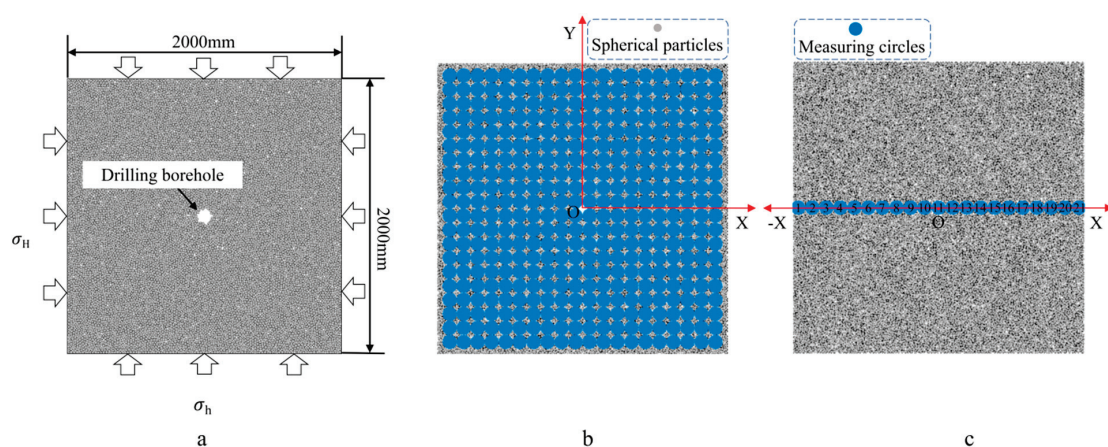


Figure 5. Borehole numerical model and distribution of measurement circles: (a) Borehole numerical model; (b) Measurement circle distribution; (c) Number of measurement circle settings per row.

3.2. Numerical Calculation Results

Figure 6 shows the mechanical state evolution around the borehole. A concentrated area of pressure chains is generated on both sides and the lower end of the borehole (Figure 6a); these force chains mainly extend toward both sides of the horizontal side and down the vertical side to the same length, and the particle contact force drops to 1–2 MPa, forming an irregular distribution of force chains [22]. Under gravity and confining pressure, the contact between particles causes extrusion deformation [23]. Figure 6c,d shows the distribution of stress and strain in the horizontal direction of the borehole, respectively, where red, green, and yellow colors indicate the surge area, plunge area, and flat area, respectively. The stress and strain data are monitored through the measurement circle. The vertical stress of the borehole rises sharply at a horizontal distance of 0.05–0.10 m from the borehole, and the maximum and minimum stresses are 13.9 MPa and 1.21 MPa, respectively, -0.1 m from the borehole. The strain values on both sides of the borehole rise rapidly at a horizontal distance of -0.1 to 0.1 m from the borehole, and peak strain is reached -0.1 m from the borehole (Figure 6d).

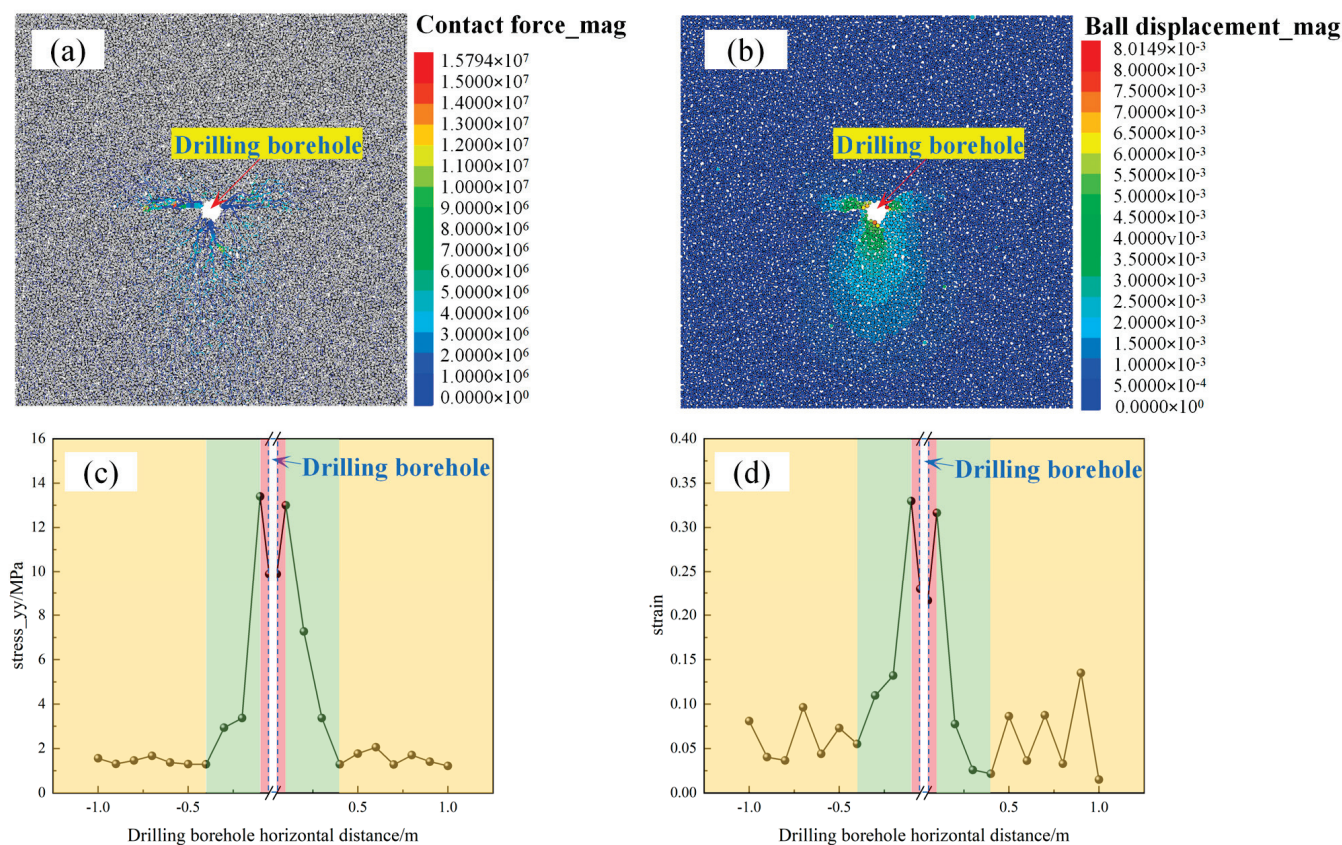


Figure 6. Schematic of the mechanical state evolution around the borehole: (a) force chains; (b) displacement vector; (c) stress; and (d) strain.

Figure 7a shows the fracture distribution characteristics around the borehole. To clearly display the distribution of fracture orders, UDScalar in PFC2D is used to draw the fracture hot-spot diagram (Figure 7b), where the size of the circle corresponds to the length of the fracture, and the color of each dot indicates the number of fractures in the area covered by that dot. The number of fractures around the borehole within 1 m of the search radius was calculated using the borehole opening as the center of the circle. It is identified that the edge of the hot-spot map predominantly shows small dots, signifying few fractures. The fractures generated after drilling can impact the porosity around the borehole [24].

Therefore, a measurement circle is used to obtain the porosity change around the coal seam after drilling, as shown in Figure 7c. The overall porosity at a horizontal distance of -0.1 to 0.2 m from the borehole is high, ranging from 0.14 to 0.35 . Outside of this region, the porosity of the coal body shows an irregular distribution but is predominantly stable at 0.15 – 0.18 .

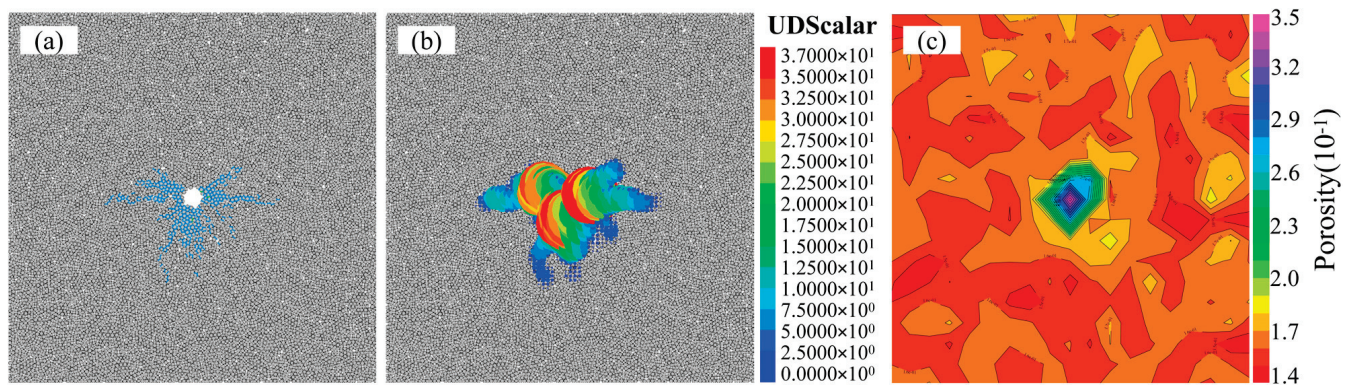


Figure 7. Distribution of fractures around the borehole: (a) fractures; (b) distribution of fracture hot spots; and (c) porosity.

4. Mechanical Properties of Boreholes in Gas-Bearing Soft Coal Seams

With the gradual shift toward deep and large-scale resource mining, the deep underground engineering environment is becoming increasingly complex [25,26]. Notably, underground drilling in coal mines is impacted by naturally formed joints and fractures in the coal and rock mass, which contribute to the initiation of borehole fractures [27]. Moreover, the heterogeneous nature of coal-seam-bearing structures leads to different lateral pressure coefficients that also impact the development characteristics of fractures [28]. In addition, the mechanical properties of coal are significantly impacted by gas pressure, which can change the stress field and fracture distribution during the drilling process in gas-bearing coal [29,30].

4.1. Mechanical Evolution of Boreholes According to the Number of Pre-Existing Fractures

To study the impact of pre-existing fractures on sample strength and fracture evolution features, a discrete fracture network is designed and built with PFC2D. Figure 8 shows coal body models with one, three, and six pre-existing fractures, where the fracture length (2d) is 400 mm. The fractures in all three models are parallel to each other, and the fracture inclination is 30° . The length of the rock bridge in Figure 8b is $2e = 200$ mm, whereas the lengths of the horizontal rock bridge and vertical rock bridge in Figure 8c are $4e = 400$ mm and $2e = 200$ mm, respectively.

Figure 9 shows the mechanical evolution of three boreholes with different numbers of pre-existing fractures, corresponding to the models in Figure 8. According to Figure 9a, the force chain gradually extends in the direction of pre-existing fractures; it increases to 7.8, 27.3, and 74.5 MPa for the three borehole types, respectively. The coal displacement at the two sides of the borehole with a single fracture is the least, and the displacement under the borehole is highly affected by the quantity of fractures. The coal displacement around the borehole keeps increasing, reaching 11, 22.6, and 23.4 mm in turn. The general trend of coal displacement is consistent with the angle of pre-existing fractures. As shown in Figure 9c, the stress and strain in the coal seam during drilling and excavation gradually increase. The peak stress and strain increase remarkably to 5.8, 9.2, and 10.6 MPa and 0.0004, 0.0029, and 0.0043%, respectively.

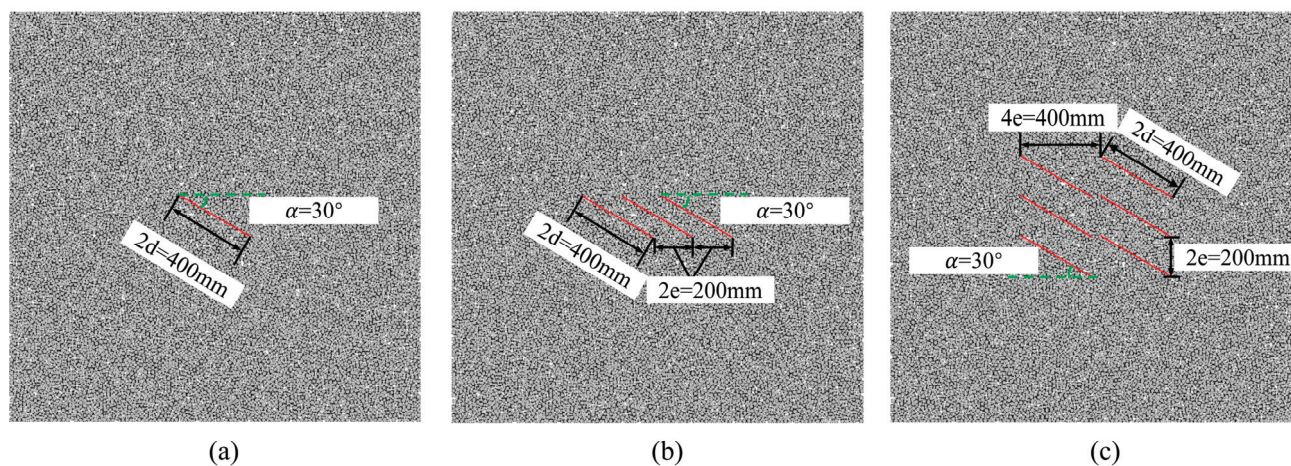


Figure 8. Discrete fracture network models for a coal body with different numbers of pre-existing fractures: (a) one; (b) three; and (c) six.

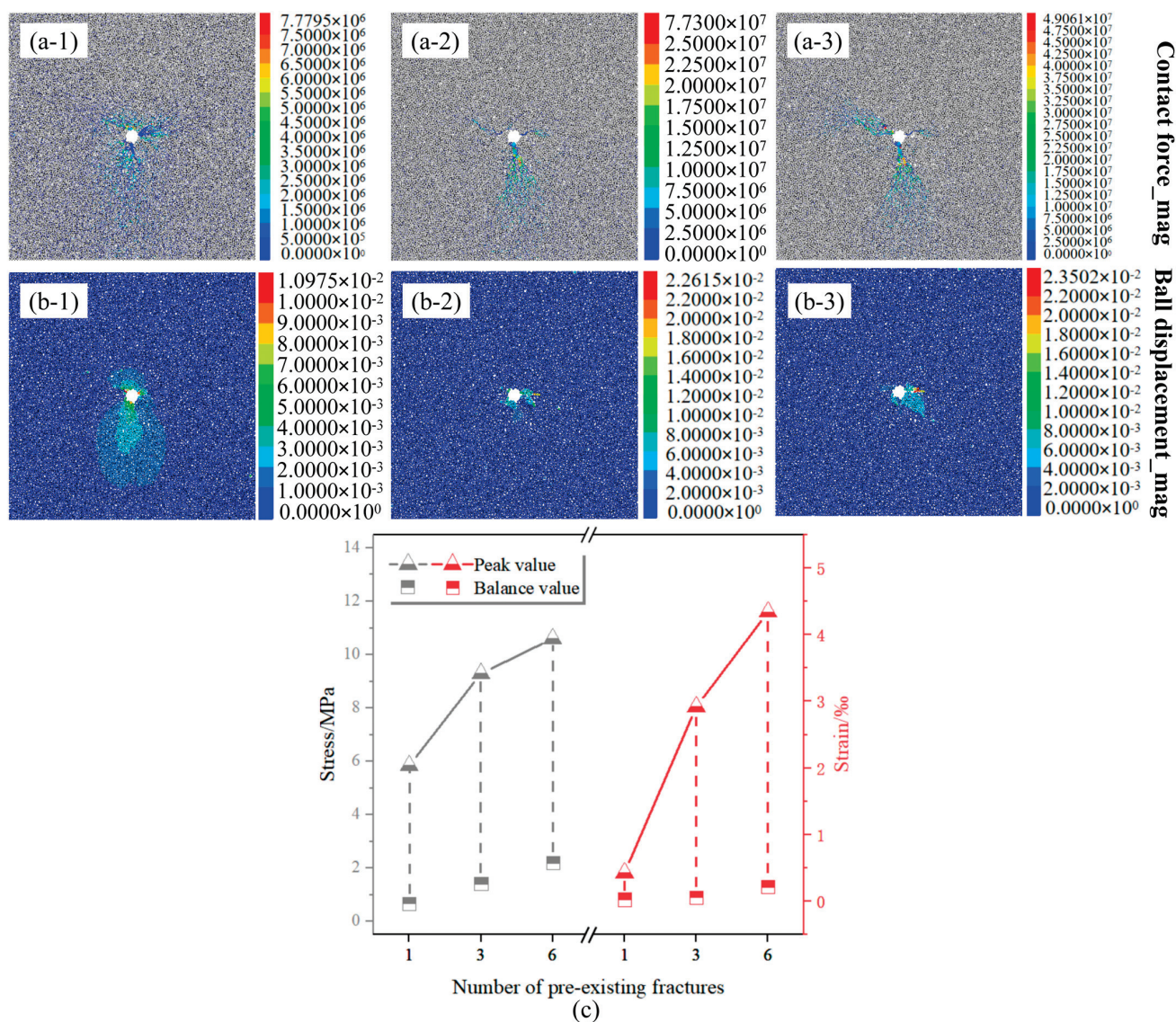


Figure 9. Mechanical state evolution around a borehole with pre-existing fractures. (a) Force chain, (b) coal displacement, and (c) stress and strain evolution in a coal body with an increasing number of pre-existing fractures. (a-1,b-1), (a-2,b-2), and (a-3,b-3) correspond to the models in Figure 8a, b, and c, respectively.

Figure 10a shows the fracture distribution characteristics around boreholes with pre-existing fractures. The fractures tend to develop in the same direction as pre-existing fractures and continually increase in length, which is mainly attributed to local failure caused by the bridge connecting the rocks adjacent to pre-existing fractures in the coal seam. The distribution law of fracture quantity is shown in Figure 10b. Fractures in the concentrated fracture area develop perpendicular to the inclination direction of pre-existing fractures, with quantities of 28, 29, and 36, respectively.

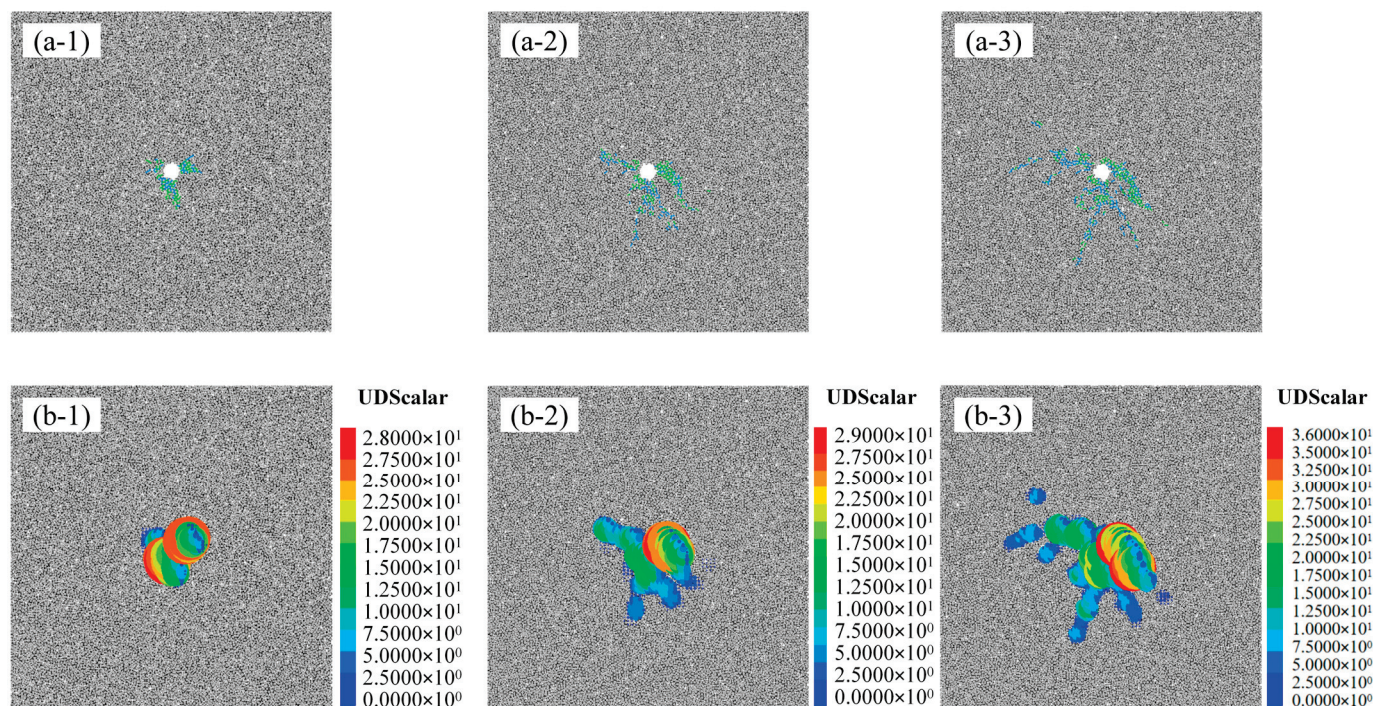


Figure 10. Fracture distribution around a borehole with pre-existing fractures. (a) Fracture distribution and (b) fracture quantity in a coal body with an increasing number of pre-existing fractures. (a-1,b-1), (a-2,b-2), and (a-3,b-3) correspond to the models in Figure 8a, b, and c, respectively.

To explore the evolution of borehole excavation fractures according to the number of pre-existing fractures, statistical analysis of the number and direction of shear and tensile fractures is performed. In Figure 11, the semicircular direction stands for the fracture direction, and the radial direction represents the number of fractures. Overall, the number of fractures in every direction rises, mainly due to the bridge connection between adjacent rocks of pre-existing fractures in the coal seam. Combined with the fracture failure patterns in Figure 10, this result indicates that the distribution of shear fractures produced by drilling in a coal seam with a single pre-existing fracture is predominantly horizontal, and the fracture inclination gradually moves closer to 30° , which is consistent with that of pre-existing fractures.

Next, the two-dimensional fractal dimension change curve is calculated (Figure 12). The numbers of shear and tensile fractures after drilling gradually increase, and pre-existing fractures evolve into secondary fractures. The fractal dimension of coal seams after drilling and excavation increases linearly to 1.294, 1.332, and 1.370, respectively, which increases the complexity of the fracture development path.

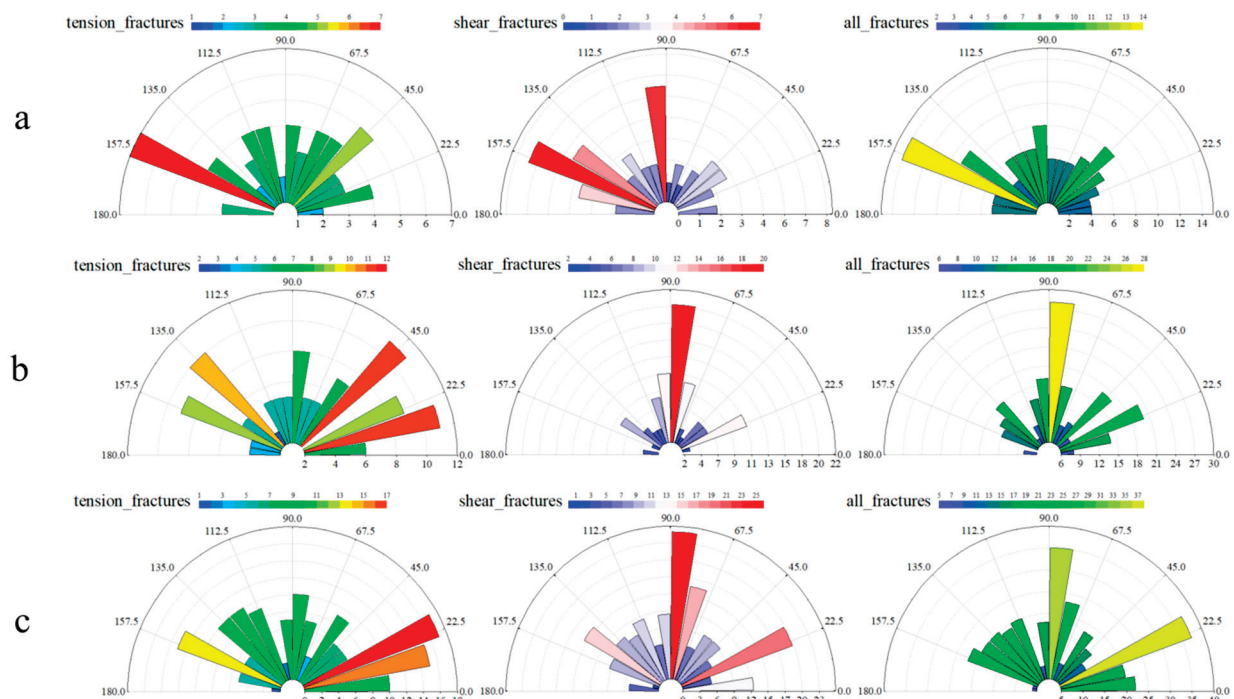


Figure 11. Rose map of tension and shear fracture development around a borehole in a coal body with (a) one, (b) three, and (c) six pre-existing fractures.

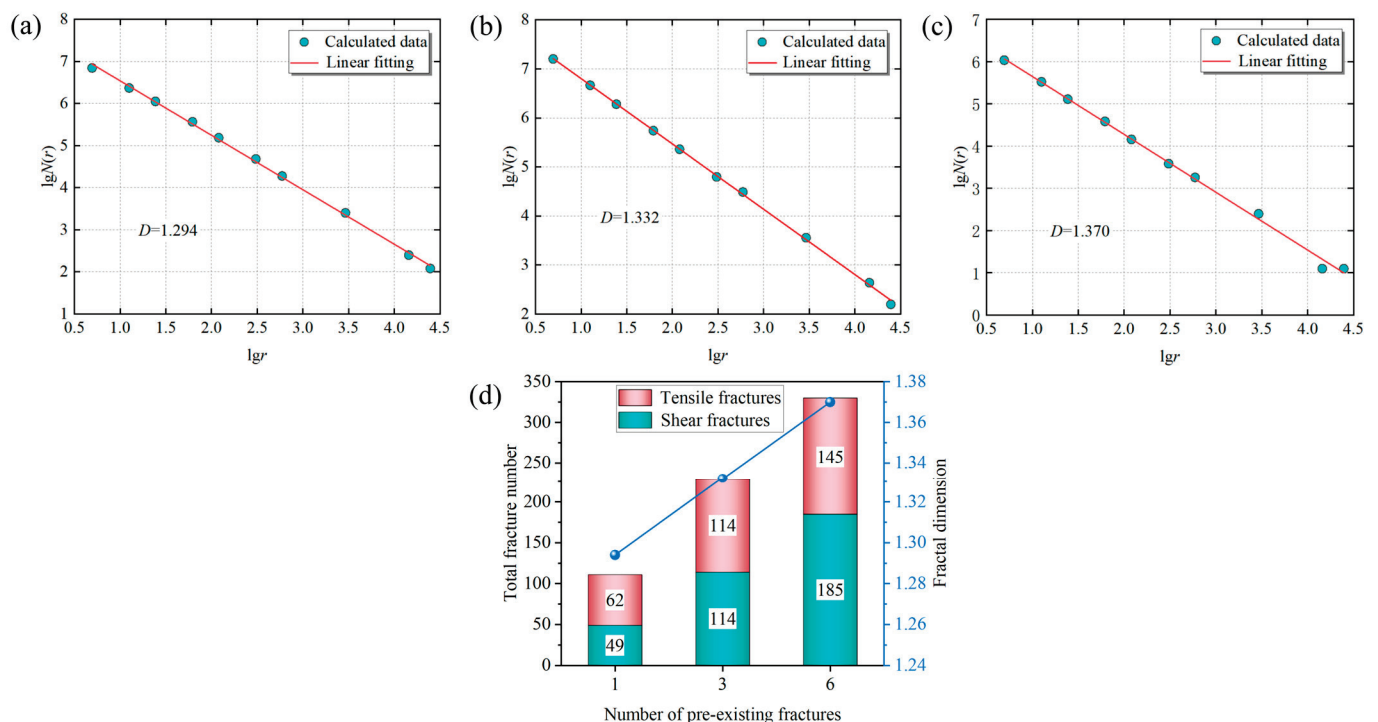


Figure 12. Fractal dimension changes in a coal body with (a) one, (b) three, and (c) six pre-existing fractures. (d) Number of tensile and shear fractures with an increasing number of pre-existing fractures.

4.2. Mechanical Evolution of Boreholes According to the Lateral Pressure Coefficient

Structural stress has a substantial influence on borehole failure and fracture development in soft coal seams. Three lateral pressure boundary conditions were set using PFC2D-5.00.25 software, where the ratio of lateral stress to axial stress was 1, 2, and 4, respectively. As the lateral pressure coefficient increases, the coal drilling force chain increases, reaching 7.7, 34.2, and 51.9 MPa, respectively (Figure 13a); the coal displacement

trend remains approximately the same, although coal displacement around the borehole gradually decreases (Figure 13b), and the stress and strain in the coal seam during drilling and excavation gradually increase, with peak stresses of 7.2, 33.8, and 37.2 MPa and peak strains of 0.0003, 0.0014, and 0.0020, respectively (Figure 13c).

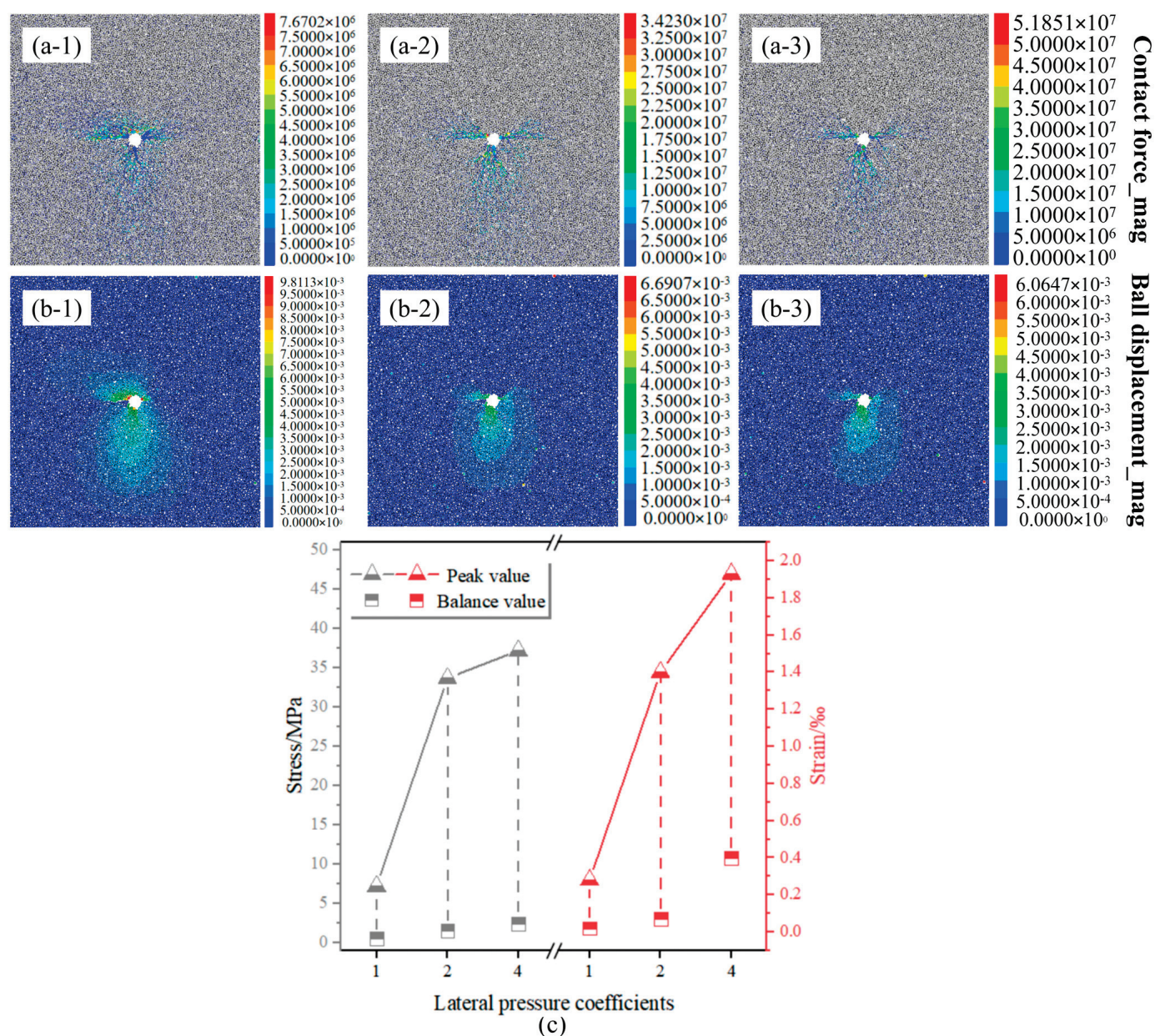


Figure 13. Mechanical state evolution around a borehole with different lateral pressure coefficients. (a) Force chain, (b) coal displacement, and (c) stress and strain evolution in a coal body with different lateral pressure coefficients. (a-1,b-1), (a-2,b-2), and (a-3,b-3) correspond to lateral pressure coefficients of 1, 2, and 4, respectively.

Moreover, the fractures around the borehole gradually expand along the horizontal direction and are distributed linearly (Figure 14a). All fracture-intensive areas are in the borehole zone, and the fracture numbers are 22, 24, and 28 when the lateral pressure coefficients are 1, 2, and 4, respectively (Figure 14b). As the lateral pressure coefficient goes up, the horizontal fracture number rises significantly.

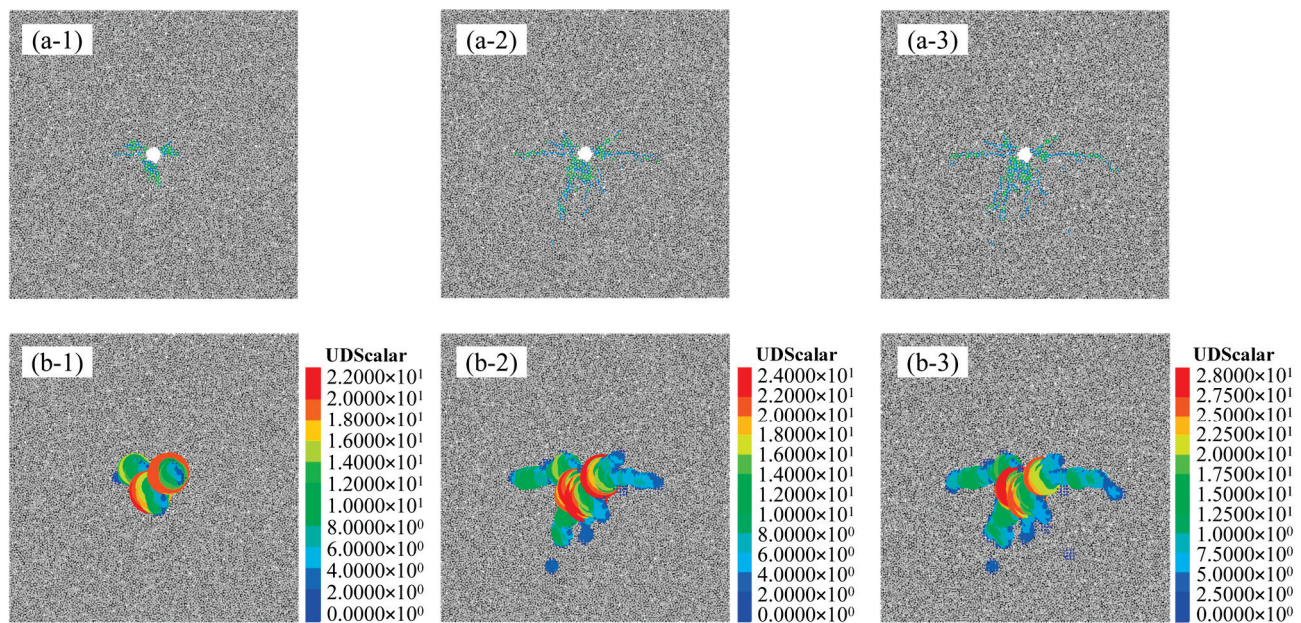


Figure 14. Fracture distribution around a borehole with different lateral pressure coefficients. (a) Fracture distribution and (b) fracture quantity in a coal body, where (a-1,b-1), (a-2,b-2), and (a-3,b-3) correspond to lateral pressure coefficients of 1, 2, and 4, respectively.

Figure 15 shows the quantity and direction of shear and tensile fractures under different lateral pressure coefficients. Usually, the upward fractures on all sides increase gradually with the higher lateral pressure coefficient, mainly because of the growing horizontal stress around the borehole; as the lateral pressure coefficient rises, the shear fracture distribution around the borehole turns mainly vertical.

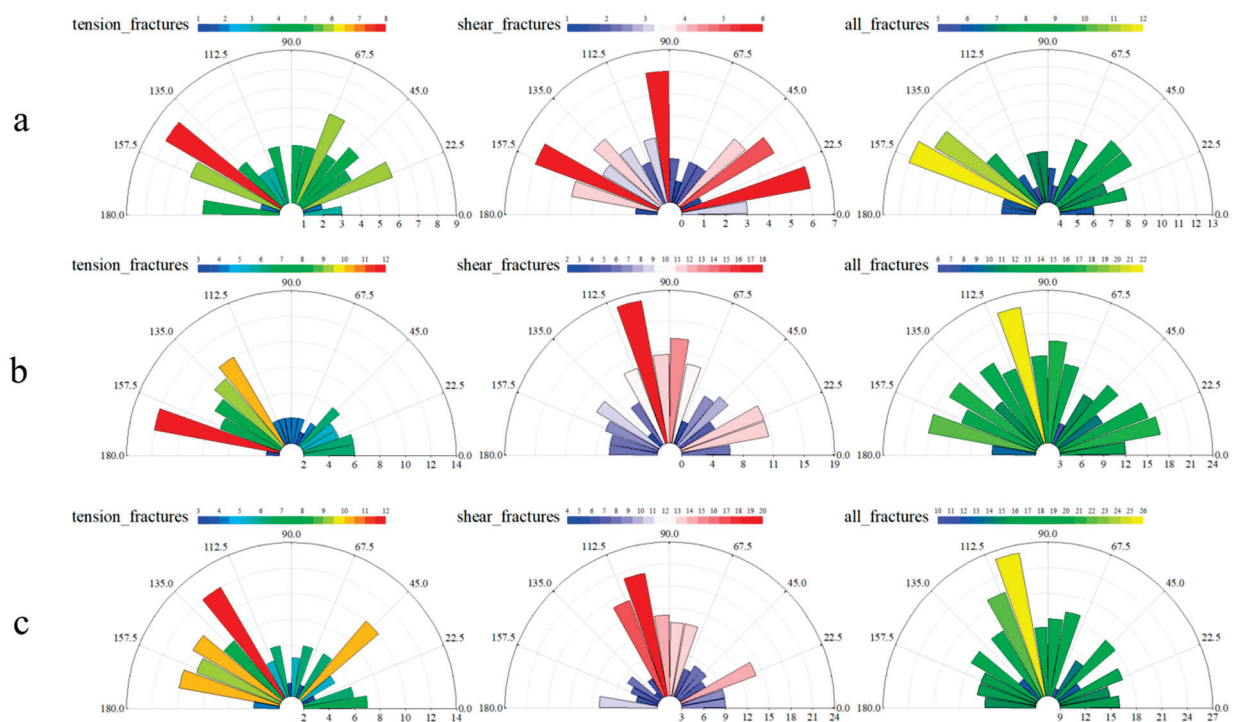


Figure 15. Rose map of tension and shear fracture development around a borehole in a coal body with lateral pressure coefficients of (a) 1, (b) 2, and (c) 4.

According to the 2D fractal dimension change curve in Figure 16, after drilling and excavation, the fracture number increases, as does the number of shear and tensile fractures, and the fractal dimension rises to 1.289, 1.362, and 1.418, respectively.

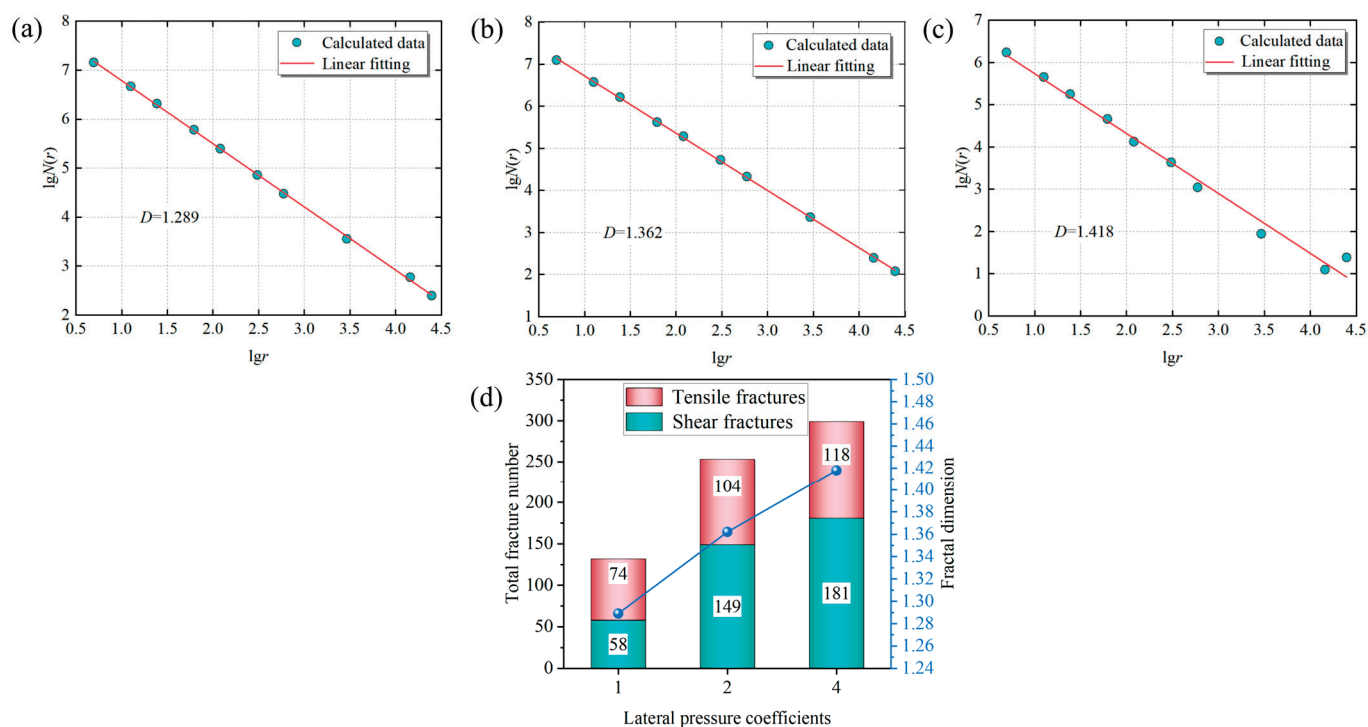


Figure 16. Fractal dimension changes in a coal body with lateral pressure coefficients of (a) 1, (b) 2, and (c) 4. (d) Number of tensile and shear fractures with increasing lateral pressure coefficient.

4.3. Mechanical Evolution of Boreholes According to Gas Pressure

Figure 17 shows the borehole mechanical evolution under three different gas pressures ($p = 0.5, 1.0$, and 1.5 MPa), where the lateral pressure coefficient is set to 4. With increasing gas pressure, the coal drilling force chain increases, reaching a peak of 3.45, 4.86, and 5.13 MPa, respectively (Figure 17a); coal displacement around the borehole gradually increases (Figure 17b), and the stress and strain gradually rise to peak values of 3.1, 3.4, and 3.9 MPa and 0.0004, 0.0006, and 0.0009%, respectively (Figure 17c).

Moreover, the fractures gradually expand horizontally and are distributed linearly (Figure 18a). All fracture-concentrated areas are in the borehole area, and the fracture numbers are 33, 34, and 35, respectively (Figure 18b). The horizontal fracture number also increases notably with the rise in gas pressure.

Based on the calculations of shear and tensile fracture numbers and directions (Figure 19), the upward fracture number on all sides gradually grows as gas pressure increases. At 0.5 MPa gas pressure, the borehole fracture inclination angle is mainly 30° – 50° ; it changes to 140° – 150° and 150° – 160° at 1.0 MPa and 1.5 MPa gas pressures, respectively.

During borehole failure in a gas-bearing coal seam, the difference between fracture number and fractal dimension is small. Both the post-drilling fracture number and fractal dimension increase and reach their peaks at 1.5 MPa gas pressure (Figure 20). The increase in fracture number and complexity in the coal body under higher gas pressure leads to more severe deterioration of coal mechanical properties and a higher risk of borehole failure.

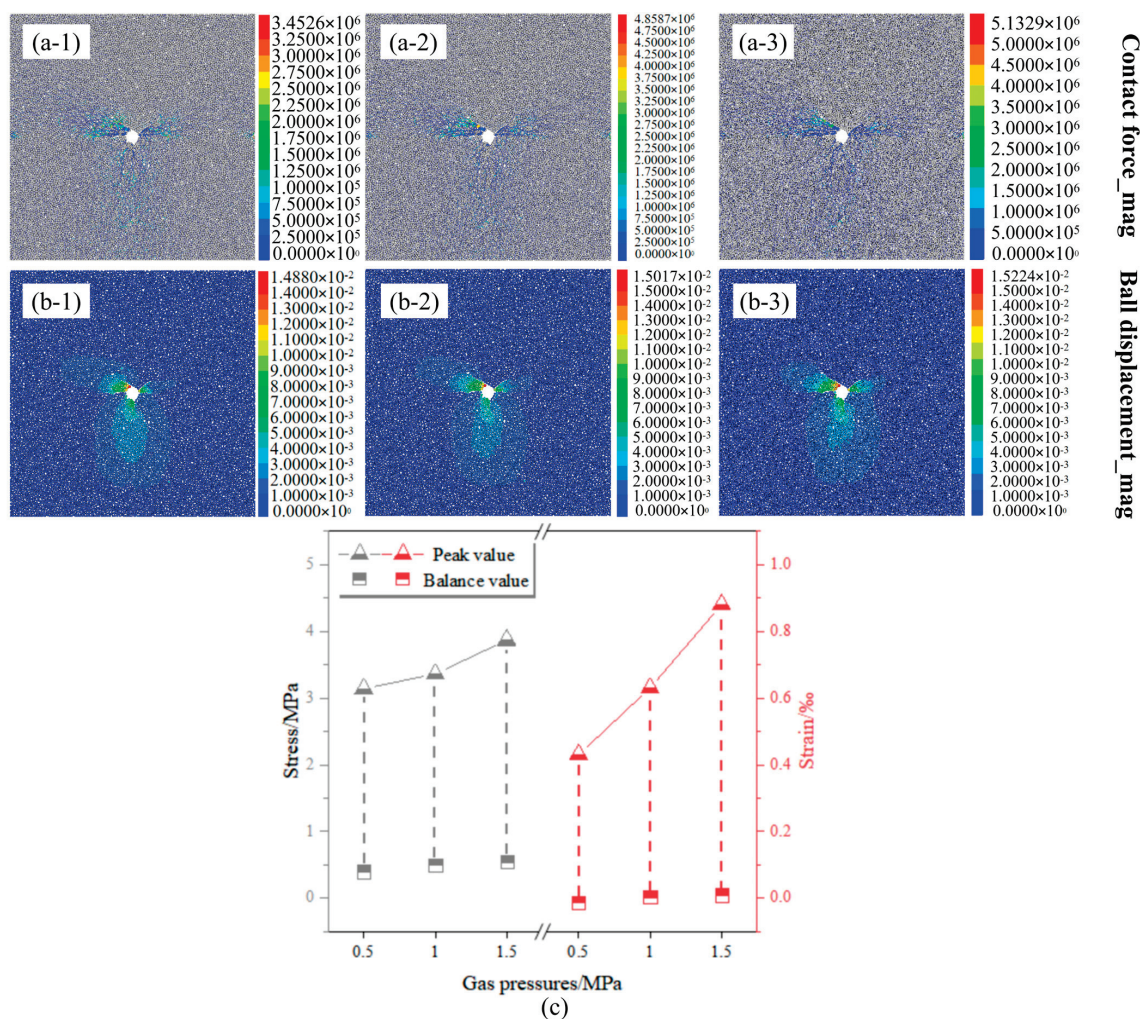


Figure 17. Mechanical state evolution around a borehole with different gas pressures. (a) Force chain, (b) coal displacement, and (c) stress and strain evolution in a coal body, where (a-1,b-1), (a-2,b-2), and (a-3,b-3) correspond to gas pressures of 0.5, 1.0, and 1.5 MPa, respectively.

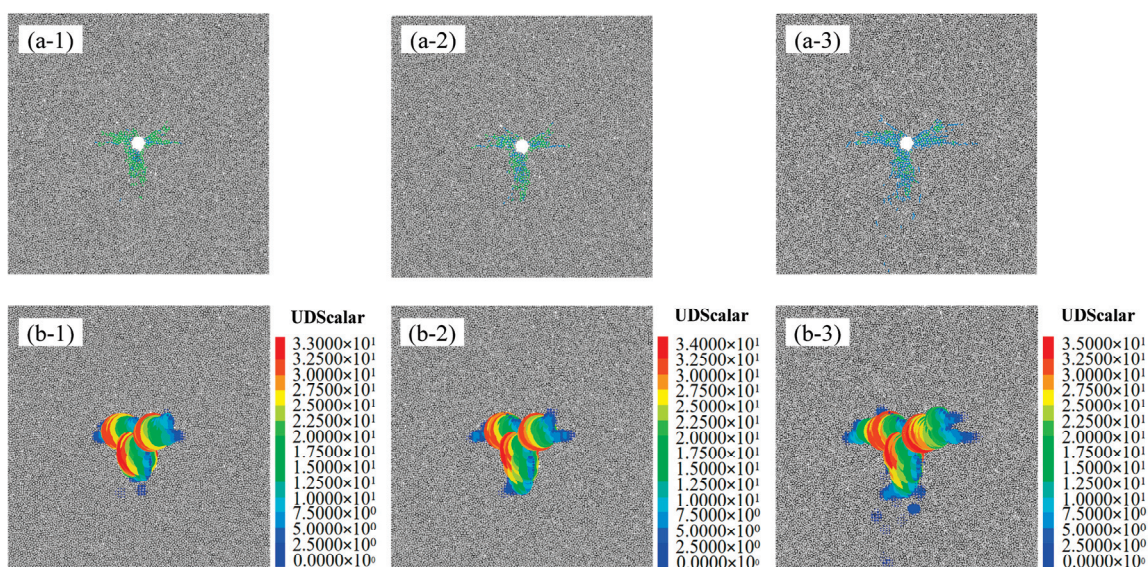


Figure 18. Fracture distribution around a borehole with different gas pressures. (a) Fracture distribution and (b) fracture quantity in a coal body, where (a-1,b-1), (a-2,b-2), and (a-3,b-3) correspond to gas pressures of 0.5, 1.0, and 1.5 MPa, respectively.

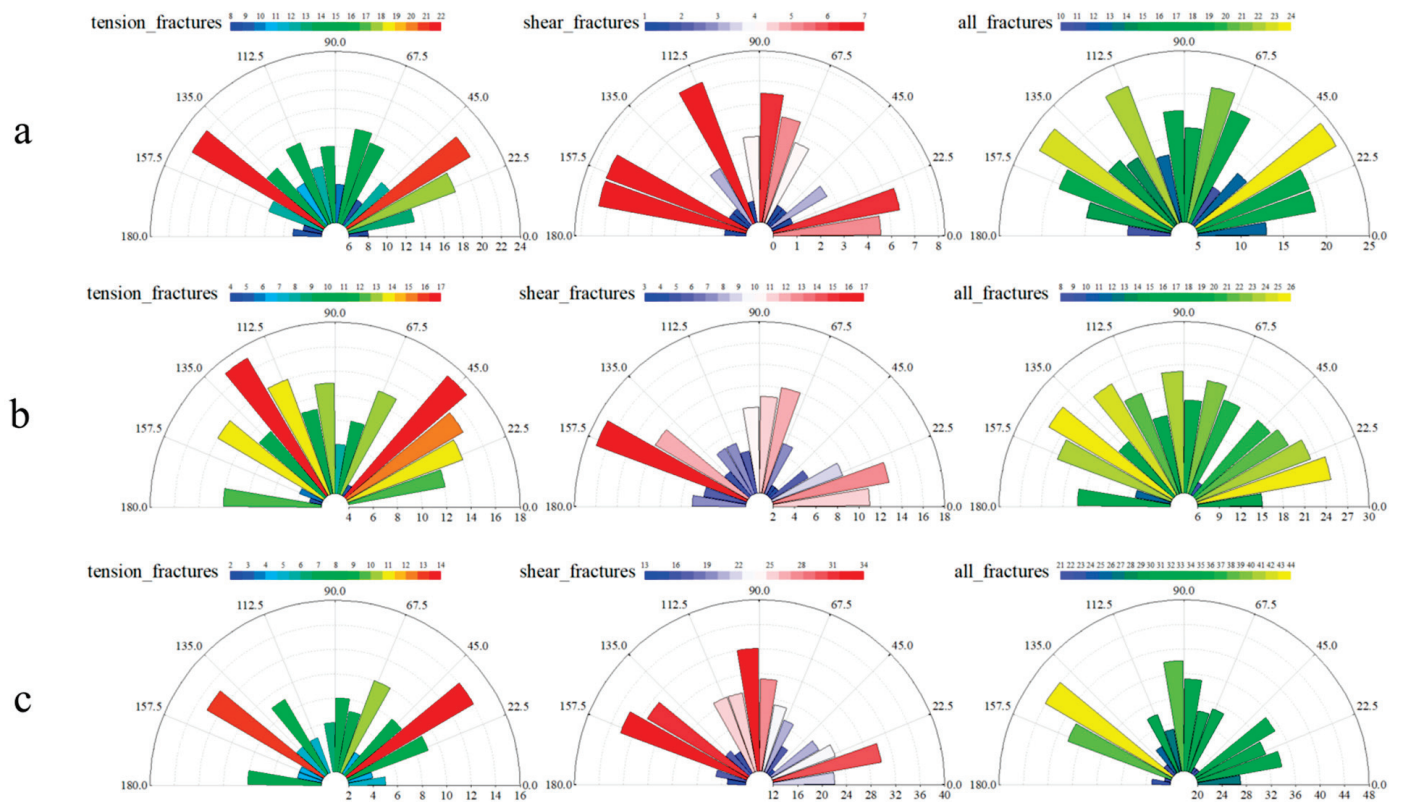


Figure 19. Rose map of tension and shear fracture development around a borehole in a coal body with gas pressures of (a) 0.5, (b) 1.0, and (c) 1.5 MPa.

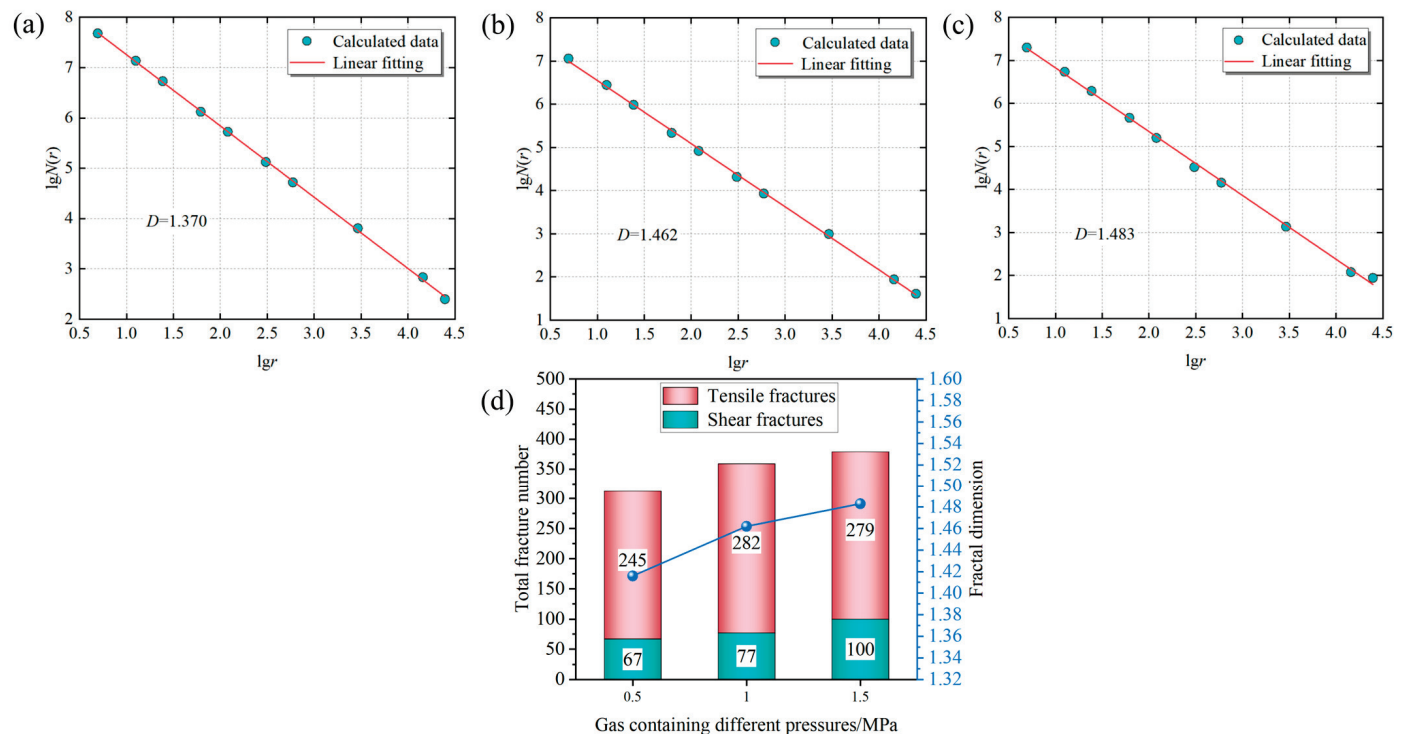


Figure 20. Fractal dimension changes in a coal body with gas pressures of (a) 0.5, (b) 1.0, and (c) 1.5 MPa. (d) Number of tensile and shear fractures with increasing gas pressure.

5. Discussion

5.1. Energy Evolution Analysis of a Gas-Bearing Soft Coal Seam Under Complex Geological Conditions

The drilling and excavation of soft coal seams are often accompanied by energy release; however, severe energy release can impact borehole stability. Energy evolution analysis can reflect changes in drilling strength and the fracture development characteristics in soft coal seams [31,32]. Therefore, the energy evolution characteristics of borehole excavation in a gas-bearing soft coal seam under complex geological conditions are analyzed through PFC2D discrete element numerical simulation. Furthermore, sensitivity analysis of the factors impacting borehole failure in soft coal seams under various complex geological factors is performed according to the number of fractures generated around the borehole, as described in Section 5.2.

Figure 21 shows the energy evolution characteristics during drilling in a soft coal seam under different conditions. The total energy reached 1.3, 3.6, and 5.2 J with one, three, and six pre-existing fractures, respectively. This is because the presence of more pre-existing fractures increases the growth rate of coal fractures around the borehole. The energy accumulated in the early stage is transformed into dissipated energy released, which eventually leads to fracturing of the coal around the borehole [33].

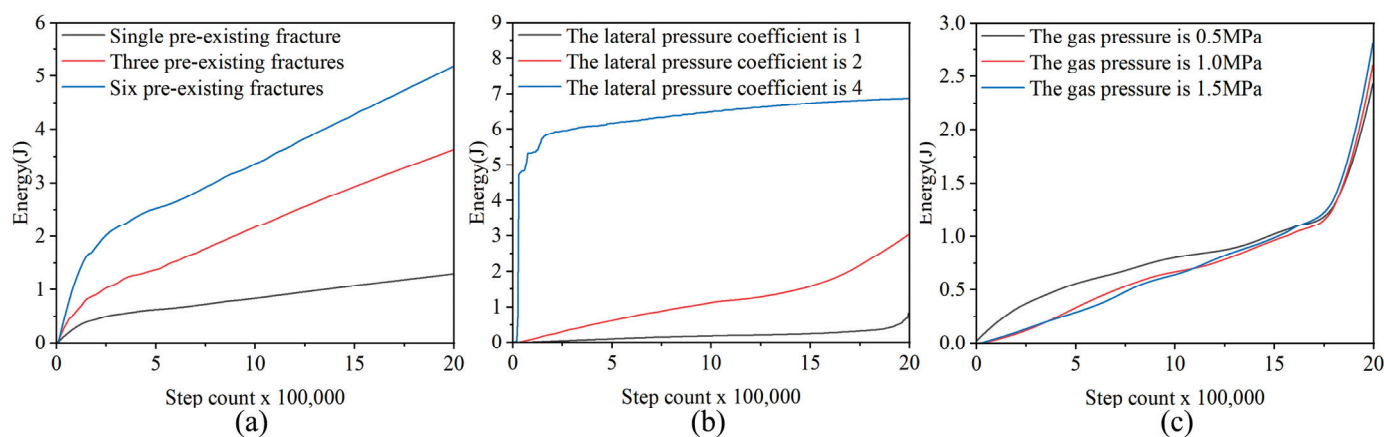


Figure 21. Energy evolution characteristics during drilling in a gas-bearing soft coal seam according to (a) different numbers of pre-existing fractures; (b) different lateral pressure coefficients; and (c) different gas pressures.

Moreover, the total energy around the borehole increases slowly during the drilling process when the lateral pressure coefficients are 1 and 2, reaching peak values of 0.8 and 3.0 J, respectively (Figure 21b). However, with a lateral pressure coefficient of 4, the total energy around the borehole first increases sharply and then flattens rapidly, reaching a peak value of 6.9 J. This is because high lateral pressure coefficients have a significant compaction effect on coal [34]; the greater the lateral pressure coefficient, the greater the increase in coal resistance to deformation damage [31], and the greater the total energy of borehole failure, resulting in stable energy around the borehole [35].

Finally, the energy of the coal body around the borehole exhibits an approximately linear increase with increasing gas pressure (Figure 21c). The initial stress of the coal body under gas pressure and the continuous accumulation of elastic energy lead to instability accompanied by the release of elastic energy during subsequent borehole excavation [31,32]. Under gas pressures of 0.5, 1.0, and 1.5 MPa, the total failure energy around the borehole reaches peak values of 2.4, 2.6, and 2.8 J, respectively.

5.2. Sensitivity Analysis of Factors Influencing Borehole Failure in a Gas-Bearing Soft Coal Seam

Based on the energy evolution features of a soft coal seam in complex geological conditions, the number of pre-existing fractures, lateral pressure coefficient, and gas pressure have different impacts on borehole stability. So, a sensitivity analysis of these three factors is performed using the orthogonal experimental design for multi-factor analysis. Then, the orthogonal test results are analyzed with the range analysis method. The range analysis method first calculates the range value of each factor's calculation results, then compares the range values to judge the influence of each factor on the test index. The larger the range value, the more sensitive the factor [36]:

$$R_j = \max\{K_{avg}\} - \min\{K_{avg}\} = \max\left\{\frac{\sum_{k=1}^n Y_{ijk}}{n}\right\} - \min\left\{\frac{\sum_{k=1}^n Y_{ijk}}{n}\right\} \quad (7)$$

Here, R_j represents the range of factor j ; n represents the total number of tests; Y_{ijk} represents the result index obtained under the first test under the value condition of the i level in factor j ; and K_{avg} is the average value of the result index obtained from n trials.

When adopting the orthogonal experimental design, the influencing factors must be clarified and the values of the influencing factors must be set at all levels. Here, the number of pre-existing fractures, the lateral pressure coefficient, and gas pressure were denoted as A, B, and C, respectively, and each was assigned three levels (Table 2). Orthogonal tables can determine the importance and interaction of each influencing factor through a small number of experiments. Each factor and level must be independent of each other and each level of each factor must occur equally frequently in the test scheme. According to the values of each influencing factor in Table 2 and the design rules of the orthogonal table, the orthogonal experimental design table shown in Table 3 is obtained. The scheme of the orthogonal design table was calculated using PFC2D software, and the statistical results are presented in Table 4.

Table 2. Set values of factors influencing borehole failure in a gas-bearing soft coal seam under complex geological conditions.

Level	A	B	C
1	1	1	0.5
2	3	2	1.0
3	6	4	1.5

Table 3. Design of the orthogonal experiment table.

Option	A	B	C
1	1	1	0.5
2	1	2	1.5
3	1	4	1.0
4	3	2	1.0
5	3	1	1.5
6	3	4	0.5
7	6	4	1.5
8	6	1	1.0
9	6	2	0.5

According to Table 4, the range value (R_j) showing the influence degree of each factor on borehole failure in a gas-bearing coal seam is the largest for the number of pre-existing fractures, followed by the lateral pressure coefficient and, finally, the gas pressure. The borehole damage is the most severe at level three (optimal level) for all factors, that is, when

there are six pre-existing fractures, a lateral pressure coefficient of 4, and a gas pressure of 1.5 MPa (Figure 22).

Table 4. Fracture number and range value for each influencing factor.

Option	Level	A	B	C
K	1	839	879	890
	2	941	946	933
	3	1010	965	967
K_{avg}	1	280	293	297
	2	314	316	311
	3	337	322	323
Optimum level		3	3	3
R_j		57	29	26
Sensitivity ranking			A > B > C	

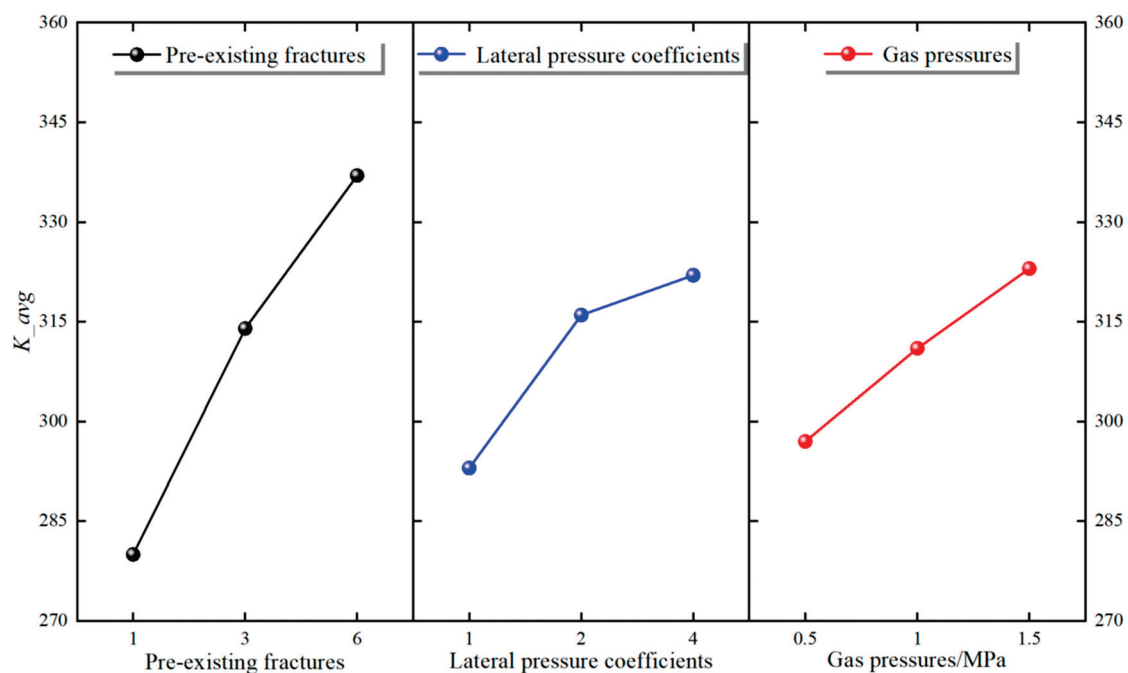


Figure 22. Plot of the average fracture number for a borehole in a gas-bearing soft coal seam under different geological conditions.

5.3. Types of Borehole Failure in a Gas-Bearing Soft Coal Seam

According to the numerical simulation results, the energy evolution under different geological conditions, and the sensitivity analysis of factors influencing borehole failure, two types of borehole failure in soft coal seams are identified: fracture-dominated failure and stress-dominated failure.

5.3.1. Fracture-Dominated Failure

Fracture-dominated failure mainly occurs when the coal seam has a complex fracture network. Here, drilling and excavation lead to many secondary fractures; the fracture density rises, the coal body strength drops, and the borehole is easily damaged. At a coal seam gas pressure of 1.5 MPa, the borehole failure with pre-existing fractures is mostly elliptical (Figure 23), and the failure zone expansion direction gradually matches the pre-existing fracture inclination angle. The horizontal deflection angles are 0° , 12° , and 30° for one, three, and six pre-existing fractures, respectively. Also, the geometric distribution gap of pre-existing fractures is the key factor affecting the fracture area in gas-bearing soft coal seams.

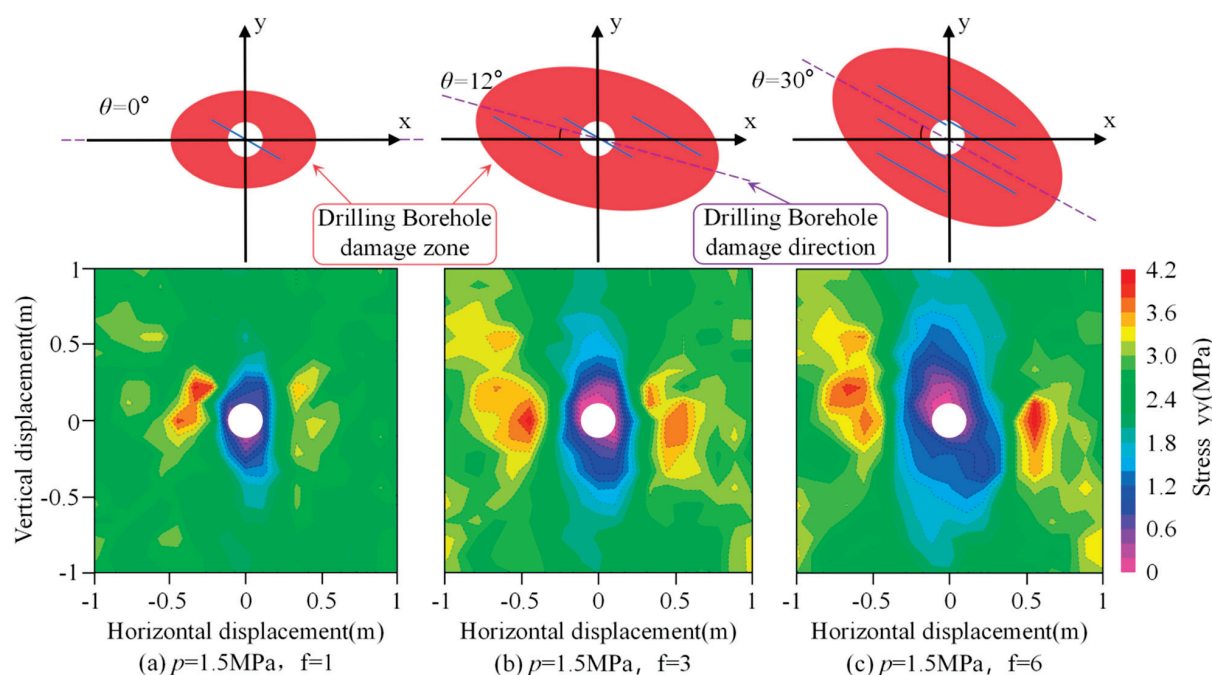


Figure 23. Fracture-dominated failure in a gas-bearing soft coal seam.

5.3.2. Stress-Dominated Failure

Under high stress, the coal body around the borehole is subjected to tangential compressive stress concentration, leading to tensile failure, as shown in Figure 24. Borehole failure is prone to occur along the direction of maximum principal normal stress. The failure area keeps expanding, and the damage is not very uniform, mainly happening above and below the borehole when the lateral pressure coefficient is 4. This is because a pressure arch structure is formed the borehole, which gradually disappears with increasing lateral pressure with damage of the arch seat, resulting in gradual vertical extension of the damage area in both directions [37].

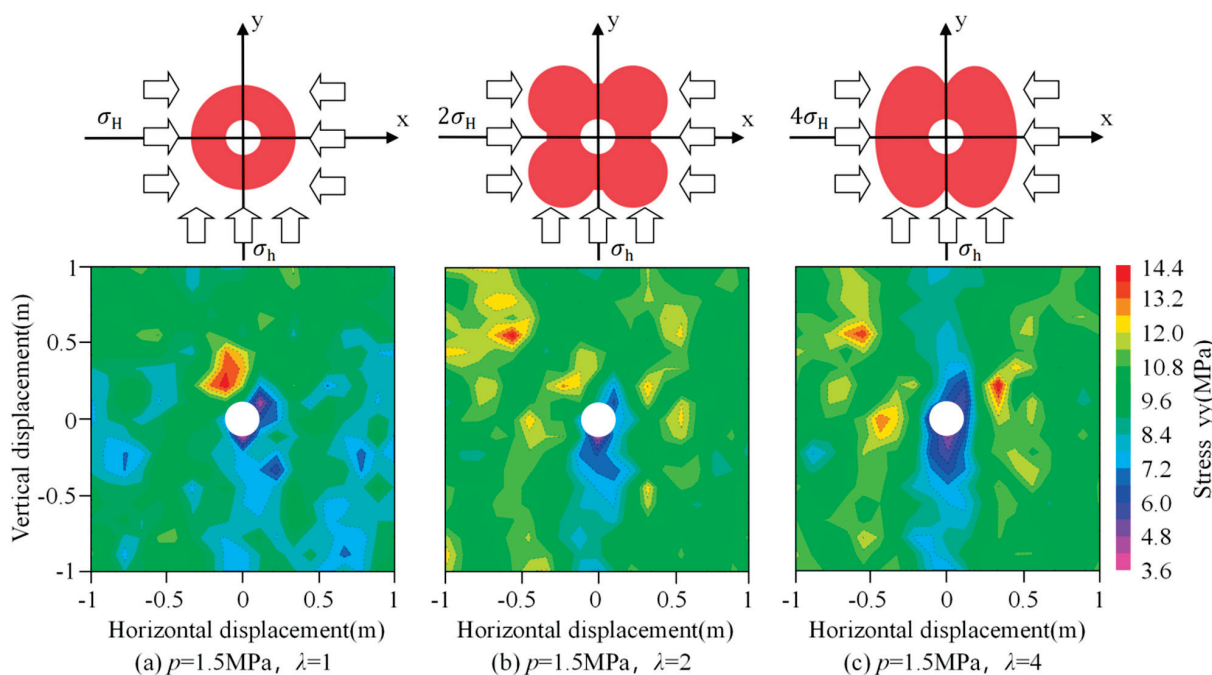


Figure 24. Stress-dominated failure in a gas-bearing soft coal seam.

6. Conclusions

In this study, the mechanical properties and energy evolution features of the coal body under various geological conditions are analyzed. The key factors affecting borehole failure are identified, and the two main kinds of borehole failure in a gas-bearing soft coal seam are classified. The following conclusions are obtained:

(1) The stress and force chain concentrate at a 0.1 m horizontal distance from the borehole, with large coal body displacement there. Fractures mainly lie on the borehole sides and bottom, influencing coal body porosity. Coal body porosity first rises and then uniformly drops within a 0.1–0.2 m horizontal distance from the borehole.

(2) As the number of pre-existing fractures, the lateral pressure coefficient, and the gas pressure increase, the number of fractures produced by drilling gradually increases, as does the fractal dimension, indicating increased fracture network complexity. Shear fractures produced in a coal body with different numbers of pre-existing fractures form mainly in the horizontal direction, whereas those produced under different lateral pressure coefficients form mainly in the vertical direction. The inclination angle of fractures increases gradually with increasing gas pressure.

(3) The total energy around the borehole increases sharply, although the three factors show varying degrees of influence on borehole stability. Sensitivity analysis of the three factors showed that the number of pre-existing fractures has the greatest impact on borehole stability, followed by the lateral pressure coefficient and then gas pressure. The degree of borehole failure is greatest when the values of these three factors are 6, 4, and 1.5 MPa, respectively.

(4) Borehole failure in a gas-bearing soft coal seam can be divided into fracture-dominated and stress-dominated failure. The former predominantly occurs in soft coal seams with a complex fracture network under high gas pressure. Stress-dominated failure mainly occurs in soft coal seams under high stress in tectonic belts. The increase in the lateral pressure coefficient makes the borehole failure zone expand vertically in two directions.

Author Contributions: X.C. (Xuexi Chen): conceptualization, funding acquisition, investigation, and methodology. Z.Y.: methodology, software, and writing—original draft. J.H.: methodology, software, supervision, and writing—review and editing. T.Y.: investigation, methodology, and supervision. J.S.: conceptualization, methodology, and project administration. Y.T.: methodology and resources. X.C. (Xingyu Chen): methodology and resources. All authors have read and agreed to the published version of the manuscript.

Funding: This research was funded by the National Natural Science Foundation of China (grant number 52174181), the S&T Program of Hebei (grant number 22375401D), the Natural Science Foundation for Youths of Hebei Province (grant number E2024508014), and the Fundamental Research Funds for the Central Universities (grant number 3142024011).

Data Availability Statement: Data will be made available on request.

Conflicts of Interest: We declare that we have no financial or personal relationships with other people or organizations that can inappropriately influence our work; there is no professional or other personal interest of any nature or kind in any product, service, and/or company that could be construed as influencing the position presented in, or the review of, the manuscript entitled “Borehole failure mechanics and influencing factors in a gas-bearing soft coal seam under complex geological conditions”.

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Article

Non-Pillar Coal Mining by Driving Roadway During Mining Period in High-Gas Top-Coal-Caving Working Face

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Abstract: To solve the problem of the inability to achieve Y-shaped ventilation in the boundary coal mining of high-gas mines and the problem of gas accumulation in the upper corner of a fully mechanized mining face, non-pillar coal mining technology is proposed by a driving roadway during the mining period. A high-gas working face requires Y-shaped ventilation to achieve upper corner gas control, but Y-shaped ventilation conditions are not available at the boundary coal body. In order to handle this challenge, studies have suggested non-pillar coal mining technology, which involves excavating roadways while mining in order to achieve non-pillar coal extraction and use recoverable wide coal pillars. During the simultaneous excavation of a working face and roadway, studies analyzed the distribution characteristics of the complicated stress environment. Following an evaluation of the impact of coal pillar width on the quality of an excavation roadway, this study's development is in terms of an effective technique for retaining coal pillars as established. During the mining period of a working face, in the goaf of the working face, the research analyzed the distribution properties of the gas flow field, and findings from the study indicate that the width of the recovered coal pillar influences the distribution of gas. Finally, the width of the coal pillar was comprehensively determined, forming non-pillar coal mining technology by a driving roadway during the mining period. The on-site practice has shown that using a wide coal pillar with a width of 70 m to protect the roadway significantly reduces the deformation of the surrounding rock in the mining roadway, the gas concentration at the return airway is lower than the safety production standard, and by decreasing the mining succession time by 15 months, studies achieved improving the working face's coal extraction rate by 12.6%.

Keywords: non-pillar coal mining; surrounding rock control; gas flow field

1. Introduction

Non-pillar coal mining technology is frequently used in underground operations by academicians and industry professionals, which can reduce the excavation workload of tunnels, improve the coal extraction rate, improve the stress environment and ventilation lines of tunnels, etc. [1,2]. Researchers and the mining industry started decreasing shallow coal deposits at the beginning of this century; the depth of coal mining and costs have been increasing, producing many safety problems such as coal pillar bursts, rock bursts, and significant rock deformation failures due to the considerable ground stress [3,4]. The studies proposed the 'cutting cantilever beam' theory [5,6] and the non-pillar longwall mining by roof cutting (also the 'N00 mining method') to address these problems. The new mining method eliminates the requirement to reserve an entry in advance, resulting in no coal pillars remaining after mining [7–9].

Scholars have also conducted multidimensional research on applying non-pillar coal mining technology and achieved preliminary results [10–12]. Zhang et al. analyzed the three critical technologies of constant-resistance extensive deformation support, directional pre-splitting blasting, and blocking-gangue support systems in non-pillar coal mining [13].

Zhou et al. studied air leakage characteristics concerning a “Y”-type ventilation in gob-side entry retaining with roof cutting, pressure relief, and the law of a resulting gas accumulation [14]. Feng et al. studied the law of airflow movement in a working face and gob under “roof-cutting and pressure-releasing” mining along gob-side entry retaining [15]. Sun et al. explored the distribution law of a stress field under the mining mode of gob-side entry retaining by roof cutting without a pillar (GERRCP) under a goaf [16]. Zhu et al. researched mining-induced ground response development of a non-pillar mining panel with gob-side entry by roof cutting [17]. Ma et al. studied the gob-side entry retaining mechanism by the roof-cutting approach and its three key technologies through a theoretical analysis, numerical simulation, and laboratory and field experiments [18]. Chen et al. established an evaluation index system for roof stability in the roof-cutting non-pillar mining method, including the roof rock integrity and the roof-surrounding rock displacement [19]. Cao et al. determined the support parameters of a roadway and flexible-formwork pre-cast partition wall through a theoretical analysis and numerical simulation [20]. Hao et al. studied pillarless mining by gob-side caving under soft rock roof conditions [21]. By using artificial supports for supporting pillars, certain technologies enable an excavation to be preserved for subsequent applications [22].

Non-pillar coal mining technology involves production processes such as roadway excavation, coal seam mining, and ventilation in mines [23]. However, current research on the application of non-pillar mining technology mainly focuses on retaining tunnels along the goaf without involving pillar-free mining related to the recovery of wide coal pillars. When faced with mining boundary coal bodies, it is impossible to achieve Y-shaped ventilation when the boundary roadway cannot be arranged. Under these conditions, in order to attain pillarless mining, studies proposed substituting Y-shaped ventilation with a W-type method. Based on the actual conditions of the mine, this article proposes a non-pillar coal mining technology that digs roadways during the mining period. By analyzing the spatiotemporal effects of the coupled stress of coal mining and roadway excavation, as well as the distribution characteristics of the gas flow field in a W-shaped fully mechanized caving face, a reasonable width of the protective coal pillar for the roadway is ultimately determined based on the stability of the roadway and gas extraction. The research results will further promote the development of non-pillar mining technology, which is significant for achieving safe, green, and efficient mining in high-gas mines.

2. Process Flow of Non-Pillar Coal Mining Technology with Width Pillar

The non-pillar coal mining proposed in this article involves setting up a reasonably wide protective coal pillar to arrange the return airway of the next working face in advance, achieving a “W”-shaped ventilation to replace the “Y + L”-shaped ventilation, and recovering the wide coal pillar. Figure 1 depicts the fundamental procedure.

- (1) 01 working face mining stage: Firstly, studies structured the highways on both sides of the 01 working face and the transportation roadway on the 02 working face. Experts have installed two ventilation systems—one for air intake and one for air return—on the 01 working face. The 01 transportation roadway–01 working face–01 air-intake roadway belongs to the “Y” ventilation system, while the 02 working face–02 transportation roadway constitutes the “L” ventilation system. Therefore, the working face is a “Y + L” ventilation system, as shown in Figure 1a.
- (2) Excavating roadway during mining: Researchers have established the ideal coal pillar size by studying the spatiotemporal effects of surrounding rock stress in tunnels and the distribution characteristics of W-shaped gas flow fields. According to studies, research constructed under discussion the return airway on the 02 working face at the same time as mining on the 01 working face, as seen in Figure 1b.
- (3) 02 Working face mining stage

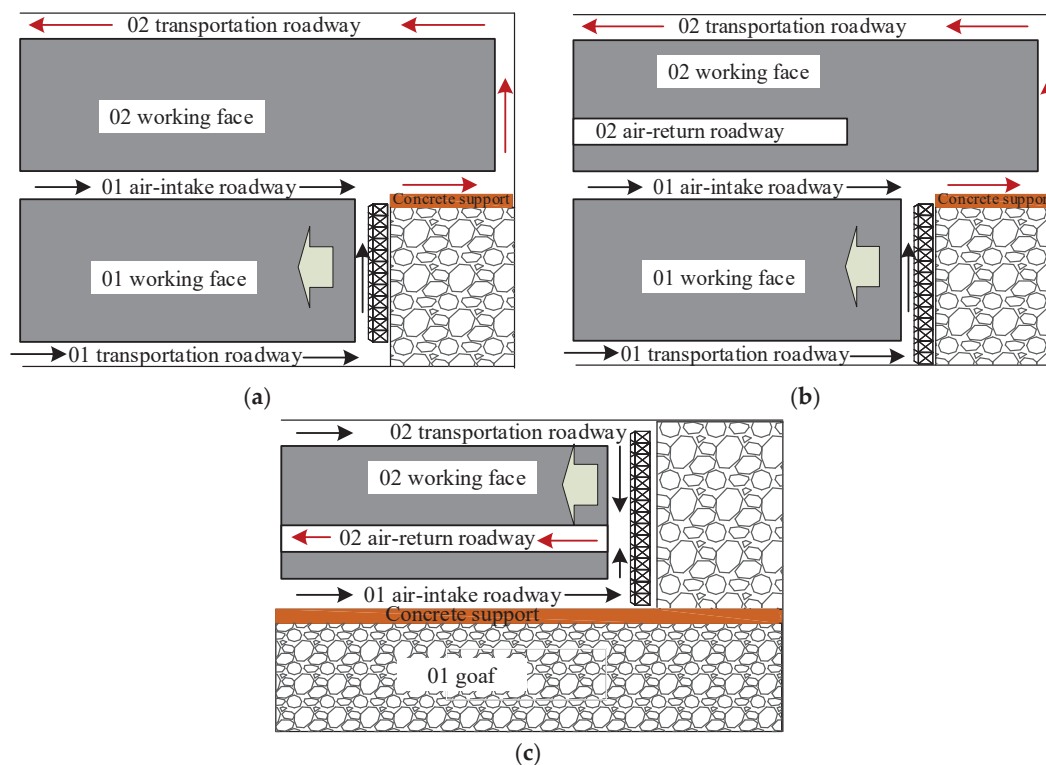


Figure 1. Process flow of non-pillar coal mining technology with width pillar. (a) 01 working face mining stage; (b) excavating roadway during mining; (c) 02 working face mining stage. Arrows indicate the direction of airflow.

The 01 air-intake roadway is retained by leaving a roadway along the goaf and used as an air-intake roadway during the mining period of the 02 working face. In order to support the transportation route for the 02 working face, experts intend to turn the 01 air-return highway into an air-intake roadway. The 02 working face also has two inlets and one return ventilation system, 01 air-intake roadway–02 working face–02 transportation roadway–02 air-return roadway, forming a “W”-shaped ventilation system, as shown in Figure 1c.

In summary, the above process is a non-pillar mining method that retains wide coal pillars and recovers them later. The stability of the road during simultaneous excavation and the gas difficulties that develop during working face mining are two critical concerns that researchers need to address concerning the coal pillar width.

3. Stability of Roadway Under Complex Mining Stress Environment

3.1. Temporal and Spatial Distribution Characteristics of Complex Stress Environment

Using FLAC3D 6.0 software, researchers designed a three-dimensional numerical model that is 500 m × 300 m × 50 m in size, as illustrated in Figure 2. The X direction of the model is the layout direction of the W4301 working face, and the Y direction is the direction of the working face advancement. Table 1 shows the mechanical and physical parameters of coal rock. For calculations, the research utilizes the Mohr–Coulomb failure criterion and turns on the significant deformation mode. The boundary conditions are that the bottom boundary restricts vertical displacement, and the peripheral boundary restricts horizontal movement. From the top of the model, the research applies a load that is equally distributed and equal to the weight of the rock layer above it. In order to mitigate the impact of boundary effects, researchers maintain a 50 m wide boundary coal pillar on one side of the 01 working face’s transportation lane. After the initial balance calculation of the model, the 01 working face is excavated from the Y = 50 m position using a staged excavation and cyclic calculation method, with each stage excavating 10 m and corresponding to a numerical calculation of 1000 time steps.

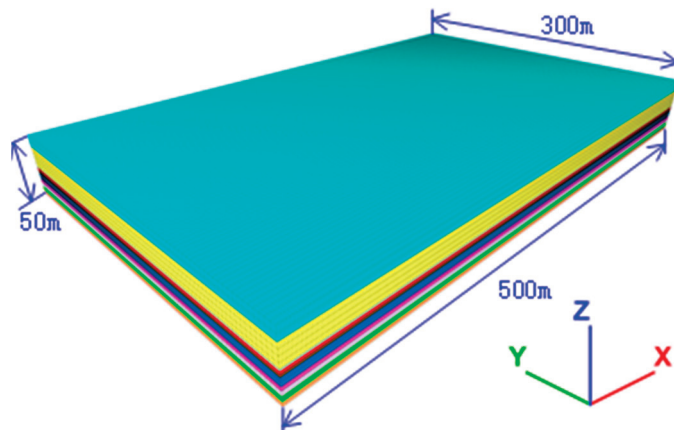


Figure 2. FLAC numerical simulation model.

Table 1. Physical and Mechanical Parameters of Coal Rock.

Rock	Thickness/m	Bulk Modulus/GPa	Shear Modulus/GPa	Tensile Strength/MPa	Internal Friction Angle/°	Adhesive Force /MPa	Density /Kg·m ⁻³
Overburden	10	5.84	3.75	3.43	36.6	4.62	2500
Middle sandstone	16.5	4.35	2.31	2.73	35.8	4.21	2450
Siltstone	1.0	6.08	3.97	4.27	37.2	4.82	2600
Sandy mudstone	1.72	5.33	3.25	3.74	36.0	4.75	2550
Mudstone	2.2	3.91	1.74	1.50	31.6	2.74	2150
3 # coal	6.08	2.32	0.9	1.20	20.0	2.02	1400
Mudstone	2.46	3.91	1.74	1.50	31.6	2.74	2150
Fine sandstone	3.52	6.08	3.97	4.27	37.2	4.82	2600
Sandy mudstone	4.0	4.24	2.14	2.32	33.2	3.26	2300
Mudstone	2.5	3.91	1.74	1.50	31.6	2.74	2100
Fine sandstone	3.5	5.33	3.25	3.74	36.0	4.75	2550
Undercover rock layer	10	5.84	3.75	3.43	36.6	4.62	2500

The complex stress environment in which roadway excavation occurs includes temporal and spatial effects [24,25]. A time effect refers to one or several stages that will be experienced during the process of roadway excavation, including the solid coal excavation stage (the stage of stress influence on the surrounding rock of the roadway), the advanced mining influence stage, the stage of roadway excavation through the working face, and the stage of mining influence on this working face, along the time gradient. A spatial effect refers to the specific range of each stage in space corresponding to the temporal effect. Figure 3 shows the schematic diagram of the complex stress environment experienced by roadway excavation.

The study reveals significant differences in the stress environment of the surrounding rock while attempting to place the 02 air-return roadway in various locations. As the width of the coal pillar increases, the roadway will no longer be affected by the stress of the surrounding rock of the 01 intake airway. The range of the solid coal roadway excavation gradually increases, and the range of the advanced mining influence stage gradually decreases. Roadway 4 has almost no advanced mining influence stage, directly transitioning from the solid coal roadway excavation stage to the working face excavation stage. The complexity of the surrounding rock's stress conditions and the difficulty in maintaining the stability of the surrounding rock increase with the breadth of the protecting coal pillar. From Figure 3, as a result, studies can determine the distribution of supporting pressure in the rock surrounding the 01 working face; the range of the advanced mining influence zone is 50 m in front of the working face, and the range of the lateral supporting pressure influence zone is within 40 m inside the coal wall.

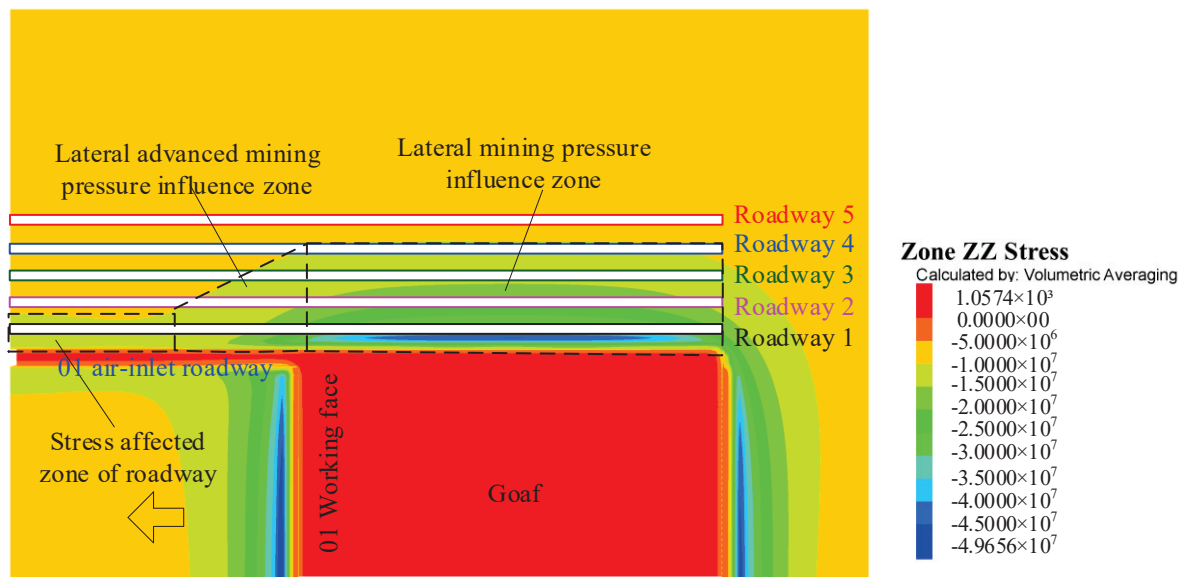


Figure 3. The complex stress environment experienced by roadway excavation during the mining period of the working face.

To further quantify the specific range of the stress-affected area by mining the stress-affected area and positioning measurement lines on one side of the solid coal, researchers can determine accuracy. Based on the distribution of bearing pressure obtained from the survey line, using a stress measurement that is 5% more than the initial rock stress, experts can identify the boundary between the advanced mining influence zone and the excavation stress influence zone, which can determine the actual range of the excavation stress influence zone and the advanced mining influence zone at different positions. Figure 4 below depicts the curve of the influence range of bearing pressure on the working face and its roadway.

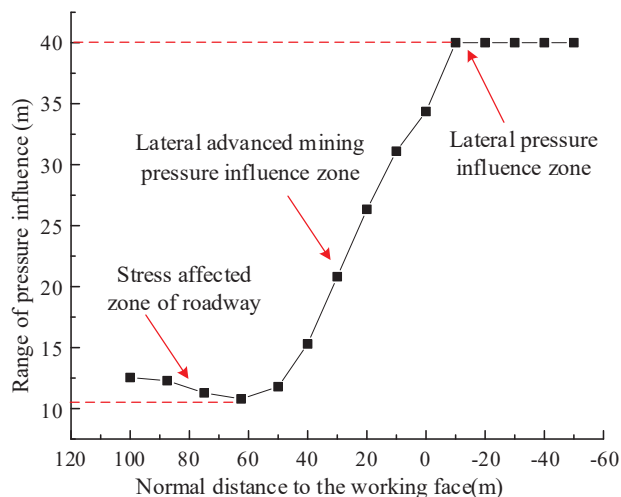


Figure 4. Scope of influence of mining pressure.

As shown in Figure 4, the range of the stress influence zone of the roadway and the lateral pressure influence zone are constant. The range of the stress influence zone of the 01 air-intake roadway is between 10.8 and 12.5 m, and the fluctuation is mainly due to the boundary effect of the model and the weak mining influence of the 01 working face. After averaging treatment, the range of the excavation stress influence zone is 11.7 m. The maximum influence range of lateral support pressure is 40 m inside the coal wall, and its range continuously changes with the expected distance to the W4301 working face,

roughly showing a linear relationship. Finally, at an average distance of 0 from the working face, it is connected to the boundary of the lateral support pressure influence zone. The segmented equations for the range of support pressure influence zones in various regions of the 01 working face can be obtained by researchers through the fitting, specifically

$$y = \begin{cases} 11.74, & (x > 50) \\ 35.12 - 0.47x & (0 \leq x < 50) \\ 40, & (x \leq 0) \end{cases} \quad (1)$$

where x is the average distance between the 02 air-return roadway and the 01 working face, m; y is the distance from the range of support pressure influence to the 01 air-inlet roadway, m.

According to Equation (1), when excavating the 02 air-return roadway in complicated stress situations, experts can determine the spatiotemporal impacts of stress distribution and deformation failure in the surrounding rock.

- (1) If the width of the coal pillar is 10 m, when $x > 50$ m, the 02 air-return roadway will be affected by the stress of the 01 air-inlet roadway. When $x = 50$ m, the roadway will enter the stage of advanced mining influence. When $x < 0$ m, it will enter the lateral support pressure influence stage.
- (2) If the width of the coal pillar is 25 m, according to Equation (1), when $x > 21.5$ m, the 02 air-return roadway is in the stage of solid coal excavation. When $0 < x < 21.5$ m, the roadway is in the stage of advanced mining influence.
- (3) If the width of the coal pillar is 50 m, there is no intersection point throughout the segmented equation, indicating that the excavation process of the 02 air-return roadway is not affected by the stress of the 01 working face and surrounding rock of the roadway; that is, the roadway excavation process is always in the stage of solid coal excavation.

3.2. Reasonable Size of Coal Pillars Based on Rock Stability

Currently, there are two main types of roadway protection methods for mining, roadway excavation along the goaf and retaining coal pillars for roadway protection, as shown in Figure 5. For traditional roadway excavation along the next goaf, according to studies, it is recommended to start excavation only after the overlying rock strata on the adjacent working face have stabilized [26]. Given that the researchers appropriately determine the narrow coal pillar's breadth, the lateral fundamental roof fracture structure protects the roadway, and in order to attain adequate support, researchers can use appropriate anchor rod support technology and place the roadway in the stress reduction zone [27]. However, when facing the mining face, the roadway undergoes a complete dynamic pressure caused by the adjacent mining process, including lateral fundamental top fracture, rotation, contact with gangue, and re-stabilization. This process will inevitably significantly impact the stability of the surrounding rock along the goaf roadway, causing significant deformation and complex maintenance. Therefore, wide coal pillars can effectively solve this problem. In order to preserve the tunnel smoothly, experts use flexible formwork support at the edge of the roadway. A support structure is formed by arranging flexible formwork bags around the roadway and pumping concrete into them.

As shown in Figure 6, when the width of the coal pillar is large enough, the stability of the coal pillar will be maintained by forming an elastic zone in the centre, in the middle of the coal pillar, to ensure its stability [28]. During the excavation of the 02 air-return roadway, one side is a goaf, and the other is solid coal, but it is subject to secondary mining stress during the mining period of this working face. The width of the coal pillar should be conducive to controlling the stability of the surrounding rock in the roadway. Therefore, in order to avoid the severe deformation of the surrounding rock, a roof collapse, floor

bulging, and other disasters, the width of the protective coal pillar should not be too small [29]. The theoretical calculation formula for setting a reasonable width is

$$B = x_1 + x_2 + x_3 \quad (2)$$

where x_1 is the width of the plastic zone generated by the mining of the 01 working face, m; x_2 is the width of the plastic zone generated by the mining of the 02 working face, m; and x_3 is the width of the middle elastic zone, which is not less than twice the height of the coal pillar.

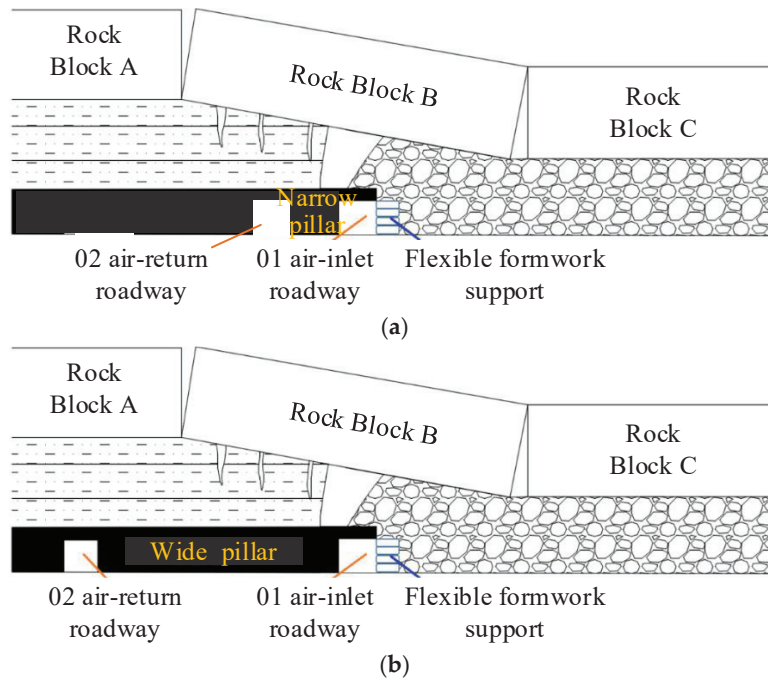


Figure 5. Roadway excavation methods for different coal pillars. (a) Roadway driving along next goaf; (b) wide-coal-pillar roadway protection.

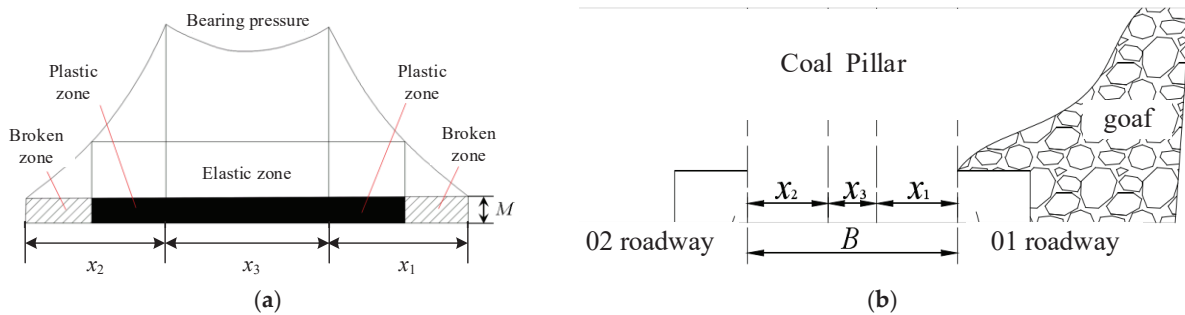


Figure 6. Theoretical model of reasonable coal pillar width. (a) Stress distribution of coal pillar; (b) schematic diagram of coal pillar width.

By implementing the limit equilibrium theory of rock masses, researchers can determine the diameter of the plastic zone as follows [30]:

$$x_1 = \frac{M}{2\bar{\xi}f} \ln \frac{K\gamma H + C \cot \varphi}{\bar{\xi}(Q_1 + C \cot \varphi)} \quad (3)$$

where K is the stress concentration factor; Q_1 is the resistance to coal support; C is the cohesive force of coal; φ is the internal friction angle of the coal body; F is the friction coefficient of the contact surface between the coal seam and the roof and floor; $\bar{\xi}$ is the

triaxial stress coefficient; γ is the bulk density of the overlying rock layer, N ; and H is the height of the overlying rock layer.

On one side of the coal pillar, researchers calculate that the plastic zone is 6.5 m wide, and considering the mining influence coefficient of 1.5, studies determined that the width of the coal pillar should not be less than 31.7 m.

Numerical simulation methods are further adopted to study the influence of coal pillar width. The simulation model adopts the model described in Section 3.1, and Figure 7 shows the excavation form. According to studies, the coal pillar's width was varied to 10 m, 20 m, 30 m, 40 m, 50 m, 60 m, and 70 m. During the excavation of the roadway in the working face, Figure 8 shows the distribution of internal support pressure of the protective coal pillar in different simulation schemes.

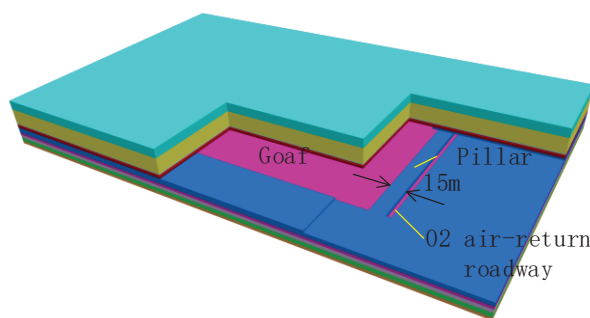


Figure 7. Simulated excavation method.

Figure 8 shows that the extreme values of the support pressure inside the protective coal pillar are consistent in different simulation schemes. Affected by the lateral advance support pressure of the 01 working face, the range of the ultimate equilibrium zone on the side of the 01 air-intake roadway increases; that is, the development range of the plastic zone increases. As the width of the coal pillar increases, the range of the impact area of the peak support pressure in the heading section gradually decreases, and the peak support pressure on both sides of the coal pillar gradually decreases. When the width of the coal pillar exceeds 30 m, the peak change is insignificant. The peak support pressure on both sides of the coal pillar has not yet overlapped, indicating that the roadway coal pillar's overall stability is still stable.

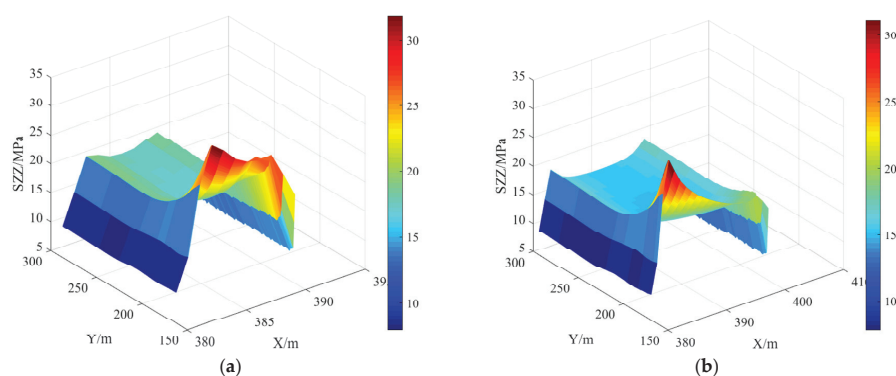


Figure 8. Cont.

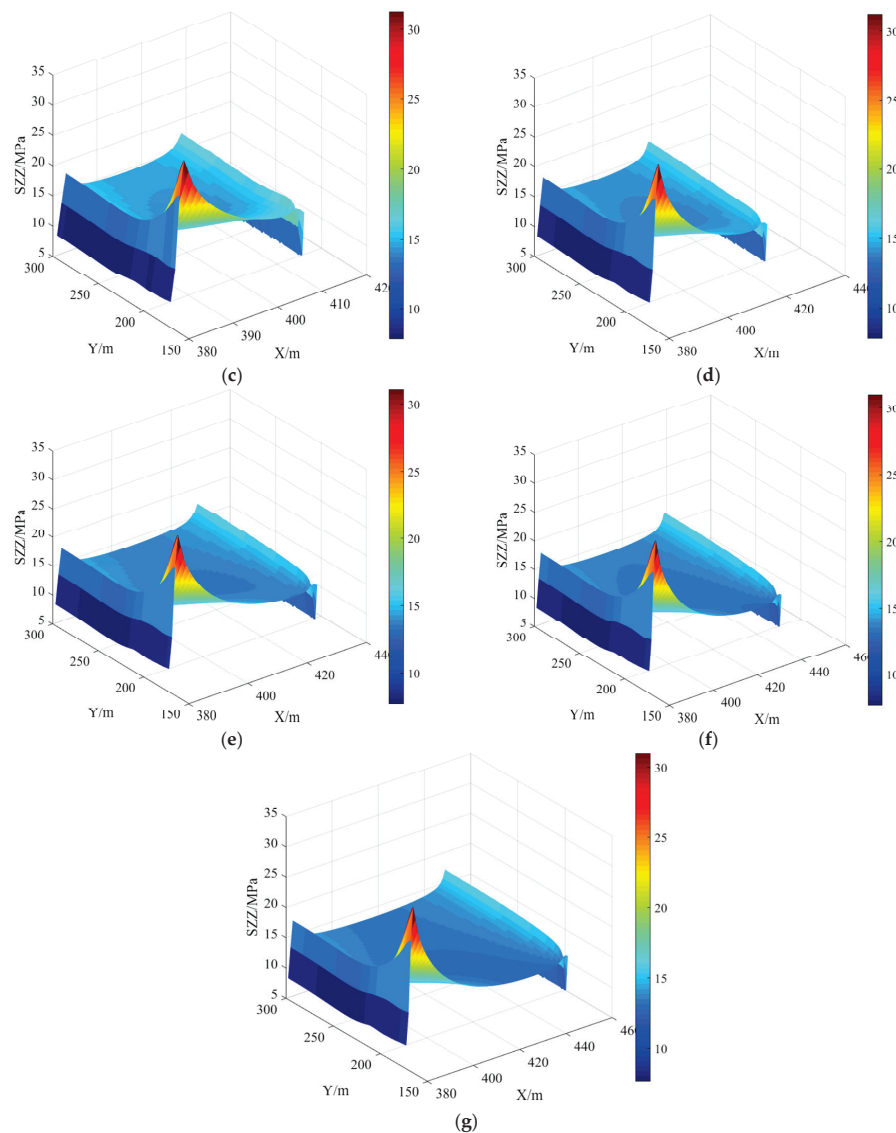


Figure 8. Bearing pressures of the protective coal pillar during the excavation process of the roadway in the working face: (a) 10 m; (b) 20 m; (c) 30 m; (d) 40 m; (e) 50 m; (f) 60 m; (g) 70 m.

Figure 9 depicts the bearing pressure in the coal pillar while mining the 02 working face. With the mining of the 02 working face, the peak support pressure inside the protective coal pillar in each simulation scheme continues to increase. When the width of the protective coal pillar is less than 20 m, the peak support pressure on both sides will overlap, and the coal pillar will undergo plastic failure as a whole. That is, when the width of the protective coal pillar is greater than or equal to 30 m, a stable elastic core zone will appear inside. There is no original rock stress zone inside the protective coal pillars of all schemes.

Simultaneously detecting the deformation displacement at different positions is shown in Figure 10. The load directly acts on the roof and side support during the coal seam excavation. Compared to other displacement curves, the overall deformation of the 02 air-return roadway floor is relatively small, indicating that the roof and two sides are critical areas for roadway maintenance. As the width of the protective coal pillar increases, the displacement gradually decreases, which is more favourable for controlling the stability of the surrounding rock in the mining roadway. When the width of the coal pillar is 10 m, the displacement of the surrounding rock is much higher than in other schemes. As the width of the coal pillar increases, the displacement gradually decreases, and the magnitude of the decrease gradually decreases. When the width of the protective coal pillar exceeds 40 m,

the displacement of the surrounding rock no longer changes significantly. Therefore, the best control impact on the surrounding rock of the mining roadway is achieved, according to research, when the protective coal pillar's width is greater than 40 m.

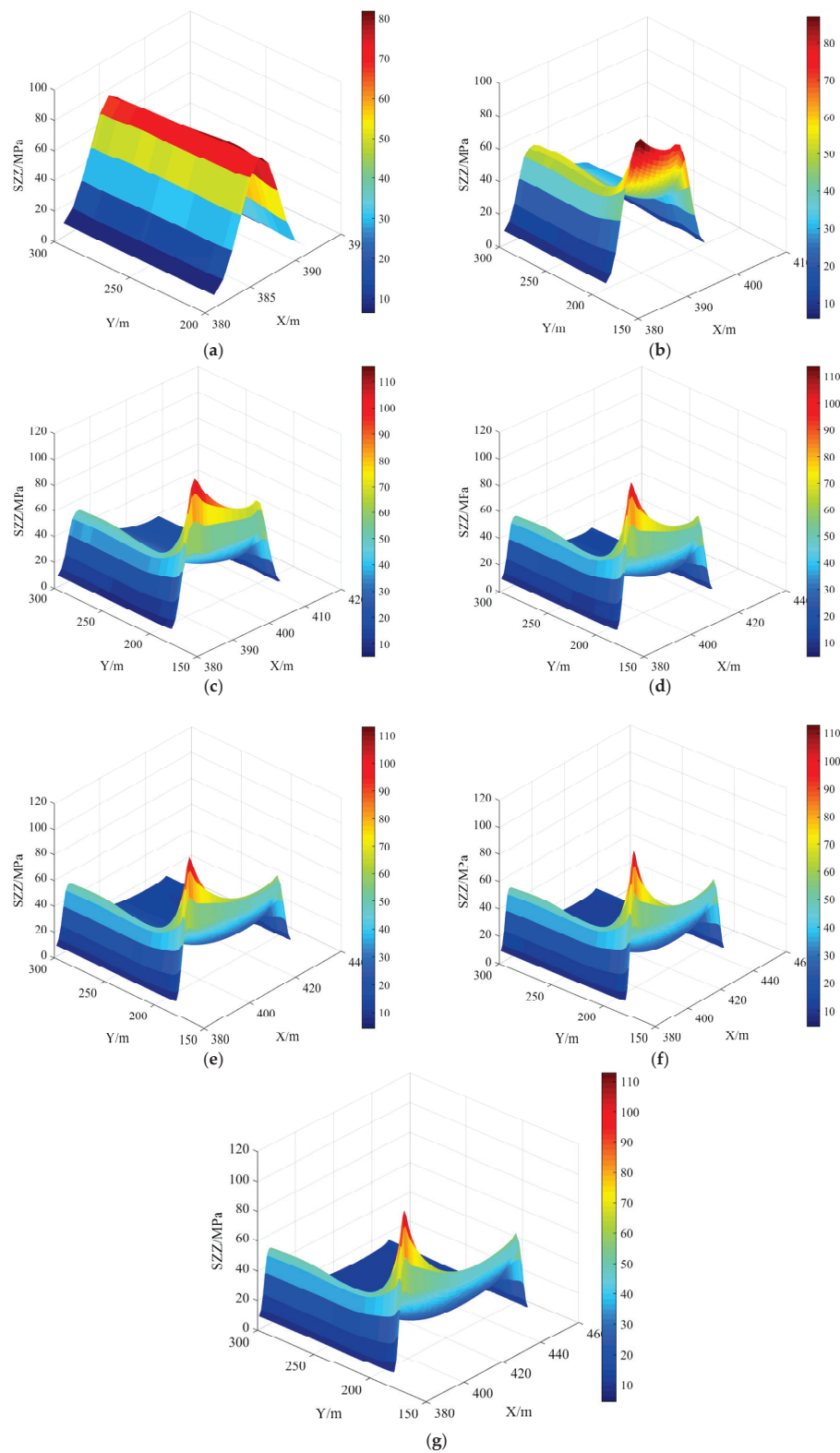


Figure 9. Bearing pressures of the protective coal pillar during 02 work face mining: (a) 10 m; (b) 20 m; (c) 30 m; (d) 40 m; (e) 50 m; (f) 60 m; (g) 70 m.

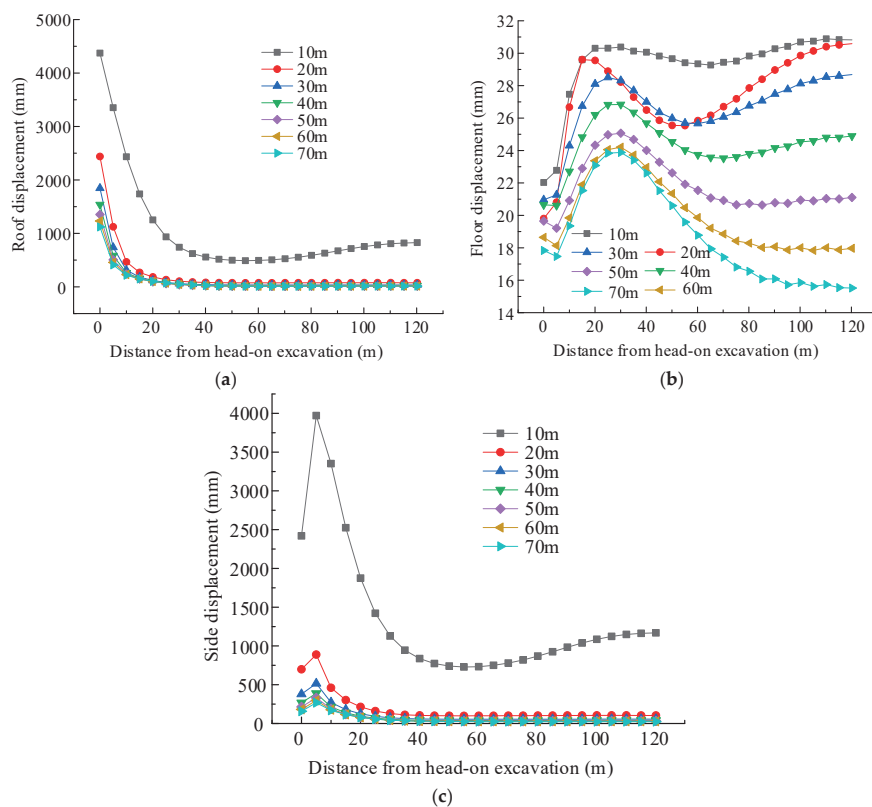


Figure 10. Deformation characteristics of the roadway under different coal pillar widths. (a) Roof displacement; (b) floor displacement; (c) side displacement.

4. Gas Control Method for Non-Pillar Coal Mining in High-Gas Top-Coal-Caving Face

4.1. Gas Concentration Law of W-Shaped Ventilation Fully Mechanized Top-Coal-Caving Face

Experts studied the migration rule of the gas flow field in the working face using FLUENT 18.2 software. Based on the design of mining parameters and gas control mode for the 02 working face, Gambit 2.4 software was used to establish models with coal pillar widths of 10 m, 30 m, 50 m, 70 m, and 90 m, as shown in Figure 11. Research analyzed how the distribution of the gas flow field in the 02 working face is affected by the width of the coal pillar.

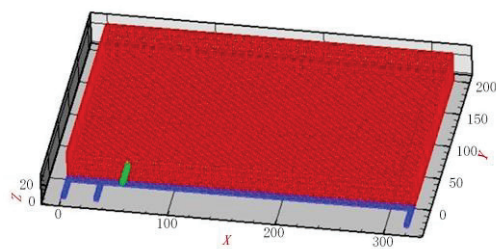


Figure 11. FLUENT simulation model of gas working face.

The 02 working face layout length is 320 m, and the height of the working face is 3.5 m. The net width and height of the return airway section are 5.2 m and 3.7 m, respectively. The net width and height of the transportation airway are 5.2 m and 3.7 m, respectively. The net width and height of the intake duct are 5.8 m and 3.8 m, respectively. The gas content is $6.0007 \text{ m}^3/\text{t}$, the residual gas content is $3.0039 \text{ m}^3/\text{t}$, and the maximum coal gas emission per cycle is $1.0 \text{ m}^3/\text{min}$.

The numerical simulation boundary conditions and calculation parameters were set in FLUENT software, with the selection of boundary conditions referring to Table 2 and the selection of discrete phase parameters referring to Table 3.

Table 2. Boundary conditions.

Boundary Name	Boundary Type	Boundary Type
Air-intake roadway	Velocity inlet	0.76 m/s
Transportation roadway	Velocity inlet	2.17 m/s
Air-return roadway	Outflow	0.9
High-pumping roadway	Outflow	0.1
Internal interface of model	Interior	-
Other external walls of model	Wall	-

Table 3. Continuous phase parameters.

Model	Definition
Solver	Pressure-Based
Viscous Model	k-Epsilon
Energy	On
Material	Methane Air

The caving zone formed during coal seam mining is the primary medium of the flow field in the goaf [31]. As the falling rock blocks and gangue are compacted, the compression of the height of the falling zone approximately follows a negative exponential function relationship [32]. Therefore, the distribution law of porosity and permeability of the porous medium in the goaf of the W-shaped fully mechanized face also approximately follows a negative exponential function relationship. Figure 12 shows the distribution characteristics of permeability in the goaf of a W shape. Along the direction of the movement, as it moves away from the working face, the porosity gradually decreases [33]. In the direction of inclination, the porosity at the inlet and outlet air ducts is higher, while the porosity at the solid coal is lower.

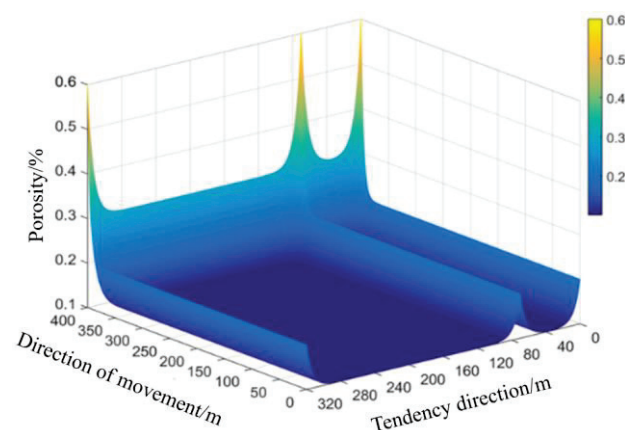


Figure 12. Distribution characteristics of permeability in goaf of W-type fully mechanized caving face.

In the W4302 goaf, researchers can obtain the distribution characteristics of the gas flow field by establishing a three-dimensional CFD model, as shown in Figure 13. The distribution characteristics of gas concentration behind the goaf are similar to those of U-shaped ventilation. Under the gas control mode of a W-shaped ventilation system combined with a high-extraction roadway, gas accumulates in the deep part of the goaf and enters an enrichment zone as the working face advances. Due to the leakage of fresh air from the air-intake roadway and transportation roadway towards the goaf, gas is blown towards the deeper part of the goaf [34]. The air-return roadway contains gas air that leaks

from the goaf, resulting in a higher gas content near the return airway in the middle of the goaf than on both sides of the goaf. Due to the gas being pumped into the high extraction roadway below the goaf, a gas reduction area appears behind the working face, which can promote safe mining of the working face [35].

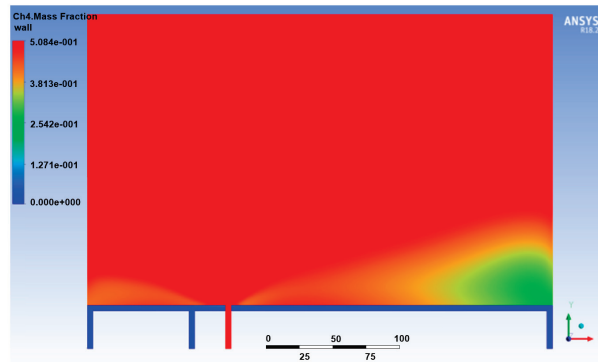


Figure 13. Distribution characteristics of gas flow field in goaf of W4302 working face.

4.2. Reasonable Coal Pillar Size Based on Gas Control

FLUENT software was used to simulate the distribution characteristics of the gas flow field in the W4302 working face with coal pillar widths of 10 m, 30 m, 50 m, 70 m, and 90 m. Figure 14 shows the sliced cloud maps of the gas flow field. In the high-extraction lane and the return-air roadway, experts found a correlation between coal pillar width and gas concentration by taking the average concentration through measuring lines at the return-air roadway and high-extraction roadway, as shown in Figure 14.

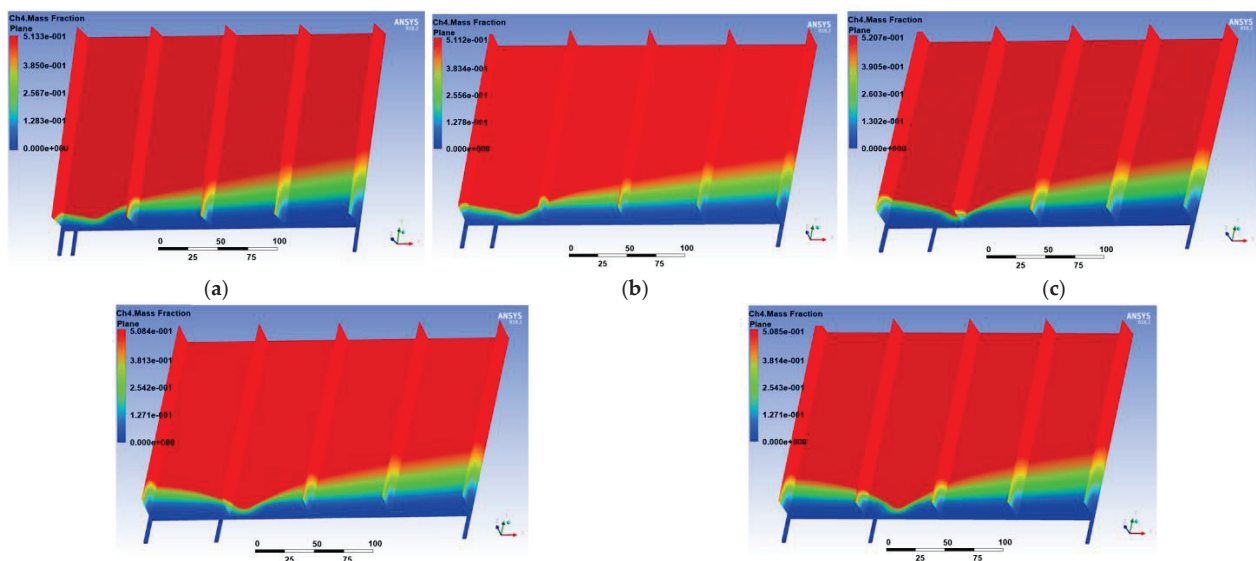


Figure 14. The sliced cloud maps of the gas flow field with different pillars: (a) 10 m; (b) 30 m; (c) 50 m; (d) 70 m; (e) 90 m.

Figure 15 shows the variation curve of gas concentration with coal pillar width. As the width of the coal pillar decreases, the gas content in the return airway gradually decreases. When the width of the coal pillar is 70 m, the gas concentration at the return airway is 0.75%, which can meet the operational requirements. When the width of the coal pillar is greater than 70 m, the rate of decrease in gas concentration at the return airway slows down. As the width of the coal pillar increases from 70 m to 90 m, the gas concentration decreases by 0.03%. The gas concentration extracted from the high-level suction roadway is between 50% and 51% and is not significantly affected by the width of the coal pillar.

Through a theoretical analysis and numerical simulation, based on the stability control of the roadway-surrounding rock and the migration law of the gas flow field, the width of the protective coal pillar should exceed 70 m to meet the requirements of gas control and roadway stability control.

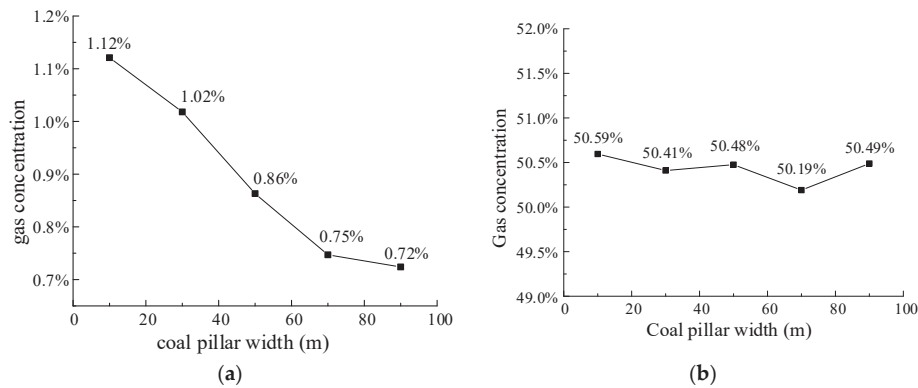


Figure 15. Gas concentration change curve. (a) Air-return roadway; (b) high-level-suction roadway.

5. On-Site Application Effect Analysis

Figure 16 shows the 02 roadway's cross-section, which is held up by cables and anchor rods. To track the deformation of the surrounding rock, experts placed eight measuring stations symmetrically on the top and bottom plates and on the left and right sides of the 02 air-return lanes. Figure 17 shows the location of the measuring points.

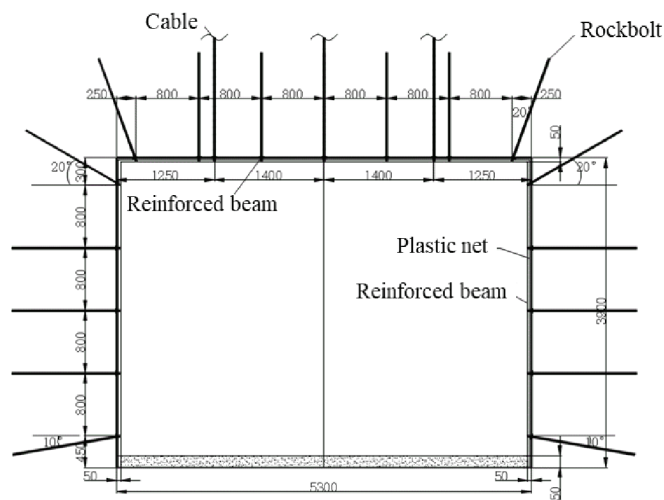


Figure 16. Section diagram of roadway support.

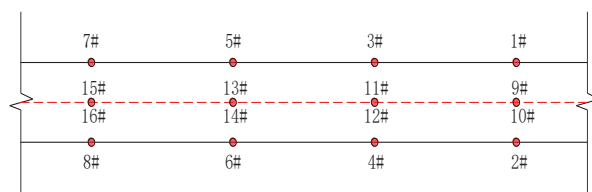


Figure 17. Distribution of monitoring points for surrounding rock.

The “cross” point method is used to observe the deformation of the surrounding rock of the roadway, as shown in Figure 18. A hole with a diameter of 30 mm and a depth of 380 mm is set in the middle of the top and bottom plates vertically and horizontally. A 400 mm long wooden pile with a diameter of 30 mm is driven into the hole, and a measuring

nail is installed at the end of the wooden pile. The two monitoring sections are spaced 0.6–1.0 m apart along the axial direction of the roadway. During measurement, tension measuring ropes are installed at points C and D, and steel tape measures are tightened between points A and B to measure A.O. and O.B. values.

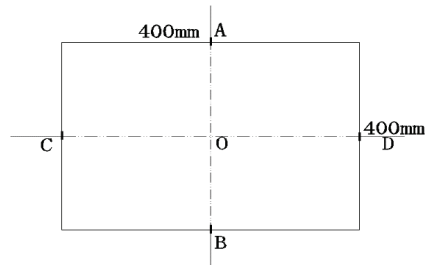


Figure 18. Diagram of surface displacement monitoring section.

Similarly, steel tape measures are tightened between points C and D to measure C.O. and O.D. values. The measurement accuracy requirement is 1 mm, and an estimated 0.5 mm is obtained. A leather tape measures the distance from the monitoring section to the excavation working face.

By monitoring the displacement of different points daily, Figure 19 shows the overall deformation of the roadway. Within the first ten days of excavation, the mining roadways showed the deformation rate of the two sides as the highest, reaching 2.25 mm/d. After 10 days, the deformation rate showed a decreasing trend of fluctuation. However, the overall movement of the two sides continued to increase until 60 days after excavation, when the movement of the two sides tended to stabilize and the deformation rate approached 0. The cumulative movement of the two sides was approximately 90 mm. On-site observations showed that the integrity of the surrounding rock in the mining roadway was high, and there were no apparent cracks or fissures on the surface of the surrounding rock.

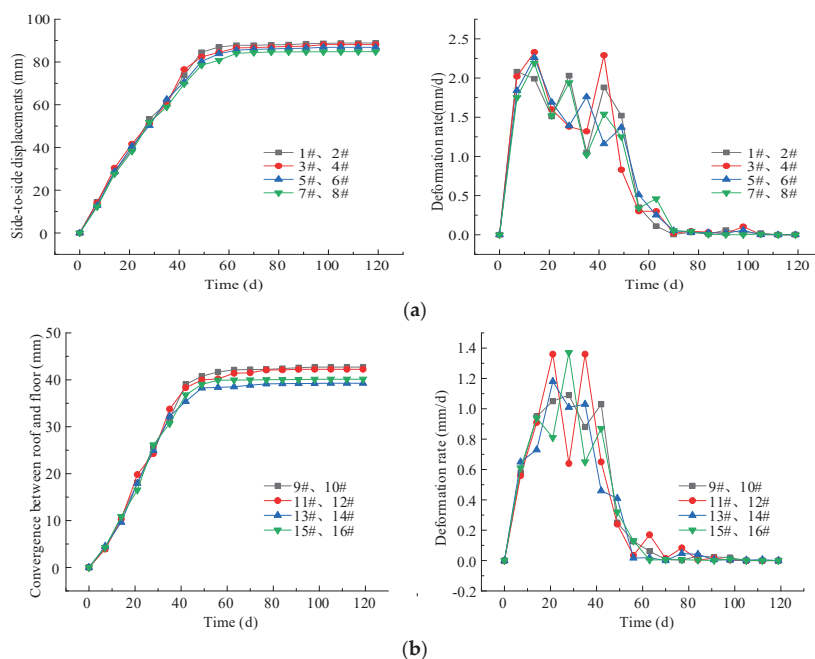


Figure 19. Displacement curve and deformation rate of measuring point in roadway. (a) Side to side; (b) roof to floor.

6. Conclusions

This article proposes an innovative non-pillar coal mining method that recovers coal pillars in a later stage. The width of the coal pillar is the key to achieving this process.

Based on the stability of the surrounding rock and gas movement, experts calculated the coal pillar's breadth, which led to the following key findings.

- (1) The proposed non-pillar coal mining technology by a driving roadway during the mining period, which reserves wide coal pillars, arranges W-shaped fully mechanized caving faces in sequence to replace Y-shaped ventilated fully mechanized caving faces, and reserves wide coal pillars for mining. The extraction rate of the working face has increased by 12.6%, and the mining replacement time has been shortened by 15 months.
- (2) The paper analyzes the complex stress environment of mining, determines the segmented equations for the boundaries of different mining influence ranges, and reveals the spatiotemporal effects of the stress environment of an excavation roadway during mining. The influence range of lateral support pressure is within 40 m from the coal wall on the side of the roadway.
- (3) To determine the reasonable width of the coal pillar, the stability of the protective coal pillar and the deformation of the surrounding rock of the roadway were studied. In addition to analyzing the effects of changes in the protective coal pillar's width on the working face gas flow field's movement law, the reasonable width of the protective coal pillar was finally determined to be 70 m.
- (4) The determined coal pillar width was successfully applied in the on-site implementation project, verifying the correctness of relevant theories. The cumulative movement of the two sides was approximately 90 mm, which meets the on-site usage requirements.

Author Contributions: Conceptualization, H.S. and Y.Y.; methodology, Z.C.; software, H.S.; validation, Y.Y.; formal analysis, Z.C.; resources, Y.Y.; data curation, Z.C.; writing, Z.C.; writing—review and editing, S.K.A.; project administration, Y.Y. and B.L. All authors have read and agreed to the published version of the manuscript.

Funding: Financial support for this work was provided by the Qing Lan Project of Jiangsu Province Universities (2022), National Natural Science Foundation of China (52204132), Hunan Provincial Natural Science Foundation of China (2023JJ40285), and Scientific Research Foundation of Hunan Provincial Education Department (22B0469).

Data Availability Statement: The original contributions presented in the study are included in the article; further inquiries can be directed to the corresponding author.

Conflicts of Interest: Author Haisheng Shen was employed by the company Shanxi Lu'an Mining Group Cilinshan Coal Industry Co., Ltd. The remaining authors declare that the research was conducted in the absence of any commercial or financial relationships that could be construed as a potential conflict of interest.

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Article

Initial Desorption Characteristics of Gas in Tectonic Coal Under Vibration and Its Impact on Coal and Gas Outbursts

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Abstract: The rapid desorption of gas in coal is an important cause of gas over-limit and outbursts. In order to explain the causes of coal and gas outbursts induced by vibration, this paper studies the gas desorption experiments of tectonic coal with different particle sizes and different adsorption equilibrium pressures under 0–50 Hz vibration. High-pressure mercury intrusion experiments were used to measure the changes in pore volume and specific surface area of tectonic coal before and after vibration, revealing the control of pore structure changes on the initial desorption capacity of gas. Additionally, from the perspective of energy transformation during coal and gas outbursts, the effect of vibration on the process of coal and gas outbursts in tectonic coal was analyzed. The results showed that tectonic coal has strong initial desorption capacity, desorbing 29.58% to 54.51% of the ultimate desorption volume within 10 min. Vibration with frequencies of 0–50 Hz increased both the gas desorption ratios and desorption volume as the frequency increased. The initial desorption rate also increased with the vibration frequency, and vibration can enhance the initial desorption capacity of tectonic coal and delay the attenuation of desorption rate. Vibration affected the changes in the initial gas desorption rate and desorption rate attenuation coefficient by increasing the pore volume and specific surface area, with the changes in macropores and mesopores primarily affecting the initial desorption rate and 0–10 min desorption ratios, while the changes in micropores and minipores mainly influenced the attenuation rate of the desorption rate. Vibration increased the free gas expansion energy of tectonic coal as the frequency increased. During the incubation and triggering processes of coal and gas outbursts, vibration has been observed to accelerate the fragmentation and destabilisation of the coal body, while simultaneously increasing the gas expansion energy to a point where it reaches the threshold energy necessary for coal transportation, thus inducing and triggering the coal and gas protrusion. The study results elucidate, from an energy perspective, the underlying mechanisms that facilitate the occurrence of coal and gas outbursts, providing theoretical guidance for coal and gas outburst prevention and mine safety production.

Keywords: tectonic coal; vibration; initial desorption rate; coal and gas outburst; gas expansion energy

1. Introduction

Coal and gas outbursts are extremely complex gas dynamic disasters that occur in underground coal mines, with China accounting for 40% of the total global outburst incidents [1–4]. As the depth of mining increases, both the gas content and gas pressure in coal seams gradually increase, making coal and gas outbursts a persistent and significant hazard threatening safe coal extraction [5]. The mechanism of coal and gas outbursts essentially involves the coupling of multiple complex physical processes, such as coal mass failure, gas desorption, gas diffusion and seepage, coal transportation in the gas flow field, and two-phase flow propagation [6]. Studies have shown that rapid gas desorption in coal, caused by external disturbances, is a major trigger of gas over-limit and outbursts [7,8].

The hypothesis of the comprehensive effect of coal and gas outburst holds that outburst is the result of the combined action of in situ stress, gas in the coal seam, the physical and mechanical properties of coal, the microstructure and macrostructure of coal, the coal seam structure, and the self-gravity of coal [9]. The adsorbed and free gas in coal directly affects the mechanical properties of coal [10]. The erosion of gas will cause the overall weakening of the mechanical strength of coal, and the greater the gas pressure, the more obvious the deterioration of dynamic parameters [11,12]. The gas in coal provides expansion energy for the destruction, pulverisation, handling, and throwing of coal during the occurrence and development of coal and gas outbursts [13]. Among them, the rapid desorption of gas in pulverised coal can provide a lot of energy for the outburst process [9,14,15]. Therefore, studying the desorption characteristics of gas in coal is an important method for preventing coal and gas outbursts and for summarising the rules governing their occurrence.

The gas desorption and diffusion capacity of coal is influenced by a multitude of factors, which can be classified into two principal categories: internal and external. The internal factors are predominantly influenced by the intrinsic characteristics of the coal itself, such as the level of coalification [16–18]. The external factors are primarily determined by external environmental conditions, which can include factors such as temperature [19,20], pressure [21,22], physical fields [23–26], and moisture [27–29]. Prior to coal and gas outbursts, operations that generate vibrations, such as drilling, blasting, and coal cutting, are often involved [30]. Some scholars have suggested that vibration acts as the energy transfer mechanism necessary for coal fragmentation, gas desorption, and the ejection of gas–solid mixtures during the incubation and development stages of coal and gas outbursts [13]. As an important physical field, vibration inevitably influences the gas desorption characteristics of coal. Previous studies have shown that low-frequency mechanical vibration affects coal seam permeability [31], porosity [32], mechanical properties [33], and the gas adsorption–desorption capacity of coal [26,30,34,35]. It was found by Li et al. [34] that the mechanical vibration at 0–40 Hz increases the adsorption sites in coal, and as a consequence of this, the process of gas molecule desorption and diffusion is impeded. Shen et al. [26,30] reported that vibration alters the pore structure of coal, increasing pore volume, specific surface area, and connectivity, which reduces resistance to gas desorption and diffusion, and promotes gas desorption and diffusion in coal. Li's [36] research indicated that after vibration, the changes in gas desorption and diffusion parameters in acid-treated granular coal were more pronounced compared to untreated coal, suggesting that vibration facilitates the desorption and diffusion of gas in acid-treated granular coal. Zhang [37] showed that vibration significantly enhances gas desorption in gas-bearing coal, increasing both the desorption volume and desorption rate. The closer the vibration frequency is to the natural frequency of the coal mass, the more effective the desorption process. Although these studies have extensively investigated the desorption characteristics of vibrated coal samples, they have not fully analysed the initial desorption characteristics of gas-bearing coal under vibration.

Following prolonged geological tectonism (faulting, folding, slip, etc.), the primary coal has undergone a range of fracturing processes, including brittle fracturing, crushing, ductile deformation, superposition failure, and even internal chemical composition and structural changes. This results in the formation of tectonic coal with typical low strength and low permeability characteristics, which provide favourable conditions for initiating coal and gas outbursts [38–41]. Accident statistics have shown that regions prone to coal and gas outbursts almost always exhibit varying thicknesses of tectonic coal, making tectonic coal a necessary condition and significant characteristic of coal and gas outbursts [13,42–44]. Compared with primary coal, tectonic coal has lower strength, more developed pore structures, weaker resistance to external stress disturbances, and a higher tendency to pulverise [42,45–47]. These factors result in desorption characteristics that differ from those of primary coal. In comparison to primary coal, tectonic coal exhibits a stronger gas adsorption–desorption capacity and a greater initial desorption–diffusion ability [21,48]. This phenomenon gives rise to a rapid rate of gas desorption during the early stages, which,

in turn, facilitates the release of gas energy [5,46,47,49,50]. Consequently, tectonic coal is more prone to destruction and pulverisation, and rapid desorption can cause a large accumulation of free gas in a short period, increasing the risk of coal and gas outbursts [15]. According to the survey results of the outburst accident site, almost all of the outburst coal in the outburst accident site is highly broken or pulverised [49,51–53]. Studies have shown that most of the outburst coal is fine pulverised coal, which is not pulverised after outburst, but has been pulverised in the coal seam before outburst [14]. Therefore, the selection of granular coal as the research object can better explain the causes of vibration induced coal and gas outbursts. Therefore, it is essential to investigate the effect of vibration on the initial desorption capacity of particle tectonic coal in order to assess the risk of coal and gas outbursts in vibrated tectonic coal.

In conclusion, in order to elucidate the underlying causes of vibration-induced coal and gas outbursts from the standpoint of altered initial gas desorption capacity in vibration-affected tectonic coal, this paper employs representative tectonic coal to conduct isothermal desorption experiments under vibration. The focus is on analysing the changes in desorption parameters during the initial desorption period in vibrated tectonic coal. Furthermore, the study examines the impact of changes in the initial desorption capacity on the occurrence and development stages of coal and gas outbursts from an energy perspective. The results of this study are significant for clarifying the reasons why vibration induces coal and gas outbursts and can provide theoretical guidance for the prevention of coal and gas outbursts in mines.

2. Materials and Methods

2.1. Selection and Preparation of Coal Samples

The pore structure of coal samples with differing particle sizes varies, as does the migration resistance of gas in granular coal [48,54]. Studies have shown that the particle size of pulverised coal thrown out at the outburst site is mainly concentrated within 3 mm [55], and the particle coal in the range of 0.25–1 mm is more likely to be pulverised twice in the process of coal and gas outbursts [14]. Accordingly, the present study has elected to examine tectonic coal with particle sizes of 0.25 to 0.5 mm, 0.5 to 1 mm, and 1 to 3 mm as the subject of its inquiry. Tectonic coal samples were selected from the Wangxingzhuang coal mine (WXZ) in Henan Province. The Wangxingzhuang coal mine is situated within the Xinmi coalfield. As a result of tectonic movement, the entirety of the No. 2-1 coal seam has developed tectonic coal. The WXZ coal samples were procured from the 15,021 working face of the No. 2-1 coal seam. Fresh samples were sealed and transported to the laboratory, where they were subjected to crushing operations to produce particle sizing of 0.25 to 0.5 mm, 0.5 to 1 mm, and 1 to 3 mm. One kilogram of each particle size was prepared for subsequent adsorption–desorption experiments and pore structure analysis. Following the preparation of the coal samples, these were subjected to a 48 h drying process at 70 °C in a vacuum oven. Following this, the samples were cooled and then sealed for storage. Figure 1 illustrates the mesoscopic characteristics of the coal samples, while Table 1 presents a list of the basic physical parameters.

2.2. Methods

In order to study the gas desorption and diffusion characteristics of granular coal under mechanical vibration, a mechanical vibration isothermal adsorption–desorption platform (Figure 2) of the MVGAD-I type was used. Experiments of the isothermal desorption type were performed on samples of granular coal of differing particle sizes (0.25–0.5 mm, 0.5–1 mm, and 1–3 mm) under varying vibration frequencies, with the equilibrium pressure set at 1 MPa. To examine the differences in gas desorption characteristics under different vibration frequencies and equilibrium pressures, the coal samples with a particle size of 1–3 mm were selected for vibration desorption experiments at different equilibrium pressures (0.5 MPa, 1 MPa, and 1.5 MPa) according to the standard “Determination Method of Gas Desorption Index by Drill Cuttings” (AQ/T 1065-2008) [56]. The adsorbed gas is

methane with a purity of 99.9%. The natural frequency of coal has been demonstrated to fall within the 50 Hz range [57], while the mechanical vibrations generated by drilling and tunnelling activities that result in gas-bearing coal seam outbursts have been shown to be below 50 Hz [34]. Additionally, the frequency range of microseismic vibrations has been established to lie between 0 and 35 Hz [30]. In light of the aforementioned vibration frequencies and the objective of this study, the selected experimental frequencies were 0, 25, and 50 Hz, and the vibration time lasted for 60 min. After the desorption experiments, coal samples of 1~3 mm in size underwent degassing treatment and were then subjected to high-pressure mercury intrusion experiments to measure pore structure.

Table 1. Basic physical parameters of tectonic coal sample.

Coal	Adsorption Constant		Components			ρ_{TRD} (g/m ³)	ρ_{ARD} (g/m ³)	ϕ_{pore} (%)
	a (m ³ /t)	b (MPa ⁻¹)	M _{ad} (%)	A _d (%)	V _{daf} (%)			
WXZ	26.32	2.74	0.88	7.73	13.81	1.43	1.38	3.19



Figure 1. Mesoscopic characteristics of the coal sample.

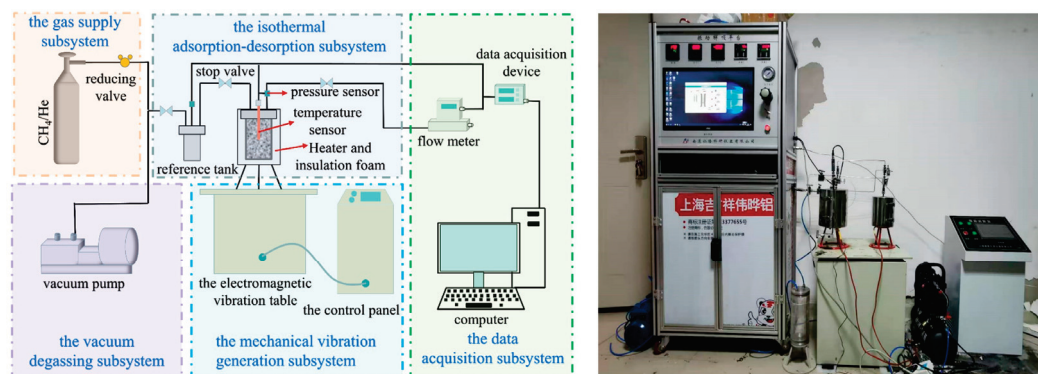


Figure 2. Schematic diagram and physical diagram of experimental equipment.

3. Results

3.1. Characteristics of Initial Desorption of Gas in Vibrated Tectonic Coal

To investigate the initial desorption characteristics of gas in vibration-affected tectonic coal under the influence of different adsorption equilibrium pressures and particle sizes, the proportion of desorbed gas volume to the ultimate desorption volume was calculated for the first minute, 1~5 min, 5~10 min, 10~30 min, and 30~60 min to reflect the desorption capacity of the coal samples at different stages. Due to the rapid occurrence of coal and gas outbursts, which typically happen within a few minutes or even seconds, and given that studies indicate that the initial desorption capacity of tectonic coal is significantly greater than that of primary coal, allowing for the desorption of substantial amounts of gas within minutes, this study focuses on analysing the desorption characteristics of coal samples during the intervals of 0 to 1 min ($Q_{0-1\text{min}}$) and 0 to 10 min ($Q_{0-10\text{min}}$). The ultimate desorption volume of the coal samples can be calculated using Formula (1) [30]:

$$Q_{\infty} = \left(\frac{abP}{1+bP} - \frac{abP_0}{1+bP_0} \right) (1 - W_f - A_f) \quad (1)$$

In Formula (1), the gas adsorption constant a represents the maximum adsorption amount of coal under the specified experimental conditions, mL/g. The value of b , also known as the gas adsorption constant, is MPa^{-1} . The moisture of the coal, designated as W_f is in %; the ash content is indicated by A_f , %. Additionally, P denotes the adsorption equilibrium pressure, while P_0 represents the atmospheric pressure at the given time. It is notable that P_0 is set to 0.1 MPa.

3.1.1. Initial Desorption Characteristics of Gas in Vibration-Affected Tectonic Coal with Different Particle Sizes

Figure 3 illustrates the gas desorption volume and desorption ratio characteristics of coal samples with different particle sizes under vibration. It can be observed from Figure 3 that the gas desorption volume of all coal samples increases monotonically over time, with the rate of increase gradually slowing down. The desorption volume is observed to decrease with increasing particle size for the same duration and vibration frequency. Additionally, the initial desorption volume is seen to increase at an accelerated rate with decreasing particle size. In the context of vibrations in the frequency range of 0–50 Hz, the desorption volume of all coal samples subjected to vibration is observed to exceed that of the non-vibrated samples. Moreover, a positive correlation is evident between the vibration frequency and the desorption volume. This suggests that mechanical vibration plays a facilitating role in the process of gas desorption. A significant rapid desorption process is evident during the initial desorption period of the coal samples. For the tectonic coal with a particle size of 0.25–0.5 mm, the desorption rate $Q_{0-1\text{min}}$ at the first minute under different vibration frequencies reaches 26.2–27%; within 10 min, it can desorb 50.66–54.51% of the ultimate desorption volume, with $Q_{0-10\text{min}}$ at 25 Hz and 50 Hz being 1.03 and 1.08 times that of 0 Hz, respectively. For the tectonic coal with a particle size of 0.5–1 mm, the desorption rate $Q_{0-1\text{min}}$ at the first minute reaches 19.1–22.9%; within 10 min, it can desorb 42.84–49.71% of the ultimate desorption volume, with $Q_{0-10\text{min}}$ at 25 Hz and 50 Hz being 1.09 and 1.16 times that of 0 Hz, respectively. For the tectonic coal with a particle size of 1–3 mm, the desorption rate $Q_{0-1\text{min}}$ at the first minute reaches 18.6–19.9%; within 10 min, it can desorb 31.51–34.67% of the ultimate desorption volume, with $Q_{0-10\text{min}}$ at 25 Hz and 50 Hz being 1.05 and 1.12 times that of 0 Hz, respectively. The desorption rates of all vibrated coal samples within the first 10 min are higher than those of the non-vibrated samples. This suggests that vibration has the effect of enhancing the initial desorption capacity of coal, resulting in the desorption of greater quantities of free gas in a shorter time period. Consequently, there is an increased potential for accumulating larger quantities of gas energy.

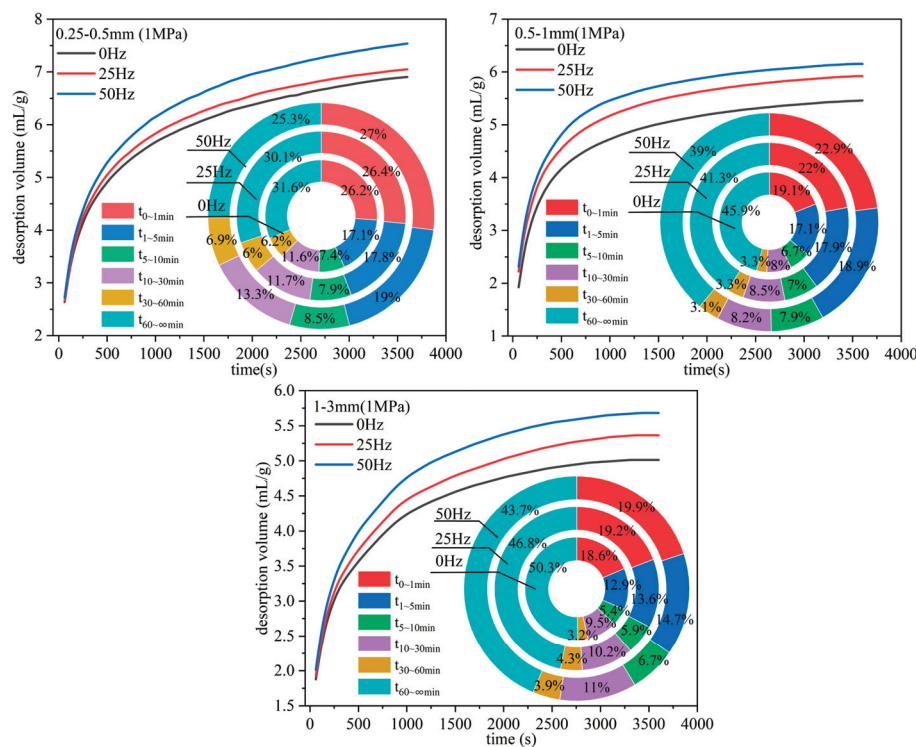


Figure 3. Gas desorption volume and desorption ratios characteristics of vibrated coal samples with different particle sizes.

To further understand the effect of vibration on the initial desorption capacity of tectonic coal, the desorption rate of vibrated tectonic coal within the first 0~10 min was analysed. The literature indicates that the gas desorption rate over time follows a power function relationship [58]:

$$V_t = V_0 \cdot t^{-k} \quad (2)$$

where V_t is the desorption rate at time t , expressed as $\text{mL} \cdot \text{g}^{-1} \cdot \text{min}^{-1}$; V_0 is the initial desorption rate, expressed as $\text{mL} \cdot \text{g}^{-1} \cdot \text{min}^{-1}$; and k is the desorption rate attenuation coefficient. The fitting results of the desorption rate data for vibrated tectonic coal with different particle sizes over 0~10 min are shown in Figure 4.

The results presented in Figure 4 demonstrate that for coal samples subjected to the same vibration frequency, the initial desorption rate (V_0) exhibits a decline as the particle size increases. Conversely, for coal samples of identical particle size, the desorption rate displays an upward trend as the vibration frequency rises. The desorption rate remains relatively high during the initial 3 min, indicating that tectonic coal has a robust initial desorption capacity. The variation in initial desorption rate and desorption rate attenuation coefficient for vibrated coal samples of different particle sizes is illustrated in Figure 5.

Figures 4 and 5 demonstrate that for coal samples at the same vibration frequency, the initial desorption rate decreases as the particle size increases. For coal samples with a particle size of 0.25~0.5 mm, the initial desorption rate at 25 Hz and 50 Hz is 1.01 and 1.03 times that of 0 Hz, respectively; for coal samples with a particle size of 0.5~1 mm, the initial desorption rate at 25 Hz and 50 Hz is 1.15 and 1.20 times that of 0 Hz, respectively; and for coal samples with a particle size of 1~3 mm, the initial desorption rate at 25 Hz and 50 Hz is 1.03 and 1.07 times that of 0 Hz, respectively. The initial desorption rates of all vibrated tectonic coal samples were observed to be higher than those of non-vibrated coal samples. Furthermore, an increase in frequency resulted in a corresponding increase in the desorption rate. In contrast, the desorption rate attenuation coefficient demonstrated a decrease with increasing vibration frequency. This finding suggests that vibration can delay the attenuation of the desorption rate. This evidence serves to reinforce the hypothesis that vibration increases the rate at which tectonic coal can desorb gas, leading to the desorption

of greater quantities of gas in a shorter time period. Consequently, conditions are created by vibration that are favourable to the occurrence of both coal and gas outbursts. It thus follows that tectonic coal, which is susceptible to pulverisation, is more likely to experience instances of coal and gas outbursts under the influence of vibration. Furthermore, the occurrence of such outbursts is more probable when the particle sizes are reduced to a smaller size, as this increases the likelihood of reaching the critical conditions required for these outbursts to occur.

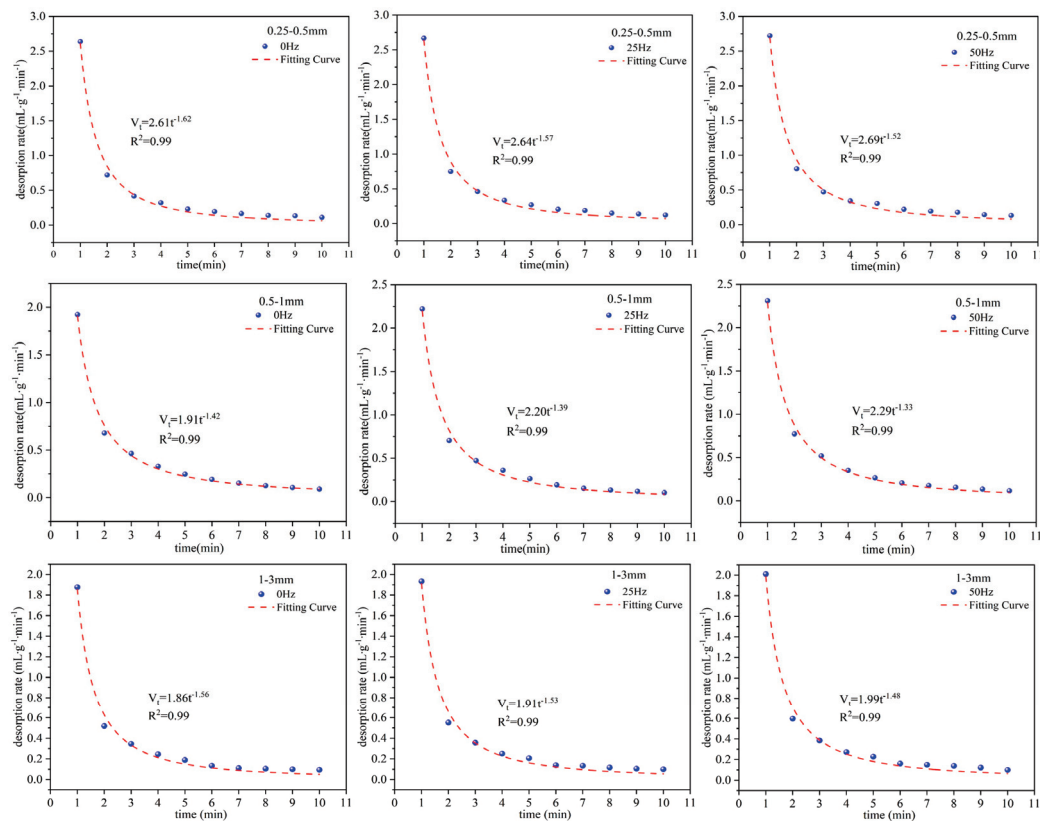


Figure 4. Fitting results and fitting formula of the desorption rate for vibrated tectonic coal with different particle sizes over 0~10 min.

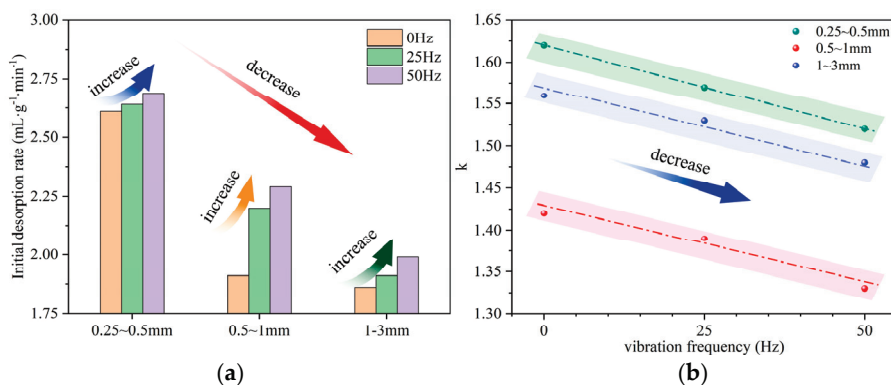


Figure 5. Variation trends in V_0 and k in vibrated tectonic coal with different particle sizes. (a) Variation trends in the initial desorption rates in vibrated tectonic coal with different particle sizes; (b) variation trends in desorption rate attenuation coefficient in vibrated tectonic coal with different particle sizes.

3.1.2. Initial Desorption Characteristics of Gas in Vibration-Affected Tectonic Coal Under Varying Adsorption Equilibrium Pressures

Adsorption equilibrium pressure is one of the key factors influencing gas desorption and diffusion. To study the effect of mechanical vibration on the gas desorption characteristics of coal samples under different adsorption equilibrium pressures, coal samples with a particle size of 1~3 mm were selected for vibration desorption experiments under equilibrium pressures of 0.5 MPa, 1 MPa, and 1.5 MPa.

Figure 6 demonstrates the gas desorption volume and desorption rate characteristics of vibration-affected tectonic coal under varying equilibrium pressures for adsorption. As shown in Figure 6, under the same vibration frequency, the cumulative gas desorption volume at the same time increases with the increase in adsorption equilibrium pressure. This is because the larger the adsorption equilibrium pressure, the stronger the concentration gradient driving rapid gas desorption and diffusion, resulting in more gas desorption within the same period. At adsorption equilibrium pressures of 0.5, 1, and 1.5 MPa, the desorption rates ($Q_{0-1\text{min}}$) of tectonic coal subjected to various vibration frequencies during the first minute were observed to be 12.78% to 23.42%, 18.6% to 19.9%, and 21.72% to 25.82%, respectively. Additionally, for coal samples at different adsorption equilibrium pressures exposed to vibrations at 25 Hz and 50 Hz, the desorption rates over the initial 10 min ($Q_{0-10\text{min}}$) were found to be 1.05 to 1.26 times and 1.12 to 1.45 times greater than those of the non-vibrated coal samples. Irrespective of the adsorption equilibrium pressure, the volume of gas desorbed and the rate of desorption of all coal samples subjected to vibration are greater than those of the non-vibrated coal samples. This provides further evidence that mechanical vibration enhances the desorption of gas from coal.

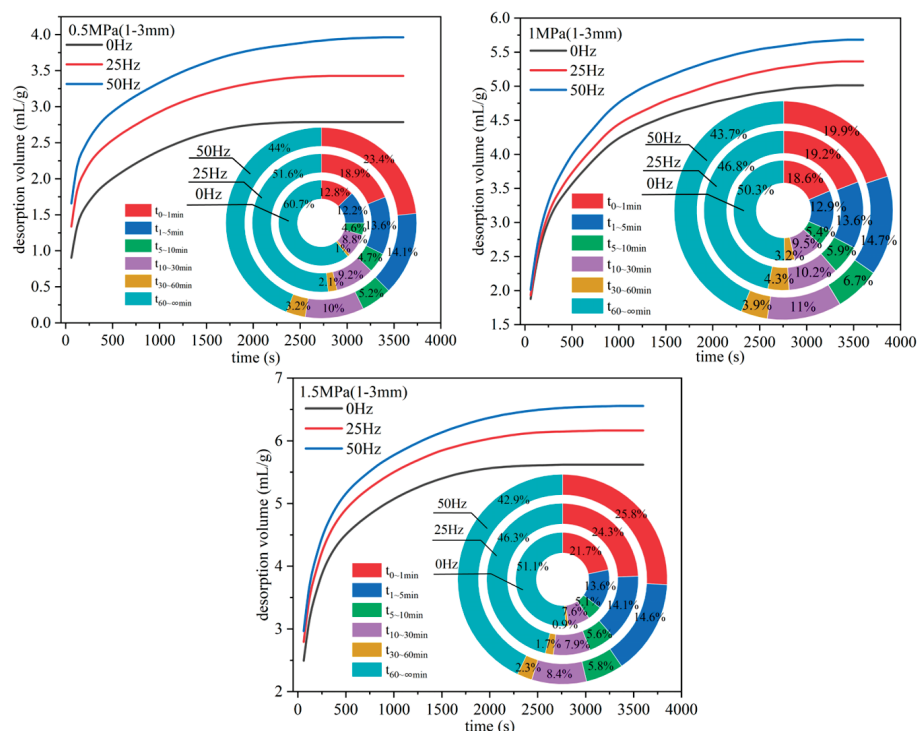


Figure 6. Characteristics of gas desorption volume and desorption rate in vibrated tectonic coal under varying adsorption equilibrium pressures.

The fitting results of the desorption rate for vibration-affected tectonic coal under different equilibrium pressures over 0~10 min are shown in Figure 7. Figure 7 illustrates that for coal samples subjected to the same vibration frequency, the initial desorption rate is observed to increase in conjunction with the adsorption equilibrium pressure. Conversely,

for coal samples subjected to the same equilibrium pressure, the initial desorption rate is observed to increase with an increase in vibration frequency.

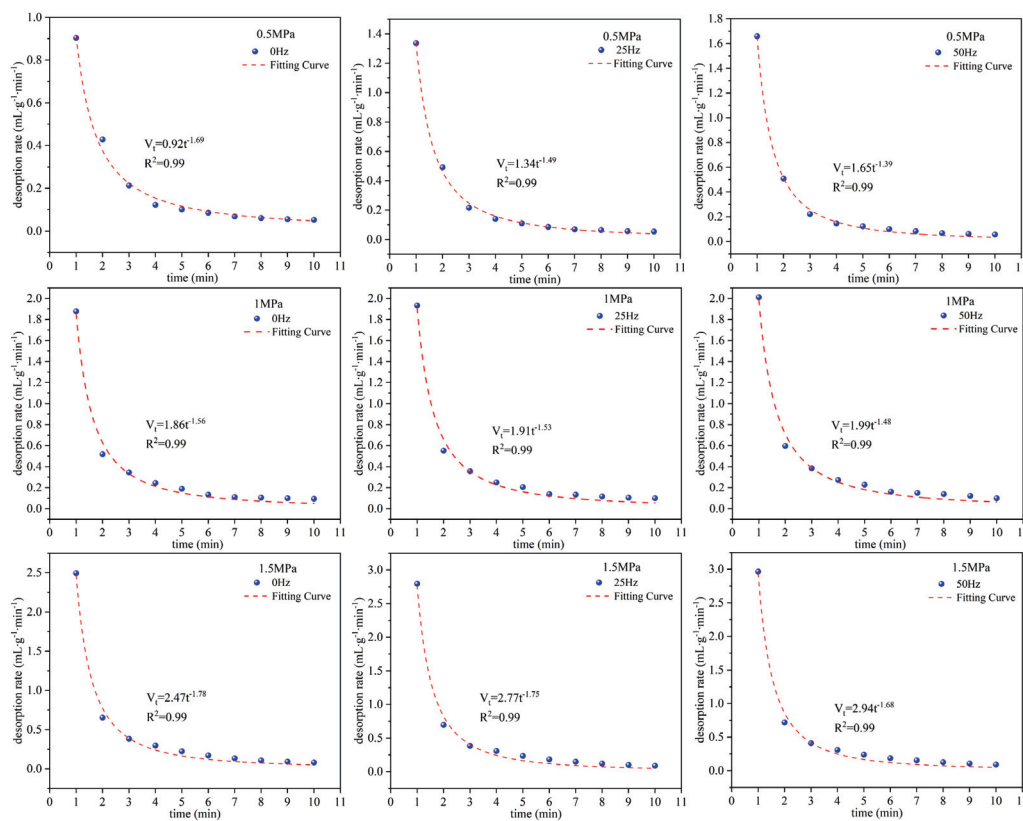


Figure 7. Fitting results and fitting formula of desorption rate in vibrated tectonic coal within 0~10 min under different equilibrium pressures.

Figure 8 shows the variation trends of initial desorption rate and desorption rate attenuation coefficient for vibrated coal samples under different adsorption equilibrium pressures. As evidenced in Figures 7 and 8, the initial desorption rate of coal samples at a consistent vibration frequency exhibits a positive correlation with increasing equilibrium pressure. For coal samples at 0.5 MPa, the initial desorption rate at 25 Hz and 50 Hz is 1.46 and 1.79 times that of 0 Hz, respectively; for coal samples at 1 MPa, the initial desorption rate at 25 Hz and 50 Hz is 1.03 and 1.07 times that of 0 Hz, respectively; for coal samples at 1.5 MPa, the initial desorption rate at 25 Hz and 50 Hz is 1.12 and 1.19 times that of 0 Hz, respectively. The initial desorption rates of all vibrated coal samples are higher than those of non-vibrated coal samples, and this increase is observed with increasing frequency. Furthermore, it was found that the higher the equilibrium pressure, the more significant the effect of vibration on the initial desorption rate. The desorption rate attenuation coefficient of vibrated coal samples decreases with increasing frequency; the lower the equilibrium pressure, the more significant the effect of vibration on the desorption rate attenuation coefficient. This is because the greater the adsorption equilibrium pressure, the higher the gas adsorption volume and concentration in the coal pores, which results in greater initial desorption kinetics and slower changes in desorption rate attenuation, thereby making the effect of vibration on gas diffusion kinetics more apparent.

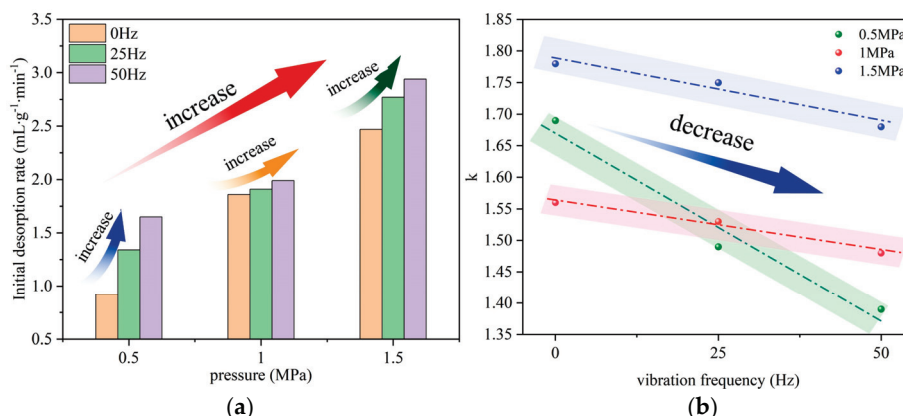


Figure 8. Variation trends in initial desorption rate V_0 and desorption rate attenuation coefficient k in vibrated tectonic coal under different adsorption equilibrium pressures. (a) Variation trends in the initial desorption rates in vibrated tectonic coal under different adsorption equilibrium pressures; (b) variation trends in desorption rate attenuation coefficient in vibrated tectonic coal under different adsorption equilibrium pressures.

In conclusion, the results of the experimental investigation into the gas desorption of coal samples with varying particle sizes and adsorption equilibrium pressures demonstrate that tectonic coal displays a pronounced capacity for gas desorption during the initial desorption phase. Vibration enhances the desorption process in tectonic coal, leading to an increase in both desorption rate and desorption volume, having a particularly significant impact on desorption capacity within the first 10 min after desorption begins. Therefore, tectonic coal that has been subjected to vibration is more prone to accumulate free gas in a relatively shorter period, which consequently increases the expansion energy of the coal mass.

3.2. Impact of Vibration on the Pore Structure of Tectonic Coals

The pore structure of coal functions as a storage and migration space for gas, and pore structure variations exert a significant influence on gas desorption and diffusion characteristics in coal. To further explore the mechanism by which vibration influences the initial desorption capacity of tectonic coal, this study analyses the control effect of pore structure changes in vibrated tectonic coal on initial gas desorption capacity from a pore structure perspective. Quantitative characterisation of the pore volume and specific surface area of macropores ($1000 < d < 10,000 \text{ nm}$), mesopores ($100 < d < 1000 \text{ nm}$), minipores ($10 < d < 100 \text{ nm}$), and micropores ($d < 10 \text{ nm}$) in vibration-affected coal by high pressure mercury intrusion porosimetry was conducted according to the B.B. Hotdot pore classification method [13,59]. The results of the test are presented in Figure 9.

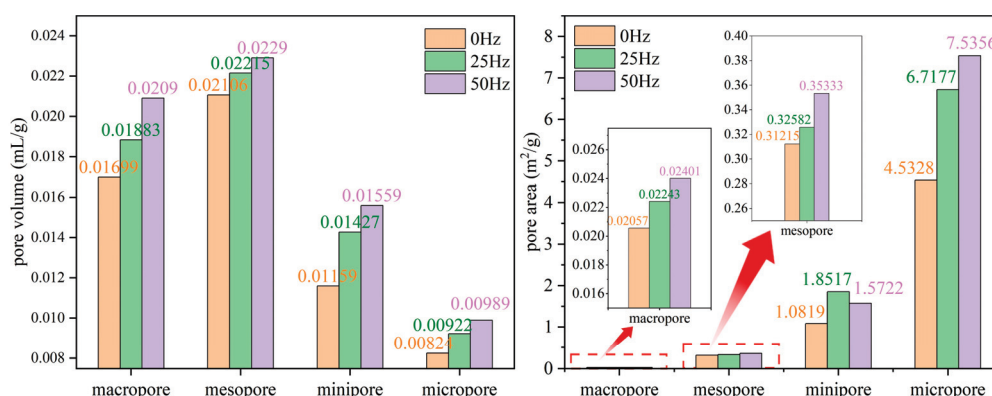


Figure 9. Pore volume and specific surface area of different pore sizes in vibrated tectonic coal.

Figure 9 shows that the pore volume of macropores and mesopores dominates in all coal samples, while the specific surface area of micropores is predominant. The pore volume and specific surface area of all pore size categories in vibration-affected coal samples are larger than those in non-vibrated coal samples, and both increase with increasing vibration frequency. This indicates that vibration expands the pore structure of tectonic coal, resulting in an increase in pore volume and specific surface area, which enlarges the storage and migration space for gas in coal and promotes gas desorption and diffusion.

4. Discussion

4.1. Control Effect of Pore Structure Changes on Gas Desorption

In order to elucidate the underlying mechanisms responsible for the alteration in gas desorption capacity in tectonic coal resulting from vibration, this paper presents a detailed analysis from the perspective of pore structure. Correlations were established between the pore structure results (pore volume and specific surface area) of the macropores, mesopores, minipores, and micropores and the desorption parameters of the coal samples ($Q_{0-10\min}$, $Q_{0-1\min}$, initial desorption rate V_0 , desorption rate attenuation coefficient k), with the fitting results displayed in Figure 10.

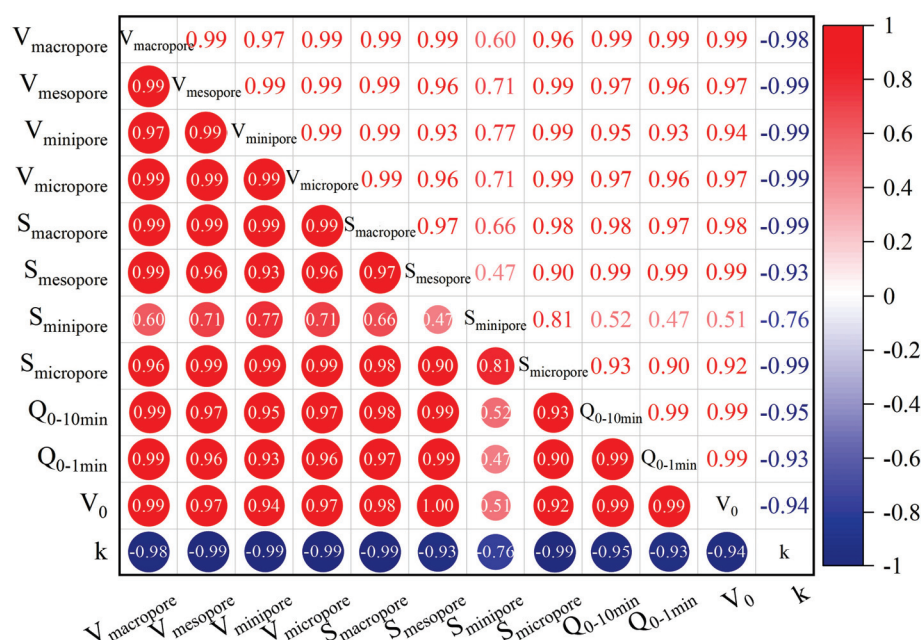


Figure 10. Results of correlation analysis between pore structure characteristic parameters and gas initial desorption capacity characteristic parameters.

The results presented in Figure 10 demonstrate a positive correlation between $Q_{0-10\min}$, $Q_{0-1\min}$, and the initial desorption rate V_0 with the pore volume and specific surface area of each pore size category. Conversely, the desorption rate attenuation coefficient k exhibits a negative correlation with pore volume and specific surface area. The pore volume and specific surface area of macropores and mesopores exert a more pronounced influence on $Q_{0-10\min}$, $Q_{0-1\min}$, and V_0 than those of minipores and micropores. Conversely, the pore volume and specific surface area of micropores and minipores exert a more pronounced influence on k . This phenomenon can be attributed to the preferential desorption of gases in macropores and mesopores to the environment during the initial stages of desorption. Additionally, the resistance to the transport of free gases in macropores and mesopores is lower than that in minipores and micropores. As the desorption time progresses, the gas adsorbed in the minipores and micropores gradually desorbs and diffuses into the environment. Since gas molecules must pass through complex pore channels to diffuse from micropores to the environment, the larger the pore volume and specific surface area

of micropores and minipores, the greater the connectivity of internal pores and the lower the complexity of pore channels, and the smaller the resistance to gas molecule diffusion, leading to a slower desorption rate attenuation. Vibration has been demonstrated to increase the pore volume and specific surface area of all pore size categories in tectonic coal. This enhances the initial desorption capacity and results in a higher initial desorption rate, which, in turn, leads to the accumulation of greater quantities of gas desorption.

4.2. Effect of Vibration on the Incubation and Triggering Stages of Coal and Gas Outbursts

4.2.1. The Energy Relationship in Coal and Gas Outbursts Process

The process of coal and gas outburst can be divided into four stages: inoculation, excitation, development, and termination [9,60,61]. Gas plays an important role in the whole process of outburst [9]. The energy conversion form in the process of outburst is mainly the elastic energy and gas expansion energy of coal into the crushing work and handling work of coal body, which follows the principle of energy conservation:

$$W_1 + W_2 = W_3 + W_4 + W_5 \quad (3)$$

where W_1 is the gas expansion energy, MJ; W_2 is the elastic energy of coal, MJ; W_3 is crushing work, MJ; W_4 is handling work, MJ; and W_5 is the residual kinetic energy of gas, MJ.

Zhao [13] and Wang [62] proposed that the crushing work is primarily derived from the elastic energy of the coal, while the expansion energy of the gas is mainly converted into the transportation work of the coal and the kinetic energy of the residual gas, as shown in:

$$W_1 = W_4 + W_5 \quad (4)$$

The expansion energy of the gas in coal can be calculated using the following equation:

$$W_1 = \frac{P_e V_0}{n-1} \left[\left(\frac{P_1}{P_e} \right)^{\frac{n-1}{n}} - 1 \right] \quad (5)$$

where P_e is the gas pressure in the roadway after the coal is ejected, MPa; V_0 is the volume of gas involved in performing the work, m^3 ; P_1 is the outburst gas pressure, in MPa; and n is the adiabatic coefficient, $n = 1.31$.

The gas involved in performing the work can be divided into two parts: free gas and desorbed gas during the outburst process. Thus, Equation (5) can be transformed into [62]:

$$W_1 = W_a + W_f = \frac{P_e}{n-1} (V_a + V_f) \left[\left(\frac{P_1}{P_e} \right)^{\frac{n-1}{n}} - 1 \right] \quad (6)$$

where V_a and V_f represent the volumes of adsorbed gas and free gas involved in the work, respectively, m^3 ; and W_a and W_f represent the energy contributed by the adsorbed gas and free gas, respectively, MJ.

In actual calculations, we can determine the original gas pressure and the volume of the original free gas, but it is difficult to determine the exact volume of gas involved in performing the work. Therefore, Equation (6) must be further transformed into:

$$W_1 = \frac{P_e}{n-1} (V'_a + V'_f) \left(\frac{P_1}{P_e} \right)^{\frac{1}{n}} \left[\left(\frac{P_1}{P_e} \right)^{\frac{n-1}{n}} - 1 \right] \quad (7)$$

where V'_a and V'_f represent the volumes of adsorbed gas and free gas before the outburst occurs, respectively.

The criterion for whether desorbed gas participates in the outburst work can be expressed as:

$$W_f < W_4 + W_5 \quad (8)$$

At this point, the adsorbed gas needs to desorb and convert into free gas to supplement the energy required. The energy that needs to be supplemented is:

$$W_{ad} = W_4 + W_5 - W_f \quad (9)$$

Based on the above analysis, when the expansion energy of the original free gas cannot meet the energy requirements of the outburst, the adsorbed gas must desorb to compensate. The rapid initial desorption of gas has been identified as a crucial factor in the genesis of coal and gas outbursts, with evidence suggesting that the energy imparted by the initial release of free gas is inadequate to satisfy the energetic requirements of outbursts [15,63]. Jin et al. [64] found that the rapid desorption of gas in pulverised coal provides 73.3% to 89.95% of the energy required for outbursts. Thus, it is evident that newly desorbed gas provides the primary energy for coal and gas outbursts.

4.2.2. Gas Expansion Energy of Vibration-Affected Tectonic Coal

The results presented in Section 3.1 of this study demonstrate that tectonic coal exhibits robust initial desorption capacity. Furthermore, it has been observed that tectonic coal accumulates greater quantities of free gas in a relatively brief span of time when subjected to vibration, resulting in an augmented expansion energy of the coal mass. Therefore, granular coal formed after tectonic coal breakage is more likely to experience a sudden increase in gas energy under the influence of vibration, and it is easier to stimulate coal and gas outbursts, which become more likely. To accurately determine the incremental gas expansion energy generated by free gas in tectonic coal under vibration, it is necessary to use desorption experimental data and calculate the corresponding gas expansion energy based on Equation (5), in order to quantitatively analyse the role of the initial gas desorption capacity of tectonic coal, influenced by vibration, in promoting the occurrence and development of coal and gas outbursts.

The statistical evidence derived from a multitude of documented instances of coal and gas outbursts indicates that the duration of such occurrences is typically less than one minute. In most cases, the outbursts last from a few seconds to several tens of seconds [8,42,47,60,65]. This research paper presents a calculation of the gas expansion energy in the process of coal and gas outbursts for a variety of coal samples of varying particle sizes and equilibrium pressures, with the influence of different vibration frequencies taken into account. The following assumptions were made [62]: (1) The duration of the coal and gas outbursts is assumed to be 0.5 min; (2) the gas desorbed in the initial 0.5 min of the desorption phase is assumed to be the sole gas involved in the outburst; (3) the environmental gas pressure is assumed to be atmospheric pressure, and the gas pressure prior to the coal and gas outbursts is assumed to be the adsorption equilibrium pressure. The results of the calculated gas expansion energy are presented in Figure 11.

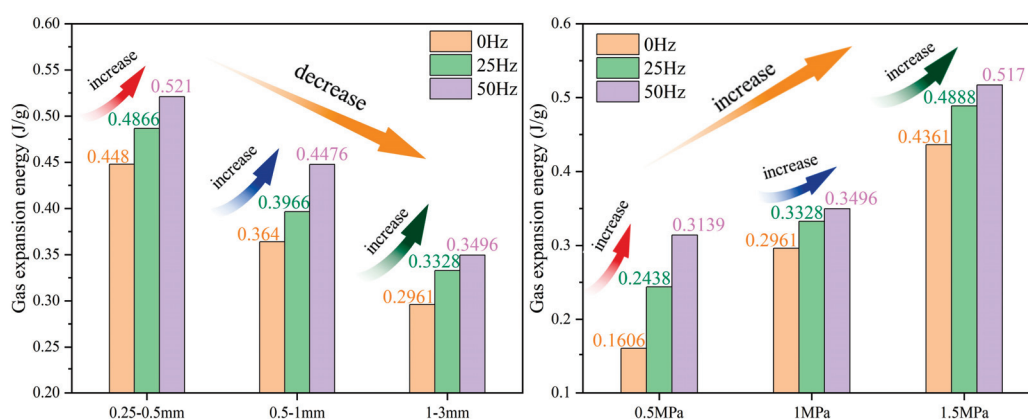


Figure 11. The variation trend of gas expansion energy in vibrated tectonic coal.

As illustrated in Figure 11, under the equilibrium pressure conditions of 1 MPa and consistent particle size, the gas expansion energy of coal samples with particle sizes of 0.25~0.5 mm, subjected to vibrations at 25 Hz and 50 Hz, increased from 0.4480 to 0.5210 J/g, representing an enhancement of 8.62% to 16.29%, respectively, in comparison to the 0 Hz samples. The gas expansion energy of coal samples with particle sizes of 0.5~1 mm exhibited an increase from 0.3640 J/g to 0.4476 J/g, representing an 8.96% to 22.97% rise. In the case of coal samples with particle sizes of 1~3 mm, the gas expansion energy increased from 0.2961 J/g to 0.3496 J/g under similar conditions, reflecting an increase of 12.39% to 18.07%. Additionally, for coal samples measuring between 1 and 3 mm in size, when subjected to an equilibrium adsorption pressure of 0.5 MPa, there was a considerable increase in gas expansion energy. From 0.1606 J/g to 0.3139 J/g, a 51.81% to 95.45% rise was observed. The gas expansion energy exhibited a comparable increase from 0.4361 J/g to 0.5170 J/g at the adsorption equilibrium pressure of 1.5 MPa. This corresponds to a notable 12.08% to 18.55% enhancement, respectively.

The above results indicate that, regardless of particle size and equilibrium pressure during adsorption, the energy of gas expansion rises with increasing frequency. After the coal mass is broken into granular coal during the incubation and development stages of an outburst, vibration significantly increases the energy of free gas expansion. This indicates that tectonic coal, which has high gas pressure and is susceptible to fracturing and pulverisation, can generate greater gas expansion energy within a shorter timeframe when subjected to mechanical vibration, thereby facilitating the fulfilment of the outburst triggering criteria and consequently inducing an outburst. Furthermore, elevated gas expansion energy will facilitate enhanced transportation of the pulverised coal throughout the outburst development process, resulting in a proportional increase in outburst intensity and severity. This provides a partial explanation for why mechanical vibration represents a significant external factor contributing to the occurrence of coal and gas outbursts.

4.3. The Role of Vibration in Triggering Coal and Gas Outbursts in Tectonic Coal Seams

Tectonic coal has lower mechanical strength than primary coal and is more susceptible to pulverisation under external forces. During the incubation stage of outbursts, the original tectonic coal is subjected to ground stress, leading to compressive deformation, shear, and fracturing, resulting in the formation of a fractured coal mass region. The original structure is damaged, and initial gas desorption occurs. When the fractured tectonic coal deposits in this region are subjected to mechanical vibration, they continue to undergo damage and breakage due to the combined effects of ground stress and vibration stress. This process leads to the formation of smaller coal particles. Concurrently, mechanical vibration facilitates the desorption of gas from these coal particles, resulting in a rapid increase and accumulation of free gas, thereby providing augmented gas expansion energy to precipitate coal and gas outbursts. As the amount of desorbed gas increases, the gas expansion energy gradually rises. When the gas expansion energy reaches the energy threshold required to transport the coal mass and the particle size of the broken coal mass decreases to the critical average size for outbursts, the crushed coal mass rapidly loses its stability under the external disturbance caused by mechanical vibration. The significant amounts of free gas can form a high-rate gas flow that carries the fractured and pulverised coal mass, completing the ejection work and triggering a coal and gas outburst. Therefore, mechanical vibration plays a role in both inducing and triggering coal and gas outbursts, as shown in Figure 12.

In this paper, only one kind of tectonic coal is used, and the difference between different tectonic briquettes (cataclastic coal, granulitic coal, mylonitic coal) is not considered. The evolution characteristics of pore structure and the variation characteristics of adsorption and desorption capacity of different types of tectonic coal under vibration should be further studied. Then, using fracture mechanics, gas adsorption theory, surface physical chemistry and other theories, the mechanism of the change of pore structure, adsorption and desorption capacity and diffusion capacity of tectonic coal caused by vibration will be

deeply explored and clarified. Based on the above research results, the theory and method of increasing the output of coalbed methane by means of vibration excitation are sought to promote the efficient extraction of coalbed methane.

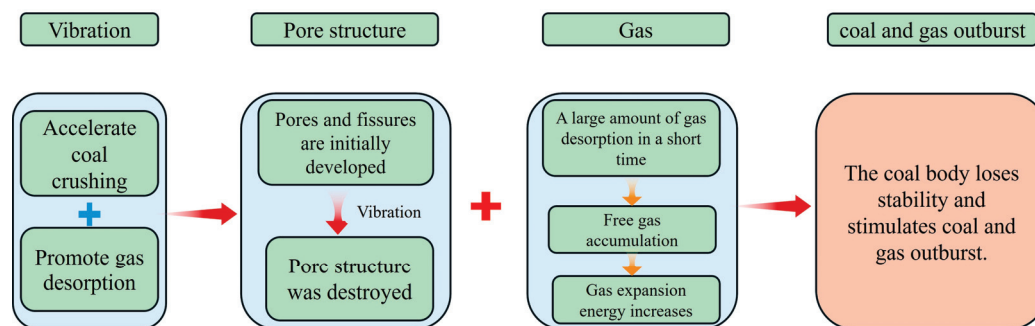


Figure 12. The process of coal and gas outburst induced and stimulated by vibration.

5. Conclusions

- (1) The desorption experiments on tectonic coal under vibration show that tectonic coal has a high initial desorption capacity, with 29.58~54.51% of the ultimate desorption volume being desorbed in the first 10 min. Additionally, the smaller the particle size and the greater the adsorption equilibrium pressure, the higher the desorption ratios in the first 10 min. Regardless of particle size and adsorption equilibrium pressure, vibration at frequencies of 0~50 Hz led to increased gas desorption rate and volume as the frequency increased, indicating that vibration promotes gas desorption in tectonic coal.
- (2) The initial desorption rate of tectonic coal increases as the particle size decreases and adsorption equilibrium pressure increases. For coal samples of different particle sizes and adsorption equilibrium pressures, the initial desorption rate after vibration was 1.01~1.2 times and 1.03~1.79 times that of the non-vibrated samples, respectively. Furthermore, the initial desorption rate demonstrated a positive correlation with vibration frequency. Furthermore, vibration also resulted in a delay in the attenuation of the desorption rate, as evidenced by a decrease in the desorption rate attenuation coefficient with increasing vibration frequency. Vibration has been demonstrated to enhance the initial desorption capacity of tectonic coal, thereby facilitating the rapid desorption of substantial quantities of free gas within a relatively brief period following the onset of desorption.
- (3) The results of the high-pressure mercury intrusion experiments on vibration-affected tectonic coal demonstrate that vibration causes pore expansion in tectonic coal. This leads to an increase in pore volume and specific surface area, which subsequently affects the initial gas desorption rate and desorption rate attenuation coefficient. The desorption rate in the first minute, the desorption rate within 10 min, and the initial desorption rate are positively correlated with the pore volume and specific surface area of each pore size in coal, with the influence of macropores and mesopores being greater than that of minipores and micropores. The desorption rate attenuation coefficient k is negatively correlated with pore volume and specific surface area, with minipores and micropores having a greater influence than macropores and mesopores.
- (4) Vibration causes the free gas expansion energy of tectonic coal to increase with vibration frequency. During the incubation and development stages of an outburst, when the tectonic coal mass is broken into granular coal and subjected to vibration, it may more easily reach the critical energy needed to trigger an outburst due to the generation of higher gas expansion energy within a short period. In the incubation and triggering process of coal and gas outbursts, vibration accelerates coal mass fragmentation and destabilisation to a certain extent, promotes the rapid desorption of gas from the coal mass, and leads to the accumulation of more gas expansion

energy, thus causing the gas expansion energy of the broken tectonic coal mass to reach the energy threshold required for transporting the coal mass, thereby inducing and triggering coal and gas outbursts.

Author Contributions: Conceptualisation, M.S.; methodology, M.S.; software, M.S. and W.W.; validation, M.S.; formal analysis, L.S.; investigation, C.Z.; resources, H.Z.; data curation, M.S.; writing—original draft preparation, M.S.; writing—review and editing, M.S. and H.Z.; visualisation, L.S.; supervision, Z.H.; project administration, Z.H.; funding acquisition, H.Z. All authors have read and agreed to the published version of the manuscript.

Funding: This study was supported by the National Natural Science Foundation of China (52304224); the Special Project for Technology Innovation and Entrepreneurship Foundation of Tiandi Science & Technology Co., Ltd. (2021-2-TD-MS001); and the China Coal Research Institute Technology Innovation Fund (2022CX-I-05).

Data Availability Statement: The data presented in this study are available on request from the corresponding author. The data are not publicly available due to containing information that could compromise the privacy of the research participants.

Conflicts of Interest: The authors declare no conflicts of interest.

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Article

Combined Effect of Multiple Slotting Spatial Parameters on Gas Extraction Efficiency: Numerical Analysis and Field Verification

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Abstract: Hydraulic slotting is an effective technology that enhances gas extraction operations and prevents gas disasters in coal mines. Slotting parameters and spatial arrangements substantially affect permeability enhancements. The pressure-release range and effective extraction area under different slotting spatial parameters were obtained by constructing a hydraulic slotting pressure-release permeability and three-dimensional (3D) slotting numerical models. These models quantitatively characterized the influence rules of multiple slotting spatial arrangement parameters on the extraction efficiency at a 3D scale, clarified the interactions of multiple slottings and their combined effects on pressure relief and permeability enhancement, and verified the results using field engineering tests. The results showed that hydraulic slotting significantly alters local stress and strain distributions, creating high-strain and high-stress zones with clear spatial attenuation. The process enhances fracture development, reducing gas pressure from 1 MPa to 0.08 MPa, thereby improving extraction efficiency. Enlarging the slot dimensions from 1.5 to 2.5 m increases the gas pressure-relief efficiency by up to 41% and nearly triples the impact radius. Wider slot spacing (1.5 m to 3.5 m) and additional slots (from one to three) further reduce the borehole gas pressure by 23% to 25%, optimizing hydraulic slotting technology for practical applications.

Keywords: gas extraction; hydraulic slotting; pressure relief; permeability enhancement; numerical simulation

1. Introduction

Coal seam gas (CSG) plays a crucial role in the global energy mix, especially in countries like the United States, Australia, and China. In the United States, while CSG held significance in the past, its importance has diminished in recent years with the rapid rise of shale gas. However, coal still remains an essential source for electricity generation, and managing methane in coal mines remains a key safety challenge. Australia, on the other hand, is one of the world's major producers of coal seam gas, particularly in the Bowen and Surat basins in Queensland. CSG not only supports the domestic energy market but also contributes significantly to liquefied natural gas (LNG) exports. In contrast, China's coal seam gas industry is rapidly expanding. In 2023, China produced 394.5 billion cubic meters of natural gas, with 43% coming from unconventional resources such as CSG (Figure 1). Despite the rapid growth, China's extraction technology and efficiency still lag behind Australia [1]. The extraction of coal seam gas relies on advanced extraction technologies, especially in China, where poor coal seam permeability complicates the process. During deep mining operations, rising gas pressure and decreasing permeability in high-gas, low-permeability coal seams increase the risks of gas-related disasters, particularly methane explosions and aerological hazards. If not properly managed, these risks can lead to severe safety incidents and environmental issues. Therefore, safely and efficiently extracting coal seam gas and mitigating these risks is a major challenge for the global coal mining industry [2–4].

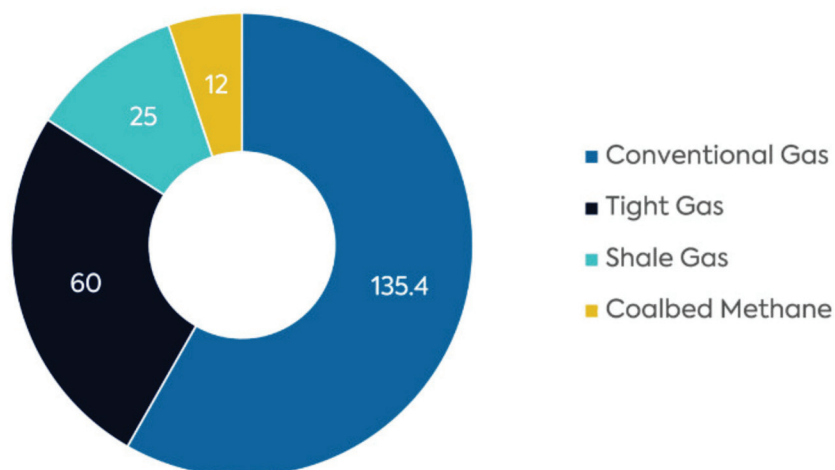


Figure 1. China gas production 2023 (Total 232.4 bcm).

Currently, improving gas extraction efficiency mainly requires permeability enhancement by using pressure-relief measures, promoting the generation and expansion of coal seam fractures, and strongly promoting coal seam gas flow to drilling holes. The most widely used measures for hydraulic permeability enhancement include hydraulic fracturing, punching, and slotting [5–7]. Extensive research has been conducted recently by using experiments, data analysis, and numerical simulations. Hydraulic fracturing involves injecting high-pressure water to fracture a coal body, promoting fracture development, increasing coal body permeability, and improving gas extraction efficiency [8–13]. Yang et al. [14] investigated the application of large-scale hydraulic fracturing technology in deep coal seam gas horizontal wells, focusing on optimizing resource extraction and enhancing gas production. Hydraulic punching uses high-pressure water jets to destroy the coal body around the borehole wall, inducing small outbursts in the borehole and using water flow to create space, allowing stress in the coal body to be released and permeability to be increased. This approach effectively mitigates poor coal seam permeability and difficult gas extraction [15–19]. Yang et al. [20] proposed the use of hydraulic jet drilling, controlled shock waves, and nitrogen injection to increase coal seam gas production. Field tests and numerical simulation results showed significant increases in gas extraction volumes. Further, Pan et al. [21] used scanning electron microscopy (SEM), low-noise amplification (LNA), and maximum intensity projection (MIP) techniques to analyze the pore structure of coal seam gas reservoirs post-hydraulic fracturing. Their study showed substantial increases in mesopore count, pore volume, and specific surface area compared with those in the pre-hydraulic fracturing measurements.

Hydraulic fracturing has a stronger permeability enhancement efficacy than the above-mentioned methods but is complex and has a long construction period. Hydraulic perforation must be conducted after the completion of drilling, resulting in low efficiency and limited permeability enhancement. Hydraulic slotting has developed rapidly as an emerging permeability enhancement technology because of its simple operation and high permeability enhancement efficacy. The principle of this technology is to use ultra-high-pressure water jets to cut the coal body internally, forming a flat slot. Around this slot, fractures are generated and rapidly propagate outward, creating a pressure-relief zone and significantly enhancing the coal body's permeability [22–25]. Research has recently focused on the physical and mechanical mechanisms of slotting pressure relief and permeability enhancement. Liu et al. [26] performed uniaxial compression numerical simulations of coal bodies with single fractures at varying angles and multiple fractures with different spatial arrangement patterns using the particle flow code (PFC2D) method. In their study, they focused on the physical mechanisms through which the widespread formation and interconnection of microfractures around hydraulic slots enhance coal seam pressure relief and permeability. Their findings led them to introduce two indicators for assessing the

efficiency of pressure relief and permeability improvement in coal seams. Based on the simulation results, the optimal spatial distribution pattern of slots that favors coal seam pressure relief and permeability enhancement was determined. Chen et al. [27] found that directional wells with multiple hydraulic slots improve gas recovery in deep coal seams by optimizing the dimensions of the slots. Hydraulic slotting enhances gas extraction efficiency over traditional hydraulic fracturing by generating new fractures within the borehole through ultra-high-pressure water jets, without compromising the integrity of the coal seam roof and floor rock formations [4,28,29].

Slotting parameters and arrangement methods significantly impact the extraction efficiency rate [30,31]. Studies have also been conducted on different hydraulic slotting construction parameters using numerical simulations and field verification. Hao et al. [32] used FLAC3D-COMSOL sequential coupling numerical solving methods to numerically analyze the three-dimensional (3D) pressure-relief gas flow field of drilling and slotting coal bodies with different slot radii, slot thicknesses, burial depths, and slotting methods. Zhanjin et al. [33] proposed a precise control technology for ultra-high-pressure hydraulic slotting by analyzing the slot width and depth control, slot coal-fall speed control, and slot area effect control; developing a ZGF-100(A) ultra-high-pressure hydraulic slotting device; and conducting precise control slotting field application tests.

However, in the abovementioned studies, only the mechanical characteristics and slotting-related permeability enhancement mechanisms under various arrangement parameters based on a two-dimensional scale were analyzed. Thus, the results of these studies do not truly reflect the characteristics of three-dimensional slotting spatial parameters. To cover this research gap, in this study, a hydraulic slotting pressure-release permeability model and a three-dimensional slotting numerical model were used to obtain the pressure-release range and effective extraction area for coal seam gas under varying slotting spatial parameters. Using the proposed models, it was possible to quantitatively characterize the effects of multiple slotting spatial arrangement parameters on the extraction efficiency based on an actual three-dimensional scale, clarify the interactions between multiple slottings and their combined effects on pressure relief and permeability enhancement, and verify the results via field engineering tests. Thus, the main goal of our study was to provide a theoretical reference for optimizing hydraulic slotting arrangement parameters and ensuring efficient gas extraction and disaster prevention.

2. Hydraulic Slotting Technology

2.1. Hydraulic Slotting Model Principle

The hydraulic slotting technique involves the use of ultra-high-pressure water jets to cut the coal seam without damaging the lithology of the roof and floor. This technique increases the exposed coal surface area, effectively reduces the pressure, improves the stress conditions in the coal seam, and promotes fracture formation within the borehole. These actions enhance permeability and accelerate gas release. During this process, the number of fractures around the slotting hole substantially increases and connects with the original fractures, thereby improving the speed of gas desorption and release, considerably increasing the gas extraction efficiency. Compared with traditional drilling, hydraulic slotting drilling creates additional channels for gas desorption, reduces the gas content in the coal seam, and ensures outburst elimination (Figure 2).

2.2. Hydraulic Slotting Coal Body Permeability and Fluid–Solid Coupling Model

After hydraulic slotting, the coal surrounding the borehole becomes fractured, causing most of the gas to dissipate, leading to a significantly reduced gas pressure. Therefore, the gas flow in this part of the coal body is considered a constant. The initial porosity of the coal body is influenced by structural changes, and the variation in coal body porosity is expressed as follows:

$$q = 1 - \frac{(1 - q_0)}{1 + \varepsilon_v} \left(1 - \frac{\Delta p}{K_s} \right) \quad (1)$$

where q_0 is the original coal body porosity, ε_v is the coal body volume strain, Δp is the change in gas pressure (MPa), and K_s is the coal body structure volume modulus (MPa).

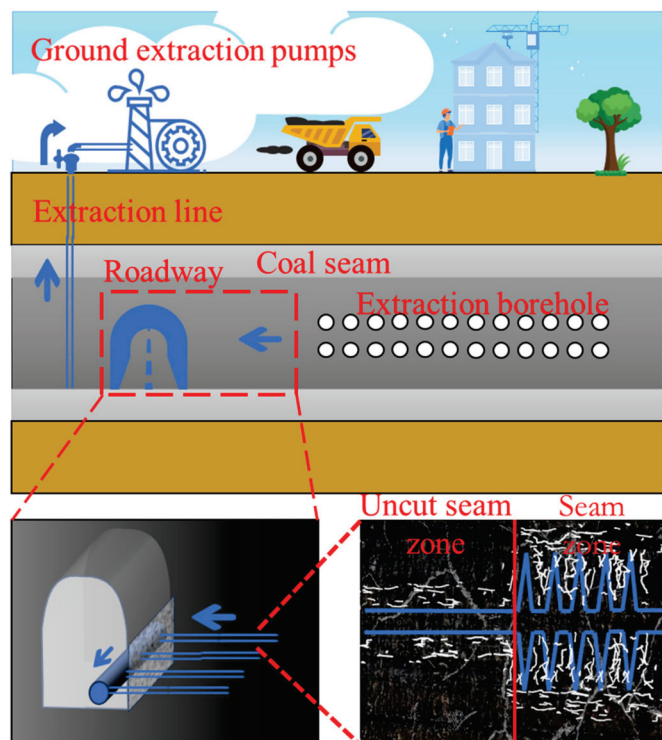


Figure 2. Schematic diagram of the hydraulic slotting principle.

The permeability of the fractured coal body can be obtained through a combination with the Kozeny–Carman equation.

$$k = \frac{k_0}{1 + \varepsilon_v} \left[1 + \frac{\varepsilon_v}{q_0} + \frac{\Delta p(1 - q_0)}{q_0 K_p} \right]^3 \quad (2)$$

where k_0 is the initial permeability of the coal body (m^2) and K_p is the solid skeleton volume modulus (MPa).

To develop a fluid–solid coupling model for gas extraction in hydraulically slotted coal seams, the coal body is assumed to be a porous, homogeneous, and isotropic medium. Additionally, we assumed that gas flow in both intact and slotted coal adheres to Darcy’s law, behaves as an ideal gas, and remains unaffected by temperature variation.

The mass conservation equation for gas flow in the coal seam is expressed as follows:

$$\frac{\partial M}{\partial t} + \nabla(\rho v) = Q_m \quad (3)$$

where M represents the gas content of the coal (kg/m^3), t denotes the time (s), ρ represents the density of the gas within the coal (kg/m^3), v represents the gas flow velocity (m/s), and Q_m is the gas source term ($\text{kg}/(\text{m}^3 \cdot \text{s})$).

The coal’s gas content was divided into adsorbed gas M_1 and free gas M_2 , represented as follows:

$$M_1 = \frac{abP}{1 + bP} \frac{100 - A - W}{100} \frac{1}{1 + 0.31W} \quad (4)$$

$$M_2 = \delta P \varphi \quad (5)$$

where a is the maximum adsorption capacity of the coal body per unit mass (m^3/kg), b is the adsorption constant of coal (MPa^{-1}), P is the gas pressure in the coal seam (MPa), A is

the ash content of coal (%), W is the moisture content of coal (%), δ is the gas compressibility coefficient, $\delta = m_g/(RT)$, and φ is the porosity of the coal body.

The gas density in the coal body can be expressed as follows:

$$\rho = \frac{m_g P}{RT} \quad (6)$$

Here, m_g represents the molecular weight of the gas (16 g/mol), R denotes the ideal gas constant (8.314 J/(mol·K)), and T represents the thermodynamic temperature (K).

According to Darcy's law, the gas flow velocity near the borehole can be described by Equation (7) as follows:

$$v = -\frac{k}{\mu} \nabla P \quad (7)$$

By integrating Equations (3)–(7), the gas migration equation in coal seam fractures is derived as follows [34,35]:

$$\left[\frac{m_g \varphi}{RT} + \frac{ab(100 - A - W)}{(1 + bP)^2 100(1 + 0.31W)} \right] \frac{\partial P}{\partial t} - \frac{km_g}{\mu RT} \nabla(P \nabla P) = 0 \quad (8)$$

The cross-coupled relationships between the aforementioned equations are illustrated in Figure 3.

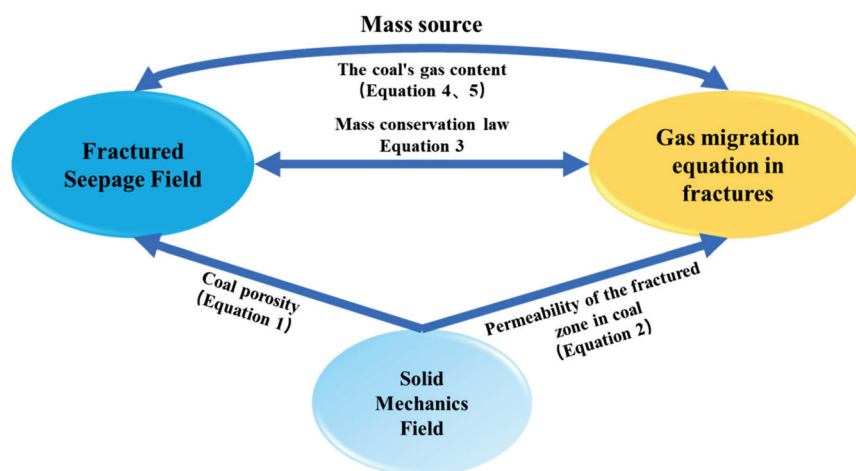


Figure 3. Numerical simulation and theoretical formula logic diagram.

3. Numerical Simulation Study of Hydraulic Slotting

3.1. Model Establishment

In this study, COMSOL Multiphysics (6.0) was chosen for the numerical simulation due to several key advantages compared to similar software tools. Firstly, COMSOL's ability to couple multiple physical processes, such as fluid flow, structural mechanics, and heat transfer, allows for a more comprehensive simulation of the complex conditions encountered in hydraulic fracturing studies. Additionally, COMSOL provides highly flexible and customizable modeling capabilities, enabling researchers to tailor simulations according to specific experimental conditions. Its powerful finite element analysis (FEA) solver handles complex geometries and boundary conditions efficiently, which is critical for accurately simulating the fractured coal body and gas flow processes. Furthermore, COMSOL's post-processing tools allow for detailed analysis and visualization of the results, making it easier to interpret and validate the outcomes. These advantages make COMSOL an ideal choice for conducting simulations in studies related to hydraulic fracturing and gas extraction.

Utilizing the geological conditions of the Wangxingzhuang Coal Mine as a case study, the COMSOL numerical simulation software was employed to develop a calculation model

for the gas pressure field in hydraulic slotting boreholes. A simplified 3D geometric model with dimensions of $10 \times 10 \times 10$ m was created based on the conditions of the 11,131 working face. The model was then refined using adaptive meshing. The borehole was located at the center of the 3D model with a designed extraction borehole radius of 50 mm and length of 10 m. The slot was randomly generated using COMSOL numerical simulation software with an initial condition of a random slot with an axis length of 1.5 m. Figure 4 illustrates the numerical simulation results of hydraulic slotting.

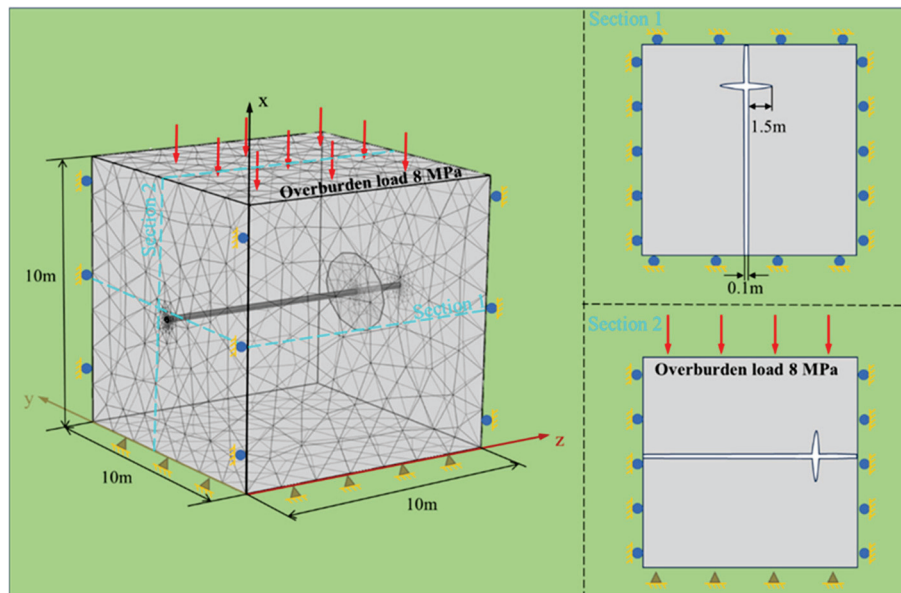


Figure 4. Schematic diagram of the numerical simulation of hydraulic slotting.

The upper boundary of the geometric model was subjected to a uniform load of 8 MPa, representing the overburden rock strata. The lower boundary was fixed, while the surrounding boundaries were constrained using roller supports. In this dual-porosity model, we treat the slot as a macroscopic fracture, and both the coal matrix and the post-slotting fractured zone are assumed to be porous, homogeneous, and isotropic media. Dirichlet boundary conditions were applied to these settings, ensuring stability in the simulation. The boundaries of the borehole and slot were left stress-free to simulate direct contact with air. The specific parameters are listed in Table 1.

Table 1. Numerical simulation parameters for hydraulic slotting.

Parameter	Value	Parameter	Value
Initial gas pressure of coal seam (MPa)	1	Coal seam temperature (K)	303.15
Adsorption constant a ($\text{m}^3 \cdot \text{kg}^{-1}$)	24	Initial permeability of coal body (m^2)	9.37×10^{-17}
Adsorption constant b	1	Dynamic viscosity of gas (Pa·s)	1.08×10^{-5}
Ash content of coal (%)	14.34	Elastic modulus of coal seam (GPa)	2.84
Moisture content of coal (%)	0.9	Internal friction angle of coal seam ($^\circ$)	18
Molecular weight of gas ($\text{g} \cdot \text{mol}^{-1}$)	16	Density of coal seam ($\text{kg} \cdot \text{m}^{-3}$)	1450
Gas constant ($\text{J} \cdot \text{mol}^{-1} \cdot \text{K}^{-1}$)	8.314	Gas density ($\text{kg} \cdot \text{m}^{-3}$)	0.7
Matrix porosity	0.06	Poisson's ratio of rock	0.29
Initial fracture rate	0.012	Poisson's ratio of coal seam	0.37

3.2. Stress Changes in Hydraulic Slotting Coal Body

To further investigate the effects of hydraulic slotting on coal body strain, the COMSOL numerical simulation software was employed to model the stress and equivalent plastic strain distributions post-slotting (Figure 5).

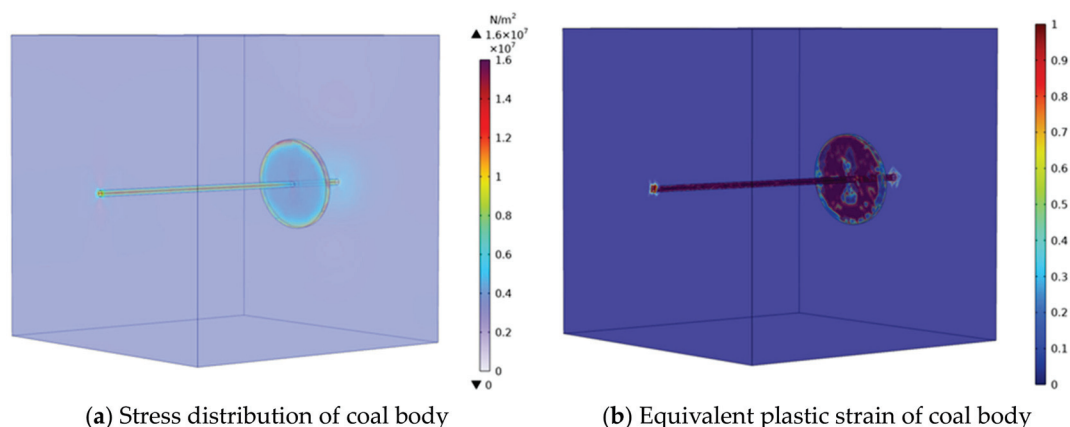


Figure 5. Stress distribution and equivalent plastic strain of the coal body. (a) shows the stress distribution after hydraulic slotting, where the color changes indicate different stress values at various positions. (b) shows the distribution of the equivalent plastic strain. The color change from blue to red indicates the distribution from low to high strains.

The results in Figure 5a show that the stress in the area around the slot significantly increased, with the maximum stress value reaching $1.6 \times 10^7 \text{ N/m}^2$. This outcome was caused by the impact and force exerted by the high-pressure water jets on the coal body during hydraulic slotting. The stress concentration area was primarily distributed in the disk-shaped area on both sides of the slot, which was consistent with the location of the high-strain zone. Simultaneously, the stress value gradually decreased with the increasing distance from the slot center, showing a clear gradient distribution. This observation suggested that the stress impact zone was closely associated with the strain impact area and was primarily localized around the slot. However, owing to the boundary conditions, the stress levels at the edges of the model were reduced, and the coal mass located further from the slot showed minimal effects owing to the impact of the hydraulic slotting.

A distinct high-strain zone formed around the slot depicted in Figure 5b, proving that the coal body in this area underwent significant plastic deformation. The high-strain zone exhibited a disk shape, which is consistent with the shape and position of the slot. With the increasing distance from the slot center, the strain gradually decreased, showing a clear spatial attenuation. This result indicates that the impact of hydraulic slotting was mainly concentrated in a limited range around the slot, with areas far from the slot being less affected than those close to the slot. Owing to the intense impact of the high-pressure water jets during hydraulic slotting, the strain was notably high at the slot edges and borehole ends, causing significant local deformation in these areas of the coal body.

3.3. Changes in Gas Pressure in Hydraulic Slotting

To shorten the calculation time of the numerical simulation, gas pressure changes were simulated after 10, 50, and 100 d of gas extraction and combined with actual field situations. Figure 6 illustrates the gas pressure distribution around the extraction borehole after 10, 50, and 100 days, respectively. The results show a clear trend of decreasing gas pressure with time, particularly in the slotted areas.

After 10 days (Figure 6a), the pressure drop is mainly localized near the slotted fractures, reducing from 1 MPa with a significant gradient near the borehole. Beyond this, the pressure remains largely unchanged, indicating limited extraction impact at this stage. By 50 days (Figure 6b), the pressure drop expands further into the coal seam, with pressure in the slotted areas approaching 0.08 MPa. The broader pressure-relief zone reflects the increased efficiency of hydraulic slotting in facilitating gas flow compared to unslotted areas, where higher pressure persists. After 100 days (Figure 6c), the pressure stabilizes around 0.08 MPa in slotted zones, confirming improved permeability and sustained gas extraction efficiency. The pressure difference between the slotted and unslotted areas becomes more

pronounced, with unslotted regions maintaining higher pressure, highlighting hydraulic slotting's critical role in enhancing gas migration and extraction efficiency over time.

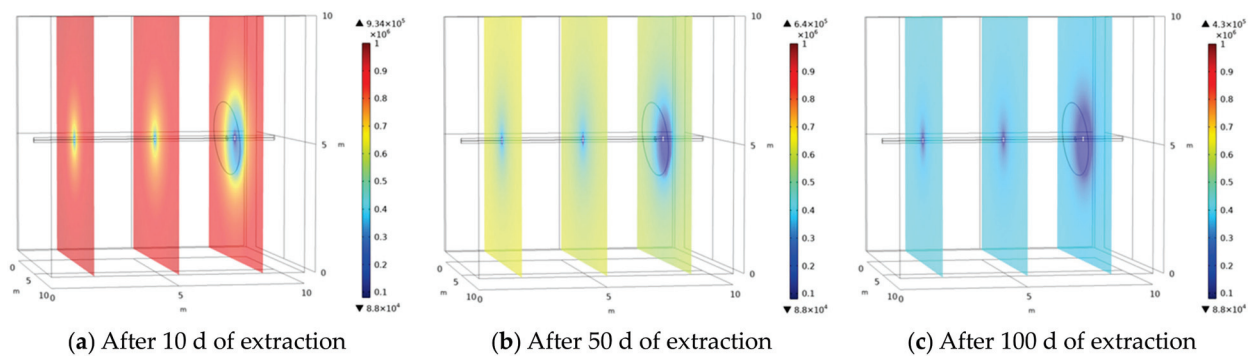


Figure 6. Gas pressure contour maps at different extraction times.

Furthermore, the pressure-relief range of hydraulic slotting and the slotting direction were clearly correlated. The gas pressure-relief range expanded along the slotting direction, ensuring a broader gas pressure-relief area (Figure 7). The high-pressure water flow within the slot dynamically affected the surrounding coal body, breaking the original coal structure and forming new permeable channels along certain paths. This event increased the permeability of the coal body and effectively mitigated the gas pressure.

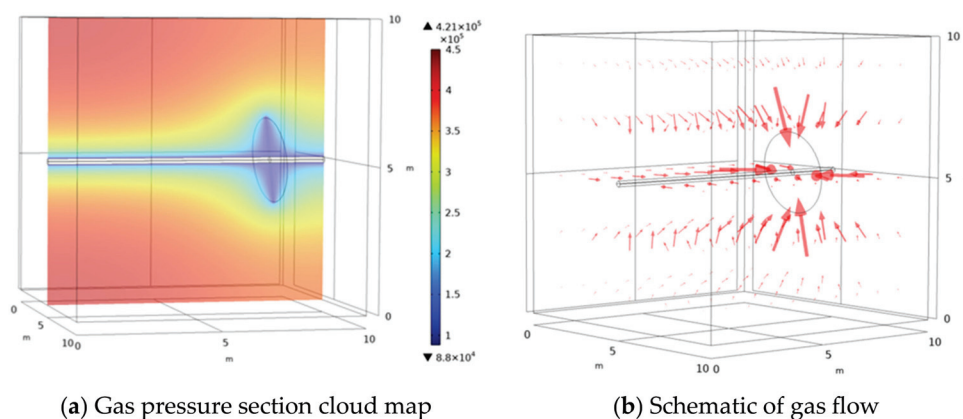


Figure 7. Gas pressure cross-sectional contour and flow schematic after hydraulic slotting.

Additionally, to thoroughly investigate variations in the gas pressure during extraction following hydraulic slotting, a detection line was set along the axial direction of the extraction borehole. Three detection lines were placed at 2.5, 5, and 7.5 m perpendicular to the borehole direction to monitor gas pressure changes along the axial and perpendicular directions of the extraction borehole at various extraction times. The layout of these detection lines is depicted in Figure 8.

Figure 9 presents the variation curves of the gas pressure along different monitoring lines over various extraction periods. As observed from the axial monitoring line (Figure 9a), the gas pressure in the slotted area decreases significantly at different extraction times. After 100 days of extraction, the gas pressure in the slotted area decreased by approximately 90% compared to the original coal seam pressure, reaching a minimum of around 0.1 MPa. In contrast, the gas pressure in the unslotted area only decreased by approximately 61%, with the pressure dropping to approximately 0.39 MPa. In the radial direction (Figure 9b), the gas pressure around the extraction borehole continued to decrease as the extraction period extended. However, in addition to the gas pressure in the slotted coal body decreasing significantly, the pressure-relief range in the slotted area was also approximately 2 m

larger than in the unslotted area, indicating that the effect of hydraulic slotting and gas management was particularly significant.

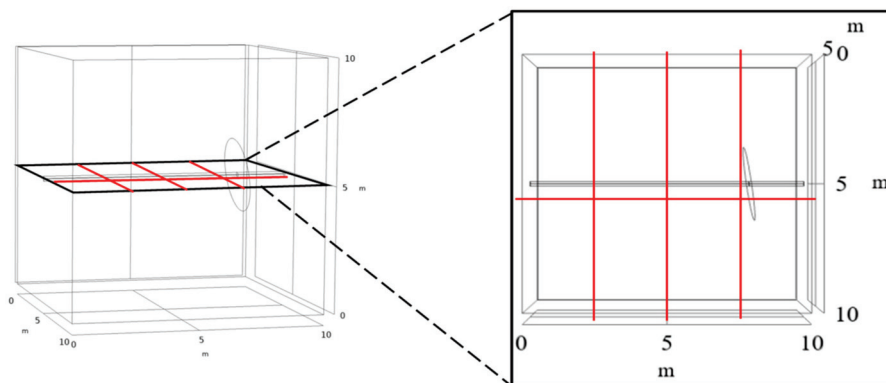


Figure 8. Schematic diagram of the detection line layout.

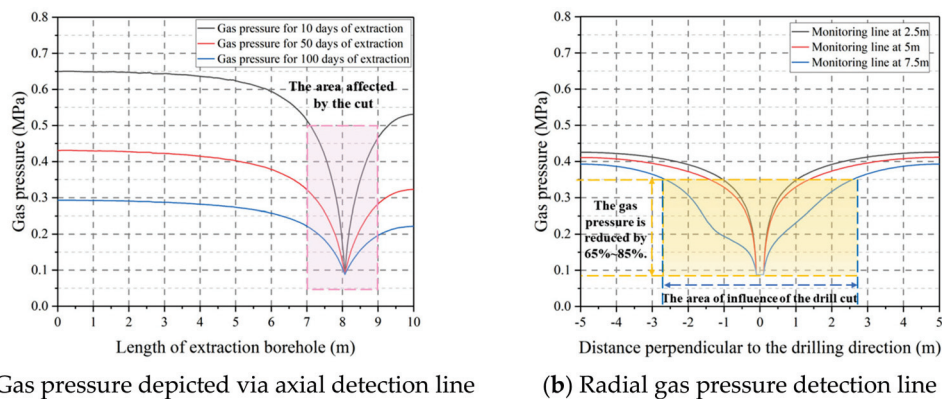


Figure 9. Gas pressure curves at detection lines.

4. Study on Different Hydraulic Slotting Construction Parameters

Based on the coal mine extraction experience, a single-borehole single slot has a limited impact range for gas pressure release. Ensuring a rapid pressure release and outburst elimination in coal seams in gas-rich areas is difficult, which leads to insignificant pressure-relief efficacies. Therefore, studying the impact of different hydraulic slotting construction parameters on the gas pressure is necessary. By simulating the impact of different slot sizes, spacings, and numbers on the gas pressure during borehole hydraulic slotting, the parameters for rapid extraction and pressure relief through hydraulic slotting can be obtained, and the regularity of the gas pressure changes after hydraulic slotting can be understood. Mastering the impact of the slotting construction parameters on the gas pressure can effectively control the effective extraction range of hydraulic slotting boreholes, fully utilize the timeliness of hydraulic slotting boreholes, reduce the cost of coal mine gas management, and shorten the gas management cycle.

4.1. Impact of Slot Size on Gas Pressure

To simulate the impact of different slot sizes on the gas pressure, hydraulic slotting models with slot sizes of 1.5, 2, and 2.5 m were set up. Under the premise of the same extraction time of 100 days, changes in the gas pressure with different slot sizes over the extraction time were simulated (Figure 10). For the same extraction duration, an increase in the slotting size leads to a decrease in the gas pressure within the coal seam (Figure 10), demonstrating that larger slot dimensions significantly influence the gas pressure reduction due to hydraulic slotting. As a result, increasing the slot size improves the effectiveness of gas pressure reduction.

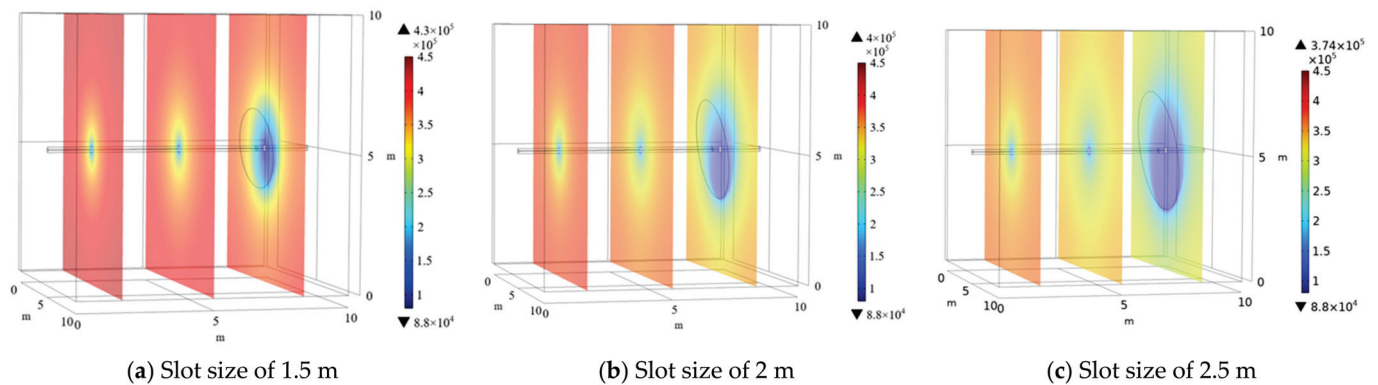


Figure 10. Gas pressure contour maps after 100 d of extraction with different slot sizes.

The gas pressure curves along the axial monitoring line under the condition of 100 days of extraction with different slotting sizes are shown in Figure 11. The gas pressure in the slotting area decreased significantly, with a larger slot size leading to a greater decrease in the gas pressure, dropping to a minimum of 0.088 MPa. Additionally, as the slot size increased, the gas pressure reduction along the borehole's axial direction gradually intensified. At a slot size of 2.5 m, the gas pressure in the slotted area around the extraction borehole dropped by approximately 75%. In contrast, when the slotting size was 1.5 m, the gas pressure in the unslotted area decreased by 71%. Additionally, the overall pressure-mitigation efficacy in the slotting area improved by 24–41% compared to the unslotted area.

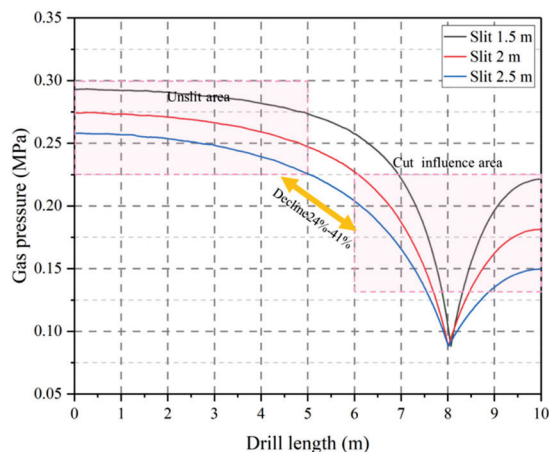


Figure 11. Gas pressure curves along the axial detection line with different slot sizes.

To intuitively observe the impact of different slotting sizes on the gas pressure, the gas pressure curves along the radial monitoring line at 7.5 m after 100 days of extraction with varying slotting sizes were compared (Figure 12). Different hydraulic slotting sizes affected the gas pressure range variably, and the extent of this impact on the gas pressure increased with the increasing slot size. Additionally, under the same extraction duration, the gas pressure around the extraction borehole with larger slotting sizes decreased to a greater extent. When the slot size was 1.5 m, the affected range was approximately 1.2 m around the borehole, whereas when the slot size increased to 2.5 m, the affected range reached approximately 3 m, nearly tripling. Therefore, the slot size mitigated the gas pressure in both the axial and radial directions, and the pressure-release rate improved with the increasing slot size.

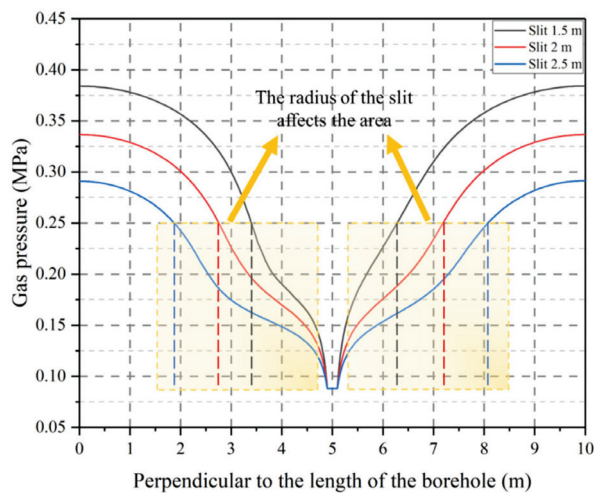


Figure 12. Gas pressure curves at the 7.5 m radial detection line with different slot sizes.

4.2. Impact of Slot Spacing on Gas Pressure

With a fixed slot size of 1.5 m and a slot number of 2, slot spacings of 1.5, 2.5, and 3.5 m were set for the numerical simulation, and gas pressure contour maps after 100 d of extraction with different slot spacings were drawn. The gas pressure changes caused by different slot spacings are shown in Figure 13.

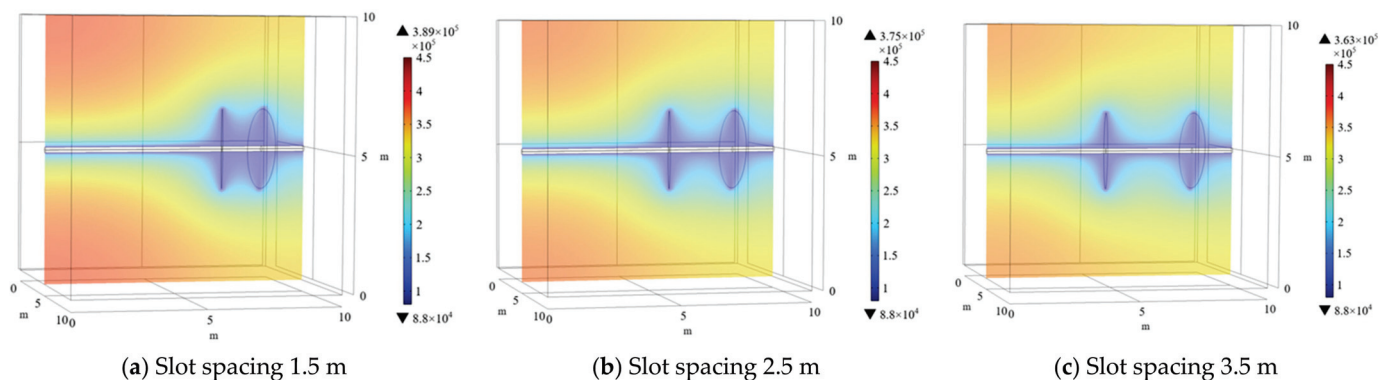


Figure 13. Gas pressure contour maps after 100 d of extraction with different slot spacings.

The greater the slot spacing, the wider the range of gas pressure relief due to hydraulic slotting. However, as the slot spacing increased, the gas pressure between the two slots also increased. To observe the impact of different slotting spacings on the gas pressure during extraction, the gas pressure variation data along the axial direction for different slotting spacings were exported. The resulting gas pressure curves after 100 days of extraction under different slotting conditions are illustrated in Figure 14.

The gas pressure curves indicate that as the slotting interval increases, the overall gas pressure shows a downward trend. The most significant decrease occurred at the hydraulic slotting locations, reaching the minimum gas pressure. When the slotting interval was 3.5 m, the average gas pressure around the extraction borehole reduced to 0.2 MPa. However, the gas pressure between the slots gradually increased with the growing slotting interval. At a 1.5 m slotting interval, the gas pressure between the slots remained relatively low, approximately 0.13 MPa. In contrast, for other slotting intervals, the gas pressure between the slots showed an increasing trend, particularly at the 3.5 m interval, where the gas pressure between the slots changed the most significantly, reaching a maximum of 0.19 MPa, representing a 50% increase.

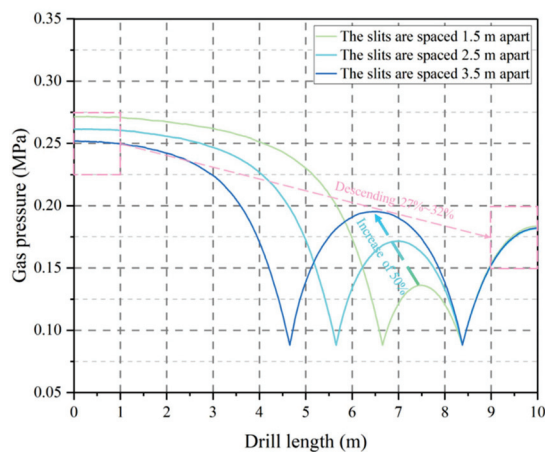


Figure 14. Gas pressure curves after 100 d of extraction with different slot spacings.

Therefore, in areas with dense slotting, the permeability of the coal body significantly increased after hydraulic slotting, effectively promoting gas flow and facilitating pressure release during extraction. With the increasing slot spacing, the permeability of the coal body between the slots did not increase significantly, although the overall gas pressure showed a decreasing trend after extraction. The gas retention time in the coal body was long, forming partially gas-rich areas. Therefore, when constructing hydraulic slotting boreholes, the spacing between each slot should be fully considered to avoid forming gas-rich areas and affecting gas management effectiveness.

4.3. Impact of Slot Number on Gas Pressure

To further study the impact of the slot number on the gas pressure based on previous numerical simulation results, the slot size and spacing were both fixed at 1.5 m. Numerical simulations were conducted with slot numbers of 1, 2, and 3, and gas pressure contour maps after 100 d of extraction with different slot numbers were drawn (Figure 15).

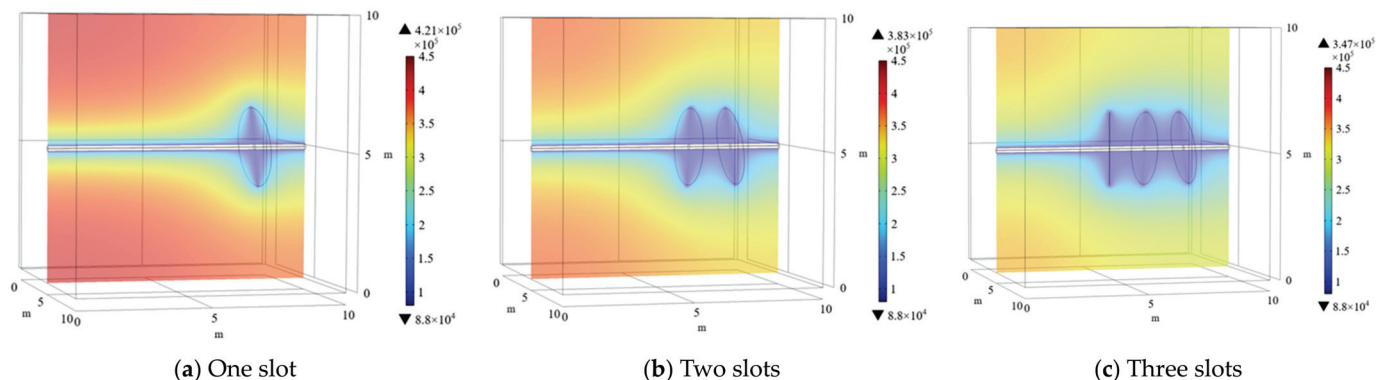


Figure 15. Gas pressure contour maps after 100 d of extraction with different slot numbers.

As illustrated in Figure 15, increasing the number of slots expanded the pressure-relief zone, indicating that more slots effectively enlarged the gas pressure-relief area, significantly reducing the coal seam gas pressure and establishing an efficient gas discharge zone, thereby accelerating the pressure-relief process. Further, the gas pressure profiles along the axial monitoring line confirmed that the number of slots favored gas pressure reduction (Figure 16). The gas pressure curves with different slot numbers show that with the increasing slot number, the peak gas pressure along the axial line of the extraction borehole decreased, and the decreasing trend in the gas pressure became evident with the increasing slot number. The gas pressure in the extraction borehole decreased by 23–25%,

indicating that the hydraulic slotting disturbance had a cumulative effect on enhancing gas desorption and permeability.

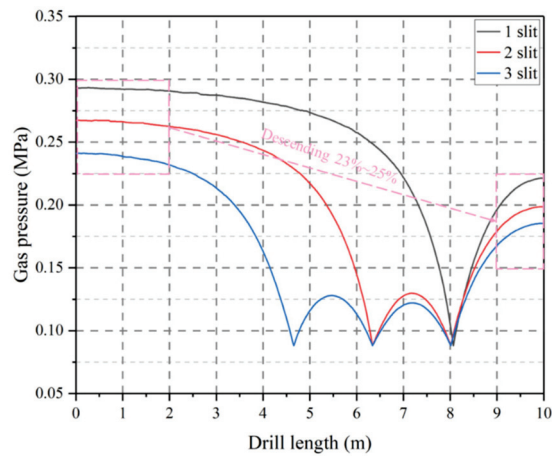


Figure 16. Gas pressure curves along the axial detection line after 100 d of extraction with different slot numbers.

Consequently, hydraulic slotting influenced the dynamic behavior of the original gas flow in the coal and altered the permeability of the coal body. Moreover, as the number of slots increased, the cumulative disturbance from hydraulic slotting on the coal body intensified, further promoting a reduction in the gas pressure within the coal seam.

5. Field Experiments

5.1. Geological Conditions

The Wangxingzhuang Coal Mine is located southwest of Xinzheng City, Henan Province (Figure 17), and the 11,131 working face of this mine, located in the 11th mining area, has an average length of 706 m (east to west) and an average width of 146 m (north to south). In this region, the 2-1 coal seam, which is susceptible to outbursts, is situated in the lower part of the Shanxi Formation, approximately 54 m below the overlying Sanguoyao Sandstone. The extraction of gas from this coal seam is primarily realized via the fully mechanized retreat longwall mining method.

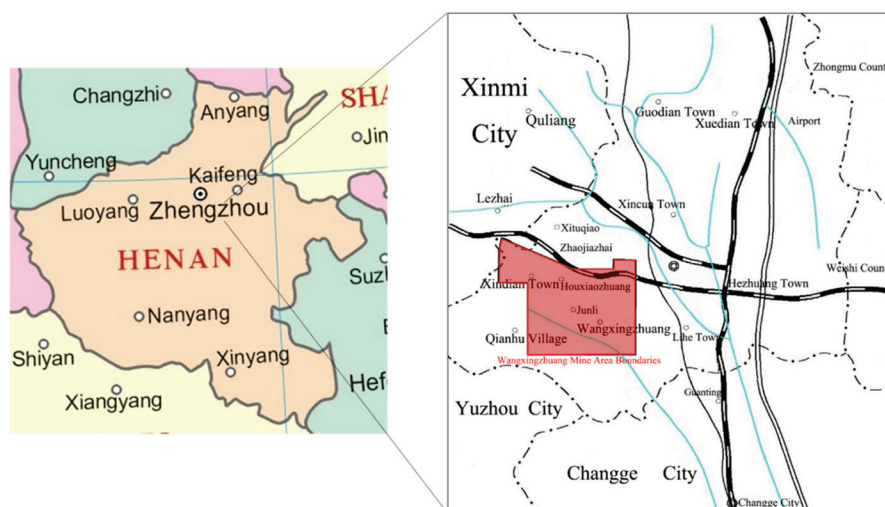


Figure 17. Location map of Wangxingzhuang Coal Mine.

The coal seam thickness ranges between 0 and 24.76 m, with an average thickness of 6.67 m, and the coal seam is relatively stable. The original gas content of the coal seam averages $3.52 \text{ m}^3/\text{t}$, and the gas pressure averages 0.19 MPa. The immediate roof of the

coal seam is medium-to-fine-grained sandstone and the floor is dark gray sandy mudstone with sandy streaks.

5.2. Field Experiments Project Overview

To determine the effective radius of gas extraction for hydraulic slotting boreholes in the floor gas extraction roadway of the 11,131 lower gateway, seven groups of hydraulic slotting test boreholes were arranged in the gateway. Each group had one extraction borehole and four gas-content test holes (Figure 18).

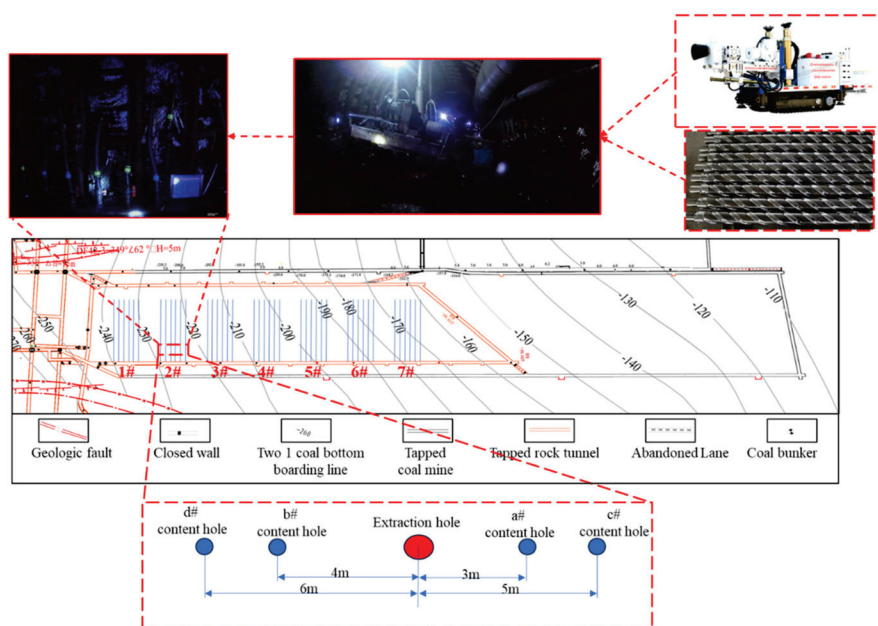


Figure 18. Layout of boreholes.

To determine the effective radius of the extraction borehole, seven groups of test boreholes were positioned both parallel and perpendicular to the roadway, taking into account the specific conditions of the Wangxingzhuang Coal Mine. The borehole diameters used in each group were 75 mm. Group 1, 2, and 3 boreholes served as the control group for different slot sizes. Group 1, 4, and 5 boreholes comprised the control group for different slot numbers, whereas group 4, 6, and 7 boreholes constituted the control group for different slot spacings. During the construction process, hydraulic slotting boreholes were first constructed, and the original gas content was measured. The extraction holes were connected for extraction. After 100 days of extraction, the coal seam gas contents were recorded at 3, 4, 5, and 6 m from the extraction borehole, corresponding to the measurement points labeled a#, b#, c#, and d#, respectively. The coal seam gas content was measured after 100 days of extraction, and the extraction radius was considered effective when the gas content decreased by 30% compared to the initial levels. During the extraction process, the gas concentration and flow rate from the extraction borehole were monitored daily.

5.3. Data Analysis of Field Experiments

The variations in the gas extraction concentration and gas extraction flow rate for boreholes with different slotting sizes, as well as the gas content in different test holes, within the same observation period, are shown in Figure 19.

The gas extraction concentrations reflected by boreholes 1# (slot size = 1.5 m), 2# (slot size = 2 m), and 3# (slot size = 2.5 m) were consistent with the rules reflected in the numerical simulation results in Figures 13 and 14. Both the extraction flow and concentration increased with the increasing slot size (Figure 19). Larger slot sizes increased the permeability of the coal seam, thereby increasing the gas extraction efficiency. Under the same observation period, with the increasing slot size, the overall gas extraction flow

of borehole 3# was 56.87 and 25.06% higher than that of boreholes 1# and 2#, respectively (Figure 19a). During the observation period, the maximum extraction flow of borehole 3# was $87.32 \text{ m}^3/\text{min}$, whereas the maximum gas extraction concentrations of boreholes 1# and 2# were 55.65 and $69.82 \text{ m}^3/\text{min}$, respectively. The maximum extraction flow of borehole 3# was 1.57 times that of borehole 1# and 1.25 times that of borehole 2#. Moreover, under the same observation period, the overall extraction concentrations of boreholes 1# and 2# were 6.38 and 3.13% lower than those of borehole 3#, respectively (Figure 19b). The overall trend in the gas extraction flow rate and concentration over time is similar across the three boreholes, but there were significant differences in the magnitudes. The 3# borehole, with the largest slotting size, consistently exhibited higher gas extraction flow rates and concentrations compared to the 1# and 2# boreholes with smaller slotting sizes, indicating greater extraction efficiency. As shown in Figure 19c, for the same extraction duration, the effective radius of gas extraction expanded as the slotting size increased. The extraction radius of the 1# borehole was approximately 3.5 m, while the extraction radius of borehole 3# was approximately 5.5 m, exhibiting an increase of approximately 2 m. This indicates that as the borehole slotting size increases, the affected area also expands.

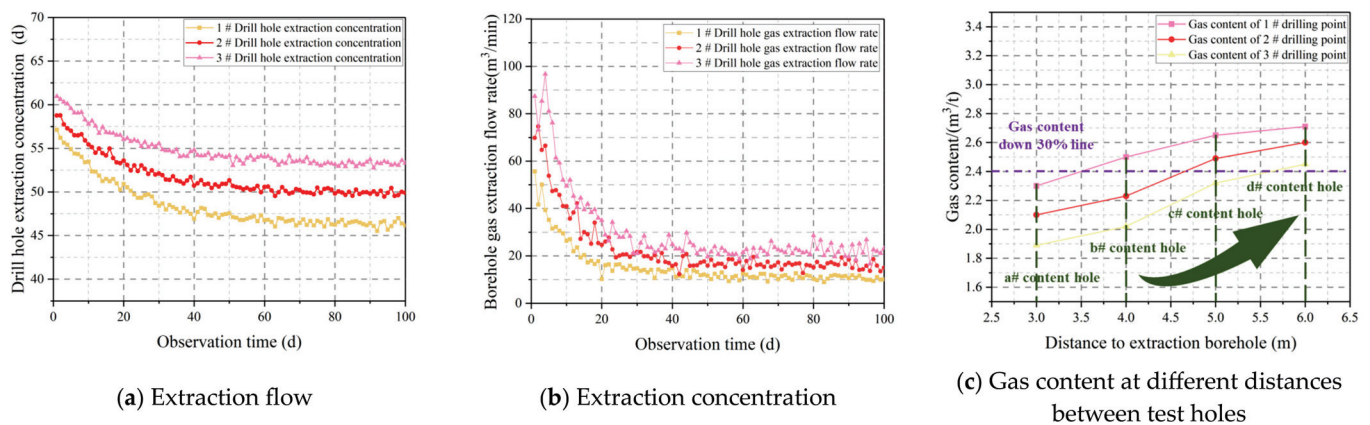


Figure 19. Observation of gas extraction flow, concentration data, and gas content in different distance test holes for different slot sizes.

The changes in the gas extraction concentration, gas extraction flow, and gas content in test holes of variable distances for boreholes with different slot numbers were recorded within the same observation period (Figure 20). Borehole 1# (single slot) had an average extraction flow of $15.24 \text{ m}^3/\text{min}$, borehole 4# (double slots) averaged $35.94 \text{ m}^3/\text{min}$, and borehole 5# (three slots) reached $46.40 \text{ m}^3/\text{min}$. Under the same observation period, the overall extraction flow of the borehole increased with the slot number (Figure 20a). When the slot number was increased from one to three, the overall extraction flow of the borehole increased from 15.24 to $46.40 \text{ m}^3/\text{min}$, an increase of 3.04-fold. This outcome indicates that as the number of slots increases, the permeability of the coal seam improves. With the increasing slot number, the overall extraction concentration of the borehole also gradually increased (Figure 20b). When the number of slots was increased from one to three, the overall extraction concentration of the borehole also increased from 48.41% for borehole 1# to 56.62% for borehole 5#, an increase of 12.13%. This result shows that the gas extraction concentrations reflected by boreholes 1#, 4#, and 5# were consistent with the rules reflected in the numerical simulation results (see Figure 16). Simultaneously, under the same observation period, the extraction radii of different slot sizes showed an increasing trend. When the number of slots increased from one to three, the extraction radius increased from 3.5 to 6 m. Increasing the number of slots can significantly improve the gas extraction volume and concentration. With multiple slots, both the flow and concentration reached higher levels.

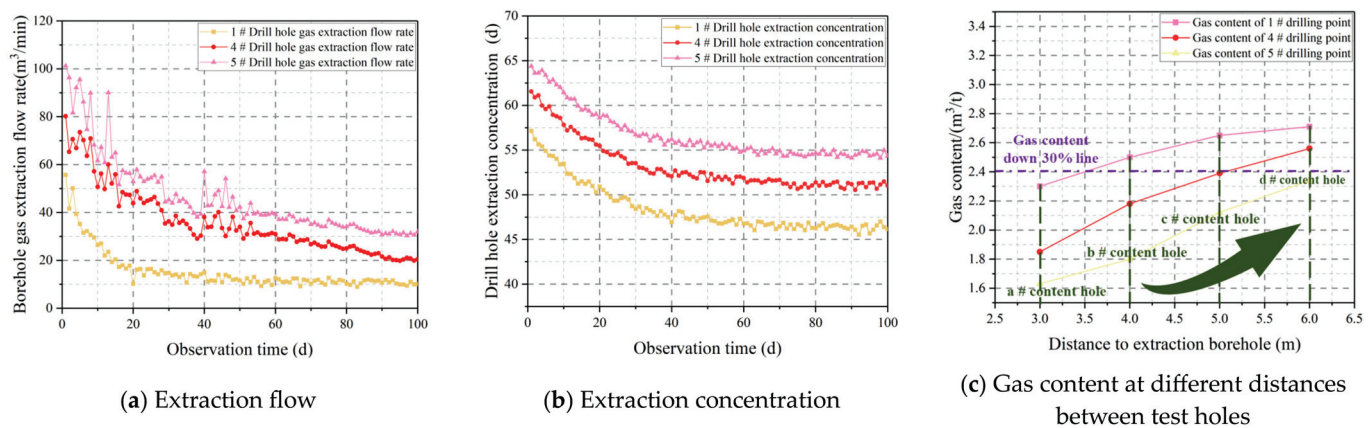


Figure 20. Observations of gas extraction flow, concentration data, and gas content in test holes of varying distances for different slot numbers.

The changes in the gas extraction concentration, gas extraction flow, and gas content in boreholes with different slot spacings were recorded during the same observation period (Figure 21). Borehole 4# (1.5 m spacing) had an average extraction flow of 35.94 m³/min, borehole 6# (2.5 m spacing) averaged 50.77 m³/min, and borehole 7# (3.5 m spacing) achieved 69.98 m³/min. As shown in Figure 21a, during the same observation period, the gas extraction flow rate of the borehole increased notably as the slotting spacing widened. When the slotting spacing increased from 1.5 to 3.5 m, the borehole's gas extraction flow rate increased from 35.94 to 69.98 m³/min, nearly a 1.95-fold increase. This outcome suggests that as the slot spacing increased, the gas flow path became less obstructed, resulting in a higher gas extraction flow rate. The overall gas extraction concentration in the borehole also gradually increased with the increasing slot spacing (Figure 21b). When the slot spacing increased from 1.5 to 3.5 m, the gas extraction concentration increased from 53.20% for borehole 4# to 58.49% for borehole 7#, an overall increase of approximately 15.81%. In addition, an increase in the slot spacing also affected the extraction radius. At a slotting spacing of 1.5 m, the extraction radius was 5 m. As the slotting spacing increased to 3.5 m, the extraction radius expanded to approximately 6 m. This result indicated that increasing the slot spacing effectively increased the permeability of the coal seam, thereby enhancing the gas extraction efficiency. Optimizing the slotting design, particularly by adjusting the slotting spacing, plays a crucial role in enhancing the coal seam gas extraction rates.

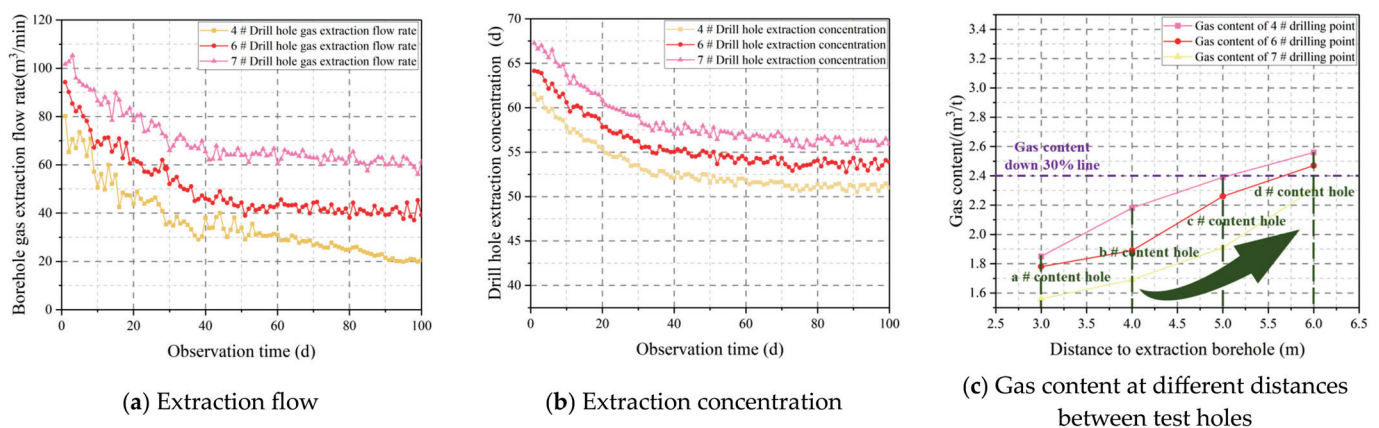


Figure 21. Observations of gas extraction flow, concentration data, and gas content in test holes of variable distances for different slot spacings.

From the above analysis, we concluded that different slotting parameters had significant effects on the extraction flow of the boreholes. Large slot sizes, additional slots, and

large slot spacings increased the extraction flow of the boreholes. These findings align with previous numerical simulation results and offer further theoretical support for the optimized design of hydraulic slotting technology in practical applications.

6. Conclusions

This study focuses on the 11,131 working face of the Wangxingzhuang Coal Mine. Using a fluid–solid coupling model with hydraulic slotting for gas extraction from coal seams, we employed a modeling process to estimate the effective radius of gas extraction for through-seam drainage boreholes in the 11,131 working face. Using COMSOL simulations, this study analyzed the impact of hydraulic slotting on coal seam stress variations and gas pressure, as well as the changes in the gas pressure and pressure-relief enhancement range under different construction parameters. The research explored the changes in coal seam stress and pressure following hydraulic slotting and examined the influence of the construction parameters. The numerical simulation results were validated by comparing them with field observations of the extraction flow rates and concentrations. The results indicate the following:

(1) Hydraulic slotting significantly affected the local stress and strain distributions of the coal body. On the 3D scale, distinct high-strain and high-stress zones formed around the slot, with both strain and stress exhibiting evident spatial attenuation characteristics. The impact is primarily concentrated near the slotting area, where the coal seam is subjected to intense impact from high-pressure water flow, resulting in significant local deformation. The stress around the slotting region increases markedly, with the maximum stress value reaching $1.6 \times 10^7 \text{ N/m}^2$, effectively enhancing the permeability of the coal seam.

(2) Hydraulic slotting treatment enhances fracture development within the coal seam, playing a critical role in gas extraction by facilitating gas flow. This process effectively reduces the gas pressure within both the matrix and the fractures. As the extraction time extended from 10 to 100 days, a widely expanded low-pressure zone developed along the slot range in the coal body. The gas pressure within the slot area dropped from 1 to $\sim 0.08 \text{ MPa}$, significantly enhancing the efficiency of coal seam gas extraction.

(3) A 3D numerical simulation study of different hydraulic slotting construction parameters showed that increasing the slot size, number, and spacing effectively released pressure in the coal body. The interaction and combined effects of multiple slots were most evident in the slot spacing, with closer spacing leading to faster extraction but a smaller range, and larger spacing leading to slower extraction but a larger range. As the slotting size increases from 1.5 m to 2.5 m, hydraulic slotting effectively relieves gas pressure in both the axial and radial directions. In the axial direction, gas pressure reduction in the unslotted area increases from 71% to approximately 75%, with the slotted area showing a 24% to 41% improvement in pressure relief. Radially, the affected range expands nearly threefold. When the slotting spacing increases from 1.5 m to 3.5 m, the overall gas pressure decreases, with the average borehole gas pressure dropping to 0.2 MPa. However, the gas pressure between slots increases, reaching a maximum of 0.19 MPa, 50% higher than the minimum. Additionally, increasing the number of slots from one to three shows a cumulative effect, reducing the borehole gas pressure by 23% to 25%.

(4) Based on field observations of the gas extraction concentration, extraction flow, and gas content in test holes at different distances combined with data analysis, under different hydraulic slotting construction parameters, the gas extraction concentration, extraction flow, and extraction radius all showed varying degrees of improvement. As the slotting size increases, the gas extraction flow rate rises up to 1.57 times, the concentration improves by 6.38%, and the extraction radius expands by approximately 2 m. Increasing the slotting spacing results in an approximate 1.95-fold increase in the extraction flow rate, 15.81% increase in the concentration, and 1 m expansion in the extraction radius. Additionally, increasing the number of slots leads to a maximum rise of 3.04 times in the flow rate, a 12.13% improvement in the concentration, and an approximate 2.5 m increase in the extraction radius.

This study successfully quantified the pressure-release range and effective extraction area under various slotting spatial parameters by constructing 3D hydraulic slotting numerical models. These models revealed the influence of the slotting arrangements on the extraction efficiency and demonstrated the combined effects of multiple slotting configurations on pressure relief and permeability enhancement. The findings were further validated through field engineering tests, providing solid scientific and practical contributions. The results not only offer new insights into optimizing hydraulic slotting parameters but also present a framework that can be applied to similar gas extraction studies, making these findings relevant for future research and practical applications in coal mining. However, further research is necessary to refine the slotting parameters and explore their integration with other extraction methods for enhanced efficiency. Additionally, the long-term stability of slotting-induced fractures and the effects of geological factors, such as coal seam heterogeneity, should be examined to ensure broader and more effective applications of hydraulic slotting technology in diverse mining environments.

7. Discussion

The results of this study confirm the significant role of hydraulic slotting in enhancing coal seam gas extraction efficiency by influencing local stress distribution, fracture development, and gas migration. The use of a fluid–solid coupling model provided an effective means of simulating the complex interactions between hydraulic slotting and gas extraction, offering new insights into optimizing the construction parameters for improved extraction outcomes. The findings align with previous studies on gas extraction, which have similarly shown the effectiveness of hydraulic slotting in creating highly permeable zones that facilitate gas migration [36]. The study highlights that hydraulic slotting significantly alters the local stress and strain distributions in the coal seam, a finding consistent with similar work by Liu et al. [37]. The development of high-strain and high-stress zones around the slots has been shown to enhance the coal seam's permeability, thereby accelerating the gas flow and extraction efficiency. This local stress concentration may also contribute to long-term pressure relief in the surrounding areas, further facilitating gas migration over extended extraction periods. However, the study also showed the impact of slotting is primarily localized, suggesting that strategic slot placement and optimized construction parameters are necessary to achieve uniform pressure relief across larger areas.

Moreover, this study reinforces the importance of construction parameters such as the slot size, spacing, and number in determining the overall extraction performance. As noted by Chen et al. [27], increasing the number of slots can lead to cumulative effects in gas pressure reduction and extraction efficiency. Our findings further suggest that slot spacing accelerates gas extraction but reduces the pressure-relief range, whereas larger spacing expands the pressure-relief zone but slows the extraction rate. These trade-offs must be carefully considered in practical applications to balance efficiency and safety concerns. Field observations corroborated the simulation results, particularly in terms of the extraction flow rates and gas concentrations, suggesting that hydraulic slotting is a robust technique for enhancing gas extraction in coal seams.

However, several challenges remain that warrant further investigation. Firstly, while the numerical simulations accurately captured the short-term effects of hydraulic slotting on gas extraction, the long-term stability of slot-induced fractures and their impact on the gas extraction efficiency over time remain uncertain. Geological factors, such as coal seam heterogeneity and the presence of natural fractures, could influence the effectiveness of slotting, as suggested by previous studies [38]. Future research should explore integrated hydraulic slotting with other extraction methods, such as hydraulic fracturing and directional drilling, to enhance efficiency and address potential limitations related to geological variability.

Additionally, while the findings of this study offer guidance for optimizing the hydraulic slotting parameters, they are based on a specific case study of the Wangxingzhuang Coal Mine's 11,131 working face. More comprehensive field studies are needed to validate

the generalizability of these results across different coal mines with varying geological conditions. Incorporating advanced monitoring techniques, such as microseismic monitoring and 3D modeling of gas migration, could further improve the understanding of hydraulic slotting's impact on long-term gas extraction performance.

In conclusion, this study successfully demonstrated the effect of hydraulic slotting in enhancing coal seam gas extraction. By optimizing the slotting parameters, significant improvements in gas pressure reduction and extraction efficiency were achieved. Future research should focus on refining slotting techniques, addressing the long-term stability of induced fractures, and exploring the integration of multiple extraction methods to maximize efficiency in diverse coal mining environments.

Author Contributions: X.C.: project administration; conceptualization; writing—original draft. X.M.: data curation; software. J.H.: supervision; visualization. T.Y.: investigation; supervision. A.Z.: investigation; supervision. R.B.: investigation; supervision. J.S.: validation. All authors have read and agreed to the published version of the manuscript.

Funding: This work was supported by the National Natural Science Foundation of China (No. 52174181), the Hebei Natural Science Foundation Youth Science Foundation Project (E2024508014), the Fundamental Research Funds for the Central Universities (No. 3142024011, 3142021008), and the S&T Program of Hebei (Grant No. 22375401D).

Data Availability Statement: Data will be made available on request.

Conflicts of Interest: We declare that we have no financial and personal relationships with other people or organizations that can inappropriately influence our work, there is no professional or other personal interest of any nature or kind in any product, service, and/or company that could be construed as influencing the position presented in, or the review of, the manuscript entitled, “Combined Effect of Multiple Slotting Spatial Parameters on Gas Extraction Efficiency: Numerical Analysis and Field Verification”.

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Article

The Pore Structure Multifractal Evolution of Vibration-Affected Tectonic Coal and the Gas Diffusion Response Characteristics

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Abstract: Vibrations caused by downhole operations often induce coal and gas outburst accidents in tectonic zone coal seams. To clarify how vibration affects the pore structure, gas desorption, and diffusion capacity of tectonic coal, isothermal adsorption-desorption experiments under different vibration frequencies were carried out. In this study, high-pressure mercury intrusion experiments and low-pressure liquid nitrogen adsorption experiments were conducted to determine the pore structures of tectonic coal before and after vibration. The pore distribution of vibration-affected tectonic coal, including local concentration, heterogeneity, and connectivity, was analyzed using multifractal theory. Further, a correlation analysis was performed between the desorption diffusion characteristic parameters and the pore fractal characteristic parameters to derive the intrinsic relationship between the pore fractal evolution characteristics and the desorption diffusion characteristics. The results showed that the vibration increased the pore volume of the tectonic coal, and the pore volume increased as the vibration frequency increased in the 50 Hz range. The pore structure of the vibration-affected tectonic coal showed multifractal characteristics, and the multifractal parameters affected the gas desorption and diffusion capacity by reflecting the density, uniformity, and connectivity of the pore distribution in the coal. The increases in the desorption amount (Q), initial desorption velocity (V_0), initial diffusion coefficient (D_0), and initial effective diffusion coefficient (D_e) of the tectonic coal due to vibration indicated that the gas desorption and diffusion capacity of the tectonic coal were improved at the initial desorption stage. Q , V_0 , D_0 , and D_e had significant positive correlations with pore volume and the Hurst index, and V_0 , D_0 , and D_e had negative correlations with the Hausdorff dimension. To a certain extent, vibration reduced the local density regarding the pore distribution in the coal. As a result, the pore size distribution was more uniform, and the pore connectivity was improved, thereby enhancing the gas desorption and diffusion capacity of the coal.

Keywords: multifractal evolution; pore structure; vibration; gas diffusion; tectonic coal

1. Introduction

China has extremely complex coal seam reserve conditions [1,2]. With the continuous increase in mining depth, the gas content and gas pressure of the working seam have gradually increased, and coal and gas outburst accidents still seriously threaten the safe production of coal mines [3–5]. Such coal and gas outburst accidents are often preceded by supporting, coal falling, drilling, blasting, coal cutting, and other on-site operations that can produce mechanical vibration. Studies have shown that the energy required for coal pulverization, gas desorption, and gas-solid mixture bursts is transmitted by the surrounding rock of the coal seam through vibration [6]. Therefore, vibration can be considered one of the main critical causes of coal and gas outburst accidents.

In the past 20 years, experts and scholars have extensively studied the effects of vibration on coal permeability, porosity, mechanical properties, and gas adsorption-desorption capacity, and some results have been achieved. Ren et al. [7,8] and Pan et al. [9] found that

vibration can alter the stress state of the coal mass, and the induced cracks in the coal mass have a certain softening effect on it. Song et al. [10,11] found through experiments and CT imaging that mechanical vibration is conducive to crack expansion and development in the coal mass, increasing its permeability. Ren et al. [12] and Wen et al. [13] studied the response characteristics of coal vibration induced by simple harmonic vibration, concluding that the mechanical parameters of coal determined its vibration response characteristics, and the peak values of the response parameters increased significantly when the excitation force frequency resonated with the natural frequency of the coal mass. Wei et al. [14,15] further explored the influence of low-frequency vibration on the permeability of coal samples, concluding that the permeability growth rate of the coal mass increased first and then decreased as the vibration frequency increased, and the permeability enhancement was greater near the resonance frequency. Li et al. [16] found that vibration can reduce the gas adsorption capacity of the coal mass and accelerate gas desorption. Meanwhile, their analysis from the perspective of fracture mechanics concluded that vibration caused unstable crack expansion and development in the coal mass, thus inducing bursts. Shen et al. [17,18] concluded that vibration could cause pore structure changes in the coal mass, which increased the gas diffusion path and reduced the diffusion resistance, thus promoting gas desorption in particulate coal.

Coal and gas outburst sites are typically characterized by the development of tectonic coal, with tectonic coal development at almost all outburst sites [19–21], i.e., the wide tectonic coal reserve plays a decisive role in the outburst [22]. The microstructure of the coal mass is closely related to its mechanical properties, and the yield and failure of the coal mass are the macroscopic manifestations of the evolution of microscopic pores and cracks [23]. At present, experts and scholars have conducted a series of studies on the differences in microstructure and physical properties of tectonic coal and primary coal. Previous research results showed that tectonic coal had significantly decreased pore compressibility due to tectonic deformation [24]. Compared with primary coal, tectonic coal has the characteristics of low mechanical strength, weak adhesion, and weak resistance to external stress disturbance, and its natural particles are smaller and more likely to break into small-particle coal powder [22,25]. Meanwhile, tectonic coal has more complex pore structure characteristics than primary coal, with more intensive pore development, a lower pore-throat ratio, and lower tortuosity than primary coal [26]. Tectonic coal has a strong gas adsorption-desorption capacity, and its initial gas desorption and diffusion capacity is greater than that of primary coal [27–29], leading to rapid gas desorption at the initial desorption stage and promoting the release of gas energy [4,30–32].

Despite the great progress in the research on how vibration influences the physical properties of the coal mass, the differences in pore structure and physical properties between primary coal and tectonic coal were not considered. Due to its low mechanical strength, easy pulverization, weak resistance to external stress disturbance, complex pore structure, and strong adsorption-desorption capacity, tectonic coal must exhibit different response characteristics from primary coal when subjected to vibration. Therefore, on the basis of previous studies, pore structure determination and gas desorption and diffusion experiments were conducted on vibration-affected tectonic coal in this research. The evolutionary characteristics of pore heterogeneity and connectivity in vibration-affected tectonic coal were analyzed using multifractal theory. A correlation analysis was performed between the pore structure characteristic parameters and the gas desorption and diffusion characteristic parameters to derive the pore fractal characteristics and gas desorption and diffusion response characteristics of the vibration-affected tectonic coal and the intrinsic connection between them. The research results can provide theoretical guidance for gas management and coal and gas outburst prevention in tectonic coal mines.

2. Methods and Theory

2.1. Coal Sample Selection and Preparation

In the experiment, tectonic coal was collected from the Wangxingzhuang Coal Mine (WXZ) in Henan and the Gaohe Coal Mine (GH) in Shanxi. The Wangxingzhuang Coal Mine is located in the Xinmi coalfield. Due to the influence of tectonic action in the coalfield area, the II₁ coal seam presents the characteristics of the development of tectonic coal in the whole coal seam. The WXZ coal sample collection site is located in the 15,021 working face of the II₁ coal seam. Structures such as faults, folds, and collapse columns were developed in the Gaohe coal mine field. The GH coal sample collection site is located in the structural area of the south wing roadway passing through the footwall of the F₃ fault. The freshly collected samples were sealed and transferred to the laboratory. After crushing, coal samples 1~3 mm in size were selected. The coal from each mine was divided into 3 even samples, 200 g each. The basic physical parameters of the coal sample are presented in Table 1.

Table 1. Basic coal sample physical parameters.

Coal Sample	Adsorption Constant		Industrial Analysis			TRD (g/m ³)	ARD (g/m ³)	φ (%)
	a (m ³ /t)	b (MPa ^{−1})	Mad (%)	Ad (%)	Vdaf (%)			
WXZ	26.32	2.74	0.88	7.73	13.81	1.43	1.38	3.19
GH	26.18	0.7184	0.28	4.97	11.84	1.41	1.37	2.84

2.2. Experimental Equipment and Processes

After the coal sample preparation, a MicroAutoPore IV type 9500 mercury piezometer and an ASAP type 2460 physical adsorption instrument were used to perform high-pressure mercury intrusion experiments and low-pressure liquid nitrogen adsorption experiments, thus characterizing the pore structure of the coal sample before vibration. In order to ensure the reliability of the test results, the instrument was calibrated using the standard samples provided by the instrument manufacturer before the high-pressure mercury injection experiment and the low-pressure liquid nitrogen adsorption experiment.

An MVGAD-I type vibration condition isothermal adsorption-desorption platform independently developed by our research team was used to perform isothermal gas adsorption-desorption experiments of coal under vibration, where the adsorption equilibrium pressures were set to 1 MPa and 1.5 MPa. The schematic diagram of the experimental equipment is shown in Figure 1. Methane (CH₄) was selected as the gas in the adsorption and desorption experiments. Vibration was applied after the coal reached adsorption equilibrium to allow desorption for 60 min. Previous research [13,14] showed that the natural frequency of coal was within 50 Hz, the mechanical vibration frequency that may induce coal and gas outburst accidents caused by underground operation was 5~100 Hz [17,18], and the vibration frequency of micro-seismic was 0~35 Hz [33]. Therefore, combining the frequency of various vibration forms and the purpose of this study, the vibration frequency in this experiment was set to 0, 25, and 50 Hz. Following the desorption experiment, the coal samples were degasified and subjected to high-pressure mercury intrusion and low-pressure liquid nitrogen adsorption experiments to determine the pore structure.

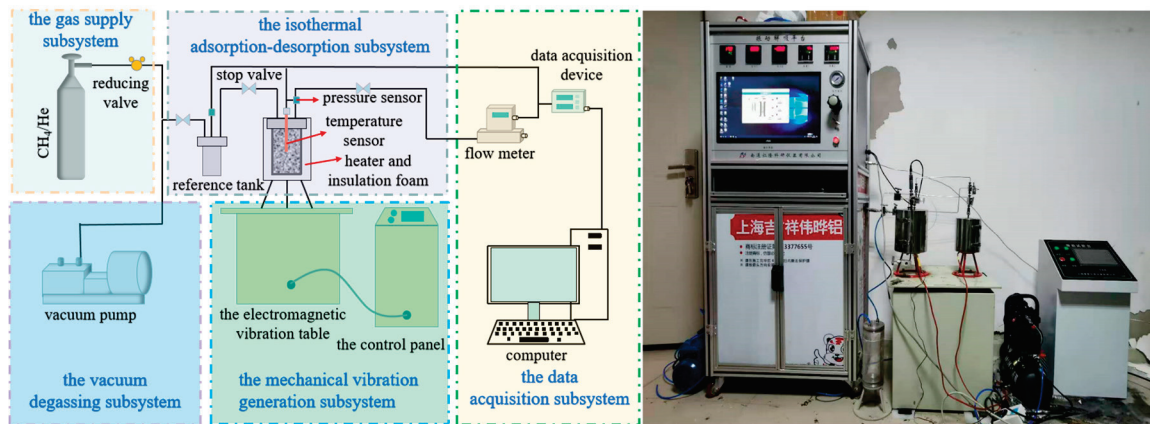


Figure 1. The schematic diagram and physical diagram of the MVGAD-I type vibration condition isothermal adsorption-desorption platform.

2.3. Multifractal Theory

The monofractal method describes or characterizes the porosity characteristics of the porous medium as a whole or as an average, thus only capable of describing the overall irregularity and heterogeneity within finite dimensions [34–37]. Unlike the monofractal method, the multifractal method can be decomposed into a series of interwoven fractal subsets that characterize the local variability and heterogeneity of the studied variables through a series of scaling indices, thus providing more accurate pore structure information [38–42]. These scaling indices can be calculated by using the pore structure test data obtained by the high pressure mercury intrusion experiment and the low pressure liquid nitrogen adsorption experiment according to Formulas (1)–(13).

The multifractal analysis on porous media with an interval of $I = [a, b]$ necessitates the equal division of the interval into a set of boxes with a length of ε [40,41]. The dichotomy method is commonly used to divide the interval $I = [a, b]$ with a length of L into $N(\varepsilon) = 2^k$ boxes with a length of $\varepsilon = L \cdot 2^{-k}$ ($k = 0, 1, 2, 3, \dots$). Among these boxes, the mass probability function of the i -th box with a length of ε can be expressed as follows [39,42,43]:

$$P_i(\varepsilon) = \frac{N_i(\varepsilon)}{N_t} \quad (1)$$

where $N_i(\varepsilon)$ is the total number of study subjects in the i -th box, and N_t is the total number of study subjects in all boxes throughout the interval.

The partition function associated with q and $\chi(q, \varepsilon)$ (also known as the statistical moment function) is an important intermediate variable in multifractal calculations and can be derived from the following equation [44]:

$$\chi(q, \varepsilon) = \sum_{i=1}^{N(\varepsilon)} P_i^q(\varepsilon) \sim \varepsilon^{\tau(q)} \quad (2)$$

where q is the statistical moment order, a real number generally within $[-10, 10]$, and $\tau(q)$ is the mass scaling function of q .

The mass scaling function $\tau(q)$ can also be expressed as follows:

$$\tau(q) = \lim \left[\ln \sum_{i=1}^{N(\varepsilon)} P_i^q(\varepsilon) / \ln(1/\varepsilon) \right] \quad (3)$$

Then, the generalized fractal dimension spectrum $D(q)$ can be determined as follows:

$$D_q = \frac{1}{q-1} \lim \frac{\log \chi(q, \varepsilon)}{\log \varepsilon} = \frac{\tau(q)}{q-1} (q \neq 1) \quad (4)$$

With $q = 0, 1$, and 2 , D_0 , D_1 and D_2 are defined as the capacity dimension, information dimension, and correlation dimension, respectively [43].

With $q = 1$, D_1 can be solved by L'Hospital's rule to make the D_q function continuous, as follows [45]:

$$D_1 = \lim_{\varepsilon \rightarrow 0} \frac{\sum_{i=1}^{N(\varepsilon)} P_i(1, \varepsilon) \log[P_i(1, \varepsilon)]}{\log(\varepsilon)} \quad (5)$$

If ε is small enough, then, $P_i(\varepsilon)$ is almost evenly distributed within each sub-interval, and $P_i(\varepsilon)$ and ε have the following relationship:

$$P_i(\varepsilon) \propto \varepsilon^\alpha \quad (6)$$

where α is the singularity exponent, reflecting the local singularity strength of the mass probability function $P_i(\varepsilon)$.

Different sub-intervals may have different α values, and the singularity exponent α can be calculated as follows [46]:

$$\alpha(q) \propto \frac{\sum_{i=1}^{N(\varepsilon)} [u_i(q, \varepsilon) \cdot \log \varepsilon]}{\log \varepsilon} \quad (7)$$

Among them, $u_i(q, \varepsilon)$ can be calculated as follows:

$$u_i(q, \varepsilon) = \frac{P_i^q(\varepsilon)}{\sum_{i=1}^{N(\varepsilon)} P_i^q(\varepsilon)} \quad (8)$$

The number of boxes with the same α value, denoted as $N_\alpha(\varepsilon)$, changes exponentially with the scale ε , i.e.,

$$N_\alpha(\varepsilon) \sim \varepsilon^{-f(\alpha)} \quad (9)$$

where $f(\alpha)$ is the multifractal singularity spectrum, representing the fractal dimension with the same singularity exponent subset.

$f(\alpha)$ can be calculated as follows [47]:

$$f(\alpha) \propto \frac{\sum_{i=1}^{N(\varepsilon)} [u_i(q, \varepsilon) \log u_i(q, \varepsilon)]}{\log \varepsilon} \quad (10)$$

The mass scaling function $\tau(q)$ and the generalized fractal dimension D_q have the following relationship:

$$\tau(q) = (q - 1)D_q \quad (11)$$

The B.B. Hotdot pore classification method [36] was adopted in this study. The mercury intrusion method was used to quantitatively characterize the macropores ($100 < d < 10,000$ nm) and mesopores ($100 < d < 100$ nm) above 100 nm, and the low-pressure nitrogen adsorption method was used to characterize the minipores ($10 < d < 100$ nm) and micropores ($d < 10$ nm) in the coal mass.

3. Results and Discussion

3.1. Vibration-Affected Tectonic Coal Pore Structure Characteristics

3.1.1. Pore Size Distribution and Volume Change Characteristics of Macropores and Mesopores in Vibration-Affected Tectonic Coal

Based on the high-pressure mercury intrusion data, the pore size distribution and pore volume change characteristics of the tectonic coal before and after vibration at different frequencies are obtained, as shown in Figure 2. As shown in Figure 2, the macropore and mesopore volumes of the WXZ and GH coal samples increase with increasing vibration frequencies. After the WXZ coal sample was subjected to 25 Hz and 50 Hz vibration, the macropores were enlarged by 11.45% and 28.84%, respectively, compared with the coal

samples without vibration, and the mesopore volume was increased by 10.06% and 28.72%, respectively. After the GH coal sample was subjected to 25 Hz and 50 Hz vibration, the macropores were 1.66 times and 3.27 times those of the coal sample without vibration, respectively, and the mesopore volume was increased by 1.46 times and 2.38 times, respectively. After the WXZ coal sample was subjected to 25 Hz and 50 Hz vibration, the total volume of macropores and mesopores increased by 12.32% and 28.74%, respectively. After the GH coal sample was subjected to 25 Hz and 50 Hz vibration, the total volume of macropores and mesopores increased by 1.09 times and 2.79 times, respectively. Thus, low-frequency mechanical vibration can increase the volume of macropores and mesopores in tectonic coal, and the effect gradually increases with the increase in vibration frequency.

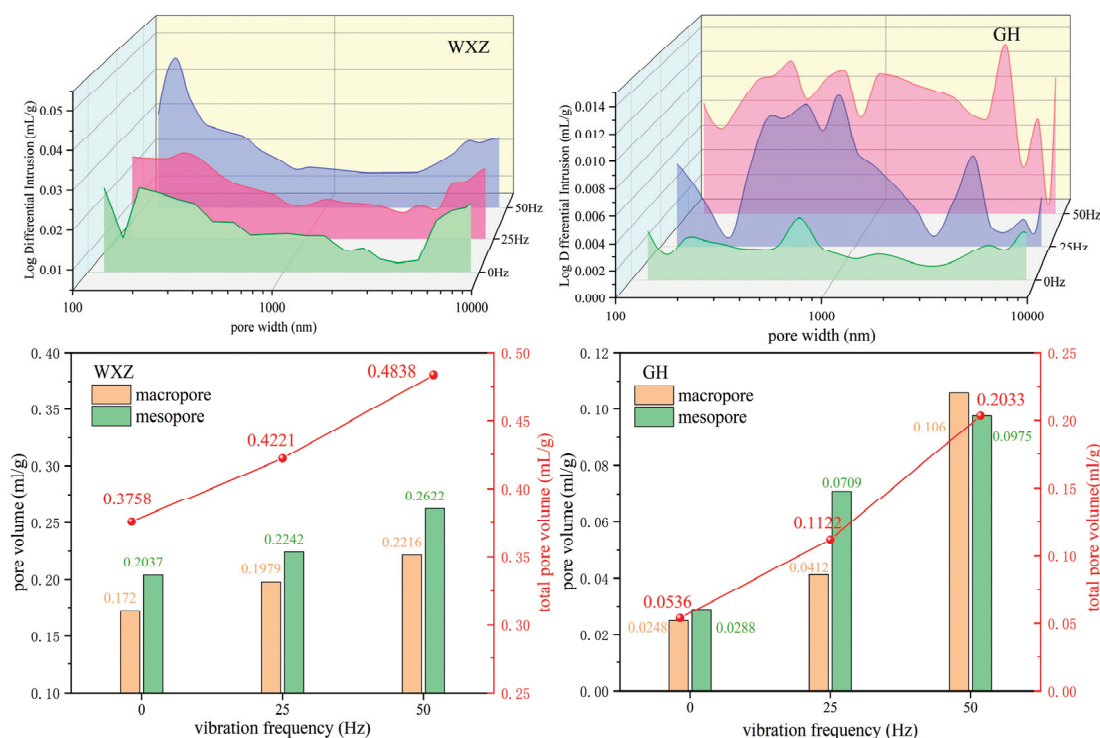


Figure 2. Size distribution and volume change characteristics of macropores and mesopores in vibration-affected tectonic coal.

According to the pore size distribution of the coal sample in Figure 2, the vibration increases in the volume of the mesopores with diameters of 100 to 200 nm and macropores with diameters of 3000 to 5000 nm in the WXZ coal sample. Among them, the 50 Hz vibration significantly increases the volume of mesopores, which range from 100 to 200 nm in size. The volume of pores with diameters of 300 to 5000 nm in the GH coal sample increased significantly after vibration. With decreased pore size, the macropore and mesopore size distributions of the coal sample change significantly after vibration, showing a unimodal and multimodal distribution. These results further indicate that vibration significantly affects the size distribution of macropores and mesopores, and the effect is stronger under greater frequencies.

3.1.2. Size Distribution and Volume Change Characteristics of Minipores and Micropores in Vibration-Affected Tectonic Coal

Figure 3 shows the size distribution and volume change characteristics of minipores and micropores in vibration-affected tectonic coal. As shown in Figure 3, the size distribution of minipores and micropores in the WXZ and GH coal samples is clearly multimodal. The pore size distribution of the WXZ coal sample increases with the increase in pore size, while that of the GH coal sample is uniformly distributed with the increase in pore size.

The vibration significantly increased the volume of minipores 40 to 100 nm in size in the GH coal sample, and the pore volume significantly increased in the coal sample after 50 Hz vibration. The effect of vibration on the size distribution of minipores and micropores in the WXZ coal sample is not obvious.

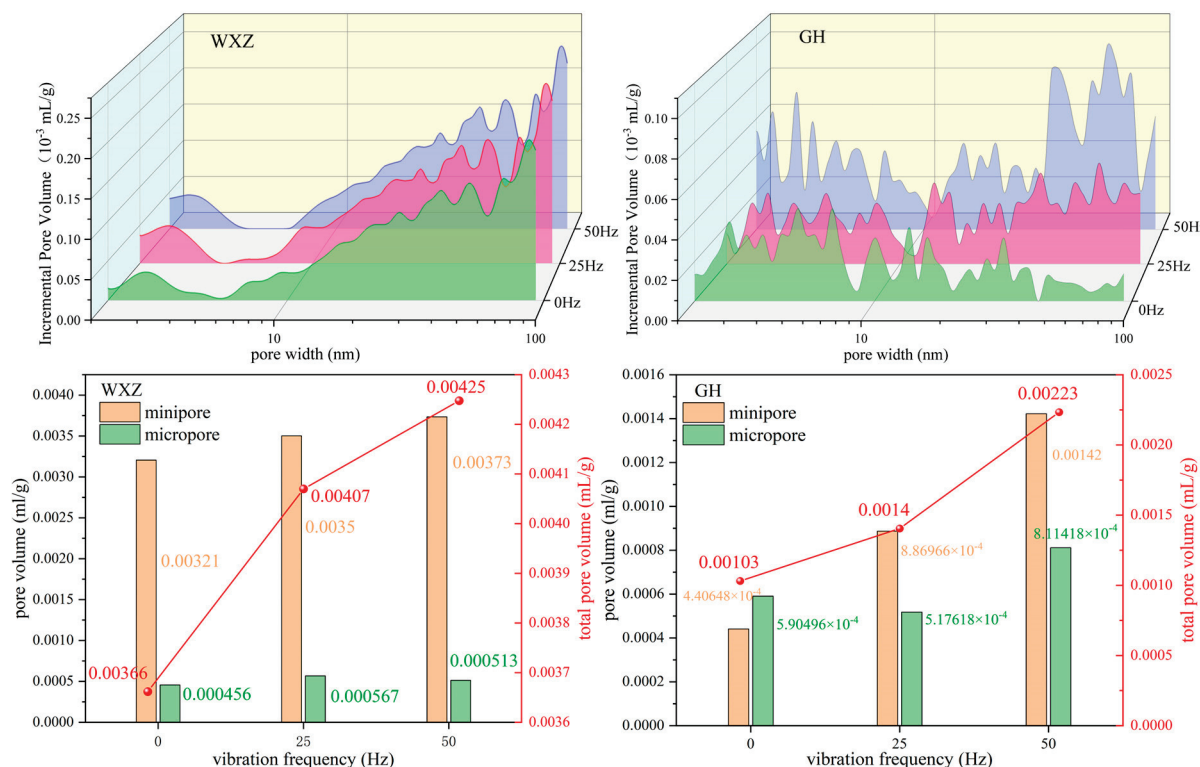


Figure 3. Pore size distribution and volume change characteristics of minipores and micropores in vibration-affected tectonic coal.

The 25 Hz and 50 Hz vibrations caused the minipore volume of the WXZ coal sample to increase by 9.03% and 16.19%, respectively, and the micropore volume by 24.34% and 12.50%, respectively. After the 25 Hz and 50 Hz vibrations, the minipores in the GH coal sample become 2.01 times and 3.22 times that in the coal sample without vibration. The 25 Hz vibration reduced the micropore volume of the GH coal sample by 12.34% but increased the micropore volume by 37.41%. After the WXZ coal sample was subjected to 25 Hz and 50 Hz vibration, the total volume of minipores and micropores increased by 11.20% and 16.12%, respectively. After the GH coal sample was subjected to 25 Hz and 50 Hz vibration, the total volume of macropores and mesopores increased by 35.92% and 116.50%, respectively. The vibration has a greater impact on the distribution and pore volume of minipores and micropores in the GH coal sample than in the WXZ coal sample. Overall, the effect of vibration on the minipores and micropores in tectonic coal increases with the increase in frequency.

3.2. Multifractal Characteristics of the Pore Structure of the Vibration-Affected Tectonic Coal

The pore interval for the multifractal calculation of macropores and mesopores was $I = [100, 10,000 \text{ nm}]$, the pore interval for the multifractal calculation of minipores and micropores was $I = [1, 100 \text{ nm}]$, and the statistical moment order q was set to all integers in $[-10, 10]$. Previous studies have confirmed that the research subjects have multifractal characteristics if there is a linear relationship between $\log(\varepsilon)$ and $\log \chi(q, \varepsilon)$. Figure 4 reflects the characteristics of the fitting between the partition function $\chi(q, \varepsilon)$ of the size distribution of macropores and mesopores in vibration-affected tectonic coal and the sub-interval length ε . Figure 5 reflects the characteristics of the fitting between the partition

function $\chi(q, \varepsilon)$ of the size distribution of micropores and micropores in vibration-affected tectonic coal and the sub-interval length ε . The fitting results showed a significant linear relationship between the $\log(\varepsilon)$ of the pore size distributions of all WXZ coal samples and GH coal samples and $\log \chi(q, \varepsilon)$, indicating that the generalized fractal dimension can be used to characterize the multifractal characteristics of the pores in the coal samples.

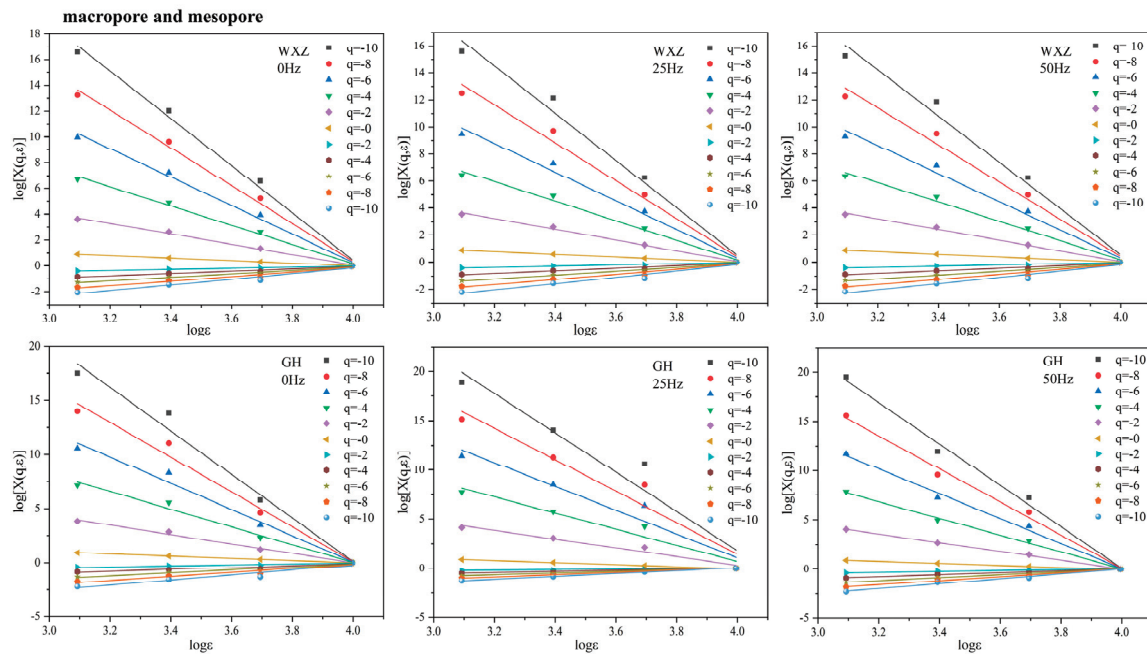


Figure 4. Fitting curves between the partition function of macropores and mesopores and the interval length of vibration-affected tectonic coal.

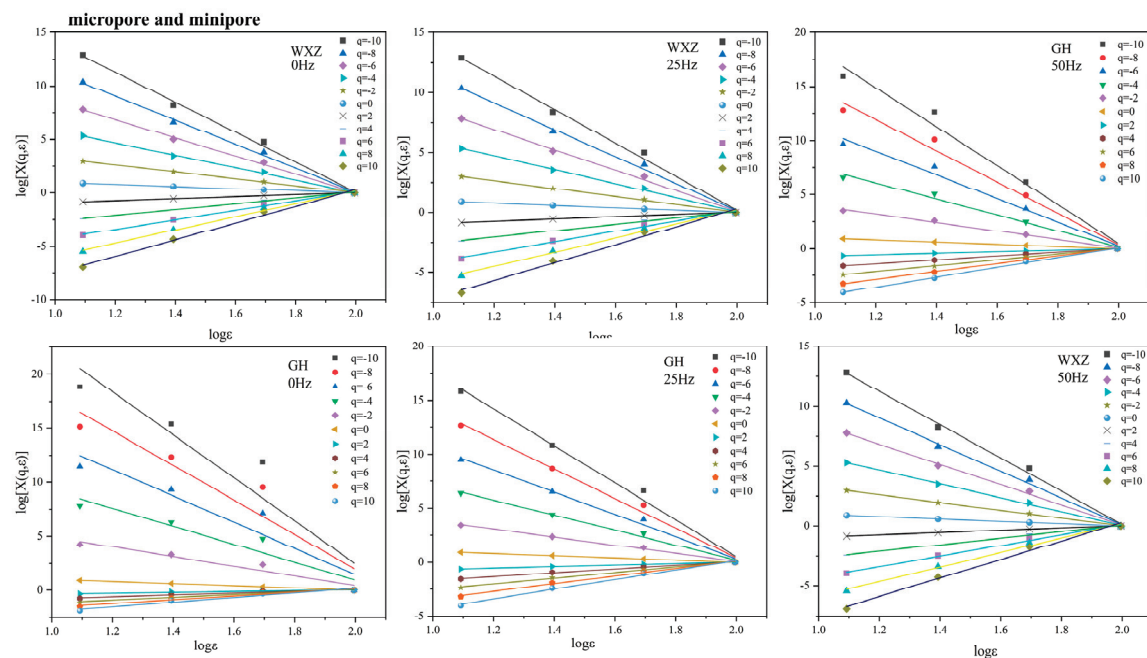


Figure 5. Fitting curves between the partition function of micropores and micropores and the interval length of vibration-affected tectonic coal.

As q gradually increased from -10 to 10 , the fitting curve gradually shifted from negative correlation to positive correlation, with gradually increased density, indicating that the pore

size of the coal was unevenly distributed and clustered in the studied small range. This is consistent with the pore size distribution characteristics shown in Figures 2 and 3.

The generalized dimension spectrum $D(q) \sim q$ of the pore size distribution of the vibration-affected tectonic coal is shown in Figure 6. The generalized dimension spectral curves of the pores of all tectonic coal samples show trends in an inverse “S” shape, with maximum and minimum values, indicating that the pore size distribution has non-uniform characteristics.

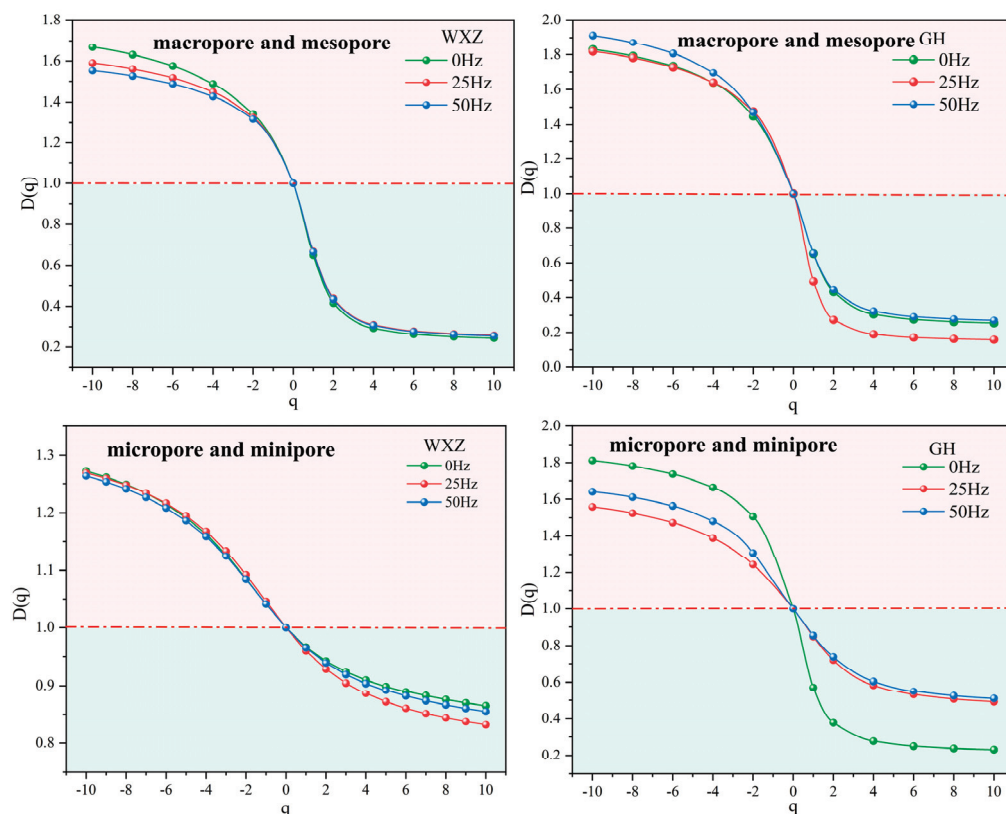


Figure 6. The generalized dimension spectra of the macropore and mesopore size distribution in the vibration-affected tectonic coal.

According to the generalized fractal dimension spectra, characteristic parameters such as capacity dimension D_0 , information dimension D_2 , correlation dimension D_2 , and spectral width $D_{-10} - D_{10}$ can be obtained. These parameters can reflect the heterogeneity information of the pore distribution in the coal, as shown in Table 2. The capacity dimension D_0 reflects the scaling characteristics of non-empty boxes containing certain pores under continuous segmentation. The data in Table 2 show that the capacity dimension of all coal sample pores is 1, indicating the presence of data in all boxes. The capacity dimensions of all coal samples are the same because the same pore size range was selected as the study interval.

The information dimension D_1 reflects the concentration degree of the pore distribution as the pore size interval changes. The correlation dimension D_2 reflects the uniformity of the pore size distribution measurement intervals. A smaller D_1 value indicates a more concentrated pore distribution, while a larger D_2 value indicates a more evenly spaced pore size distribution [27]. The D_1 value is selected for detailed analysis in this study. The D_1 values of the macropores and mesopores in the WXZ coal samples after vibration are greater than those of the coal samples without vibration, and the 25 Hz vibration causes the D_1 values of the minipores and micropores to fall below those of the coal samples without vibration. The D_1 values of the minipore and micropore distributions of the GH coal sample after vibration are greater than those of the coal sample without vibration.

The 25 Hz vibration significantly decreased the D_1 values of the macropore and mesopore distributions in the GH coal sample, indicating that the 25 Hz vibration increased the degree of local concentration of the macropore and mesopore distributions. The 50 Hz vibration slightly increased the D_1 values of the macropore and mesopore distributions in the GH coal sample. Thus, vibration changes the local concentration of pore distribution in tectonic coal. Overall, the 50 Hz vibration causes the pore distribution in coal to become more uniform.

Table 2. Characteristic parameters of the generalized fractal dimension spectra of the pore structure of the vibration-affected tectonic coal.

Coal Sample	Pore Type	Frequency (Hz)	D_0	D_1	D_2	$D_{-10}-D_{10}$
WXZ	macropore and mesopore	0	1	0.6488	0.4155	0.7567
		25	1	0.6668	0.4377	1.3321
		50	1	0.6639	0.4335	1.2978
	micropore and minipore	0	1	0.9656	0.9390	0.4076
		25	1	0.9600	0.9284	0.4369
		50	1	0.9669	0.94202	0.4086
GH	macropore and mesopore	0	1	0.6520	0.4322	1.5747
		25	1	0.4930	0.2733	1.6603
		50	1	0.6536	0.4454	1.6411
	micropore and minipore	0	1	0.5714	0.3783	1.5806
		25	1	0.8482	0.7192	1.0662
		50	1	0.8535	0.7374	1.1304

The spectral width $D_{-10}-D_{10}$ reflects the discrepancy characteristics of the pore size distribution in the local interval, which can characterize the heterogeneity of the pore structure in the coal sample. A larger $D_{-10}-D_{10}$ value indicates greater local fluctuations in the pore size distribution and a more complex pore structure [46]. The increased $D_{-10}-D_{10}$ of macropores and mesopores in the WXZ coal samples and the GH coal samples after vibration indicates that the vibration increased the local fluctuations of the macropore and mesopore size distribution of the tectonic coal. After vibration, the $D_{-10}-D_{10}$ of the minipores and micropores in the WXZ coal sample increased slightly, and the $D_{-10}-D_{10}$ of the GH coal sample decreased. Thus, vibration can change the complexity of the minipore and micropore size distributions.

The multifractal singularity spectra of the pore size distribution in the coal samples are shown in Figure 7. In the coal samples after vibration at different frequencies, the multifractal singularity spectra of pore size distribution are characterized by an inverted U-shaped unimodal convex function, which is another typical feature of the study subjects consistent with multifractal distribution. According to the multifractal singularity spectrum curve, the multifractal singularity spectrum characteristic parameters can be calculated, as shown in Table 3. The pore size distribution mass scaling function $\tau(q)$ curve (Figure 8) of vibration-affected tectonic coal is an obvious two-stage convex function, once again proving the multifractal characteristics of the pore structure of vibration-affected tectonic coal.

The singularity exponent α_0 , also known as the Hausdorff dimension, can reflect the degree of local concentration of the pore size distribution, and a larger value indicates a smaller degree of local concentration [44]. The singularity exponents α_0 of the macropores and mesopores in the WXZ and GH coal samples after vibration are greater than those of the coal samples without vibration, indicating that the vibration increased the local distribution fluctuations and local concentration degrees of the macropores and mesopores of the tectonic coal. Except for the GH coal sample subjected to 25 Hz vibration, the singularity exponent α_0 of the minipores and micropores of other coal samples with vibration is greater than that of the coal samples without vibration. Overall, the vibration reduces the local concentration of the pores in the tectonic coal, rendering their distribution more uniform.

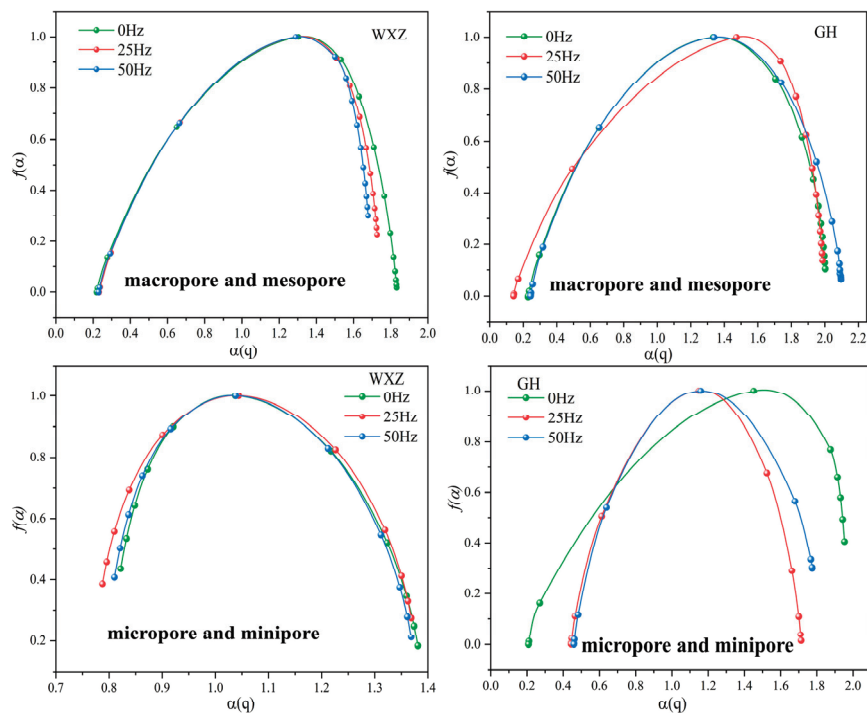


Figure 7. Generalized dimension spectra of the pore size distribution of vibration-affected tectonic coal.

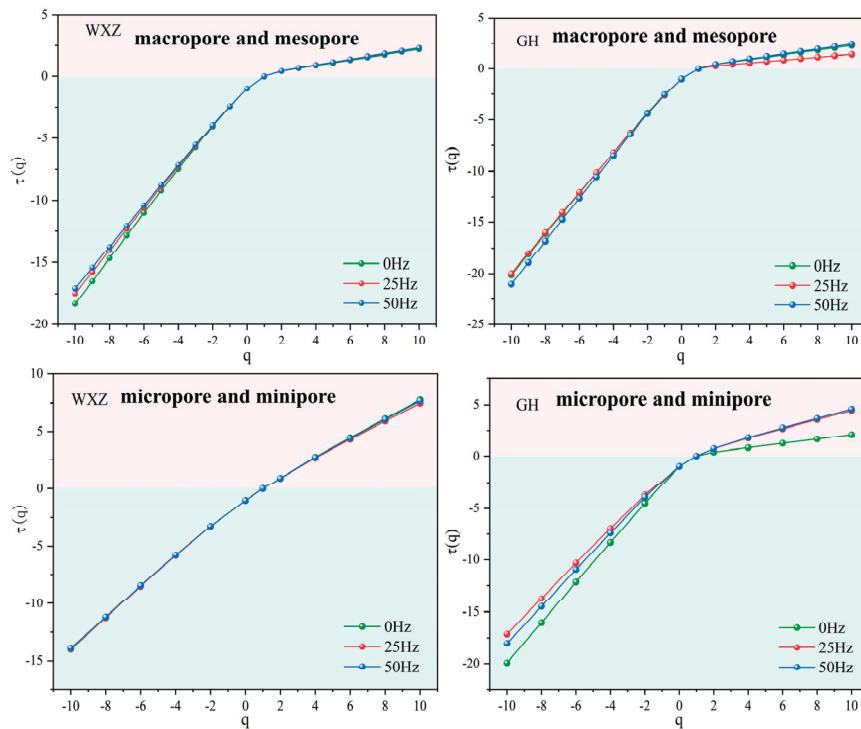


Figure 8. Mass scaling function $\tau(q)$ of pore size distribution of vibration-affected tectonic coal.

Table 3. Characteristic parameters of the generalized dimension spectrum of the pore size distribution of vibration-affected tectonic coal.

Coal Sample	Frequency (Hz)	Macropore and Mesopore			Micropore and Minipore		
		α_0	$\Delta\alpha$	Δf	α_0	$\Delta\alpha$	Δf
WXZ	0	1.2891	1.6127	0.0166	1.0375	0.5584	−0.2509
	25	1.3034	1.4943	0.2262	1.0438	0.5810	−0.1120
	50	1.2894	1.4485	0.3004	1.0383	0.5592	−0.1947
GH	0	1.3337	1.7727	0.1046	1.4511	1.7444	0.4051
	25	1.4722	1.8442	0.1374	1.1479	1.2698	0.0135
	50	1.3392	1.8531	0.0632	1.1591	1.3154	0.2983

The spectral width $\Delta\alpha$ ($\alpha_{\max} - \alpha_{\min}$) of the multifractal singularity spectrum can characterize the complexity of the probabilistic measure distribution of the entire pore size space, and a larger value indicates more significant differences within the multifractal measurement system and a more uneven distribution [40]. The vibration reduced the $\Delta\alpha$ of macropores and mesopores in the WXZ coal samples, and that of the minipores and micropores in the GH coal samples showed the opposite trend. The changes in the spectral width $\Delta\alpha$ of the pores in the GH coal samples after vibration are opposite to those of the WXZ coal samples. These results show that vibration can change the uniformity of the pore distribution in tectonic coal, altering the degree of difference in the pore size distribution. Since coal is a complex porous medium, a single parameter cannot effectively reflect the changes in its pore structure. Therefore, it is necessary to continue to analyze another characteristic parameter Δf of the multifractal singular spectrum and the parameter characterizing pore connectivity.

$\Delta f(f(\alpha_{\max}) - f(\alpha_{\min}))$ characterizes the ratio of the maximum and minimum number of elements in the subset of relevant physical parameters [41]. When the small probability subset predominates, the multifractal singularity spectrum shows a right-hook shape, that is, $\Delta f > 0$. The Δf of the macropore and mesopore distributions in the WXZ and GH coal samples showed obvious right hook shapes, and the difference was not obvious. Thus, the macropores and mesopores in the vibration-affected tectonic coal have small-range aggregation characteristics, and this characteristic plays a dominant role in the heterogeneity of the pore size distribution, but the vibration does not cause significant changes in its heterogeneity. The Δf of the minipore and micropore distributions in the GH coal sample were above 0, but the obvious right hook-shaped trend gradually weakened after vibration. The Δf of the minipore and micropore distributions in the WXZ coal sample were below 0, but the changes before and after vibration are not obvious. These results show that vibration can change the heterogeneity and aggregation characteristics of pore distribution in tectonic coal, but the effect is not significant.

The Hurst index can be used to characterize pore connectivity and can be derived using the multifractal mass scaling function $\tau(q)$ and the multifractal singularity spectrum $f(\alpha)$. The calculation is as follows [42,46]:

$$H(q) = \frac{\tau(q) + 1}{q} \quad (12)$$

$$\tau(\alpha) = q\alpha(q) - f(\alpha) \quad (13)$$

where $H(q)$ is the Hurst index, which characterizes the pore connectivity.

With $q = 2$, the classical Hurst index can be calculated using the above two equations, and a smaller value indicates poorer pore connectivity in the medium [39,44]. The Hurst indices of the pores of all grades in the tectonic coal after vibration at different frequencies are presented in Figure 9. The Hurst indices of macropores, mesopores, minipores, and micropores in all WXZ coal samples and GH coal samples after vibration were greater than those of the coal samples without vibration, showing increasing trends with the

increase in vibration frequency. Thus, vibration increases the pore connectivity in the coal. Studies have shown that pore connectivity affects coal permeability, and the permeability is higher with better pore connectivity. Therefore, vibration can increase the permeability of tectonic coal.

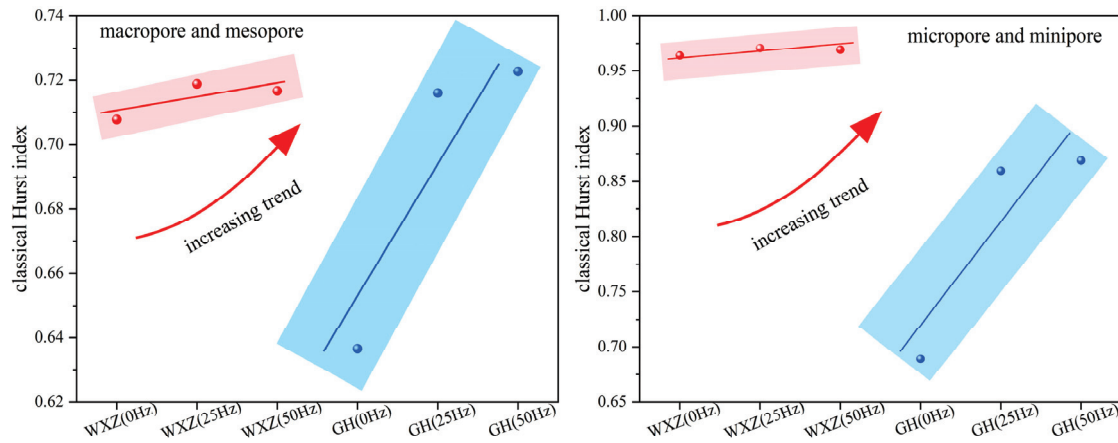


Figure 9. Classic Hurst index of vibration-affected tectonic coal pores.

3.3. Characteristics of Gas Desorption in Vibration-Affected Tectonic Coal

The gas desorption characteristics of tectonic coal samples with different adsorption equilibrium pressures at different vibration frequencies are shown in Figure 10. When the equilibrium pressure is 1 MPa, the desorption of the WXZ coal sample under 25 Hz and 50 Hz vibration increases by 7.02% and 13.38% compared with 0 Hz, respectively. The desorption of GH coal samples subjected to vibration at 25 Hz and 50 Hz increases by 18.83% and 32.31% compared with 0 Hz, respectively. At an equilibrium pressure of 1.5 MPa, the desorption of the WXZ coal sample under 25 Hz and 50 Hz vibration increases by 9.70% and 16.67% compared with 0 Hz, respectively. The desorption of GH coal samples subjected to vibration at 25 Hz and 50 Hz increases by 26.21% and 39.51% compared with 0 Hz, respectively. At the same vibration frequency, the cumulative gas desorption of the same coal sample at the same moment increases with increasing adsorption equilibrium pressure. A higher adsorption equilibrium pressure and a concentration difference drive the rapid desorption and diffusion of the gas, resulting in more gas being desorbed in the same period. The gas desorption amount of all coal samples subjected to vibration is greater than that of coal samples without vibration, indicating that mechanical vibration can promote gas desorption in tectonic coal.

The adsorption equilibrium pressure of gas in coal determines the size of its internal gas concentration, with a higher pressure resulting in a higher concentration of desorbed gas inside the coal pore fissures during the desorption process. Consequently, the gas diffusion process becomes stronger, and the gas desorption rate is higher. Figure 11 shows the initial gas desorption rate of tectonic coal samples subjected to vibration at different adsorption equilibrium pressures. It can be seen that a larger adsorption equilibrium pressure leads to a higher initial gas desorption rate when the coal sample is subjected to the same frequency vibration. However, the initial gas desorption rate of vibration-affected coal samples is greater than that of unaffected coal samples, regardless of the adsorption equilibrium pressure, with a larger frequency resulting in a higher value. These results indicate that vibration improves the initial gas desorption capacity of tectonic coals.

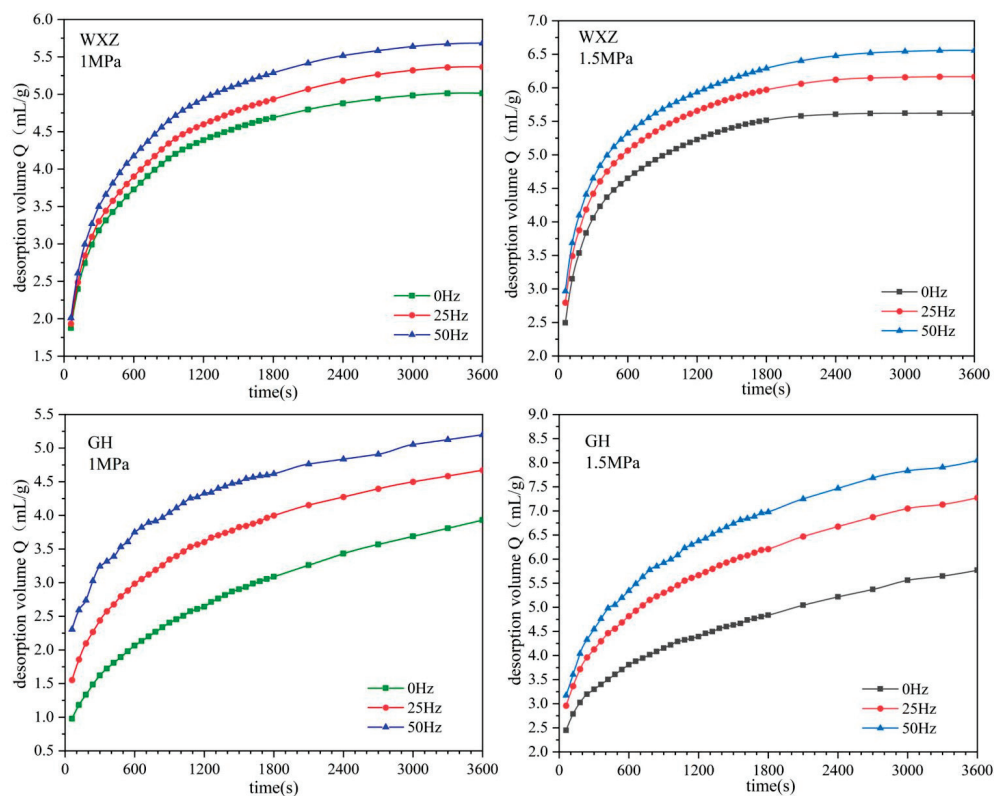


Figure 10. Gas desorption curves of coal samples.

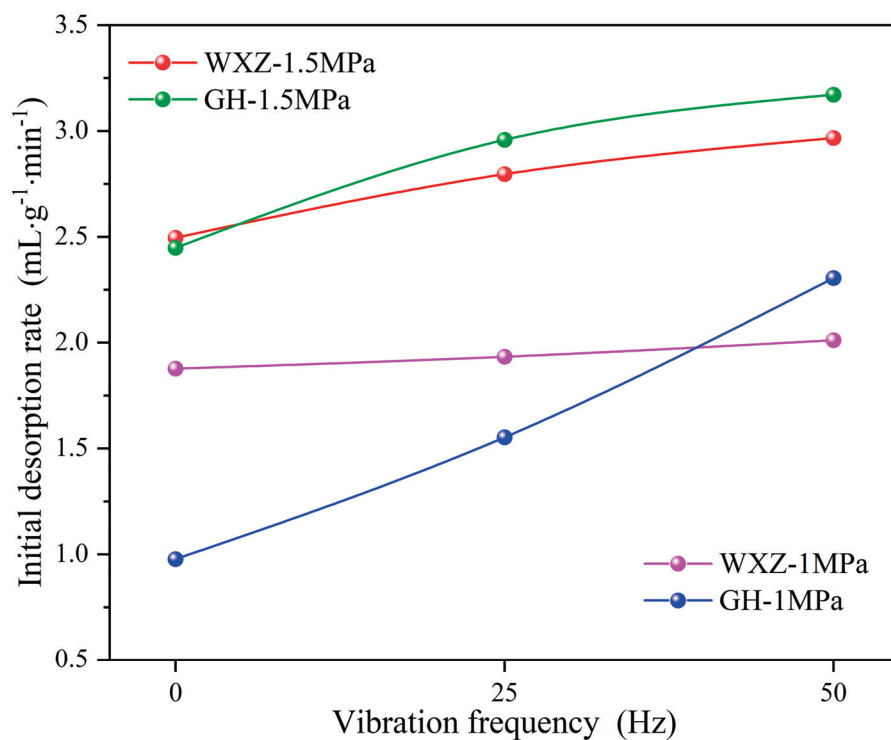


Figure 11. Initial gas desorption rate of vibration-affected tectonic coal.

3.4. Characteristics of Gas Diffusion in Vibration-Affected Tectonic Coals

The transport of gas (the main component is methane CH_4) from coal to the external environment involves a desorption-diffusion-seepage process [28,29,48,49]. For granular coal, scholars generally agree that the diffusion of gas from the coal matrix pores to the

particle surface needs to undergo rapid gas desorption on the pore surface and diffusion in the matrix pore fracture system. Li et al. [50] proposed a kinetic diffusion coefficient model for gas diffusion in multiscale, multilevel pores of coal particles based on the classical single-pore model for unsteady diffusion in homogeneous granular coal. The analytical solution of the dynamic diffusion coefficient model is as follows [18,48–50]:

$$\frac{Q_t}{Q_\infty} = 1 - \frac{6}{\pi^2} \sum_{n=1}^{\infty} \frac{1}{n^2} \exp\left(-\frac{n^2 \pi^2 D_0}{r_0^2 \beta} (1 - e^{-\beta t})\right) \quad (14)$$

$$D(t) = D_0 \exp(-\beta t) \quad (15)$$

where Q_t is the total amount of gas desorption from coal particles at time t , mL/g, and r_0 is the radius of coal particles, cm. The average value of the corresponding particle size is taken. Q_∞ is the maximum gas desorption amount, mL/g; $D(t)$ is the kinetic diffusion coefficient, cm^2/s ; D_0 is the initial gas diffusion coefficient at time $t = 0$, cm^2/s ; and β is the diffusion coefficient attenuation coefficient, s^{-1} .

The initial effective gas diffusion coefficient D_e can better reflect the gas diffusion capacity at the initial stage of desorption, and its expression is as follows:

$$D_e = \frac{D_0}{r_0^2} \quad (16)$$

Q_∞ can be calculated using the following equation:

$$Q_\infty = \left(\frac{abP}{1+bP} - \frac{abP_0}{1+bP_0} \right) (1 - W_f - A_f) \quad (17)$$

where a is the gas adsorption constant, indicating the maximum adsorption amount of coal under the experimental conditions, mL/g; b is the gas adsorption constant, MPa^{-1} ; W_f is the moisture of coal, %; A_f is the ash of coal, %; and P and P_0 are the adsorption equilibrium pressure and atmospheric pressure, respectively, MPa, which, at this time, is $P_0 = 0.1$ MPa.

The fitting effect between the measured gas diffusion of the coal sample and the dynamic diffusivity model is shown in Figure 12. The fitted correlation coefficients for WXZ coal samples at equilibrium pressures of 1 MPa and 1.5 MPa are 0.9993 to 0.9997 and 0.9556 to 0.9902, respectively. The fitted correlation coefficients for GH coal samples at equilibrium pressures of 1 MPa and 1.5 MPa are 0.9442 to 0.9929 and 0.8999 to 0.9593, respectively. The above results show that the dynamic diffusion coefficient model is applicable to the gas diffusion characteristics of vibration-affected tectonic coal.

The kinetic parameters of gas diffusion in vibration-affected tectonic coal D_0 , D_e , and β were obtained by fitting the kinetic diffusion coefficient model, as shown in Table 4. Figure 13 reflects the trend of the initial gas effective diffusion coefficient of the vibration-affected tectonic coal. Both the initial gas diffusion coefficient and the initial gas effective diffusion coefficient of coal samples with different equilibrium pressures at the same frequency exhibit an increasing trend, as evidenced by Table 4 and Figure 13. This trend arises from the positive correlation between adsorption equilibrium pressure and gas adsorption in coal, resulting in an increased gas concentration. The diffusion process in the early desorption stage mainly occurs in pores with large pore sizes, and its diffusion resistance is relatively small. Subsequently, a higher adsorption equilibrium pressure leads to a greater concentration difference between inside and outside the coal particles. Consequently, the diffusion drive becomes greater, resulting in a greater initial effective gas diffusion coefficient.

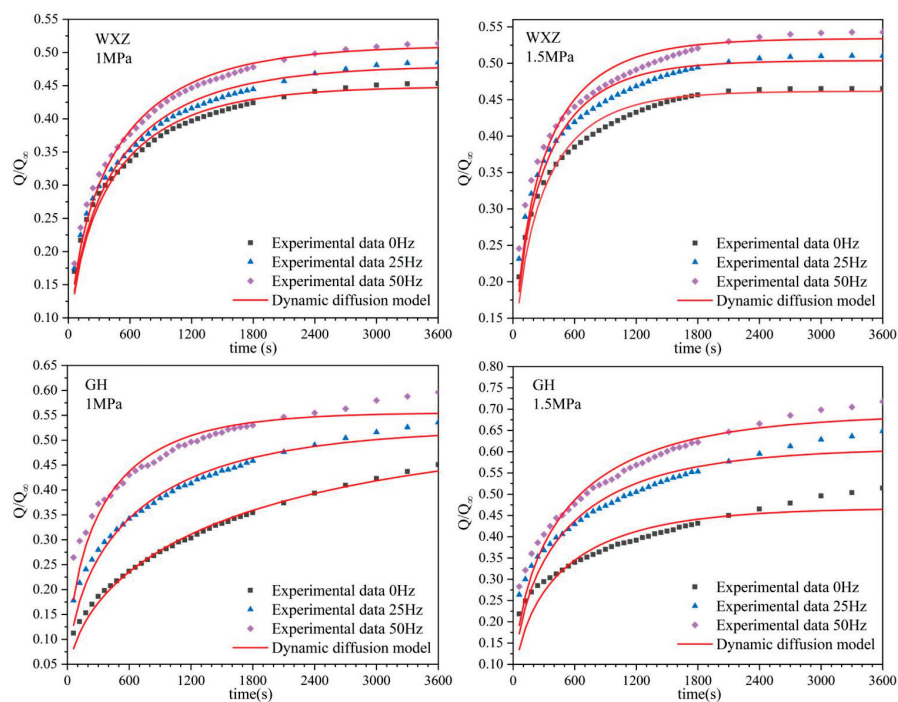


Figure 12. Measured gas diffusion rate of coal samples fitted with the kinetic diffusion model.

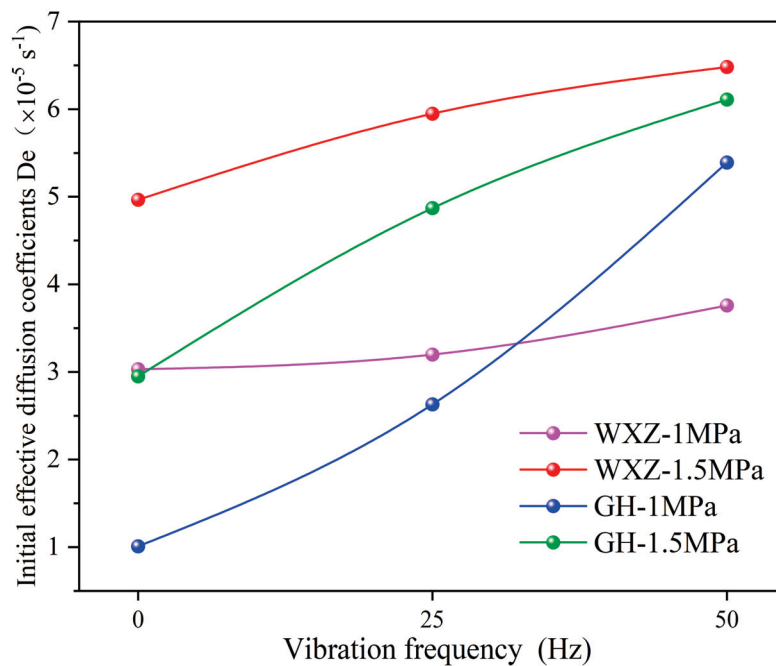


Figure 13. Initial effective diffusion coefficient of vibration-affected tectonic coal.

The attenuation coefficients of gas diffusion in coal samples do not show a uniform trend with increasing vibration frequency, owing to the intricate dynamics involved in the process of gas diffusion. Variations in pore connectivity, tortuosity of pore channels, complexity of pore structure, and pore sizes collectively contribute to disparities in resistance encountered during gas diffusion within coal. The energy required for diffusion, the length of the diffusion path, and the effective diffusion area also affect the diffusion capacity of the gas. The decay of diffusion capacity during diffusion is a multifactorial, coupled process.

Table 4. Kinetic parameters of gas diffusion in vibration-affected tectonic coals.

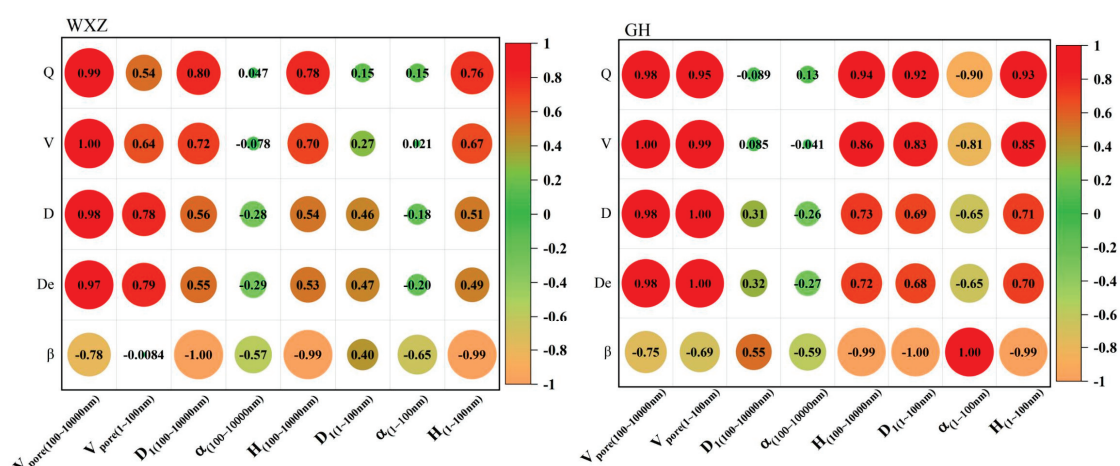
Coal Sample	Frequency (Hz)	1 MPa			1.5 MPa		
		D_0	D_e	β	D_0	D_e	β
WXZ	0	10.8	3.03	12.9	17.7	4.97	19.7
	25	11.4	3.19	11.6	21.2	5.95	19.1
	50	13.4	3.76	11.7	23.1	6.48	18.0
GH	0	4.04	1.01	3.01	11.8	2.95	11.3
	25	10.5	2.63	7.72	19.5	4.87	9.69
	50	21.6	5.39	13.6	24.4	6.11	8.65

Notes: $D_0 (\times 10^{-7} \text{ cm}^2/\text{s})$; $D_e (\times 10^{-5} \text{ s})$; $\beta (\times 10^{-4} \text{ s}^{-1})$.

At the same equilibrium pressure, the initial gas diffusion coefficient and initial effective gas diffusion coefficient of the same coal under vibration are greater than those of the coal sample under non-vibration conditions. It indicates that vibration enhances the initial gas diffusion ability in tectonic coal, and the overall diffusion ability is stronger with higher frequency. The diffusion of gas (CH_4) molecules is affected by a series of physical effects of mechanical vibrations on the gas molecules and by the energy transfer resulting from this process. In addition, the increase in pore parameters, such as macroporous volume in granular coal due to mechanical vibration, facilitates the diffusion of initially desorbed gas.

3.5. Analysis of the Correlation between Multifractal Characteristics of Pore Space and Gas Desorption Capacity in Vibration-Affected Tectonic Coals

The pore volume, the degree of pore connectivity, the homogeneity of pore distribution, and the tortuosity of pore channels all determine the ease of gas diffusion and transport from coal particles to the outside environment. These pore characteristics determine the magnitude of parameters characterizing the gas desorption and diffusion capacity, such as the gas desorption volume Q , the initial desorption velocity V_0 , the initial gas diffusion coefficient D_0 , the initial effective diffusion coefficient D_e , and the diffusion attenuation coefficient β . Therefore, this paper conducts Pearson correlation analysis between the pore volume, multifractal characteristic parameters, and desorption parameters at various levels. The results are shown in Figure 14.

**Figure 14.** Correlation matrix between pore structure parameters and desorption-diffusion parameters.

It can be seen that Q , V_0 , D_0 , and D_e of the WXZ and GH coal samples have significant positive correlations with pore volume and Hurst index at all levels. It indicates that with a larger pore volume and pore connectivity, the initial gas desorption and diffusion capacity in coal is stronger. Q , V_0 , D_0 , and D_e of WXZ and GH coal samples have positive

correlations with the information dimension D_1 of all levels of pores, but Q , V_0 , D_0 , and D_e of GH coal samples show less significant correlations with D_1 of their large and medium pores. These results indicate that a larger D_1 leads to a more uniform pore distribution, which is conducive to the desorption and diffusion of gas. In contrast, V_0 , D_0 , and D_e of WXZ and GH coal samples and the Hausdorff dimension of all levels of pores α_0 have a negative correlation, but the differences are not significant. With a greater local concentration of pores in coal, the initial gas desorption and diffusion abilities of gas are weaker. Therefore, the multifractal parameters can reflect the compactness degree and uniformity of the pore distribution in coal, which affects the ability of gas desorption and diffusion. Finally, the β of the coal samples exhibits a negative correlation with the volume of pores at all levels, the Hausdorff dimension, and the Hurst index. This correlation suggests that a higher pore volume in tectonic coals results in a more even distribution of pores and greater connectivity among them, leading to a lower diffusion attenuation coefficient and enhanced gas (CH_4) diffusion ability.

According to the research results in Section 3.2, vibration increases the volume of pores in tectonic coal to a certain extent and reduces the density of the pore distribution in the coal. Consequently, the pore distribution in the coal is more uniform, with reduced complexity and increased connectivity of pores at all levels. This further proves that vibration can facilitate the gas desorption and diffusion abilities of tectonic coal.

4. Conclusions

- (1) The results of high-pressure mercury intrusion and low-temperature liquid nitrogen adsorption experiments show that vibration affects the pore size distribution and pore volume characteristics of tectonic coals. Vibration causes an increase in pore volume in coal, and the expansion effect becomes stronger as the vibration frequency increases within the frequency range of 0–50 Hz. The increase in pore volume in coal provides more space for gas storage and transportation, and vibration is conducive to gas migration in coal.
- (2) The pore structure of the vibration-affected tectonic coal has multiple fractal characteristics, and the pore size distribution has a small range clustering feature. Vibration changes the degree of concentration, heterogeneity, local fluctuations, complexity, and connectivity of the pore distribution in coal. The change in the multifractal characteristic parameters of the pore structure indicates that vibration reduces the local concentration of pore distribution in coal. As a result, the pore size distribution is more uniform, and the connectivity of pores increases at all levels, promoting gas desorption and diffusion in coal.
- (3) The diffusion coefficient model is applicable to the gas diffusion characteristics of seismic coal structures. Vibration increases the gas desorption amount, initial desorption velocity, initial gas diffusion coefficient, and initial effective gas diffusion coefficient in tectonic coal. In addition, the desorption and diffusion ability of gas in structural coal is increased with vibration, and this ability becomes stronger as the vibration frequency increases within the frequency range of 0–50 Hz.
- (4) Multifractal parameters can affect gas desorption and diffusion ability by reflecting the density, uniformity, and connectivity of pore distribution in coal. Gas desorption volume, initial desorption velocity, initial gas diffusion coefficient, and initial gas effective diffusion coefficient have a significant positive correlation with pore volume and Hurst index. In contrast, the initial gas desorption velocity, initial gas diffusion coefficient, and initial gas effective diffusion coefficient are negatively correlated with the Hausdorff dimension. In general, a higher pore volume in tectonic coals results in a more even distribution of pores and greater connectivity among them, leading to a lower diffusion attenuation coefficient and enhanced gas diffusion ability. These results demonstrate that vibration can effectively improve gas desorption and diffusion in coal.

Author Contributions: Conceptualization, M.S.; methodology, M.S.; software, M.S. and W.W.; validation, M.S.; formal analysis, L.S.; investigation, Q.L.; resources, Q.L.; data curation, M.S.; writing—original draft preparation, M.S.; writing—review and editing, M.S. and P.Z.; visualization, L.S.; supervision, Z.H.; project administration, Z.H.; funding acquisition, Q.L. All authors have read and agreed to the published version of the manuscript.

Funding: This study was supported by the National Natural Science Foundation of China (52304224), the Special Project for Technology Innovation and Entrepreneurship Foundation of Tiandi Science & Technology Co., Ltd. (2021-2-TD-MS001), and the China Coal Research Institute Technology Innovation Fund (2022CX-I-05).

Data Availability Statement: The data presented in this study are available on request from the corresponding author. The data are not publicly available because they contain information that could compromise the privacy of research participants.

Conflicts of Interest: The authors declare no conflicts of interest.

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Article

Experimental Study on the Effect of Unloading Paths on Coal Damage and Permeability Evolution

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Abstract: Coal seam cavitation is one of the most effective techniques for gas disaster control in low-permeability coal. Due to the difference in cavitation method and process, the damage degree and fracture development range of the coal body around the cavern are greatly different, and the effect of increasing the permeability of the coal body is further changed. In order to further understand the permeability enhancement mechanism of cavitation technology on low-permeability coal and effectively guide engineering applications, this paper conducted experimental research on the unloading damage and permeability evolution characteristics of coal under different cavitation paths using a coal-rock “adsorption-percolation-mechanics” coupling test system. Through the analysis of coal strength and deformation characteristics, coal damage characteristics, and the evolution law of coal permeability combined with the macroscopic damage characteristics of coal, the strength degradation mechanism of unloaded coal and the mechanism of increased permeability and flow were revealed. The results show that unloading can significantly reduce the strength of coal, and the greater the unloading rate, the more obvious the reduction. The essence of this is that unloading reduces the cohesion and internal friction angle of coal—damage and breakage are the most effective ways to improve the permeability of the coal body. Unloading damaged coal bodies not only significantly improves the permeability of the coal body but also improves the diffusion ability of gas, and finally, shows a remarkable strengthening effect of gas extraction.

Keywords: cavitation; coal damage; gas extraction; increase permeability; unloading stress

1. Introduction

The annual production of nearly 4 billion tons of coal has strongly supported the rapid development of China’s economy in recent years [1,2]. However, the strong demand for coal has led to the extension of mining at an average rate of 10–30 m per year [3,4]. Many mines now have depths exceeding 800 m, particularly in regions such as Shandong, Anhui, Heilongjiang, and Henan, where some mines exceed 1000 m. As mining depth increases, the coal seam’s in situ stress, gas pressure, and gas content increase sharply while permeability decreases significantly [5,6]. This “three highs and one low” situation has caused more and more non-outburst mines to become coal and gas outburst mines, posing severe safety challenges to coal mining enterprises [7–10]. As one of the world’s largest producers and consumers of coal, China heavily invests each year in managing coal mine gases. Despite significant achievements, gas accidents remain one of the primary causes of major casualties in coal mines [11,12].

Practical experience shows that gas extraction is the most effective measure to solve coal mine gas accidents and achieve energy conservation and environmental protection. However, more than 95% of high-gas coal seams and protruding coal seams in China are low-permeability coal seams, with a permeability of only 10^{-4} – 10^{-3} mD, which is 3–4 orders of magnitude lower than that in the United States [13–15]. The widespread low-permeability coal seams in gas extraction present the problems of low extraction efficiency

and great difficulty in extraction. The high ground stress during deep digs into a coal mine can cause many permeability enhancement technologies applied in the shallow part of the mine to gradually fail [16–18]. The reason is that such measures have failed to effectively release the stress of the coal mass, resulting in the gradual closure of damaged cracks under the action of high-ground stress. Pressure relief and permeability enhancement technology is a method based on the relief of ground stress and the derivative cracks, which is a way to solve the low permeability problem of high-stress coal seams [19,20]. Some studies have proved that the relief of ground stress will be an effective means and a necessary path for managing gas disasters in deep coal seams [21,22].

The pressure relief and permeability enhancement technology mainly includes technologies such as mining protective layers, hydraulic cutting, and hydraulic cave-making. Among them, hydraulic cavitation technology has become more widely used for gas control in high-stress and low-permeability coal seams due to its simple operation and good pressure relief effect, coupled with continuous progress in equipment and theory [23–25]. In particular, mechanical cavitation technology has been successfully applied, and the defects of water jet coal destruction in submerged environments and hard coal seams have been overcome, leading to significant improvements in the efficiency of coal destruction and the successful application of cavitation technology in soft and hard coal [26,27]. Thus, the surrounding coal body undergoes the mechanical path of increasing vertical stress and decreasing horizontal stress because the essence of cavitation permeability enhancement is to relieve pressure and damage the coal mass around the cavern [28–30].

However, differences have gradually emerged between the methods and processes of cavitation (collectively referred to as cavitation paths) due to the differences between the coal destruction processes and techniques. For example, it is generally believed that under the conditions of the same water pressure and equipment, the diameter of the high-pressure water jet cavitation is positively correlated with the flushing time [31,32]. Under a specific cavitation diameter, the cavitation diameter can be gradually expanded by repeatedly pushing and pulling the drill rod for repeated flushing (as shown in Figure 1a), or the drill rod can be rotated for repeated flushing until the cavitation diameter reaches the target and then gradually advances forward (as shown in Figure 1b). The two water jet cavitation methods have different coal destruction sequences, resulting in different unloading paths for the coal mass around the cavern. Correspondingly, the mechanical cavitation technology cuts and breaks the coal with a mechanical knife. After the cavitation diameter is completed once, the length of the cavern is extended by pushing the drill rod, as shown in Figure 1c.

Compared with hydraulic coal destruction, mechanical coal destruction is highly efficient; that is, the coal mass around the cavitation experiences a higher unloading rate. In other words, different coal destruction efficiencies and coal destruction sequences cause the coal mass around the cavern to experience different unloading paths, which in turn have a significant difference in the degree of coal damage and the range of fracture development and ultimately show a significant effect of the cavitation method on the permeability enhancement effect of the coal mass.

Extensive research has been conducted on the damage of coal and rock during unloading confining pressure in recent years. Xue et al. [33] studied the effect of the unloading rate on the mechanical properties and permeability evolution (including energy evolution and fractal dimension) of coal and rock. Zhao et al. [34] studied the energy conversion and dissipation characteristics of rocks during different unloading confining pressure processes and obtained the effect of the unloading rate on both. Chen et al. [35] studied the effect of the unloading rate of confining pressure on the crack propagation characteristics of sandstone through experiments and believed that with the increase in the unloading rate of confining pressure, the crack volume expansion strain and crack propagation rate of sandstone would increase. Liang et al. [36] studied the post-peak mechanical characteristics and deformation characteristics of rock mass based on the demand for underground engineering support and believed that the rate of unloading confining pressure has a significant

effect on the bearing capacity and deformation characteristics of rock mass. Wang et al. [37] studied the effect of the unloading rate on coal damage and permeability during unloading confining pressure based on the stress state of coal mass during mining. Zhang et al. [38] established a theoretical model of permeability evolution of gas-bearing coal considering the change of loading and unloading rate based on previous research and verified the effectiveness of the model through experimental data. Chen et al. [39] studied the damage and permeability evolution characteristics of coal bodies under stress relief conditions based on the application background of mining protective layers. These research results have proved that unloading has a significant impact on the strength, deformation and permeability of coal and rock mass. They also proved theoretically that pressure relief has obvious significance for gas extraction but did not provide an explanation as to how pressure relief is more beneficial. In addition, they did not clearly explain the evolution law of local pressure relief damage of a coal body around cavitation and cannot effectively guide the application of cavitation technology.

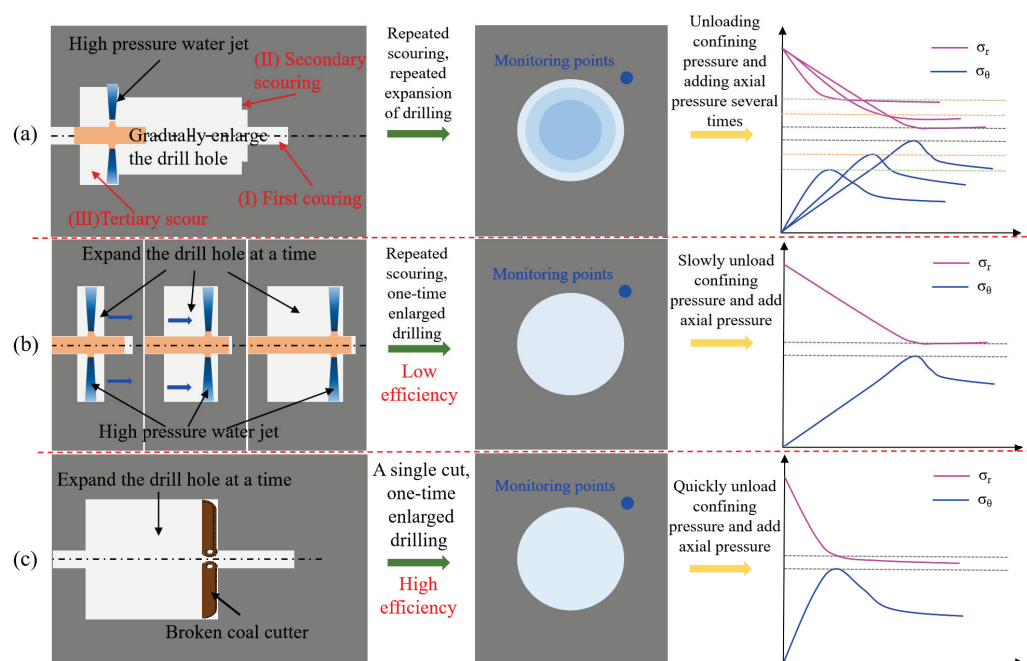


Figure 1. Schematic diagram of stress evolution of coal mass around the cavern with different hole expansion paths: (a) Hydraulic cavitation process-repeated scouring into holes; (b) Hydraulic cavitation process-one-time scouring into holes; (c) Mechanical cavitation process.

In order to further understand the permeability enhancement mechanism of cavitation technology on low-permeability coal and effectively guide engineering applications, this paper is based on the effect of different cavitation methods on the permeability enhancement effect of coal in field applications and condenses the idea of conducting different unloading process and unloading rate tests to study the effect of unloading behavior on coal damage and permeability enhancement effect. On this basis, the mechanism of cavitation paths' effect on the unloading permeability enhancement effect is revealed. The research results help strengthen the understanding of the mechanism of cavitation technology for coal body pressure relief and permeability increase. The stress evolution characteristics and permeability increase characteristics of coal after cavitation are clarified, which is very helpful for guiding site construction and improving gas extraction efficiency.

2. Materials and Equipment

2.1. Coal Sample Preparation and Experimental System

The coal samples were taken from the E₉₋₁₀ coal seam of Pingmei No. 8 Mine in Pingdingshan City, Henan Province. The coal seam thickness is 2.5~2.7 m, the structure

is uniform, and the Proctor coefficient is about 0.5. Large pieces of coal with a complete structure were selected from the underground site and sent to the laboratory for coal sample preparation. According to the standard methods for engineering rock tests, large coal blocks were processed into standard cylindrical specimens with dimensions of $\varnothing 50 \text{ mm} \times 100 \text{ mm}$. Intact and undamaged specimens were selected and encapsulated for testing, as shown in Figure 2a.

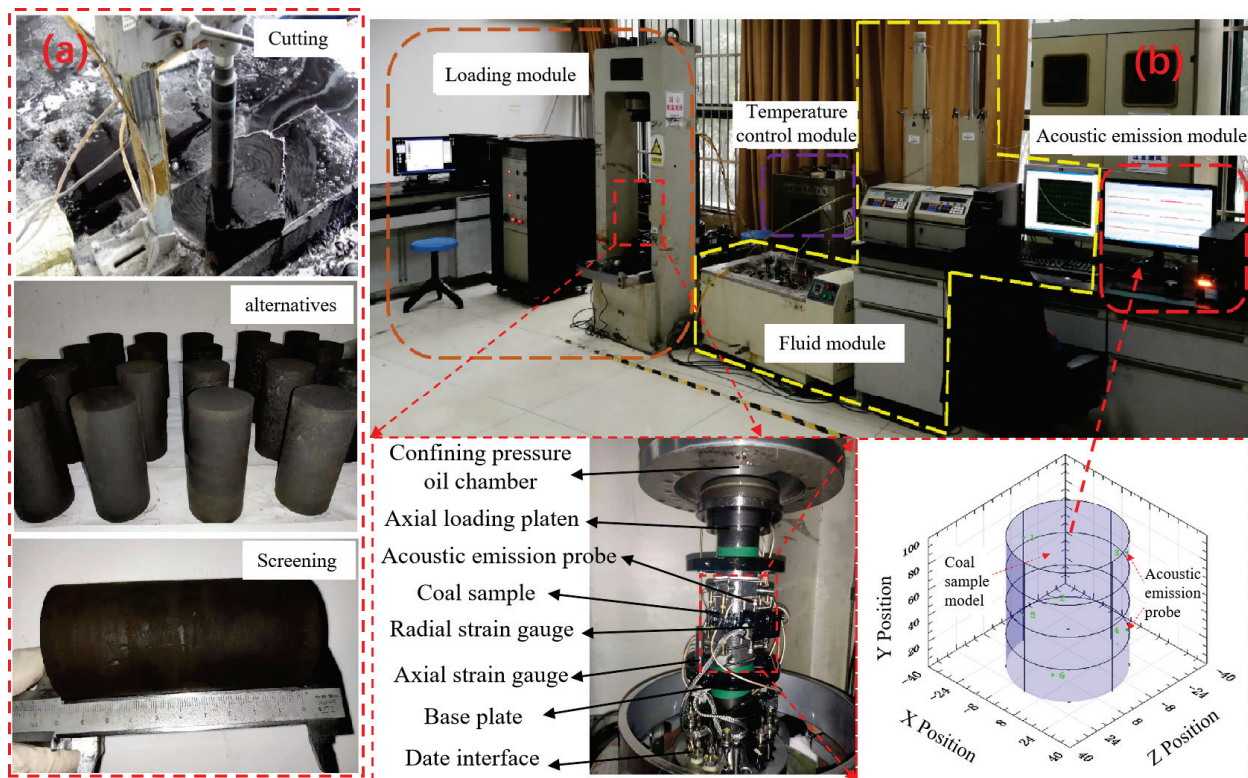


Figure 2. Experimental coal sample and test system: (a) Coal sample preparation process; (b) The test system.

The experimental system used a coal-rock “adsorption–percolation–mechanics” coupling test system, including mechanical testing, permeability testing, acoustic emission monitoring, and temperature control modules, as shown in Figure 2b. The mechanical testing module can perform conventional uniaxial and triaxial loading tests on standard specimens. The permeability testing module measures the permeability of standard coal samples under different stress states and temperatures. The acoustic emission monitoring module analyzes the damage characteristics of standard coal samples during mechanical tests. The system can apply axial loads up to 600 kN with control precision $\geq \pm 1\%$ and confining pressure up to 60 MPa with control precision $\geq \pm 1\%$. The strain gauge is series non-contact micro displacement sensor, which is an axial range of 25.0 mm, a radial range of 5.0 mm, and a linear error $\leq 0.25\%$. Permeability measurement uses the transient method with helium as the non-adsorptive gas source, and the metering pump’s maximum driving pressure is 40 MPa. The acoustic emission module uses the Micro-II acoustic emission acquisition system (Version number: TEST0000/2014) produced by MISTRAS Group, Inc. Products & Systems Division Princeton Junction, West Windsor Township, NJ, USA, including a 24-channel high-speed data acquisition system that receives acoustic signals at frequencies up to 0.1 μs and monitors parameters such as energy, amplitude, waveform, and count of acoustic emission signals. At the same time, according to the difference of signals received by multiple monitoring probes, the 3D location of the fracture in the sample is realized, and then the damage and fracture situation inside the sample and the macro failure characteristics are studied. In order to detect a weak break signal, the probe

and the sample are closely fitted, and the probe surface is coated with a coupling agent and then contacted with the sample wall and tightened by a strong spring ring.

Before the experiment, strain sensors and acoustic emission monitoring probes were fixed on standard coal samples. The strain sensors include axial and radial strain gauges, and acoustic emission monitoring probes are evenly arranged at both ends of the coal samples, with three probes at 120° intervals on the upper and lower ends. The prepared coal samples are fixed in the testing chamber to conduct mechanical and permeability tests.

2.2. Experimental Scheme

To study the effect of unloading paths on coal damage and permeability evolution characteristics, the following three types of experiments were designed: conventional tri-axial loading experiment (I), low-rate unloading confining pressure and loading experiment (II) and high-rate unloading confining pressure and loading experiment (III), respectively, to carry out the mechanical damage and permeability evolution experiments of coal samples under three mechanical paths. (I) The loading rate is 50 N/s, and the experiment concludes after the coal sample is damaged while loading continues for an additional 3~5 s. (II) After the confining pressure reaches the target value, the axial load is gradually increased. When the axial load is equal to the confining pressure ($\sigma_1 = 2\sigma_3$), the confining pressure is unloaded. To ensure that the axial pressure loading rate and the confining pressure unloading rate are the same, the axial pressure loading rate and the confining pressure unloading rate are 50 N/s and 25 N/s, respectively. The experiment concludes after the coal sample is damaged, with loading continuing for an additional 3~5 s. Experiment (III) has the same preparation stage as Experiment (II) but with an axial loading rate and confining pressure unloading rate of 100 N/s and 50 N/s, respectively. Similarly, the experiment ends 3 to 5 s after the coal sample is damaged. All three types of experiments are stress-controlled and are conducted with 5 MPa, 10 MPa, and 15 MPa confining pressures. During the tests, acoustic emission monitoring is used to detect damage signals in the coal samples. The specimen parameters and experimental conditions are listed in Table 1.

Table 1. Coal sample specifications and experimental parameters.

Test Classification	Coal Samples S.N.	Height/mm	Diameter/mm	Confining Pressure/MPa	Axial Pressure/KN	Loading Method
I	1-1	100.0	48.5	5	Load to damage	Constant confining pressure and axial pressure loading
	1-2	100.08	48.5	10		
	1-3	100.1	48.5	15		
II	2-1	100.1	48.6	5	Load to $\sigma_1 = 2\sigma_3$, then Unload confining pressure and load to damage	Confining pressure unloading at 25 N/s, Axial pressure loading at 50 N/s
	2-2	100.05	48.5	10		
	2-3	100.1	48.6	15		
III	3-1	100.1	48.5	5	Load to $\sigma_1 = 2\sigma_3$, then Unload confining pressure and load to damage	Confining pressure unloading at 50 N/s, Axial pressure loading at 100 N/s
	3-2	100.1	48.5	10		
	3-3	100.08	48.6	15		

3. Results and Analysis

3.1. Coal Strength and Deformation Characteristics of Different Unloading Paths

Different confining pressures are applied first, and then the loading tests are carried out on the coal samples as per the experimental scheme. During the loading process, the coal samples are continuously compressed and deformed until they become unstable and damaged. The relationship between the axial strain ε_1 , radial strain ε_2 , and volumetric strain ε_v of the coal samples and the stress is plotted, with the deviatoric stress ($\sigma_1 - \sigma_3$) as the ordinate and the strain as the abscissa, and the deviatoric stress–strain curve of the coal sample is obtained (where compression is a positive value and expansion is a negative value).

The stress–strain characteristics of the coal samples under different confining pressures at the same unloading rate are shown in Figure 3. Also shown are the conventional triaxial load-in, confining pressure unloading at 25 N/s, and loading and confining pressure unloading and loading at 50 N/s. As can be seen from Figure 3a, when the confining pressure is fixed at 5 MPa, 10 MPa, and 15 MPa, the peak deviatoric stress of the coal

samples is 21.99 MPa, 31.22 MPa, and 50.23 MPa, respectively, the axial strain (mm) is 0.01709, 0.05687, and 0.07262, respectively, the radial strain (mm) is -0.00932 , -0.03181 , and -0.05818 , respectively, and the volumetric strain (mm) is -0.00155 , 0.00675 , and -0.04373 , respectively. The results show that except for volumetric strain, which first increases and then decreases with the increase of confining pressure, the strength, axial strain, and radial strain of the coal samples exhibit a significant increasing trend. As shown in Figure 3b, for the initial confining pressures of 5 MPa, 10 MPa, and 15 MPa with a confining pressure unloading rate of 25 N/s, the peak deviatoric stresses of the coal samples are 12.88 MPa, 22.95 MPa, and 32.99 MPa, respectively. The axial strains (mm) are 0.01534, 0.02345, and 0.01723, respectively, the radial strains (mm) are -0.02206 , -0.03659 , and -0.00933 , respectively, and the volumetric strains (mm) are -0.02878 , 0.04974 , and -0.00142 , respectively. The overall trend indicates that as the initial confining pressure increases, the strength, axial strain, radial strain, and volumetric strain of the coal samples also increase. However, after the unloading point, the stress–strain curve shows a significant downward trend, indicating a notable decrease in the strength of the coal samples. As shown in Figure 3c, for initial confining pressures of 5 MPa, 10 MPa, and 15 MPa with a confining pressure unloading rate of 50 N/s, the peak deviatoric stresses of the coal samples are 10.01 MPa, 17.83 MPa, and 20.53 MPa, respectively. The axial strains (mm) are 0.01471, 0.02094, and 0.02717, respectively, the radial strains (mm) are -0.01505 , -0.01416 , and -0.02102 , respectively, and the volumetric strains (mm) are -0.0154 , 0.00738 , and -0.01488 , respectively. The patterns observed are similar to those in Figure 3b, but due to the higher unloading rate, the coal samples quickly reach their ultimate strength and damage after unloading. From Figure 3, it can be seen that regardless of whether unloading occurs, the ultimate damage strength of the coal samples significantly increases with increasing confining pressure. However, unloading significantly reduces the ultimate strength of the coal samples.

In order to further compare the effect of unloading rate on coal strength, the stress–strain characteristics of coal samples under the same confining pressure and different unloading rates were analyzed, as shown in Figure 4. It contains an initial confining pressure of 5 MPa, an initial confining pressure of 10 MPa, and a loading and initial confining pressure of 15 MPa. It can be seen from the figure that when the confining pressure is 5 MPa, the peak point deviatoric stress of the coal sample under the conditions of constant confining pressure, 25 N/s unloading confining pressure and 50 N/s unloading confining pressure are 21.99 MPa, 12.88 MPa and 10.01 MPa, respectively, which are 41.42% and 54.47% lower than the peak point strength of 5 MPa constant confining pressure; when the confining pressure is 10 MPa, the peak point deviatoric stress of the coal sample under the three unloading rates are 32.33 MPa, 22.95 MPa, and 17.83 MPa, respectively, which are 29.01% and 44.84% lower than the peak point strength of 10 MPa constant confining pressure; similarly, when the confining pressure is 15 MPa, the peak point deviatoric stress of the coal sample under the three unloading rates are 50.23 MPa, 32.99 MPa, and 20.53 MPa, respectively, which are 34.32% and 59.12% lower than the peak point strength of 15 MPa constant confining pressure. It shows that under different unloading rates, the ultimate strength of the coal sample decreases with the increase of the unloading rate, and the greater the unloading rate, the more obvious the attenuation of the ultimate strength of the coal sample. In addition, from the evolution trend of the stress–strain curve, it can be seen that the slope of the strain curve drops suddenly after the unloading point, and the sudden drop trend slows down with the increase of the confining pressure. The reason for the sudden drop in the slope of the strain curve is that the unloading causes the coal sample's ability to resist deformation to suddenly decrease. However, with the increase of the confining pressure, the confining pressure has a relatively high value at the beginning of unloading, and the coal sample still has a strong ability to resist deformation, but it is reduced compared with the constant confining pressure. The changes in the curves after the peak point are similar. The curves of the coal samples at different unloading rates basically fall rapidly after the peak point, especially under low confining pressure conditions. This

shows that under the condition of unloading confining pressure, the friction force in the fracture surface decreases significantly as the confining pressure decreases after the coal sample is damaged so that the damaged coal mass almost loses its bearing capacity. In general, when the unloading rate is large, the strain value required to destroy the coal mass is smaller, and the ultimate strength is lower.

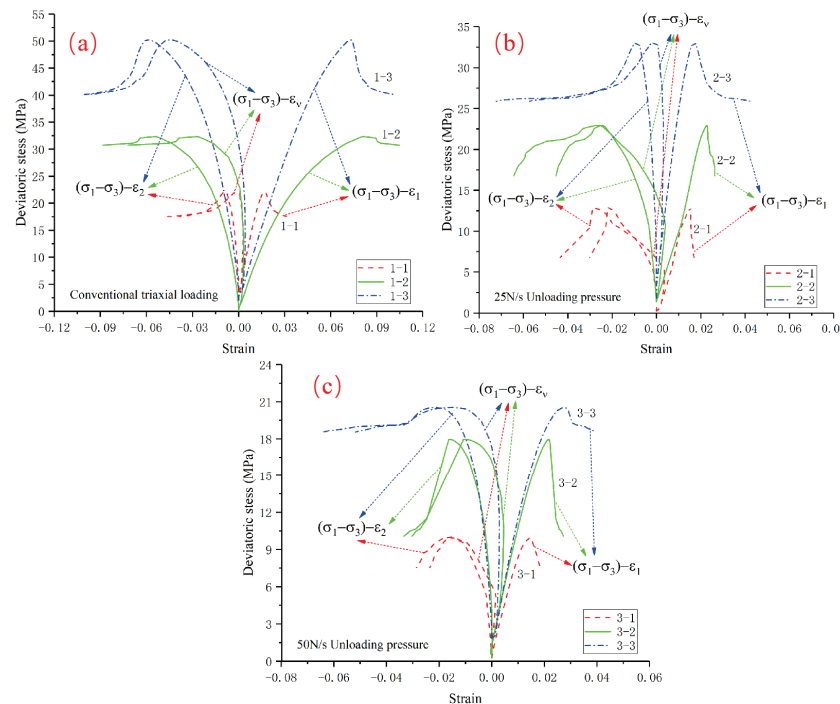


Figure 3. Stress–strain characteristics of coal under different confining pressures at the same unloading rate: (a) conventional triaxial loading; (b) confining pressure unloading at 25 N/s and loading; (c) confining pressure unloading at 50 N/s and loading.

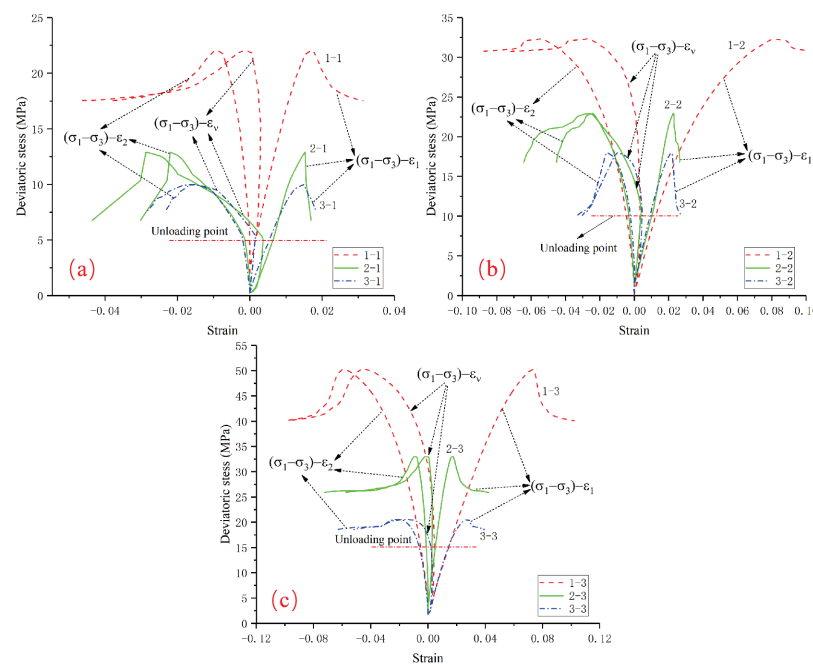


Figure 4. Stress–strain characteristics of coal under the same confining pressure and different unloading rates: (a) initial confining pressure of 5 MPa; (b) initial confining pressure of 10 MPa; (c) initial confining pressure of 15 MPa.

In order to further analyze the intrinsic mechanism of the effect of unloading rate and initial confining pressure on the damage and destruction of coal samples, the peak stress and initial confining pressure of coal samples under different unloading conditions were statistically analyzed, and the corresponding strength parameters were regressed and calculated. The fitting of peak stress and initial confining pressure data under different unloading behaviors is shown in Figure 5. The peak stress and initial confining pressure data under different unloading behaviors are fitted to obtain the slope and intercept of the lines using the least squares method. Based on the physical meaning of the slope and intercept of the lines and the Mohr–Coulomb strength criterion, the cohesion and internal friction angle of the coal samples for the corresponding paths can be calculated. According to the fitted data in Figure 5, it can be obtained:

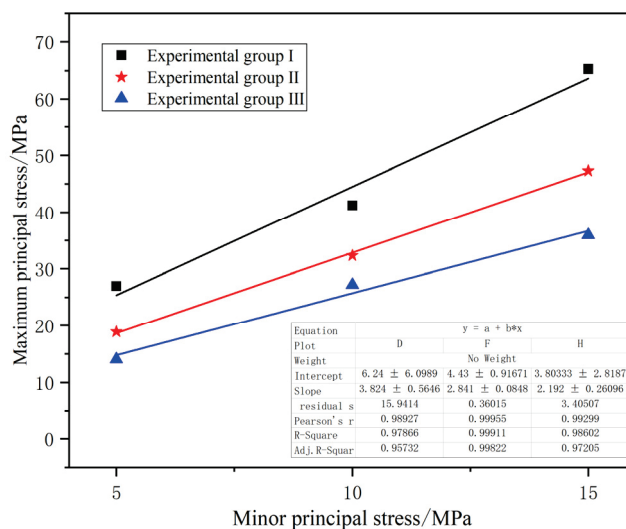


Figure 5. Changes of peak stress of coal samples under different initial confining pressures.

Under the condition of constant confining pressure and axial pressure loading, the relationship between the peak stress and initial confining pressure of the coal sample is:

$$\sigma_1 = 6.240 + 3.824\sigma_3 \quad (1)$$

When confining pressure unloading at (25 N/s) and axial pressure loading to the coal sample, the relationship between the peak stress and the initial confining pressure is:

$$\sigma_1 = 4.430 + 2.841\sigma_3 \quad (2)$$

When confining pressure (50 N/s) unloading and axial pressure loading to the coal sample, the relationship between the peak stress and the initial confining pressure is:

$$\sigma_1 = 3.803 + 2.192\sigma_3 \quad (3)$$

The linear regression results and strength parameters of coal sample experimental data under different unloading conditions are shown in Table 2. It can be seen from the table that the fitting accuracy of the three groups of data is high. The results show that compared with the constant confining pressure, the cohesion and internal friction angle calculated for the coal sample under unloading confining pressure is smaller. According to the Mohr–Coulomb criterion, it can be known that internal friction and cohesion constitute the shear strength of the coal mass. Therefore, the cohesion and internal friction angle comprehensively reflect the strength characteristics of the coal mass. It shows that the mechanical essence of unloading confining pressure making the coal mass more easily damaged and destroyed is that unloading confining pressure reduces the cohesion and internal

friction angle of the coal mass, and the cohesion and internal friction angle of the coal mass decreases more obviously with the increase of the unloading rate of confining pressure.

Table 2. Calculation results of strength parameters of different unloading conditions.

Stress Path	Intercept	Slope	Cohesion/MPa	Internal Friction Angle/°	R ²
Constant confining pressure and axial pressure loading	6.240	3.824	1.595	35.83	0.978
Confining pressure unloading at 25 N/s and axial pressure loading	4.430	2.841	1.314	28.64	0.999
Confining pressure unloading at 50 N/s and axial pressure loading	3.803	2.192	1.284	21.93	0.986

3.2. Coal Damage Characteristics in Different Unloading Paths

In order to study the effect of different unloading paths on the damage characteristics of coal mass during destruction, acoustic emission (AE) was used to monitor the AE energy and AE count of the coal sample destruction process. Where AE energy refers to the acoustic emission energy detected during the experiment, and AE count refers to the number of acoustic emissions during the experiment. Both reflect the damage situation inside the coal sample monitored at a certain moment during the test, while AE cumulative energy and AE cumulative count reflect the cumulative damage situation inside the coal sample before a certain moment.

The acoustic emission signals of coal samples obtained during the mechanical experiment were processed and analyzed to obtain the acoustic emission AE counts (as shown in Figure 6) and AE energy (as shown in Figure 7) of coal samples under different unloading paths. The number of experimental coal samples corresponds to the number in the picture. As can be seen from Figure 6, the AE cumulative counts of all coal samples during the mechanical experiment showed a trend of first slowly increasing, then sharply increasing, and finally steadily increasing. Combined with the axial stress curve of the coal sample, it was found that the AE count increased slowly during the compression deformation stage, and the AE count increased sharply before the coal sample was damaged until the AE count incidence rate decreased after the coal sample was damaged and a stable increase appeared. In addition, by comparing the acoustic emission AE counts of coal samples with different unloading paths, the AE counts of coal samples loaded under constant confining pressure increased more slowly, and the AE counts of coal samples under confining pressure unloaded at 25 N/s and 50 N/s increased faster, and the time for the coal samples to reach destruction due to unloading was faster. For example, when the initial confining pressure was 5 MPa, the time for the coal samples to reach destruction under constant confining pressure, 25 N/s unloading confining pressure, and 50 N/s unloading confining pressure was 1200 s, 300 s, and 200 s, respectively. In addition, the final AE cumulative counts of coal samples under different unloading paths have no obvious regularity, which may be due to the heterogeneity of the coal sample itself.

As can be seen from Figure 7, the AE energy of the coal sample is basically the same as the AE counting law, but the AE cumulative energy has obvious regularity. Under the same initial confining pressure, the AE cumulative energy decreases with the increase of the unloading rate when the coal sample is damaged. It is confirmed that unloading reduces the energy required for damage and destruction of the coal sample, and the greater the unloading rate, the more obvious the reduction.

In order to quantitatively analyze the damage characteristics of coal samples under different unloading paths, the AE cumulative counts and AE cumulative energies of each group of coal samples were statistically analyzed, as shown in Figure 8. Under the same unloading path, the AE cumulative counts and AE cumulative energies of coal samples both showed an increasing trend with the increase of confining pressure. Under the same

initial confining pressure, except for a few discrete points, the cumulative AE (Acoustic Emission) counts and cumulative AE energy of the coal samples generally show a pattern where constant confining pressure is greater than that at 25 N/s unloading rate and greater than that at 50 N/s unloading rate. This proves that unloading conditions can reduce the strength of the coal, and the greater the unloading rate, the more significant the strength reduction.

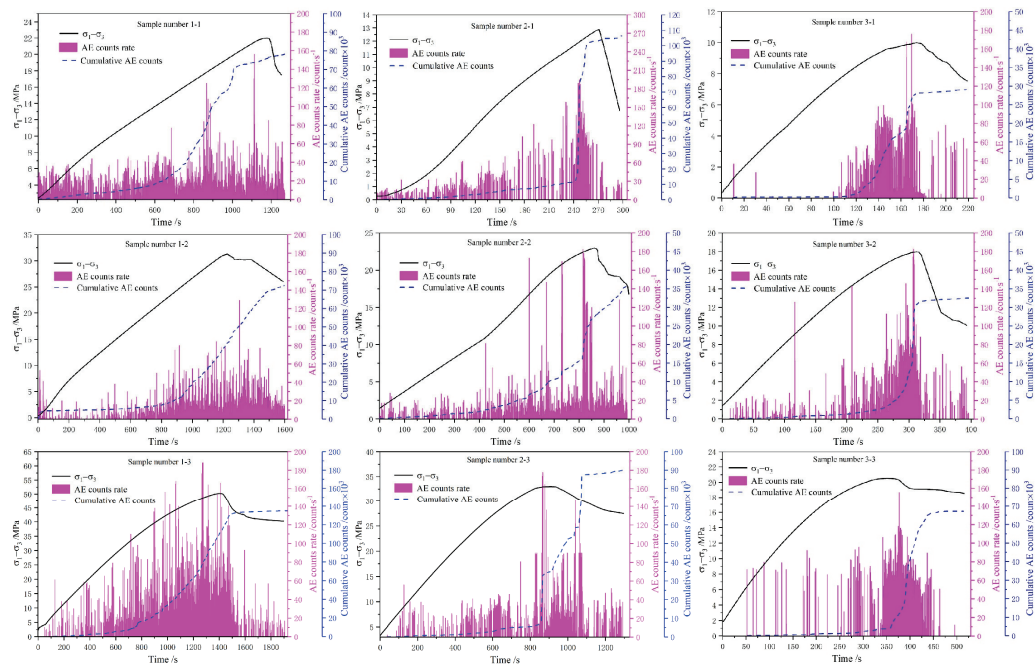


Figure 6. Characteristics of acoustic emission signals (AE counts) of coal samples in different unloading paths.

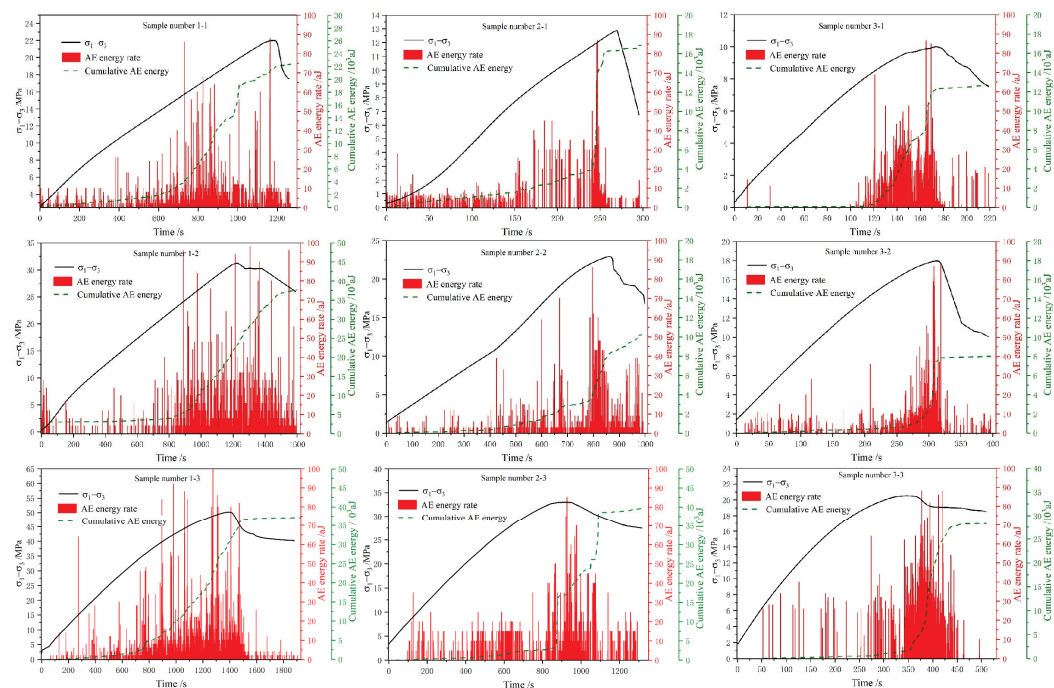


Figure 7. Characteristics of acoustic emission signals (AE energy) of coal samples in different unloading paths.

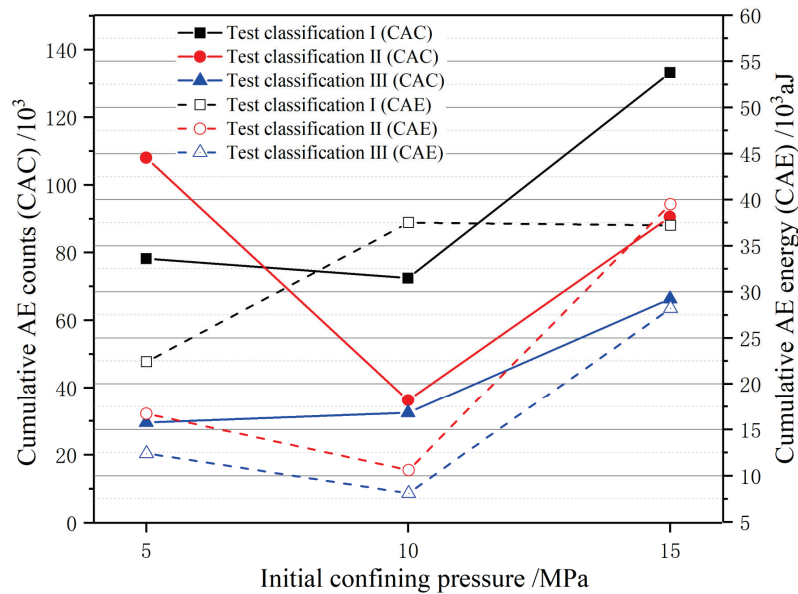


Figure 8. Comparison of coal acoustic emission data in different unloading paths.

3.3. Effect of Different Unloading Behaviors on Coal Permeability Evolution

In order to obtain the effect of different unloading paths on the damage and permeability enhancement of coal, the permeability of the coal samples was measured during the experiment. Before the coal samples reached their peak strength, pulse gas was released at randomly selected test points to measure the permeability of the coal samples. After the coal samples reached their peak strength, one or more measurements were continued according to their bearing capacity. The permeability measurement results of each group of coal samples were matched with the stress–strain curve, as shown in Figure 9. The number of experimental coal samples corresponds to the number in the picture. The initial permeability of the coal samples and the permeability at the time of damage are shown in Table 3. It can be seen from the experimental results that during the full stress–strain process of the coal samples, the permeability first decreased and then increased. For example, when the initial confining pressure is 5 MPa, the initial permeabilities of the coal samples under constant confining pressure, 25 N/s unloading confining pressure, and 50 N/s unloading confining pressure are 0.064 mD, 0.042 mD, and 0.046 mD, respectively. The minimum permeabilities of the coal samples measured during the loading process are 0.003 mD, 0.0065 mD, and 0.0018 mD, respectively. As the accumulated permeability of the coal samples due to plastic damage gradually increases, the permeability shows a jump growth after reaching the peak stress point. The permeabilities at the time of instability and damage are 7.625 mD, 12.953 mD, and 15.668 mD, respectively.

Table 3. Sudden increases in permeability of coal mass after damage under different unloading behaviors.

Test Classification	Sample Serial Number	Confining Pressure/MPa	Initial Value/mD	Damage Value/mD	Sudden Increase by Times
I	1-1	5	0.064	7.625	119.1
	1-2	10	0.022	1.655	75.2
	1-3	15	0.0082	0.712	86.8
II	2-1	5	0.042	12.953	308.4
	2-2	10	0.028	7.632	272.6
	2-3	15	0.0063	1.153	183
III	3-1	5	0.046	15.668	340.6
	3-2	10	0.036	11.336	314.9
	3-3	15	0.00506	1.735	342.9

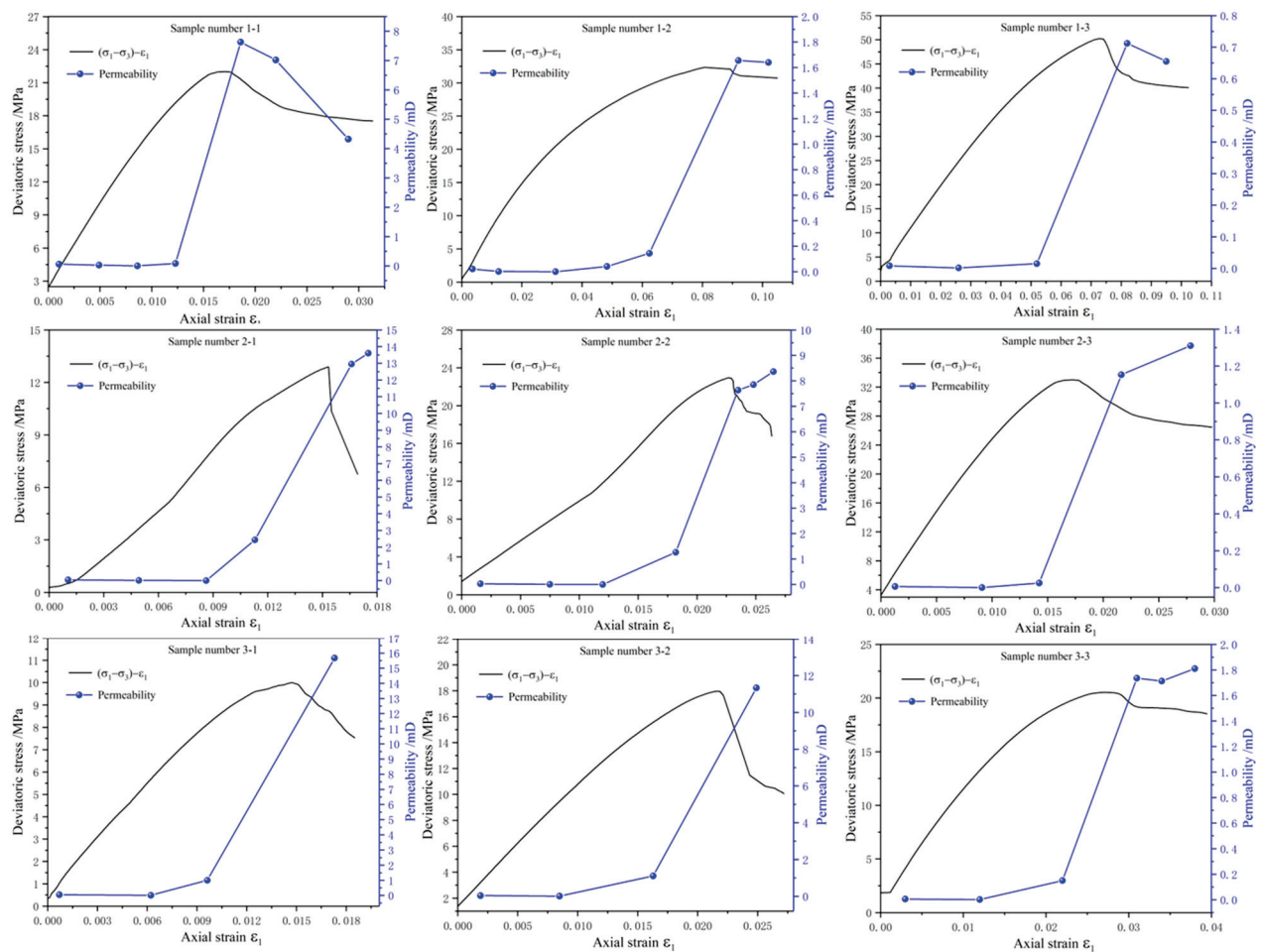


Figure 9. Permeability evolution of coal mass during full stress–strain process under different unloading behaviors.

Similarly, when the initial confining pressure is 10 MPa and 15 MPa, the permeability of the coal samples changes in the same way, and they all experience a process of first decreasing and then increasing. This is because at the beginning of the experiment, the load on the coal sample increased, and the internal cracks were compressed, resulting in a decrease in permeability. As the load increased, the coal sample gradually produced plastic strain and damage, the internal cracks of the coal sample increased, and the permeability began to rise slowly; when the strength limit was reached, the coal sample was damaged, macro cracks were generated, and the permeability suddenly increased. In addition, under the condition of constant confining pressure, the permeability of the coal sample showed a downward trend after the peak point. This is because the existence of confining pressure limits the continued increase of macro damage cracks and instead gradually shrinks under the action of axial load. This shows that high stress leads to a significant decrease in coal permeability, and damage is the most effective way to increase permeability.

As can be seen from Table 3, when the confining pressure is 5 MPa, 10 MPa, and 15 MPa, the permeability of the coal sample after instability and damage in the constant confining pressure experiment increased by 119.1 times, 75.2 times, and 86.8 times, respectively, compared with the initial state; the permeability of the coal sample after instability and damage in the 25 N/s unloading confining pressure experiment increased by 308.4 times, 272.6 times, and 183 times, respectively; the permeability of the coal sample after instability and damage in the 50 N/s unloading confining pressure experiment increased by 340.6 times, 314.9 times, and 342.9 times, respectively. It reflects that under constant confining pressure or a low unloading rate, the permeability enhancement effect after coal mass

damage is significantly affected by the initial stress. The initial stress is relatively large, and the permeability enhancement effect is relatively low, but the unloading condition can significantly reduce the effect of the initial stress on the permeability enhancement effect. This may be because the cracks after coal mass damage at a high unloading rate will not reclose under stress. It can be inferred that unloading damage can significantly improve the permeability of low-permeability coal seams, and the greater the unloading rate, the better the permeability enhancement effect.

3.4. Macroscopic Damage Characteristics of Coal Mass under Different Unloading Behaviors

The macroscopic damage mode of coal samples is influenced by their physical and chemical properties and the mechanical loading pattern. Numerous experiments have confirmed that coal rock under confining pressure conditions will only undergo shear damage, whereas, under uniaxial compression conditions, it will undergo tensile damage along the axial direction, which is more complex in form. Comparing the macroscopic damage modes of coal samples under different mechanical paths helps to gain a deeper understanding of the influence of mechanical paths on the damage characteristics of coal. The macroscopic damage modes of the coal samples from each group after the experiments are shown in Figure 10. The number of experimental coal samples corresponds to the number in the picture.

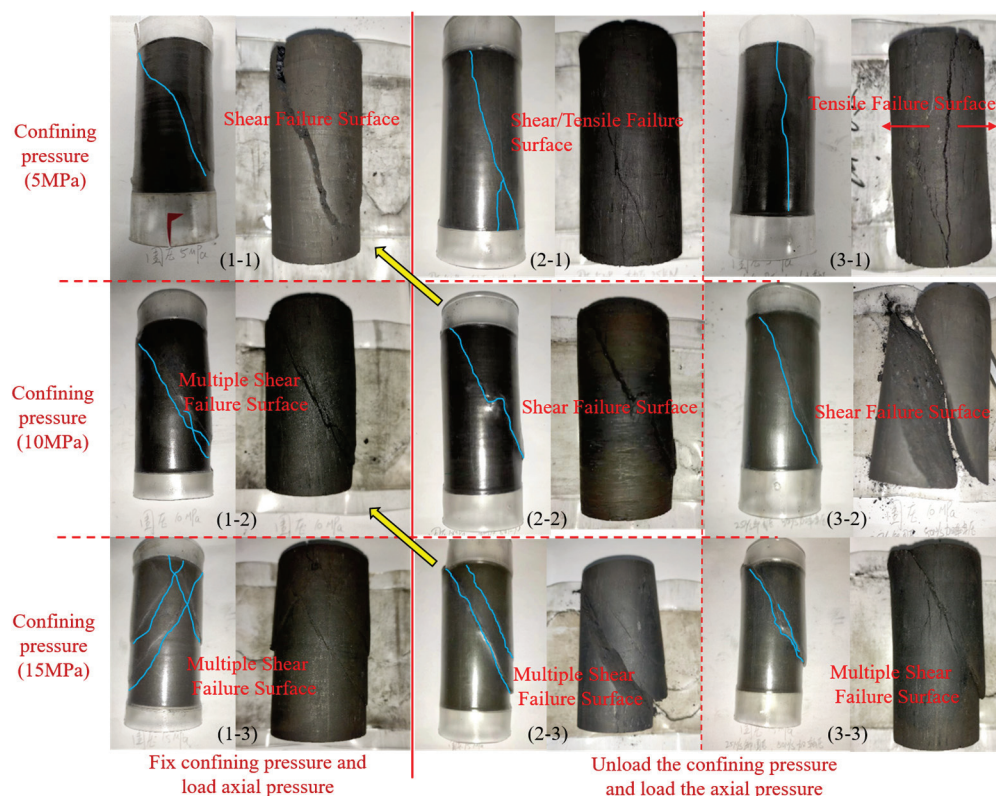


Figure 10. Macroscopic damage and destruction characteristics of coal under different unloading behaviors.

As can be seen from the figure, under the constant confining pressure and axial pressure mechanical path, the coal sample damage is all in the form of shear damage. When the confining pressure is 5 MPa, the coal sample has only one macroscopic fracture surface. When the confining pressure increases to 10 MPa, in addition to a main shear fracture surface, the coal sample also produces multiple associated small fracture surfaces around it; that is, the coal sample begins to show multiple damage. When the confining pressure is 15 MPa, the coal sample produces multiple independent macroscopic fracture

surfaces. This shows that when the confining pressure is further increased, the damage form of the coal sample is obviously different from that at low confining pressure. When the high confining pressure coal sample is damaged, it presents typical multiple shear fractures. This is because of the significant confining pressure that creates higher levels of friction on both sides of the coal sample fracture surface, which in turn leads to secondary fracture surfaces or independent fracture surfaces on both sides of the fracture surface.

There is also a certain regularity in the damage of coal samples under unloading confining pressure conditions. Under the conditions of 5 MPa confining pressure, the coal sample damage forms shear damage and tensile damage when the confining pressure is unloaded at 25 N/s, while the form of coal sample damage is tensile damage that only occurs under uniaxial compression when the confining pressure is unloaded at 50 N/s. Under the conditions of 10 MPa confining pressure and 15 MPa confining pressure, the damage forms of coal samples under 25 N/s unloading confining pressure and 50 N/s unloading confining pressure are basically consistent with those under constant confining pressure of 5 MPa and 10 MPa, respectively. The macroscopic damage forms of coal samples show that the damage forms of coal samples under the mechanical path of unloading confining pressure and loading axial pressure are downgraded compared with conventional triaxial loading; that is, the damage forms of coal samples under unloading confining pressure and loading axial pressure are similar to those of conventional triaxial under a lower level of constant confining pressure-loading.

3.5. Discussion

According to the results of the mechanical strength characteristics analysis of coal samples under different unloading paths, the coal damage and destruction criterion can be understood from the perspective of stress Mohr's circle as follows (as shown in Figure 11): under the path of constant confining pressure and loading axial pressure, the minimum principal stress remains unchanged while the maximum principal stress continues to increase. When the stress circle formed by the maximum principal stress and the minimum principal stress is tangent to the strength curve, the coal is damaged, while the intercept and slope of the strength curve formed under the path of unloading confining pressure and loading axial pressure are smaller than those under the path of constant confining pressure and loading axial pressure, and the minimum principal stress continues to decrease due to the unloading of the confining pressure. As the maximum principal stress increases, the strength of the coal is quickly reached; that is, the stress circles formed by the blue dotted line and the pink dotted line are tangent to the corresponding strength curve. In addition, as the unloading rate increases, the coal is more likely to reach the ultimate strength. It can be explained that the unloading behavior is equivalent to changing the mechanical properties of the coal and reducing its strength.

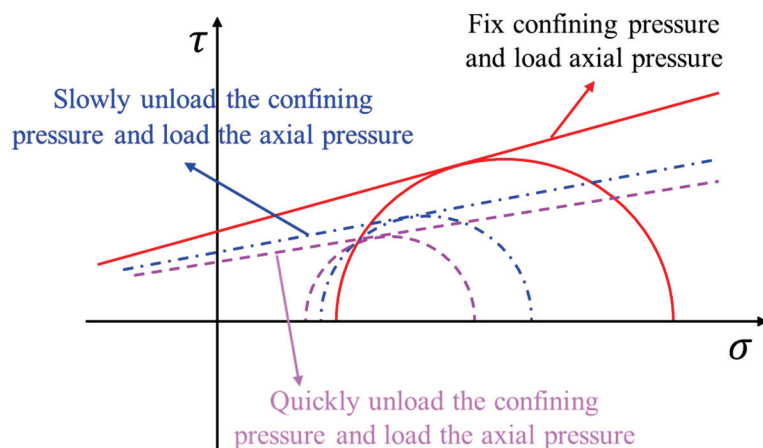


Figure 11. Coal damage and destruction mechanism under different unloading conditions.

The permeability evolution during the full stress–strain process of the coal sample shows a law of first decreasing and then increasing. The stress–strain and permeability evolution trends during the experiment are simplified and combined with the acoustic emission monitoring results, which can be described as shown in Figure 12. The acoustic emission signal shows that the coal sample damage is a cumulative process, and when the damage exceeds the influence of fracture compression, the seepage gradually increases. As shown in the figure, during the experiment, as the load on the coal sample increases, the interior is gradually compacted. The stress–strain evolution of this process corresponds to the AC stage in the figure, showing that the coal sample is compressed but has not been damaged. The compression crack causes the permeability to gradually decrease. The permeability evolution corresponds to the EF stage in the figure. Afterward, the coal sample enters the plastic deformation stage, and the coal sample begins to produce plastic damage, but the coal sample at this time does not show macroscopic damage and still has good integrity. The stress–strain evolution corresponds to the CD stage in the figure. The gradual increase in new cracks causes the permeability of the coal sample to rebound and gradually increases with the accumulation of damage. The permeability evolution is shown in the FH stage of the figure. When the pressure on the coal sample reaches the limit, instability and damage occur, and the permeability of the coal sample increases in a jump-like manner after the peak point, as shown in the HG stage in the figure. The main reason for the jump-like increase in permeability at this stage is that macro cracks appeared in the coal sample, and the unloading damage after the peak point caused the cracks between the coal matrix to expand further.

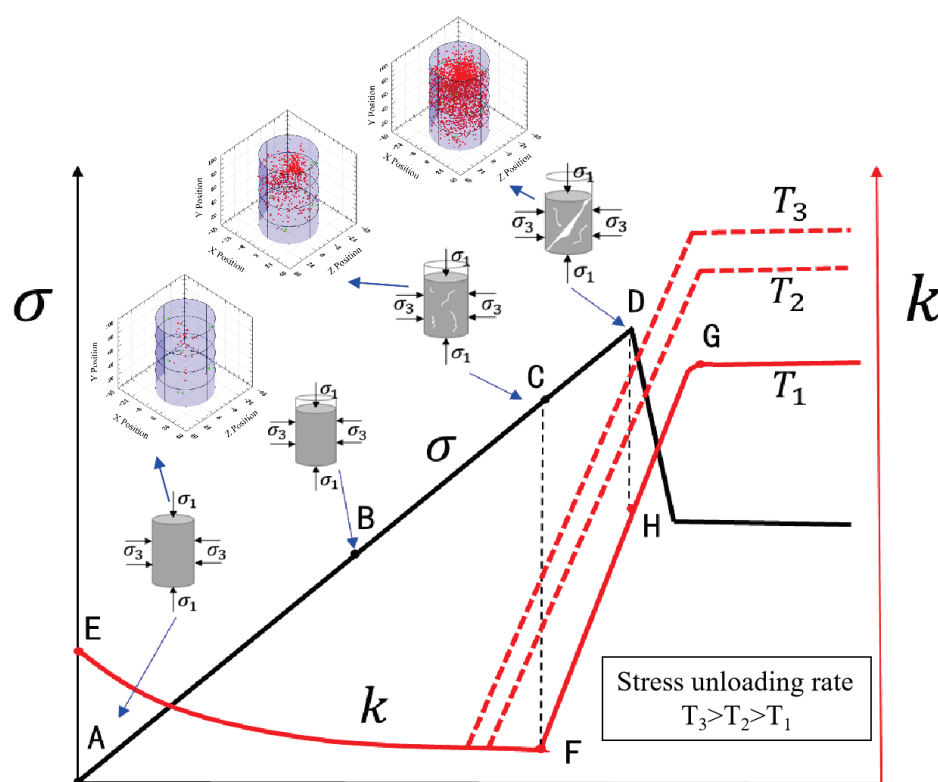


Figure 12. Permeability evolution model of coal mass during full stress–strain process under different unloading behaviors.

According to the experimental results, the permeability will increase in a jump-like manner as the coal mass is damaged during the constant confining pressure test, but the permeability will increase earlier during the unloading confining pressure test, and the permeability will be greater after the coal mass is broken. In addition, the permeability increases more with the increase in the unloading rate. In the figure, T1 is the constant

confining pressure loading process, T2 and T3 are the unloading confining pressure loading processes, but the unloading rate of T3 is greater.

As a typical dual-porosity medium, the cubic structure model established by simplifying the fracture network and coal mechanism has been widely recognized and applied. Combined with the study of the mechanical permeability evolution law of coal samples, the mechanism of coal unloading damage permeability enhancement can be described as follows (as shown in Figure 13): unloading increases the crack opening between the coal matrix units and increases the gas flow channels in the cracks, which results in an increase in the permeability of the coal. At the same time, the reduction in confining pressure greatly reduces the ultimate strength of the coal, making the coal more susceptible to damage and destruction. The continuous damage causes the internal matrix of the coal to be further divided into smaller matrix units by the new cracks, and the diffusion path of methane molecules adsorbed in the coal matrix into the cracks becomes smaller; that is, the desorbed methane molecules are more likely to enter the cracks, showing an increase in the diffusion capacity of the coal. Therefore, under the action of coal unloading and damage, the flow enhancement effect of gas extraction is achieved, which reduces the difficulty of coal seam gas extraction and improves the extraction efficiency.

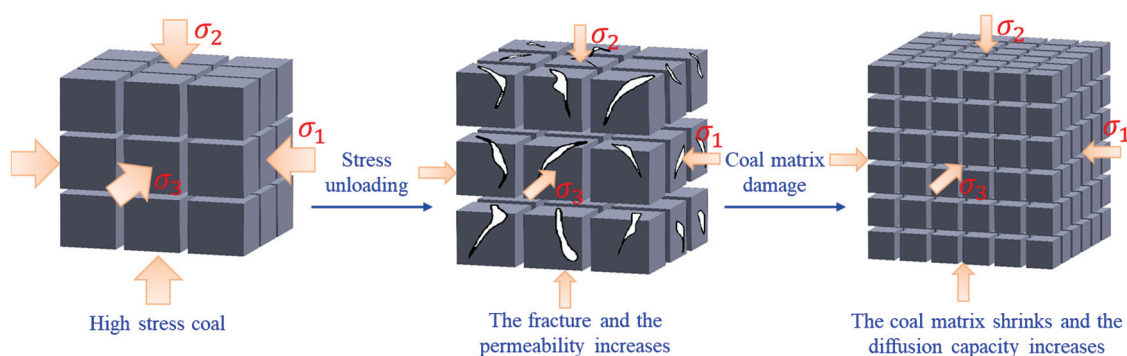


Figure 13. The path of permeability increase caused by pressure relief and damage of coal-mass.

4. Conclusions

In order to explore the effect of the unloading path on coal damage and permeability evolution, experiments with different loading methods and unloading rates were designed. The strength, deformation, and damage characteristics of coal and the evolution of coal permeability under different unloading paths were studied. Combined with the macroscopic damage characteristics of coal, the strength degradation mechanism of unloaded coal and the mechanism of increased permeability and flow were revealed. The research results further clarify the permeability enhancement mechanism of low-permeability coal seam cavitation technology and provide help in guiding on-site engineering practice. The main conclusions are as follows:

- (1) When the initial confining pressure was 5 MPa, 10 MPa, and 15 MPa, the peak differential stress of the coal samples in the constant confining pressure test was 21.99 MPa, 32.33 MPa, and 50.23 MPa, respectively. The peak differential stress of the coal samples in the unloading confining pressure (25 N/s) test was 41.4%, 29.0%, and 34.3% of that in the constant confining pressure and loading axial pressure path, respectively, indicating that the unloading confining pressure significantly reduced the strength of the coal mass. The peak differential stress of the coal samples in the 50 N/s unloading confining pressure test was 77.7%, 77.6%, and 62.2% of that in the 25 N/s unloading confining pressure test, respectively, indicating that the higher the unloading rate, the smaller the peak strength when the coal mass is damaged, that is, the high unloading rate is more likely to cause the destruction of the coal mass. The essence is that the unloading confining pressure reduces the cohesion and internal friction angle of the coal mass.

- (2) The results of acoustic emission monitoring show that the statistics of AE cumulative counts and AE cumulative energy show that the greater the unloading rate of confining pressure, the smaller the energy required for coal sample destruction, and the process of mechanical destruction of coal mass is the result of continuous accumulation of internal damage. This reflects that the unloading environment can significantly reduce the strength of coal mass, and the greater the unloading rate, the more obvious the strength reduction.
- (3) Permeability experiments show that at the initial confining pressures of 5 MPa, 10 MPa, and 15 MPa, the permeability of coal samples under constant confining pressure tests increased by 119.1 times, 75.2 times, and 86.8 times, respectively, before and after damage. For the unloading confining pressure (25 N/s) experiments, the permeability of the coal samples increased by 308.4 times, 272.6 times, and 183 times, respectively, before and after damage. This indicates that the unloading path results in a more significant increase in permeability after coal damage. For the unloading confining pressure experiments at 50 N/s, the permeability of the coal samples increased by 340.6 times, 314.9 times, and 342.9 times, respectively, before and after damage. This shows that the higher the unloading rate, the more significant the increase in permeability after coal damage. Essentially, the unloading damage to the coal reduces the paths for adsorbed gas to diffuse into the fractures, thereby increasing the channels for gas flow.
- (4) The results show that the coal mass is more easily damaged and has a better permeability increase under high-speed pressure relief. In engineering practice, mechanical cavitation technology or hydraulic cavitation with one reaming method should be preferred, as it will obtain better gas extraction efficiency under the same cavitation chamber and lower the risk of coal after pressure relief.

Author Contributions: Investigation, G.L.; writing—original draft, C.H.; writing—review & editing, Y.W. All authors have read and agreed to the published version of the manuscript.

Funding: The authors are grateful to the National Natural Science Foundation of China (52204211) and the Fundamental Research Funds of China Academy of Safety Science and Technology (2023JBKY13).

Data Availability Statement: The data that support the findings of this study are available from the corresponding author upon reasonable request.

Conflicts of Interest: The authors declare that we do not have any commercial or associative interests that represent a conflict of interest in connection with the work submitted.

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Article

Study on the Damage Mechanism of Coal under Hydraulic Load

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Abstract: Hydraulic fracturing is extensively utilized for the prevention and control of gas outbursts and rockbursts in the deep sections of coal mines. The determination of fracturing construction parameters based on the coal seam conditions and stress environments merits further investigation. This paper constructs a damage analysis model for coal under hydraulic loads, factoring in the influence of the intermediate principal stress, grounded in the octahedron strength theory analysis approach. It deduces the theoretical analytical equation for the damage distribution of a coal medium subjected to small-flow-rate hydraulic fracturing in underground coal mines. Laboratory experiments yielded the mechanical parameters of coal in the study area and facilitated the fitting of the intermediate principal stress coefficient. Leveraging these datasets, the study probes into the interaction between hydraulic loads and damage radius under assorted influence ranges, porosity, far-field crustal stresses, and brittle damage coefficients. The findings underscore that hydraulic load escalates exponentially with the damage radius. Within the variable range of geological conditions in the test area, the effects of varying influence range, porosity level, far-field stress, and brittle damage coefficient on the outcomes intensify one by one; a larger hydraulic load diminishes the impact of far-field stress variations on the damage radius, inversely to the influence range, porosity, and brittle damage. The damage radius derived through the gas pressure reduction method in field applications corroborates the theoretical calculations, affirming the precision of the theoretical model. These findings render pivotal guidance for the design and efficacy assessment of small-scale hydraulic fracturing in underground coal mines.

Keywords: hydraulic fracturing; permeability enhancement; three-parameter failure criterion; intermediate principal stress; damage model; porosity

1. Introduction

As China's shallow coal resources gradually deplete, the depth of underground mining increases, posing significant challenges for an increasing number of mines faced with deep coal mining issues [1–4]. The original state of in situ stress in coal seams is disrupted by various mining engineering disturbances, manifesting as various forms of stress loading and unloading [5–8]. Currently, strength analyses of coal–rock mass failure primarily utilize the Mohr–Coulomb strength criterion, which ignores the impact of the intermediate principal stress on material strength and is inadequate for describing the strength failure of the underground coal–rock medium under true triaxial stress conditions [9]. Shi [10] introduced the intermediate principal stress coefficient β into the Hoek–Brown criterion, improving the two-dimensional strength criterion to a three-dimensional one. The revised strength criterion has been verified by experiments and compared with three other strength criteria. Zhang [11] established a unified solution for the size of the plastic zone and the stress field before translational impact instability, based on the Unified Strength Theory, finding significant effects of changes in the intermediate principal stress coefficient on

coal seam translational impact instability. Adam [12] innovated a generalized method that transforms two-dimensional failure criteria (i.e., Mohr–Coulomb and Hoek–Brown failure criteria) into three-dimensional criteria to predict true triaxial experimental data. Du [13] demonstrated through experiments that the twin-shear unified strength criterion can accurately describe the strength characteristics of sandstone under different stress paths. Yin [14] believes that the loading or unloading of the intermediate principal stress has a significant impact on the evolution of the original coal permeability, which cannot be ignored in practical conditions. In the high-stress environment of deep coal mining, disasters such as coal and gas outbursts and rockbursts are more likely to occur. Without effective preventive measures, these can significantly impact production and severely limit capacity release [15–17]. Hydraulic fracturing [18–23] and hydraulic slotting [24–28] measures have been applied to more mines and mining group companies, and have proven to be applicable to more coal production enterprises in terms of environmental protection and low risk. Hydraulic fracturing [29,30] is a technique that involves the continuous injection of fluid into the coal–rock medium to induce fractures. This process not only enhances the permeability of the coal–rock medium to facilitate gas extraction but also can cause the fracturing of the overburden to reduce the likelihood of rock bursts. Since the first hydraulic fracturing operation in the United States in 1947, the technology has undergone significant advancements over more than 50 years, evolving from theoretical research to practical field applications.

Due to the unique underground three-dimensional spatial environment of coal seams, the hydraulic measures employed cannot solely focus on increasing hydraulic pressure to enhance regional coal permeability. However, how to reasonably set and match the hydraulic pressure value according to the occurrence of coal and its own properties can currently only be determined based on experiments in the mine, with the determination method having a certain randomness and lag, and thus not providing good preliminary guidance for hydraulic fracturing. Since the coal medium will be damaged and destroyed by external loading, Liu [31] used the Gurson damage model to establish a continuous damage model for hydraulic fracturing, with numerical simulation results showing that the damage zone is mainly concentrated near the perforation, presenting a band-like distribution. Cai [32] conducted numerical simulations of hydraulic fracturing using the multi-porous medium control equation and the cohesive damage model, which were consistent with the observed field influence range. Yi [33] defined a damage variable influenced by the maximum normal stress, with coal permeability changing with damage, and established a fully coupled seepage-damage numerical model. In the field of water inrush from shaft lining, Rong [34] based on the Unified Strength Theory and the Bui elastoplastic damage model, derived the internal stress expression of shaft lining and discussed the impact of porosity and modulus reduction ratio on the critical inrush pressure. Guo [35,36] established a damage constitutive model based on energy dissipation to study the effects of gas pressure on the mechanical properties, permeability, and energy response characteristics of coal–rock composites under true triaxial stress paths. Wei [37] derived the stress equation of rock microelements in the damage and failure zones without ignoring shear stress. Studies on water inrush from both shaft lining and hydraulic fracturing involve fluid–solid coupling problems under loading conditions on circular holes, but they have completely different liquid and solid phase boundary conditions, and therefore the research objects have significant differences in internal stress fields and damage deformation laws.

Previous studies have primarily focused on analyzing the initiation of hydraulic fracturing from the perspective of fracture mechanics, or on qualitatively describing the propagation patterns of fractures. This paper derives the relationship between the damage radius of coal and the hydraulic load considering the intermediate principal stress from the perspective of coal body damage, and provides a theoretical analytical formula to guide field practice. We conducted uniaxial compression tests, triaxial compression experiments, and mercury penetration experiments to determine the mechanical parameters of the coal medium, and substituted the experimentally obtained data into equations to investigate

the influence mechanisms of water pressure, porosity, geostress, and their ranges on coal permeability enhancement during hydraulic fracturing. These findings provide a reference and basis for the design of small-displacement hydraulic fracturing permeability enhancement applications in coal mines.

2. Analysis of Coal–Rock Medium Damage under Hydraulic Loads

Following drilling operations, the redistribution of internal stress within the coal medium leads to the damage of parts of the coal mass, subsequently resulting in the formation of plastic and elastic zones. Assuming that the hydraulic load is uniformly distributed and acts on the interior of the borehole, which has a regular circular shape, let the radius of the borehole be a , the overall influence area radius be b , and the plastic (damage) influence area radius be c . The schematic diagram of the stress around the borehole is illustrated in Figure 1. The surrounding coal medium of the borehole is an isotropic porous medium, with an effective porosity of φ .

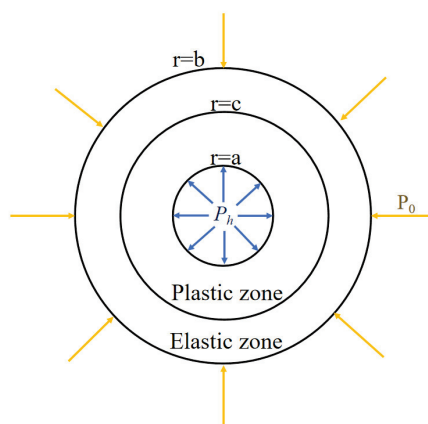


Figure 1. Influence zone of the coal under hydraulic loading.

According to the expression for the damage variable of coal rock defined by Zhou [38], it can be seen that

$$D = \begin{cases} 0 & \varepsilon < \varepsilon_c \\ \left(\frac{\varepsilon}{\varepsilon_t}\right)^n & \varepsilon \geq \varepsilon_c \end{cases} \quad (1)$$

where ε is the strain, ε_c is the peak strain, ε_t is a constant, and n represents the material damage factor. Thus, the constitutive equation for coal–rock medium damage is as follows:

$$\sigma = \begin{cases} E\varepsilon & \varepsilon < \varepsilon_c \\ E\left(1 - \left(\frac{\varepsilon}{\varepsilon_t}\right)^n\right)\varepsilon & \varepsilon \geq \varepsilon_c \end{cases} \quad (2)$$

According to the Lemaitre equivalent strain principle and extending the uniaxial constitutive model to the three-dimensional state, then

$$\sigma_i = E(1 - D')\varepsilon_i, \quad D' = \left(\frac{\varepsilon_i}{\varepsilon_t}\right)^n \quad (3)$$

Assuming the coal–rock medium in the plastic zone is incompressible, then the volumetric strain is zero. The equivalent strain of the coal–rock medium in the plastic zone under triaxial conditions is as follows:

$$\begin{aligned} \varepsilon_i &= \frac{\sqrt{2}}{3} \sqrt{(\varepsilon_r - \varepsilon_z)^2 + (\varepsilon_r - \varepsilon_\theta)^2 + (\varepsilon_z - \varepsilon_\theta)^2} \\ \varepsilon_r + \varepsilon_\theta &= 0, \\ \varepsilon_z &= 0 \end{aligned} \quad (4)$$

Considering that the boundary between the elastic and plastic zones is the boundary of damage, the expressions for displacement, strain, and equivalent volumetric strain within the plastic zone under the boundary condition $\varepsilon_i|_{r=c} = \varepsilon_c$ can be obtained as

$$\begin{cases} u = \frac{\sqrt{3}}{2r} c^2 \varepsilon_c \\ \varepsilon_r = -\frac{\sqrt{3}}{2r^2} c^2 \varepsilon_c, \varepsilon_\theta = -\frac{\sqrt{3}}{2r^2} c^2 \varepsilon_c, \varepsilon_i = \frac{c^2 \varepsilon_c}{r^2} \end{cases} \quad (5)$$

From this, the expression for the damage variable D' within the plastic zone in relation to the borehole radius r can be derived as

$$D' = \left(\frac{\varepsilon_c c^2}{\varepsilon_t r^2} \right)^n \quad (6)$$

3. Force Analysis of Hydraulic Loads Acting on the Elastic Zone of the Coal–Rock Medium

Based on the principles of elasticity and Darcy's law, the hydraulic pressure load P_h within the borehole can be obtained as

$$\begin{cases} \frac{d^2 p}{dr^2} + \frac{1}{r} \frac{dp}{dr} = 0 \\ p|_{r=a} = p_h, p|_{r=b} = 0 \end{cases} \quad (7)$$

Solve the differential equation:

$$p = \frac{p_h}{\ln(a/b)} \ln(r/b) \quad (8)$$

From Equation (8), it can be determined that the permeation force within the coal body under the action of hydraulic load is

$$F = \varphi \frac{dp}{dr} = \varphi \frac{p_h}{r \ln(a/b)} \quad (9)$$

Assuming the stress distribution due to hydraulic load within the borehole is plane-symmetric, with compressive stress being negative and tensile stress being positive, the stress distribution within the elastic zone is as follows:

$$\frac{d\sigma_r}{dr} + \frac{\sigma_r - \sigma_\theta}{r} - F = 0 \quad (10)$$

The deformation of the elastic zone of the coal body is in accordance with Hooke's law, so the constitutive equation of the plane strain problem is

$$\begin{cases} \varepsilon_r = \frac{1-\mu^2}{E} \left(\sigma_r - \frac{\mu}{1-\mu} \sigma_\theta \right) \\ \varepsilon_\theta = \frac{1-\mu^2}{E} \left(\sigma_\theta - \frac{\mu}{1-\mu} \sigma_r \right) \end{cases} \quad (11)$$

The geometric equation is

$$\varepsilon_r = \frac{du}{dr}, \varepsilon_\theta = \frac{u}{r}, \gamma_{r\theta} = 0 \quad (12)$$

After substituting Equations (9), (11), and (12) into Equation (10), we can obtain

$$\frac{d^2 u}{dr^2} + \frac{1}{r} \frac{du}{dr} - \frac{u}{r^2} = \frac{F}{E} \frac{(1+\mu)(1-2\mu)}{1-\mu} \quad (13)$$

Solving the differential equation in Equation (13) yields

$$u = \frac{\varphi}{2E} \frac{p_h}{\ln(a/b)} \frac{(1+\mu)(1-2\mu)}{1-\mu} r \ln r + C_1 r + \frac{C_2}{r} \quad (14)$$

Therefore, the stress distribution in the elastic zone under the influence of hydraulic loading is as follows:

$$\begin{cases} \sigma_r = \zeta \left(\zeta (\ln r + 1 - \mu) + C_1 + \frac{2\mu-1}{r^2} C_2 \right) \\ \sigma_\theta = \zeta \left(\zeta (\ln r + \mu) + C_1 + \frac{1-2\mu}{r^2} C_2 \right) \end{cases} \quad (15)$$

where

$$\zeta = \frac{E}{(1+\mu)(1-2\mu)}, \quad \zeta = \frac{\varphi}{2E} \frac{p_h}{\ln(a/b)} \frac{(1+\mu)(1-2\mu)}{(1-\mu)}$$

Assuming that the radial stress at the boundary between the elastic and plastic zones is $-\sigma_r^c$, the boundary conditions are as follows:

$$\sigma_r|_{r=b} = -P_0, \quad \sigma_r|_{r=c} = -\sigma_r^c \quad (16)$$

Substituting the boundary conditions into Equation (14) yields its coefficients as

$$\begin{cases} C_1 = \frac{c^2}{b^2-c^2} \left[\left(\frac{(\sigma_r^c - P_0)}{\zeta} - \zeta \cdot \ln(b/c) \right) \right] \\ \quad - \zeta (\ln b + 1 - \mu) - \frac{P_0}{\zeta} \\ C_2 = \frac{b^2 c^2}{b^2 - c^2} \left[\frac{(\sigma_r^c - P_0)}{\zeta(1-2\mu)} - \frac{\zeta}{1-2\mu} \ln(b/c) \right] \end{cases}$$

4. Force Analysis of Hydraulic Loads Acting on the Plastic Zone of the Coal-Rock Medium

Based on the unified strength theory, it can be inferred that hydraulic loads acting on boreholes can be considered plane strain problems. Assume the intermediate principal stress to be

$$\sigma_z = \sigma_2 = \frac{\beta}{2} (\sigma'_1 + \sigma'_3), \quad 0 \leq \beta \leq 1 \quad (17)$$

The coefficient β represents the intermediate principal stress coefficient, and for plastically incompressible materials, $\beta = 1$. A comparative analysis of reference [34] indicates that

$$\sigma_1 = \sigma'_r; \quad \sigma_2 = \sigma'_z; \quad \sigma_3 = \sigma'_\theta; \quad \sigma_1 > \sigma_2 > \sigma_3$$

In the equation, σ'_i represents each principal stress within the plastic zone, MPa.

According to the spatial stress theory of regular octahedron in [39], the strength criterion of the coal-rock mass under hydraulic loading in the plastic zone is as follows:

$$\frac{\tau_{oct}}{f_c} + \alpha \frac{\sigma_{oct}}{f_c} + \beta = 0 \quad (18)$$

In the equation, σ_{oct} is the positive stress in the octahedral stress space:

$$\sigma_{oct} = \frac{1}{3} (\sigma'_1 + \sigma'_2 + \sigma'_3), \text{ MPa};$$

τ_{oct} is the octahedral stress space shear stress:

$$\tau_{oct} = \frac{1}{3} \sqrt{(\sigma'_1 - \sigma'_2)^2 + (\sigma'_1 - \sigma'_3)^2 + (\sigma'_2 - \sigma'_3)^2}, \text{ MPa};$$

α, β are constant coefficients of the equation;

f_c is the uniaxial compressive strength of the coal, MPa.

Simplify Equation (18):

$$\left(\frac{\sqrt{6}}{6f_c} + \frac{\alpha}{2f_c} \right) \sigma'_r - \left(\frac{\sqrt{6}}{6f_c} - \frac{\alpha}{2f_c} \right) \sigma'_\theta + \beta = 0 \quad (19)$$

Assign $A = \frac{\sqrt{6}}{6f_c} + \frac{\alpha}{2f_c}$, $B = \frac{\sqrt{6}}{6f_c} - \frac{\alpha}{2f_c}$, and then

$$A\sigma'_r - B\sigma'_\theta + \beta = 0 \quad (20)$$

Assuming that the damage in the plastic zone of the coal is isotropic, the axial and circumferential stresses satisfy the following relation:

$$\sigma'_r = \frac{\sigma_r}{1-D'}, \quad \sigma'_\theta = \frac{\sigma_\theta}{1-D'}$$

It is possible to derive a strength criterion for the plastic zone of coal and rock under hydraulic loading:

$$A\sigma_r - B\sigma_\theta + \beta \left(1 - \left(\frac{c^2 \varepsilon_c}{\varepsilon_t r^2} \right)^n \right) = 0 \quad (21)$$

Assign $\omega_1 = \frac{B-A}{B}$, $\omega_2 = \frac{\beta}{B}$, $\omega_3 = \left(\frac{c^2 \varepsilon_c}{\varepsilon_t} \right)^n$, and the above equation can be transformed by substituting it into the balanced Equation (10):

$$\sigma_r = C_1 r^{-\omega_1} + \frac{\omega_2 \omega_3}{2n - \omega_1} r^{-2n} + \frac{\omega_2}{\omega_1} + \frac{\varphi p_h}{\omega_1 \ln(a/b)} \quad (22)$$

Combined with the boundary condition $\sigma_r|_{r=a} = -p_h$, Equation (22) can be solved to finally obtain the radial and circumferential stresses in the plastic zone of the coal-rock medium under hydraulic loading:

$$\begin{cases} \sigma_r = -a^{\omega_1} \left[\frac{\omega_2 \omega_3}{2n - \omega_1} a^{-2n} + \frac{\omega_2}{\omega_1} + p_h \left(1 + \frac{\varphi}{\omega_1 \ln(a/b)} \right) \right] r^{-\omega_1} \\ \quad + \frac{\omega_2 \omega_3}{2n - \omega_1} r^{-2n} + \frac{\omega_2}{\omega_1} + \frac{\varphi p_h}{\omega_1 \ln(a/b)} \\ \sigma_\theta = \frac{A\sigma_r + \beta[1 - \omega_3 r^{-2}]}{B} \end{cases} \quad (23)$$

At the boundary between the elastic and plastic zones within the coal affected by the hydraulic load, the radial stress in the plastic zone is equal to the radial stress in the elastic zone, which is

$$\sigma^e_r|_{r=c} = \sigma^p_r|_{r=c} \quad (24)$$

Meanwhile, the damage variable at the boundary is 0, that is, $D' = 0$. Since the stresses in the elastic and plastic zones are equal, substituting the stress expression at $r = c$ presented in Equation (15) and $D' = 0$ into the strength criterion in Equation (21) yields

$$A\sigma^e_r|_{r=c} - B\sigma^e_r|_{r=c} + \beta = 0 \quad (25)$$

Joining the above two equations yields the following equation:

$$p_h = \frac{\omega_1 \gamma_3 \ln(a/b) \left[\gamma_3 \left(1 - \frac{\gamma_4}{a^2} \right)^n + \gamma_2 (1 - \gamma_4) \right]}{\omega_1 \gamma_2 \gamma_3 + \varphi \left[\frac{\gamma_1 \gamma_4 + B \gamma_3 \gamma_5 (\ln c + \mu - 1) - c^2 B \gamma_5}{\ln(b^2/c + 1 - \mu)} + (2\mu - 1) b^2 \gamma_5 \ln(b/c) \right]} \quad (26)$$

$\gamma_1, \gamma_2, \gamma_3, \gamma_4, \gamma_5, \omega_5$ in the equation are determined by the following equations:

$$\begin{aligned} \gamma_1 &= \frac{\omega_2 \omega_3}{\omega_1 - 1} \\ \gamma_2 &= \frac{\omega_2}{\omega_1} \\ \gamma_3 &= b^2 - c^2 \\ \gamma_4 &= a^{\omega_1} c^{\omega_1} \\ \gamma_5 &= \omega_1 \omega_4 \\ \omega_4 &= \frac{(1+\mu)(1-2\mu)}{2E(1-\mu)} \\ \omega_5 &= \frac{\sigma^c_r(1+\mu)}{E} \end{aligned}$$

Under certain conditions of fixed compressive strength, porosity, brittleness index, and damage factor for the coal or rock, the analytical solution for the radius of influence in the coal–rock medium zone under the action of hydraulic load within a borehole can be derived from the above theoretical analysis. Similarly, the radius of the damage zone under the influence of hydraulic load can also be determined.

5. Analysis of the Mechanism of Coal Damage under Hydraulic Load

5.1. Analyzing the Mechanism of Hydraulic Load Damage through Theoretical Analysis

5.1.1. Laboratory Determination of Coal's Physical Dynamics Parameters

To analyze the mechanisms of hydraulic-load-induced damage through analytical solutions derived from the aforementioned theoretical derivations, and to guide the field practices at Yangmei Wukuang in Shanxi, China, experiments were conducted on coal samples from the mine. These included uniaxial compressive loading tests, triaxial compressive loading tests, and porosity determination experiments to measure their mechanical parameters.

Two larger coal samples were extracted from the coal face, and four cores were taken from each piece. These cores were processed into cylinders of 50×100 mm and polished to ensure flatness at both ends. Then, 10 g of cinder was carved with a knife at each of the four different locations on the original coal and kept for later.

The TAW-2000 electro-hydraulic servo testing machine was employed for uniaxial compression tests with a displacement loading method at a rate of 0.1 mm/min. Axial and radial displacements were measured by an extensometer, and a computer was used to record data on loads, stresses, strains, and displacements during the test. Four experimental groups were set up for the uniaxial experiments, with all samples derived from the same coal seam. The measured mechanical parameters are shown in Table 1, and the stress–strain curves for the four samples are depicted in Figure 2. These curves show some differences, especially the one from sample #4. In fact, coal is a highly heterogeneous rock. It has low material strength and a rich pore structure, which leads to the easy formation of cracks and defects under the influence of strata movement and mining disturbances. These cracks and defects are the reasons for the differences in the stress–strain curves. The trends of curves #1, #2, and #3 are more consistent with the ideal full stress–strain curves of rocks. Sample #4 has natural joint planes, and when the stress reached 9.13 MPa, it underwent large-scale fracturing, causing a sudden change in the curve. The average results of the four tests indicate that the coal body's modulus of elasticity $E = 1820$ MPa, Poisson's ratio $\mu = 0.32$, and uniaxial compressive strength $f_c = 10.52$ MPa, with the hydraulic fracturing drilling radius being 0.047 m.

Table 1. Partial mechanical parameters of the coal.

Specimen Number	f_c/Mpa	E/GPa	μ
1#	11.47	1.93	0.307
2#	10.88	2.52	0.341
3#	10.59	1.18	0.321
4#	9.12	1.66	0.319
avg	10.52	1.82	0.322

The damage coefficients $\varepsilon_c/\varepsilon_t$ and n have a corresponding relationship in Equation (6), and the constant ε_t value was obtained by using the method described in reference [38]. Using MATLAB R2022a, the full stress–strain curve data obtained from the uniaxial compression test were input into the damage constitutive equation to fit the corresponding value of n . Combining the peak strain ε_c , the $\varepsilon_c/\varepsilon_t$ and n values for the four specimens were statistically analyzed to fit the relationship curve between $\varepsilon_c/\varepsilon_t$ and n , as illustrated in Figure 3.

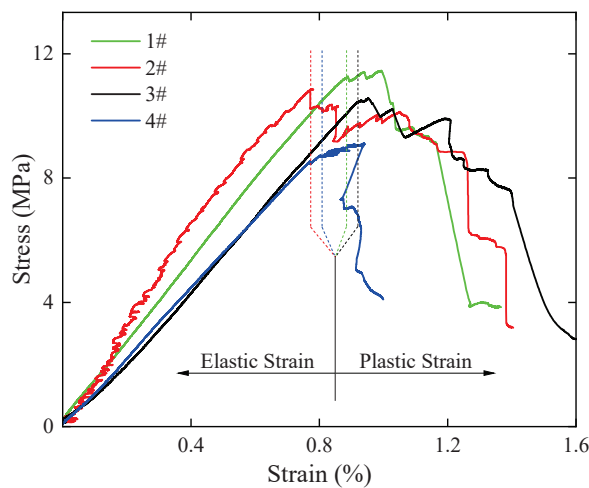


Figure 2. Stress–strain curve of coal.

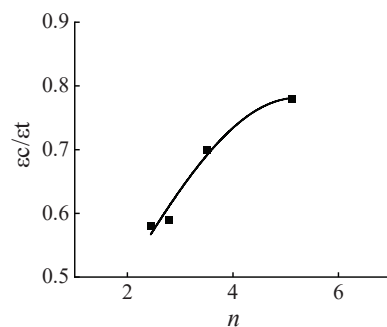


Figure 3. Coefficient curve of $\varepsilon_c/\varepsilon_t$ vs. n .

Triaxial compression tests were conducted on the GCTS RTR-4600 high-performance rock mechanics testing system at the National Key Laboratory for CCRI. Four experimental groups were established, with confining pressures set at 5 MPa, 15 MPa, 20 MPa, and 25 MPa, respectively (Figure 4). During the experiments, the confining pressure was first increased to the set value at a rate of 50 KPa/s, followed by axial pressure at the same rate until specimen failure, recording the peak axial stress at failure. By fitting Equation (18) with the obtained three principal stresses under triaxial stress conditions (as shown in Table 2) and the uniaxial compressive strength of the coal specimen $f_c = 10.52$ MPa, the equation constants $\alpha = 0.3413$, $\beta = -0.1231$ were obtained.

Table 2. Results of triaxial compression coal bursting test.

Specimen Number	σ_1 /MPa	σ_2 /MPa	σ_3 /MPa	$\frac{\sigma_{gct}}{f_c}$	$\frac{\tau_{gct}}{f_c}$
SS-01	−14.83	−5.00	−5.00	−1.01	0.57
SS-02	−29.87	−15.00	−15.00	−1.91	0.94
SS-03	−43.00	−20.00	−20.00	−3.39	1.33
SS-04	−53.40	−25.00	−25.00	−4.22	1.64

Porosity determination experiments were carried out using the PoreMaster GT-60 mercury intrusion porosimeter (Quantachrome Instruments, Boynton Beach, FL, USA). Samples were collected by carving particles of 1–3 mm in size from the original coal specimens with a knife. The pore volume distribution density function of the four groups of samples is depicted in Figure 5, and the corresponding porosity values are listed in Table 3, with an average porosity of $\varphi = 12.6\%$.

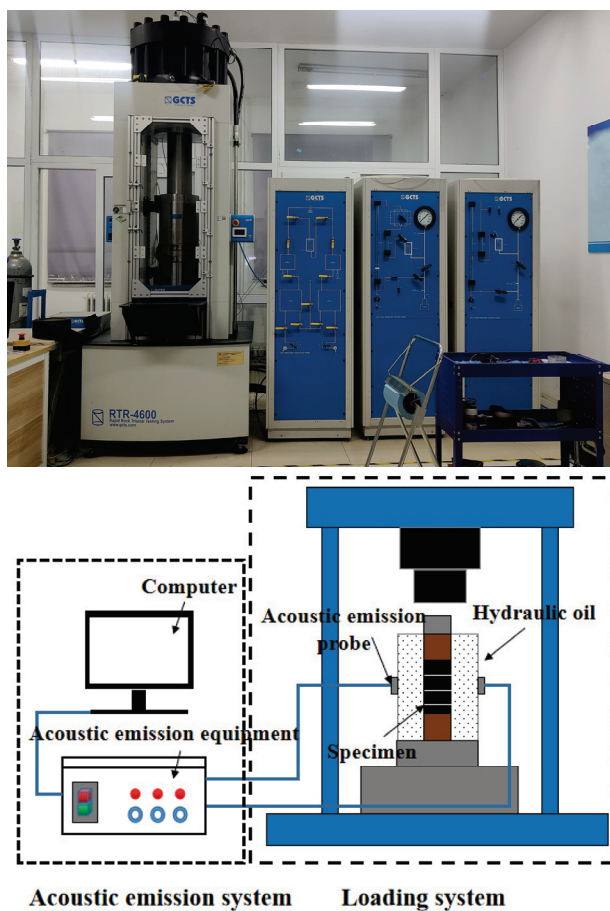


Figure 4. GCTS RTR-4600 high-performance rock mechanics testing system and schematic diagram.

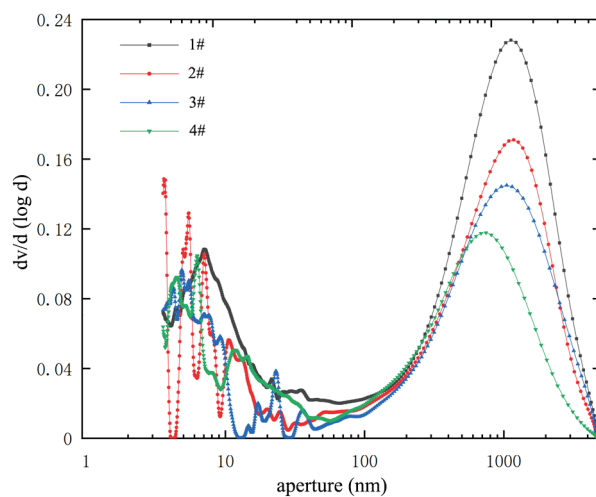


Figure 5. Pore volume distribution density function diagram.

Table 3. Porosity distribution table.

	1#	2#	3#	4#
Porosity/%	10.22	14.42	14.47	11.32
Adsorption pore/%	3.30	3.44	4.28	3.75
Seepage pore/%	6.92	10.98	10.19	7.57

5.1.2. Effect of Hydraulic Water Pressure on Damage Radius at Different Influence Ranges

The damage radius under different hydraulic pressure conditions can be obtained from Equation (26), as illustrated in Figure 6. The enlarged views at locations $c = 1$ and $c = 1.2$ are shown in Figures 6b and 6c, respectively. As indicated in the figure, within the same influence range, the required hydraulic load exhibits an exponential growth trend as the damage radius of the coal mass increases. Under the same damage radius condition, as the influence range expands, the gradient of hydraulic pressure change shows a slightly increasing trend. This implies that the larger the influence range, the greater the hydraulic pressure needed to cause the same degree of damage. When the damage radius is set at 1 m and the influence ranges exerted by the hydraulic load on the coal are 2 m, 3 m, and 4 m, respectively, the required hydraulic pressures will be 12.6 MPa, 13.1 MPa, and 13.2 MPa. When the damage radius is set at 1.2 m and the influence range is the same, the hydraulic pressures needed are 48.1 MPa, 49.4 MPa, and 49.8 MPa. As depicted in Figure 7, under a fixed damage radius, the greater the zone of influence, the smaller the incremental increase in the required water pressure. Comparing the cases of damage radii of 1 m and 1.2 m laterally, it is obvious that the larger the damage radius, the more pronounced the effect of the influence range on the hydraulic pressure.

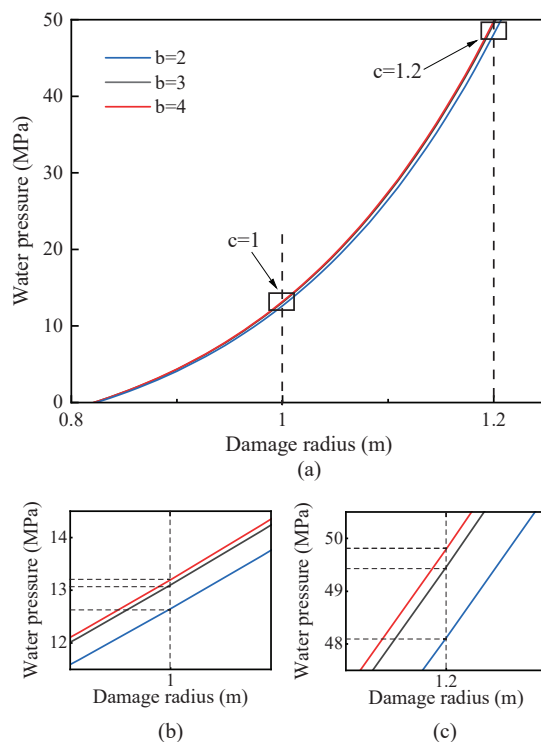


Figure 6. Curve of damage radius c versus hydraulic water pressure P_h for different influence ranges b (a), and local magnification diagrams at $c = 1$ m (b) and $c = 1.2$ m (c).

The maximum pressure of the mud pump car matched with the drilling rig is generally 15 MPa, but it can reach 20 MPa when pumping with small flow, while the maximum pressure of the dedicated fracturing pump car underground can reach 50 MPa at present. In the practical application of low-displacement hydraulic fracturing in coal mines, measures that solely consider increasing water pressure to promote coal fracturing and expand the range of influence have limited effects on the permeability enhancement of coal subject to triaxial stress conditions. With known basic mechanical parameters of the coal, one can nearly match the hydraulic pressure parameters for the implementation of hydraulic fracturing and the corresponding damage and influence range in the coal. This provides guidance and reference for the design and evaluation of the effectiveness of small-displacement hydraulic fracturing applied in coal mines underground.

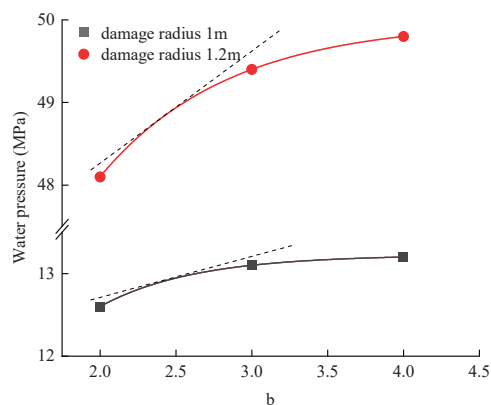


Figure 7. Relation between influence range and hydraulic pressure.

5.1.3. Effect of Hydraulic Water Pressure on Damage Radius at Different Porosity Levels

It is generally believed that the porosity of the same coal seam is a fixed value. However, as roadways are excavated, the surrounding rock around the roadways develops cracks and expands under crustal stress, leading to changes in its internal porosity, which is related to coal strain or deformation [40,41]. When hydraulic fracturing boreholes are arranged perpendicular to the roadway walls of the coal seam, the porosity of the coal at different depths of the boreholes varies slightly. Taking the results of the porosity determination experiments as intervals, the porosity was set as $\varphi = 0.08, 0.10, 0.12$, and 0.14 . The relationship between the damage radius c and hydraulic water pressure P_h for different porosities φ at an influence range of 2 m is calculated and plotted, and the curves are shown in Figure 8. Among them, Figures 8b and 8c are the enlarged plots at the positions of $P_h = 15$ MPa and $P_h = 45$ MPa, respectively.

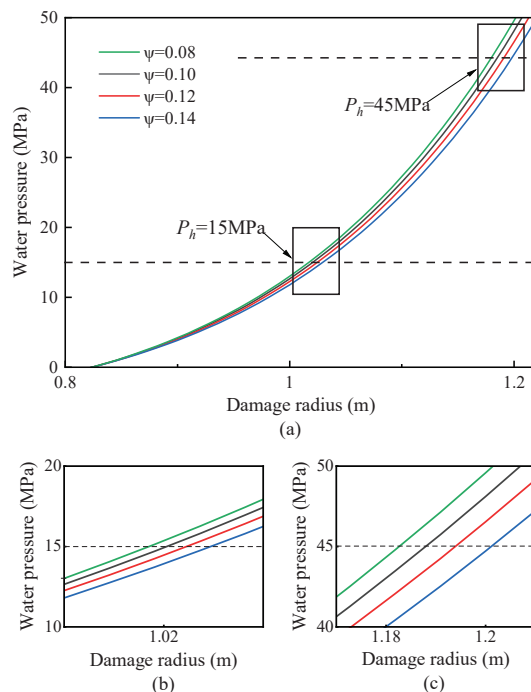


Figure 8. Curve of damage radius c versus hydraulic water pressure P_h for different porosity values φ (a), and local magnification diagrams at $P_h = 15$ MPa (b) and $P_h = 45$ MPa (c).

The observation in Figure 8 reveals that, when P_h is held constant, an increase in coal porosity means more extensive development of internal micro-cracks, which consequently reduces the seepage resistance loss of fracturing fluid inside the pores. This results in

higher pressure accumulation at the crack tip. The macroscopic appearance is that the damage radius increases with increasing porosity. Specifically, for $P_h = 15$ MPa, the damage radii corresponding to four different porosity values are 1.016 m, 1.020 m, 1.024 m, and 1.028 m, respectively, while for $P_h = 45$ MPa, they are measured as 1.182 m, 1.188 m, 1.194 m, and 1.201 m, respectively. It can be inferred that during hydraulic fracturing aimed at enhancing permeability, greater water ingress occurs within coal's internal micro-cracks and micropores due to increased porosity levels. This intensifies the coupling between the seepage field and the deformation field, ultimately leading to reduced mechanical properties of coal. The effect of porosity on the damage radius is roughly linear, and better penetration can be obtained by fracturing near the mouth of the hole or in the fracture zone of the coal seam; this phenomenon will be more obvious if the fracturing pressure is increased.

5.1.4. Effect of Hydraulic Water Pressure on Damage Radius at Different Far-Field Crustal Stresses P_0

The far-field crustal stress P_0 is taken as 7 MPa, 11 MPa, and 15 MPa, and the porosity is taken as the average value of $\varphi = 0.126$. The curves of the damage radius c versus hydraulic water pressure P_h for different far-field crustal stresses P_0 are plotted as shown in Figure 9.

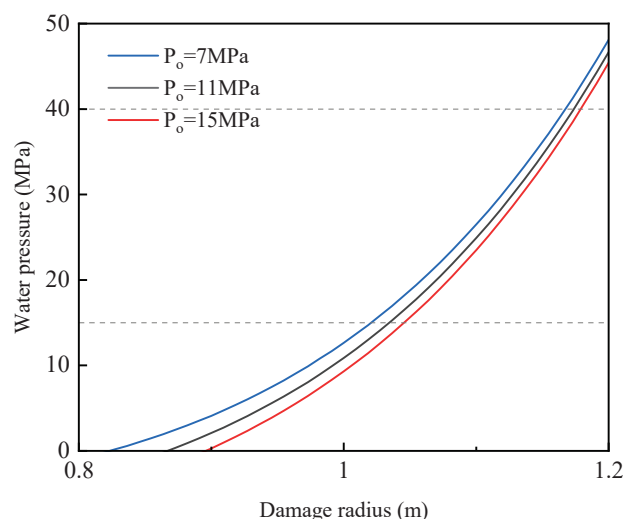


Figure 9. Curve of damage radius c versus hydraulic water pressure P_h for different far-field crustal stresses P_0 .

The increase in P_0 from 7 MPa to 15 MPa produced a larger damage radius at the same water pressure. Fracturing with a water pressure of 15 MPa in coal seams under far-field crustal stress of 7 MPa, 11 MPa, and 15 MPa resulted in damage radii of 1.045 m, 1.036 m, and 1.018 m, respectively. Damage radii of 1.189 m, 1.123 m, and 1.083 m were obtained from fractures with a water pressure of 40 MPa in coal seams under far-field crustal stresses of 7 MPa, 11 MPa, and 15 MPa.

In the practice of hydraulic fracturing, the deformation and damage of the surrounding rock around the borehole can simply be considered as a result of a combination of far-field crustal stress, induced stresses, and pore pressures. Higher far-field crustal stress produces wider and more damaged plastic zones for pore water intrusion. But at the same time, for the coal in the elastic zone, its cracks are compressed more densely, and fracturing is more difficult. This is why the effect of far-field crustal stress on the damage radius is larger when the hydraulic pressure is lower, while the effect of far-field crustal stress on the damage radius is relatively smaller when the hydraulic pressure is higher. In addition, when fracturing under high far-field crustal stress, the fracture initiation pressure is greater,

more energy is accumulated, and the fracture extends further after initiation to produce a larger damage radius.

5.1.5. Effect of Hydraulic Water Pressure on Damage Radius for Different Brittle Damage Coefficients n

The value of porosity was taken as an average value of $\varphi = 0.126$. The far-field crustal stress P_0 is 7 MPa. To obtain the equivalent $\varepsilon_c/\varepsilon_t$, the constant n is taken as 3, 3.5, and 4 and substituted into Figure 3. The results are obtained by substituting the data into Equation (26) to plot the relationship curves between the damage radius c and the hydraulic pressure P_h under various brittleness damage coefficients n , as displayed in Figure 10.

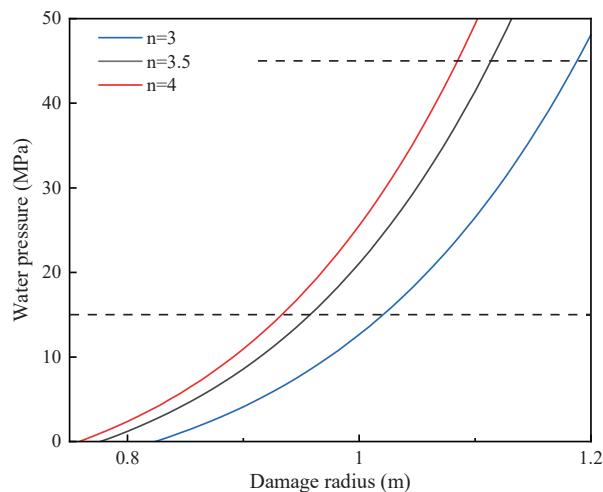


Figure 10. Curve of damage radius c versus hydraulic water pressure P_h for different n .

Constant n represents the brittleness of the coal. The greater n , the denser the coal, and the greater the tendency to undergo elastic deformation and accumulate energy under external load. As shown in the figure, the process from $n = 3$ to 4 resulted in a smaller damage radius at the same water pressure. Fracturing in $n = 3$, $n = 3.5$, and $n = 4$ coals with 15 MPa water pressure resulted in damage radii of 1.045 m, 1.036 m, and 1.018 m, respectively, with 6.0% and 8.2% reductions in damage radius for the $n = 3.5$ and $n = 4$ conditions relative to the $n = 3$ condition. Fracturing in $n = 3$, $n = 3.5$, and $n = 4$ coals with 45 MPa water pressure resulted in damage radii of 1.189 m, 1.123 m, and 1.083 m, respectively, with an increase in damage radius of 6.5% and 10.4% for the $n = 3.5$ and $n = 4$ conditions relative to the $n = 3$ condition. Therefore, compared with the other conditions mentioned above, the influence of different n values on the damage radius is greater, and the higher the water pressure, the greater the influence of n values.

5.2. Practical Validation Study of the Range of Hydraulic Load Damage Effects

The test area is located at No. 15 coal seam 15311, Yangmei Wukuang, Shanxi Province, with an average thickness of 5.12 m, a coal seam permeability coefficient of $0.156 \text{ m}^2/(\text{MPa}\cdot\text{d})$, a coal sturdiness factor of 0.45, a gas flow attenuation coefficient of 0.329 and gas pressure of 0.5 MPa. According to the Coal Mine Gas Drainage Specification, the coal seam in which the test is carried out belongs to the category of coal seams with high gas extraction difficulties. The mechanical parameters of coal are detailed in Section 5.1.

Five gas pressure measurement boreholes were pre-positioned in the hydraulic fracturing test area with spacing of 0.5 m, 1.0 m, 1.5 m, 2.0 m, and 2.5 m, respectively. Hydraulic fracturing was performed at a hydraulic fracturing borehole after the pressure measured in the boreholes stabilized. During the test, the water pressure was set to 5 MPa, and then increased to 10 MPa after confirmation of sealing, stabilizing for 5 minutes. If there is no leakage from the surrounding boreholes, the pressure will gradually increase to 4 MPa for each layer and 2 min for each layer, until the surrounding boreholes return water steadily,

with a final extreme water pressure of 35.2 MPa. The ultimate purpose of the hydraulic fracturing is to establish a smoother gas seepage network, improve the permeability of the coal seam, and enhance the regional gas extraction effect. Therefore, borehole gas pressure is an important evaluation index for evaluating hydraulic fracturing damage. The pressure gauges were installed in the gas pressure measurement boreholes to monitor and record the pressure status of the pressure measurement point after hydraulic penetration enhancement. The pressures at each pressure measurement point after the implementation of hydraulic penetration enhancement are shown in Figure 11. The gas pressure decrease ratio of the pressure measurement boreholes is shown in Figure 12. After comprehensively analyzing the proportion of gas pressure decrease after 70 days of pumping in the permeability enhancement area, basically, the proportion of gas pressure decrease decreases with the increase in the distance from the borehole of the hydraulic fracturing. Within this, the biggest decrease ratio is 17.8% in borehole #3. It is considered that the decrease in gas pressure by more than 15% is the critical condition for the determination of the damage impact radius. Under the condition of hydraulic pressure of 35.2 MPa, the limit value of the coal damage radius is 1.5 m. The data obtained from the field practice are the same as the limit value of the coal damage radius calculated theoretically under the same condition, which further verifies the correctness of the theoretical analytical formula obtained.

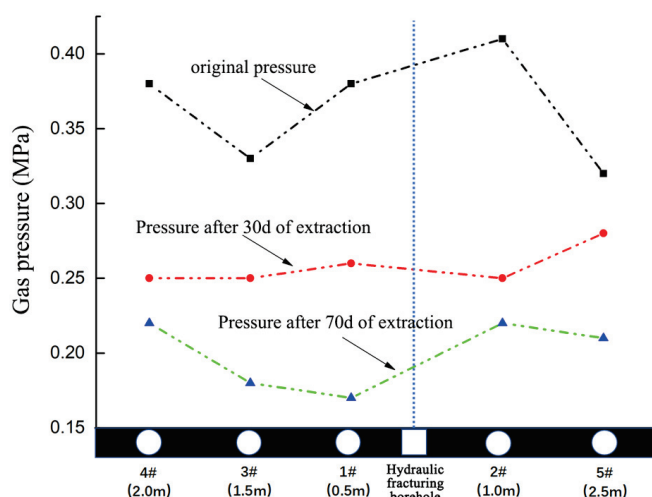


Figure 11. Examination of the coal body influence range by the pressure-drop method.

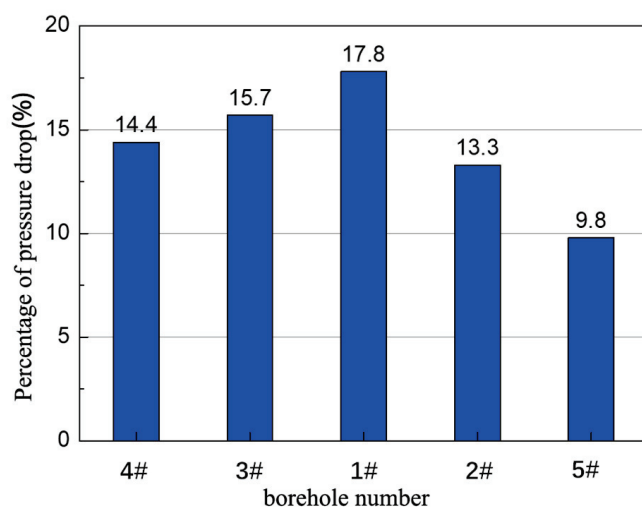


Figure 12. Proportional decline in gas pressure around the drilling hole, measured by hydraulic fracturing.

6. Conclusions

- (1) A theoretical analysis has been established to determine the relationship between the damage radii of coal bodies under hydraulic loads, considering the intermediate principal stress and water pressure. This provides a design and reference for the application of small-flow-rate hydraulic fracturing in coal mines to enhance permeability.
- (2) A quantitative analysis was conducted based on the in situ conditions of the coal seam, examining the relationship between hydraulic load, damage radius, and the area of influence, which revealed the impact mechanisms of hydraulic fracturing involving water pressure, porosity, geostress, and the range of influence on coal seam permeability enhancement. The experimental results show that the hydraulic load increases exponentially with the damage radius, indicating that continuously increasing the water pressure when it is already high does not yield better results. Within the range of geological conditions in the test area, different areas of influence and porosity levels have a limited impact on the results, while far-field geostress, particularly the brittle damage coefficient, plays a significant role in controlling the damage radius. The effectiveness of hydraulic fracturing is greatly influenced by the stress conditions and rock properties of the coal seam; the larger the hydraulic load, the less the change in far-field stress affects the damage radius, whereas the area of influence, porosity, and brittle damage coefficient have the opposite effect.
- (3) The theoretical analysis's findings on the damage range have been validated through data analyses of field measurements, thereby proving the accuracy and feasibility of designing hydraulic fracturing pressure based on theoretical analysis.

Author Contributions: Methodology, H.L.; validation, Z.S.; formal analysis, Y.F.; writing—original draft, W.W.; writing—review & editing, Y.L. (Yaolong Li); visualization, S.H.; funding acquisition, Y.L. (Yang Li). All authors have read and agreed to the published version of the manuscript.

Funding: Supported by the National Natural Science Foundation of China (52304225) and the Technology Innovation Fund of China coal Research Institute (2023CX-I-05).

Data Availability Statement: The original contributions presented in the study are included in the article, further inquiries can be directed to the corresponding author.

Conflicts of Interest: The authors declare no conflict of interest.

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Review

Theories, Techniques and Materials for Sealing Coalbed Methane Extraction Boreholes in Underground Mines: A Review

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Abstract: To further enhance the intelligent technology, platformisation, and systematisation of coalbed methane extraction sealing technology, this paper analyses the research progress of theories, technologies, and sealing materials related to coalbed methane extraction sealing and systematically summarises the latest achievements of the basic theories, key technologies, and sealing materials of coalbed methane extraction. Considering the increasing mining depth, advancements in intelligent technology, and the evolving landscape of coalbed methane development, it is particularly important to establish a more comprehensive coalbed methane extraction borehole sealing system. Based on this, future development trends and research prospects are proposed: In terms of coalbed-methane-extraction-related theories, there should be a stronger focus on fundamental research such as on gas flow within the coal matrix. For coalbed methane extraction borehole sealing technologies and devices, efforts should be made to enhance research on intelligent, platform-based, and systematic approaches, while adapting to the application of directional long borehole sealing processes. In terms of coalbed methane extraction borehole leakage detection, non-contact measurement and non-destructive monitoring methods should be employed to achieve dynamic monitoring and early warning of methane leaks, integrating these technologies into coalbed methane extraction system platforms. For coalbed methane extraction borehole sealing materials, further development is needed for liquid sealing materials that address borehole creep and the development of fractures in surrounding rock, as well as solid sealing materials with Poisson's ratios similar to that of the surrounding rock mass.

Keywords: coalbed methane extraction; gas flow; coalbed methane extraction boreholes; sealing materials; sealing extraction boreholes

1. Introduction

The “*Statistical Review of World Energy 2024*” reports that global primary energy consumption increased by 2% in 2023 compared to 2022, reaching 620 EJ (Figure 1). Of this, coal accounted for 31.61% of global primary energy consumption. In 2023, coal remained a major fuel for power generation, maintaining a stable share of about 35%. Global coal production reached a historic high of 179 EJ (Figure 2), with the Asia-Pacific region contributing nearly 80% of the global coal production, primarily concentrated in Australia, China, India, and Indonesia. In 2023, global coal consumption surpassed 164 EJ for the first time, a 1.6% increase from 2022 and seven times the average growth rate of the past decade. Although China is the largest coal consumer, accounting for 56% of global consumption, India's coal

consumption for the first time exceeded the combined total of Europe and North America in 2023 [1]. Gas, as a byproduct of the coal formation process, poses widespread risks due to the reliance of most coal production capacity on underground mining [2–5]. This year, gas accidents have occurred in almost all major coal-producing countries. For instance, on 29 June 2024, a gas explosion occurred at the Grosvenor mine of Anglo American in Queensland, Australia; on 26 June 2024, a gas explosion happened at the Solikamsk mine in Perm Krai, Russia; on 10 March 2024, a gas explosion at the Hoshth mine in Balochistan, Pakistan, resulted in 12 fatalities; and on 1 February 2024, a gas explosion at a coal mine in Cundinamarca, Colombia, led to 2 deaths.

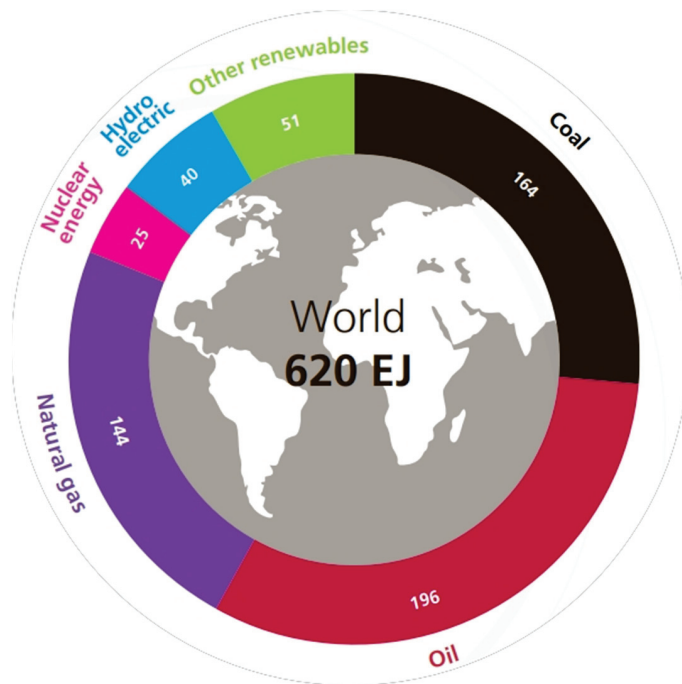


Figure 1. The 2023 global primary energy consumption chart [1].

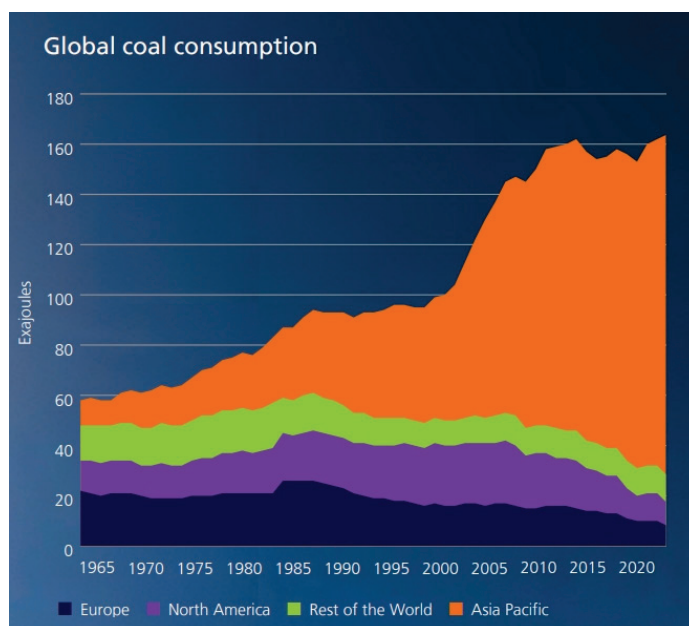


Figure 2. The 2023 global coal consumption graph [1].

In China, greenhouse gas emissions from energy use, industrial processes, and methane amounted to 12.6035 billion tons in 2023, representing a 6% increase and accounting for 31.2% of global emissions. Under China's "carbon peak and carbon neutrality" goals, coal consumption still constitutes 55.3% of total energy consumption. It is projected that by the end of the 14th Five-Year Plan, the share of coal in energy consumption will remain around 50% [6]. During the 2022 "Two Sessions" of China, President Xi Jinping emphasised that China's energy structure is predominantly coal-based, and this fundamental reality will be difficult to change in the short term. Currently, China has over 3000 coal mines, most of which see gas concentration in extraction boreholes drop below 10% after two months of extraction. With the increase in mining depth, the pressure and content of coal seam gas rise, intensifying the risks of gas outbursts and explosions [7–23]. From January 1 to 19 March 2023, there was one coal and gas outburst accident, four dangerous coal and gas outburst incidents, and forty-eight high gas concentration exceedance accidents across China. As of 15 July 2024, three coal and gas outburst accidents and one gas explosion accident have occurred in China in 2024, resulting in 29 fatalities.

The frequent gas accidents worldwide pose significant challenges to the safe development of the coal industry. Therefore, effectively preventing and controlling gas-related accidents has become a top priority for the sustainable development of the global coal industry [2–5,7–12,17–20,24–26].

In order to achieve the "dual-carbon" goal as soon as possible, China has incorporated new requirements, such as the digital and intelligent transformation of traditional industries, into the process of energy revolution and high-quality development. China's National Development and Reform Commission (NDRC) and Bureau of Energy (BEA), among others, have successively promulgated the "14th Five-Year Plan for Scientific and Technological Innovation in the Energy Sector", the "14th Five-Year Plan for a Modern Energy System", and the "Benchmarking Levels in Key Areas of Clean and Efficient Utilisation of Coal (2022)" and other documents. At the same time, with the development of science and technology, gas is receiving more and more attention as a clean energy source. "The Notice of the Comprehensive Department of the National Energy Administration on Organising and Carrying Out Demonstration Work of Efficient Extraction and Utilisation of Coal Mine Gas and Coalbed Methane Exploration and Development" and "the Implementation Plan of Green and Low Carbon Advanced Technology Demonstration Project" both take the efficient extraction and utilisation of coal mine gas as a key direction of the green and low-carbon advanced technology demonstration project. According to "the 14th Five-Year Plan for Coal", it is expected that by 2025, the amount of coalbed methane (CBM) developed and utilised in China will reach 10 billion cubic metres.

At present, high-gas mines, prominent mines, and high-gas areas of low-gas mines in China are mostly treated or utilised by pre-extraction. However, according to incomplete statistics, more than 80% of the coalbed methane extraction concentration of down-hole down-grade boreholes in China will decay to 6–20% in a short time, and the pre-extraction rate of coal seam gas is less than 23% [7,9,10,12,27–29]. How to improve the coalbed methane extraction rate will directly affect the effectiveness of gas management and utilisation and consequently the timeframe for achieving China's "double-carbon" goal.

In this context, this paper compiles the research history of coalbed methane extraction hole sealing theory, technology, and materials, summarises the gas-flow-related theory and the current status of research on coalbed methane extraction borehole sealing technology and sealing materials, and puts forward the future development direction to explain the research progress in related fields and provide references to prove the effect of coalbed methane extraction.

2. Coalbed Methane Flow Theory

Coalbed methane extraction is the process of pumping free gas and adsorbed gas desorbed by the extraction process from the coal seam into the coalbed methane extraction pipeline under negative pressure through surface drilling or downhole drilling [30]. The

engineering theoretical basis of coalbed methane extraction is the theory of coal seam gas flow.

The coal body is divided into countless substrates by a large number of crisscrossing fracture networks, and a large number of pores exist within the coal substrate, forming a kind of dual-media structure [31–34], which makes the process of coal seam gas flow very complicated (Figure 3). Coal seam gas flow theory, also known as coal seam gas transport theory, mainly includes adsorption–desorption, diffusion, and seepage, which are the basis for describing the flow or transport law of gas in coal seams. Different coal seam gas flow laws are based on different models of gas flow in homogeneous or non-homogeneous media [33,35–40]. In terms of the mechanism of gas flow, although there is a basic consensus that the gas flow law follows Darcy’s law in fissures, there is still a great deal of controversy about the theory of gas flow in coal matrix, especially the driving force of gas flow [31,33,41–44]. Most scholars believe that the driving force for gas transport in coal matrix is the concentration gradient, and therefore Fick’s law is used to describe the gas diffusion process. However, some scholars believe that the driving force for gas transport in the coal matrix is the pressure gradient and argue that gas flow in the coal matrix is more in line with Darcy’s law; and some scholars believe that Darcy flow and Fick’s diffusion coexist in gas flow within the coal matrix [31,33]. In order to facilitate the differentiation, this paper divides the gas adsorption and desorption law of coal matrix into two parts according to the spatial distance: “Gas adsorption and desorption law near the surface of coal matrix” and “gas migration theory away from the coal matrix surface”.

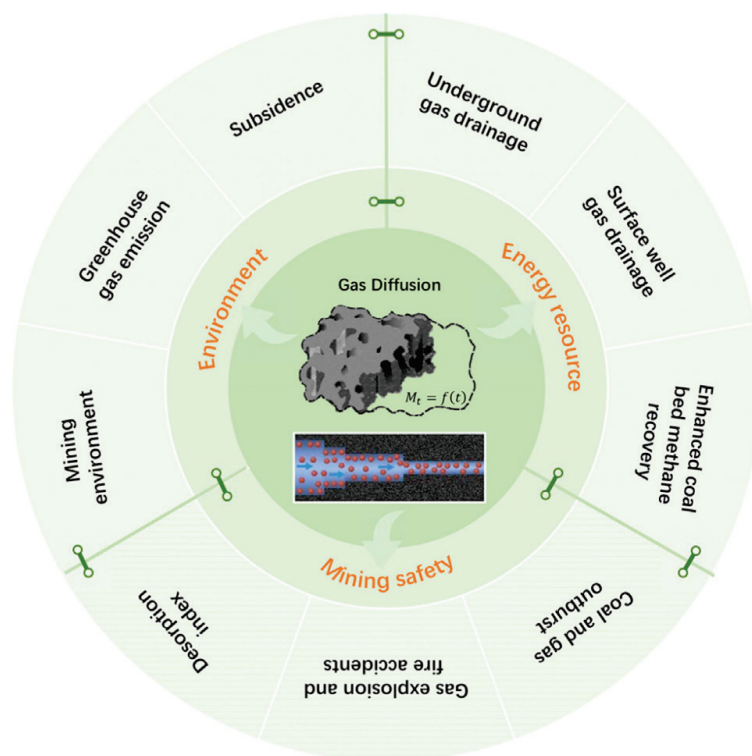


Figure 3. Applications related to gas diffusion in coal [33] (reproduced with permission from Zhao, W. et al. Fuel; published by Elsevier, 2019).

2.1. Gas Adsorption–Desorption Pattern near the Surface of Coal Matrix

The gas in the coal body is divided into free and adsorbed states. In the coalbed methane extraction process, the free state gas is firstly extracted under negative pressure, and the adsorbed state gas is subsequently desorbed to become extractable gas. The gas adsorption and desorption process of coal matrix, especially the desorption process, is the key step of gas output. The study of gas adsorption and desorption laws is an important

theoretical basis for constructing gas flow models and predicting the effect of coalbed methane extraction. Gas adsorption desorption models mainly include theoretical models and semi-empirical/empirical models [33].

2.1.1. Adsorption and Desorption Modelling

Coalbed methane adsorption models or desorption models are used to describe the adsorption and desorption processes of gas in coal and have attracted extensive attention from scholars, and a variety of gas adsorption and desorption models have been proposed [33,45–53]. The gas diffusion model of coal is an important tool for theoretical description of the adsorption and desorption process of gas in coal. With the deepening of research, new diffusion models have been proposed continuously.

In 1951, Barrer [54] derived the classical diffusion model by studying the diffusion of natural gas in zeolites and derived the exact solution of the diffusivity and its simplified equation. Nandi et al. [55,56] carried out a study of gas diffusion in coal and calculated the diffusion coefficients by using the classical diffusion model. Ruckenstein et al. [57] proposed a double-diffusion model to describe gas adsorption and desorption on the basis of the classical model. Smith et al. [52] fitted the measured gas diffusion data of coal with a dual-porosity model and a single-pore model, respectively, and found that the dual-porosity model had a higher fitting accuracy. Clarkson et al. [58] proposed an improved dual-porosity model in order to enhance the adaptability of the dual-porosity diffusion model. The classical model and the dual-porosity model explain the gas diffusion phenomenon better due to their strict derivation process and clear physical meaning and have been widely recognised and applied. Some scholars also introduced time-dependent or pressure/concentration-dependent kinetic diffusion coefficients and combined them with the Fickian diffusion model to describe the desorption process over the whole timescale, which resulted in a better match with the experimental desorption data [33].

Most of the above models assume that the gas flow in coal particles is driven by a concentration gradient, and it is valid to use Fick's law to describe the gas diffusion in the coal matrix. In addition, a mathematical model for coalbed methane adsorption and desorption based on Darcy's law has also been developed in the literature, and the numerical solution and experimental results of the gas adsorption and desorption model were compared through fixed-pressure and variable-pressure adsorption and desorption experiments, and it was found that the computational results of Darcy's model were more consistent with the experimental data than the Fick's classical model, which led to the assumption that Darcy's law is more effective than Fick's law in describing gas diffusion. Due to the extraordinarily complex structure of the pore system within the coal matrix, the pore size range extends from the angstrom scale to the micrometre scale [33,57,59–64]. The medium flow mechanisms in pores of different scales may be different, so the process of gas diffusion in coal is a multi-scale and multi-mechanism flow phenomenon, and the validity of describing gas diffusion using only Fick's law is questionable. Many scholars believe that gas transport in coal matrix is a multi-mechanism process. Lunarzewski [65] argued that only the application of Darcy's law can effectively describe gas transport in lump coals unless the coal suffers from excessive damage.

Gas diffusion refers to the movement of gas molecules from an area of higher concentration to an area of lower concentration, driven by the concentration gradient. It occurs on a molecular level and is not influenced by external forces like pressure differences or the presence of cracks.

Gas seepage refers to the slow movement of gas through porous materials or small openings, often driven by pressure differences. Seepage usually occurs in materials like rock or soil, where gas passes through tiny voids or pores, following Darcy's law in most cases.

Gas leakage refers to the uncontrolled escape of gas through larger openings or cracks, usually due to a defect or failure in the sealing system. Leakage is often more rapid than seepage and typically results from structural flaws or pressure differences.

In summary, diffusion is molecular movement due to concentration differences, seepage is slow movement through porous materials due to pressure differences, and leakage is uncontrolled gas escape through visible cracks or openings.

Lu et al. [66] argued that the release of methane from coal matrix is a combination of gas diffusion and gas seepage, and that either diffusion or seepage dominates the process, which is mainly dependent on coal pore structure characteristics within the matrix. Experimental studies have shown that two types of pores exist within the coal matrix, one is diffusion pores that control methane desorption and diffusion, and the other is permeation pores that control methane permeation [67–70]. The triple-porosity/dual-permeability model assumes that methane in the coal matrix is transported through desorption and diffusion from micropores into meso/macropores, and then transported through meso/macropores and fissures by Darcy flow, and this type of model exhibits higher accuracy than the double-porosity/single-permeability model, suggesting that not only diffusive flow but also Darcy flow exist in the process of gas flow in the pore space of the coal matrix [33,67–71].

2.1.2. Semi-Empirical/Empirical Mathematical Modelling

In addition to the above mathematical models of gas adsorption and desorption derived from the basic theory, researchers have proposed a number of relational equations to represent the change in adsorption and desorption with time based on measured gas adsorption and desorption data or measured field data in order to facilitate the rapid calculation of gas adsorption and desorption. These empirical or semi-empirical relational equations mainly have two forms: Power function form and exponential form.

(1) Power function equation

Barrer [54] obtained a simplified equation for gas adsorption versus time through adsorption experiments on zeolites based on the analytical solution of Fick's classical model. This relationship equation usually has a power function form and can describe the variation of gas adsorption with time:

$$\frac{Q_t}{Q_\infty} = \frac{2s}{V} \sqrt{\frac{Dt}{\pi}} \quad (1)$$

where Q_t is the cumulative gas desorption volume at time t , cm^3/g ; Q_∞ is the limiting adsorption volume, cm^3/g ; s is the external specific surface area of the coal sample, cm^2/g (the specific surface area refers to the total surface area of a material per unit mass. It can be categorised into two types: External surface area and internal surface area. The external specific surface area refers to the area of the outer surface of a solid material, which is the part of the solid in contact with the external environment. The external specific surface area does not include the pores or microporous structures inside the solid and only accounts for the outer surface); V is the volume per unit mass of the coal sample, cm^3/g ; D is the diffusion coefficient, cm^2/min ; t is the adsorption time, min.

Nandi et al. [55,56] and Sevenster et al. [72] found that there are some limitations in the application of Barrer's formula by fitting the data from adsorption–desorption experiments. In particular, when the adsorption amount is close to or more than half of the limiting adsorption amount, the prediction accuracy of Barrer's formula will be significantly reduced, and the prediction error increases with time.

To overcome these limitations, Smith et al. [52] referred to Barrer's formula and proposed an improved relation by studying experimental data:

$$\frac{Q_t}{Q_\infty} = \frac{6}{\sqrt{\pi}} \sqrt{D_e t} \quad (2)$$

where D_e is the effective diffusion coefficient, cm^2/s .

(2) Exponential

Bolt et al. [73] carried out gas desorption experiments on coal samples with various degrees of coal alteration, and by processing the experimental data, they proposed the equation for the relationship between cumulative gas desorption and time as:

$$Q_t = Q_\infty (1 - Ae^{-\lambda t}) \quad (3)$$

where A, λ are empirical constants.

Airey [74] argued that coal gas flow can be represented by Darcy's law and stated that the expression for the amount of gas desorbed from coal versus time is:

$$Q_t = Q_\infty \left[1 - e^{-(t/t_0)^n} \right] \quad (4)$$

where t_0 is an empirical constant.

2.2. Theory of Gas Migration Theory Away from the Coal Matrix Surface

Some studies have divided the pores in the coal matrix into an interconnected macroporous system and a disconnected microporous system, suggesting that gas desorbs and diffuses from microporous matrix to macroporous matrix, and then seeps out of the matrix macroporosity into the fissures. This approach uses two sets of equations to describe gas flow processes in the coal matrix [33,57,58,60]. However, this model has difficulties in accurately distinguishing between large and small pores in the coal matrix, the physical significance of the parameters is not sufficiently clear, and the complex calculation of the equations limits its generalised application. Therefore, scholars generally assume that all pores in the coal matrix have the same diameter and adopt a set of gas flow equations to describe the gas flow in the coal matrix. However, even under the assumption of the same pore diameter, there is still controversy about the driving force of gas flow.

2.2.1. Diffusion Theory

Some scholars have argued that the main driving force for gas transport in coal matrix is the concentration gradient, and thus gas flow conforms to Fick's law [42,55,75–79]. Ertekin et al. developed a pseudo-steady-state gas transport model based on Fick's first law by assuming that gas concentration varies with the spatial location of the coal matrix, while ignoring the effect of time on concentration [75]. However, gas transport in coal matrix is a non-stationary process related to both space and time, and ignoring the effect of time on concentration may lead to the physical properties of gas transport described by the model not matching the actual situation. Subsequently, some scholars have developed non-stationary gas transport models based on Fick's second law, which integrates the effects of time and spatial location on gas concentration [33,78–80]. Some studies found that the model based on Fick's law could not accurately describe the whole process of gas diffusion, especially in the initial stage of gas diffusion or during the stage of the maximum cumulative amount, where the issue of being unsuitable for Fick's law arises [33,77,78,81].

In order to solve these problems, scholars have established a gas transport model based on Fick's law with diffusion coefficient varying with time through experimental and theoretical studies [33,45,60,82–91]. Taking the diffusion coefficient as a variable makes the simulation results of the model more consistent with the experimental data and ensures the accuracy of the calculation results. However, the treatment of the diffusion coefficient change with time contradicts the initial assumption of Fick's law, and scholars have not yet clarified the physical significance of this change. As a result, some scholars have become sceptical about using Fick's law to describe the gas flow in coal matrix and try to use new theories to describe the gas transport process in coal matrix [33,67,92–96]. For example, based on the fractional dynamics approach, some scholars improved the unsteady gas transport model and established a time–space fractional order anomaly model for gas adsorption and desorption in coal matrix and validated the model by using

mercury intrusion porosimetry (MIP), scanning electron microscopy (SEM) techniques, and adsorption and desorption experiments. This model contains several uncertain parameters, and the calculation process is complicated and needs to be determined with the help of a fractal dimension, which is not easy to apply. In addition, Fick's law considers the total gas content in the coal body as the diffusion concentration of the gas, which leads to the calculated value of the gas transport being higher than the actual value. Liu et al. [97] corrected the defects in Fick's law by distinguishing the free state gas from the adsorbed state gas in the coal matrix and established a non-linear gas flow model for coal matrix based on the difference of densities.

Meanwhile, simulation tools play an important role in describing the complex transport phenomena associated with CBM. Wang [98] summarised the basic theory of gas transport in nanopores, covering the mechanisms of gas transport in porous media and their simulation methods, including molecular dynamics and lattice Boltzmann simulation methods, as well as pore reconstruction methods (e.g., porosity, pore size, and pore shape) in diffusion and transport applications.

2.2.2. Theory of Seepage

Another group of scholars argued that the main driving force for gas transport in coal matrix is the pressure gradient, and thus the gas flow conforms to Darcy's law [33,67,96,99–102]. Earlier, Airey [74] applied Darcy's law to describe gas flow in a coal body; Zhou et al. [102,103] concluded that the gas flow laws in both coal matrix and fissures follow Darcy's law and established a mathematical model of gas leakage from drilling holes, which provided theoretical guidance for improving the efficiency of drilling extraction and drilling sealing technology. Liu et al. [103] used different particle sizes and pressures under different conditions for the gas adsorption and desorption experiments, the gas flow models were established by Darcy's law and Fick's law, respectively, and the results of the solved model were compared with the experimental data. It was found that the mathematical model based on Darcy's law was more in line with the experimental results. However, with further research, it was found that when the permeability coefficient of coal grains was inverted [103–105], the permeability coefficient decreased exponentially with the increase in pressure, and this change was quite different from the experimental results in the literature [106,107]. That is, the permeability coefficient of the coal matrix gas flow model based on Darcy's law needs to change with pressure in order to match the experimental results with the simulation results. However, this is similar to the problem of the diffusion coefficient changing with time in Fick's law, where the key constants in the model should not change with time or pressure.

Therefore, using either Fick's law or Darcy's law to describe the coal matrix gas flow process is problematic. In the development of coal matrix gas transport theory, the gas flow mechanism is still controversial, and new theories need to be proposed to accurately describe the coal matrix gas flow process through systematic and comprehensive theoretical and experimental studies and to establish a general, accurate, and easy to promote application of coal matrix gas flow models based on these new theories.

2.3. Solid–Gas Coupled Modelling of Gas Flow

The seepage characteristics of coal seam gas are affected by factors such as ground stress field, ground temperature field, ground electric field, acoustic field, pore pressure, sliding effect, and moisture of the coal body. Currently, there are several models, including “single-pore–single-seepage” model, “dual-porosity–single-seepage” model, “dual-porosity–double-seepage” model, the “single-pore–single-permeability” model, the “dual-porosity–single-permeability” model, the “dual-porosity–double-permeability” model, and the “triple-pore–double-permeability” model (Figure 4). Among them, the triple-porosity/double-permeability model takes into account the simultaneous existence of diffusion flow and Darcy flow during the flow of gas in the pores of the coal matrix and has a higher accuracy [108].

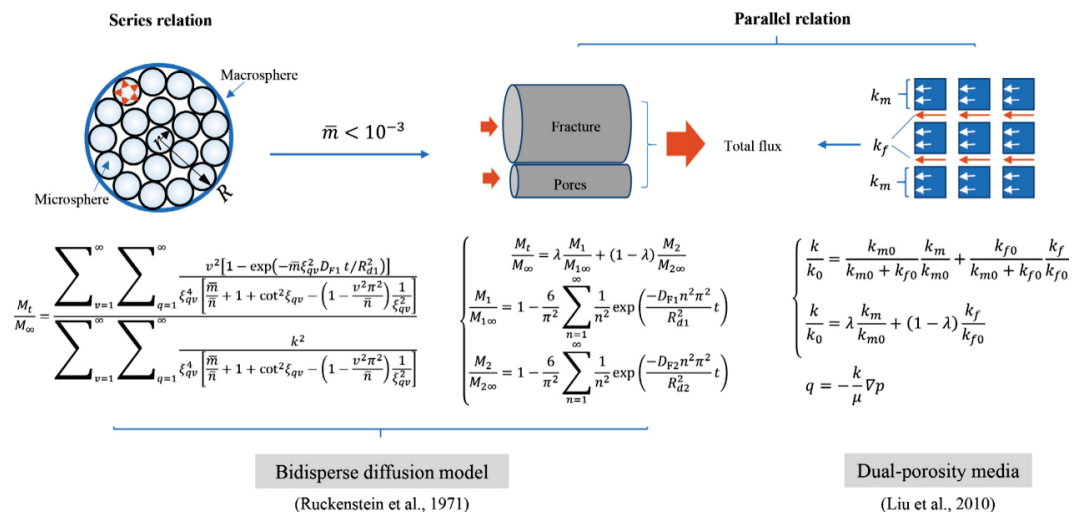


Figure 4. Comparison between the geometrical models of the bidisperse diffusion and the dual-porosity media [33] (reproduced with permission from Zhao, W. et al. Fuel; published by Elsevier, 2019).

In terms of mathematical models of fluid permeability, scholars have achieved many important results. For example, some scholars proposed the capillary model, which is based on the assumption of capillary bundles and replaces the pore structure of the core with capillary bundles [109,110]; Fatt et al. established the spherical particle stacking model, which stacks spherical particles of certain diameters in a specific way and is used to simulate the pore space of reservoir rocks [111]; and some scholars used the lattice method to study the gas seepage in the Klinkenberg slip effect in gas seepage and the relative permeability curves of oil and water phases [111,112]. In recent years, with the rapid development of computer technology and image scanning and analysis technology, more and more researchers have established pore network models that are closer to the real pore structure of the core by reconstructing the pore network [113], such as the SD permeability model [114].

According to the pore structure of coal, the form of gas diffusion, and seepage characteristics, it can be classified into the following types from the point of view of the idealised model and the form of mutual coupling of coal and gas by the micro- and macrostructure of the coal body [33,66–71,74–77,81,83,115–117].

2.3.1. The “Monoporous–Monotonic” Model

The “single-pore–single-permeability” model idealises the pore structure of a gas-containing coal body as a homogeneous coal matrix model and considers only the flow of gas in the pores of the matrix. After the adsorption and desorption dynamic equilibrium of the gas in the coal matrix is broken, the gas is desorbed on the surface of the matrix, and then flows directly in the coal matrix; the gas pressure acts on the surface of the matrix, which affects the deformation of the solid matrix, and the solid–gas coupling model reflects the Darcy seepage law of gas based on the single-pore medium.

2.3.2. Dual-Porosity–Single-Permeability” Model

The “dual-pore–single-seepage” model idealises the coal pore system as a dual-pore medium structure, i.e., the coal matrix and the fissures. In the dual-pore structure, the scale of the fissures is much larger than that of the pores of the coal matrix, so only the seepage of gas in the fissures of the coal body is considered. When the dynamic equilibrium adsorbed on the microporous surface is broken, the gas desorbs on the microporous surface of the coal matrix, and then diffuses from the surface of the high-concentration coal matrix to the low-concentration coal body fissures, where it flows in the fissures. During this process, there is an influence of adsorption-induced deformation and gas pore pressure on coal matrix deformation; conversely, the volumetric and surface forces of the coal matrix

also impact the gas pressure. This process is a strongly coupled process in which the deadsorption of gas from the pore surfaces of the coal matrix acts as a source for gas flow in the fractures.

2.3.3. The “Dual-Porosity–Double-Seepage” Model

The “dual-pore–dual-seepage” model idealises the pore structure of the coal body as a dual-pore structure of matrix pores and fissures. The main flow spaces of fluids in the coal seam are matrix pores and coal body fissures, i.e., matrix seepage and coal body fissure seepage are considered at the same time. When the dynamic equilibrium of gas adsorption in the coal body is broken, the gas adsorbed on the surface of the matrix microporosity of the coal body is desorbed, and a part of the desorbed gas flows in the pore space within the coal matrix, while another part of the desorbed gas is transported to the cracks of the coal body through diffusion, and then flows in the cracks. In this process, the interaction between the solid medium and the gas is also strongly coupled, and the desorbed gas from the matrix is the source and sink of both the pore flow of the gas within the matrix and the flow in the coal body fissures.

2.3.4. Triple-Porosity and Dual-Permeability Model

The “triple-porosity and dual-permeability” model idealises the coal body as a porous medium consisting of impermeable matrix, seepage matrix, and fissures. The main seepage space in the coal body is the seepage matrix and fissures, and the main factors affecting the permeability are the seepage pores and fissures.

The gas flow process in the coal body involves gas desorbing from the micropore surfaces of the coal matrix and diffusing into the pores of the flowing matrix under the influence of concentration gradients. The gas then flows through the pores of the coal flowing matrix driven by pressure gradients. Generally, gas always moves towards the space with the least flow resistance. Eventually, most of the free gas flows into the fracture structure of the coal body, and under the pressure gradient in the fractures, it continuously moves towards larger and macropores and is finally extracted to the surface.

Sang [115] proposed a typical triple-porosity and dual-permeability model, which considers a three-porosity medium composed of kerogen matrix (organic matrix), inorganic matrix, and fractures. The gas is mainly transported by diffusion in the casing matrix and seepage transport in the inorganic matrix and fissures; the amount of gas in the casing is then the source–sink term for the gas flow in the inorganic matrix and fissures, and the fissure gas flow is in accordance with the cubic law.

In summary, the research on solid–gas coupling theory has achieved relatively rich results. However, the impermeable matrix, permeable matrix, and fractures have the same computational domain in most of the coupling models, which is somewhat different from the gas diffusion and transport in the actual reservoir. Therefore, it is necessary to establish a solid–gas coupling model that distinguishes the computational domains of fractures and matrix to better investigate the gas flow law in reservoirs.

3. Sealing Technology for Coalbed Methane Extraction

After the coal body is exposed, the fissure is the main path of gas transport, its free gas gushes out under the action of a pressure gradient, the place of storing gas (coal matrix) is affected, and its adsorbed gas is desorbed from the inner surface of the pore and transported to the fissure within the pore space of the matrix and then gushes out of the coal seam [7,8,14–16,20–22,25–29,36,118–120]. Coalbed methane extraction is the process of extracting the free gas from the coal seam by negative pressure (Figure 5). The effect of coalbed methane extraction is not only related to the gas storage law, adsorption and desorption of the coal seam itself, but also related to the sealing effect of the coalbed methane extraction borehole. The level of coalbed methane extraction sealing technology determines the effect of coalbed methane extraction. The key to coalbed methane extraction sealing technology lies in coalbed methane extraction sealing parameters, sealing technology, and sealing devices.

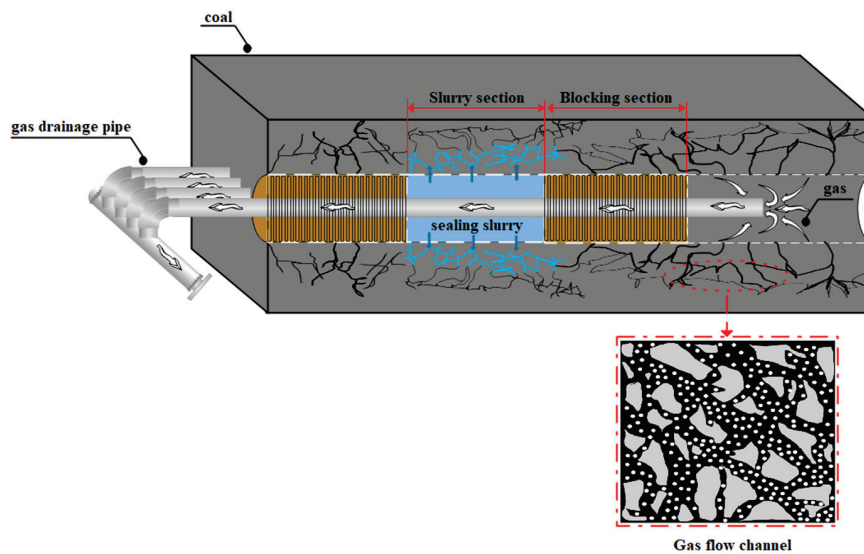


Figure 5. Schematic diagram of coalbed methane extraction blocking.

3.1. Theory of Fracture Development in Drilling Surrounding Rock

The development of fissures in the surrounding rock of coalbed methane extraction boreholes directly affects the flow path and flow rate of gas. By studying the theory of fissure development, we can understand the distribution, scale, and connectivity of the fissures in the surrounding rock of the coalbed methane extraction boreholes, so as to optimise the reasonable sealing parameters of the boreholes, to ensure that the gas can be released and flowed smoothly from the coal seam, and to improve the efficiency of coalbed methane extraction.

3.1.1. Current Status of Coal Rock Drilling Perimeter Rock Stability Research

The construction of coalbed methane extraction boreholes is equivalent to the excavation of a small roadway or chamber, and the stress distribution around the borehole is similar to that of a circular roadway. At present, theoretical analysis and numerical simulation methods (Figures 6 and 7) are mainly used to study the stress distribution of the surrounding rock in the excavation body [7,115].

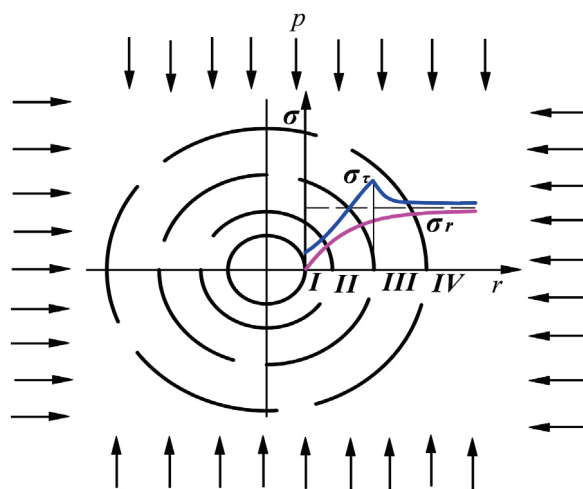


Figure 6. Distribution of borehole deformation zones under the surrounding rock stress. I—fractured zone; II—plastic zone; III—elastic zone; IV—original stress zone [25] (reproduced with permission from Wang, K. et al. Process Safety and Environmental Protection; published Elsevier; 2019).

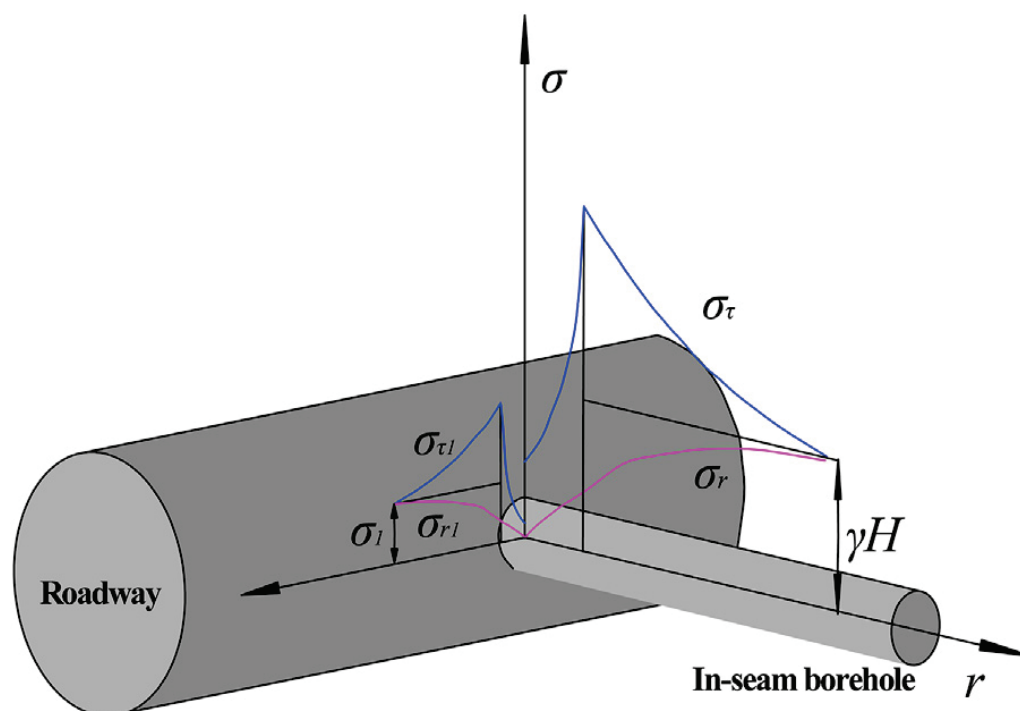


Figure 7. Stress distribution of the in-seam borehole around the roadway [25] (reproduced with permission from Wang, K. et al. Process Safety and Environmental Protection; published by Elsevier; 2019).

Due to the complex field conditions of the mine, the mechanical properties and damage characteristics of the coal rock body by the roadway excavation and digging, back-mining, and drilling construction are more difficult to obtain, so some researchers have devoted themselves to the methods of obtaining the above parameters under laboratory conditions. To this end, some scholars have carried out research on the mining mechanical behaviour of coal rock bodies under different mining conditions, obtained the change in stress state experienced by the coal body in front of the workings, and investigated the strength characteristics, deformation characteristics, and infiltration evolution law of the coal samples during the loading and unloading process [121–125].

(1) Classical mechanical analysis based on the Mohr–Coulomb criterion

Hu [121], based on classical elasto-plastic mechanics, equated the borehole stress field to a plane-strain circular hole problem, derived an expression for the stress distribution around the borehole, and combined with the empirical radius of influence, gave a relational equation for the relationship between the plastic zone of the borehole and the radius of influence as well as the diameter of the borehole. Lu et al. [126] conducted a theoretical analysis of the elasto-plastic stresses around the circular roadway. Hashemi [127] investigated the variation of particle displacement of the borehole wall under different factors.

Zhang et al. [128] proposed a new analytical solution for calculating the stress distribution around a circular excavation body. On the basis of the research results on the damage deformation of the roadway and coal rock borehole, the radius of the crushing zone and damage expansion zone is given in combination with the damage expansion characteristics of the coal body, and it is pointed out that it is too conservative to use only the Mohr–Coulomb criterion to determine whether the coal body is in a plastic state or not. Even if the stress state is unchanged, the coal body may be damaged after a certain period of time after the yield state.

(2) Classical mechanics analysis based on Hoek–Brown criterion

Most of the above studies are based on the linear Mohr–Coulomb criterion, but in engineering practice, due to the inherent non-linear characteristics of rocks and rock bodies,

as well as the structural surfaces, the effect of stress state on strength, and anisotropic rock bodies, the linear Mohr–Coulomb criterion is not applicable, especially in the case of coal rock bodies which are mostly jointed, and the damage characteristics tend to be more in line with the non-linear Hoek–Brown criterion [129]. Brown et al. [130] and Lee et al. [131] investigated and obtained numerical solutions for the stress field of strain softening in the chamber enclosure.

Zhou et al. [132], based on the Hoek–Brown criterion, studied the strength damage patterns of sandstone under different unloading rates. They proposed an index to describe the changes in rock strength due to unloading, introducing the concept of the unloading strength damage factor, and modified the Hoek–Brown criterion through engineering examples. Sheng [133] used the non-linear Hoek–Brown criterion with the non-correlated flow method to derive the formulae for the deformation characteristics of the tunnel-enclosing rock and the Romberg method to give the stress solution, and finally the deformation solution of the tunnel-enclosing rock is obtained, and the reasonableness of the theoretical derivation and the final solution is verified by comparing the engineering examples with the Carranza-Torres method.

Tu et al. [134] proposed an elastic–plastic calculation method for deep coal seam borehole collapse pressure based on the radius of the engineering-permissible plastic zone based on the Hoek–Brown criterion and verified the correctness of the method by combining it with engineering examples and comparing with the Hoek–Brown criterion and the Mohr–Coulomb criterion.

At the same time, due to the Hoek–Brown criterion, related parameter selection is difficult, and a large number of scholars have conducted related research and achieved significant results.

Xia et al. [135] gave the formulae of GSI and D parameters in the Hoek–Brown criterion based on rock wave velocity and compared the Hoek–Brown criterion with the rock wave velocity method based on the actual engineering slopes, and the results showed that the rock wave velocity method can give the mechanical parameters of the rock body quickly and accurately in the case of insufficient test data. Yang et al. [136] estimated the rock body properties in the excavation damage zone based on the generalised Hoek–Brown damage criterion and a varying perturbation factor to estimate the rock mass properties in excavation damage zones. The results showed that the perturbation factor was quantified by using acoustic P-wave velocities tested in both undamaged and damaged rock mass, rather than a single constant value selected from a descriptive guide. Chang et al. [137], in order to address the traditional method involved in the determination of the perturbation factor (D) when considering mining damage, which cannot reflect the anisotropic characteristics of jointed rock strength, proposed a numerical analysis method to determine the D value using the jointed rock damage tensor and mining damage evolution simulation.

Sun et al. [138] constructed a new method for numerical modelling of jointed rock bodies based on the geological strength index (GSI) and the principles of damage mechanics. With the help of the basic concept of damage mechanics, the joint equivalence coefficient was proposed, and the relationship between GSI and joint spacing was established. Sun et al. [139] quantitatively described the quantitative relationship between GSI and the integrity coefficient of the rock body and the degree of rock weathering in the Hoek–Brown criterion, and then determined the relevant rock mechanical parameters in the Hoek–Brown criterion, and the parameters obtained by this method were compared with the actual engineering to obtain a small error, which verified the reasonableness of the method. Compared with the actual engineering, the error obtained by this method is relatively small, which verifies the reasonableness of this method. Bertuzzi et al. [140] constructed a new quantitative table of GSI using the interval number theory, which solves problems such as the number of volumetric joints of the rock body being unable to be determined accurately and provides a new basis for the reasonable selection of parameters in indoor rock mechanics experiments. Liu et al. [141] introduced the spacing of the nodal surfaces in the rock body and JRC into the quantitative relationship of GSI in the Hoek–Brown criterion,

and then determined the related rock body mechanical parameters in the Hoek–Brown criterion. Brown’s criterion of quantitative GSI grading and parameter values established a quantitative GSI evaluation system and successfully predicted the related rock mechanical parameters by combining with the actual engineering fissured rock. Fischer et al. [142], based on the quantitative GSI perimeter rock rating system and the theory of continuum media, established a mechanical model of post-peak strain softening of fractured rock considering the influence of the perimeter pressure and used the elasto-plasticity. The intrinsic model developed explicit structural models for intact rocks and provided realistic parameter ranges for the explicit structural properties. Some scholars [140–144] quantified the formulae describing the GSI with the volume nodal number of the rock body and the structural surface condition factor, combined them with the Hoek–Brown criterion to estimate the mechanical parameters of the rock body, and verified the reasonableness of the results through specific examples.

Some scholars [143,144] summarised the strength discounting method based on the Hoek–Brown criterion, derived the stress-invariant expression form of the Hoek–Brown criterion, proposed the equal proportional discounting scheme of σ_{ci} and m_b , and proved the correctness of the scheme through numerical simulation.

(3) Enclosure stability analysis by other methods

Chuanqing Zhang et al. [145] used yield proximity and its phase-complementary covariates as random variables and proposed an inversion algorithm to derive the relevant parameters and proposed an analysis method for the safety of the surrounding rock based on the statistical law of rock strength.

Fu et al. [146] introduced the concept of local energy release rate and established an unloading instability analysis model for the goaf. Based on catastrophe theory, they developed a local energy criterion for identifying rock instability. They concluded that in the goaf, energy is primarily released in the horizontal direction, while energy is concentrated in the vertical direction.

Zhou Hui et al. [147] investigated the relationship between the non-yield point and yield point in the principal stress space for four yield codes, defined the yield proximity index, and gave the expression of yield proximity under the four yield codes, and the research results can be better used in the evaluation of stability of the surrounding rock of underground engineering. Li [148], based on the Mohr–Coulomb criterion, Drucker–Prager criterion, and elastic–plasticity theory, derived the safety factor of tunnel-surrounding rock, and combined with numerical simulation, the corresponding support measures were selected through a tunnel-surrounding rock example. Gaede et al. [149] analysed the stresses around the borehole in anisotropic media.

(4) Analyses of the stability of the surrounding rock considering the case of drilling or roadway support

Wang et al. [150] derived a formula for calculating the circle of perimeter rock relaxation based on the Hoek–Brown criterion combined with elastic–plasticity theory when the pressure measurement coefficient is not equal to 1 and the support force is considered. The displacement monitoring of the perimeter rock in deep tunnels is close to the theoretical calculation results, which indicates the correctness of the derived formula for calculating the circle of perimeter rock relaxation.

In common construction practices, gas extraction boreholes typically require grouting for sealing. The grouting material in the sealing section provides some support to the borehole, which has a certain impact on the stability of the surrounding rock. A few scholars have also studied the stability of the surrounding rock under the support effect of the sealing section.

Some scholars [151–153] analysed the stability of the drilled hole sealing section by drawing on the relevant results of roadway support and introduced the analogy of roadway support in the drilled hole sealing section in the plastic radius of the surrounding rock and the deformation of the surrounding rock to measure the stability of the drilled hole

sealing section and ultimately the Mohr–Coulomb criterion to determine that the sealing section is in the plastic zone or the crushing zone. Some other scholars [154–157], based on the results of roadway support, gas-containing coal constitutive model, elastic–plastic mechanics, etc., derived the expression of effective stress of coal body around the borehole wall of horizontal and inclined boreholes under the consideration of pore pressure and gave the corresponding collapse pressure of the borehole wall by combining it with the Mohr–Coulomb criterion and established the viscoelastic non-linear creep constitutive model of soft coal and combined it with the numerical simulation to analyse the stability of borehole collapse.

3.1.2. Current Status of Research on Fissures in the Surrounding Rocks of Coal Rock Drill Holes

Field practice and experimental research show that the interaction between the ground stress and the surrounding rock of the roadway will produce different sizes of rupture zones. Many researchers mainly use field measurements or indoor tests to study the distribution characteristics of the rupture zones in the surrounding rock of the excavation body.

Kong et al. [158] numerically simulated the development of fissures in actual thin coal mining workings in the void zone based on the discrete element software UDEC 6.0. Liu et al. [159] combined the rock mechanics with the gas flow law of coal rock in the process of full stress–strain and used a deep displacement automatic detector and a fissure channel circuit recorder to obtain the change in gas fissure channel with the working’s back mining, from primary fissure to five stages of fissure channel generation, expansion, maturity, and closure.

Yu et al. [160] studied the generation and development process of perimeter rock fissures in deep roadways, put forward the concepts of plate-like structure and block structure, and classified the perimeter rock fissures into four partitions, namely, macrotensile fissure zone, macroscopic shear fissure zone, high-incidence and microscopic fissure zone, and weakly influenced fissure zone. Xu et al. [161] used UDEC to simulate the distribution characteristics, expansion law, and relative evolution tendency of the perimeter rock fissure field of roadways under different spans. Li et al. [162] analysed the fissure development and destabilisation characteristics of a mudstone roof under different water content conditions by means of borehole peeping equipment, revealing that there is a saturation phenomenon in the fissure development and that controlling the development of the microfissure zone into the fissure development zone is the key to the support of the anchor rod technology. Li et al. [163] studied the deformation characteristics of randomly distributed through fissures generated after the excavation of underground chambers and established a computational model for the deformation of the rock body with randomly distributed through fissures, which showed that the equivalent modulus of elasticity gradually decreased with the increase in the average spacing of the fissures, and the equivalent Poisson’s ratio gradually increased with the small average inclination angle.

Fu et al. [164] introduced the definition and calculation method of fissure strain based on the CD fissure strain model, derived the fissure volumetric strain formula, defined the corresponding characteristic stresses at each stage of coal body fissure development as fissure closure stress, crack initiation stress, damage amplification stress, and peak stress, and concluded that the process of damage amplification of the coal body is the process of gradual compaction of fissures within the coal body, the concentration of stress at the tip that causes the expansion of primary fissures, newborn fissure generation, and then the development, connection, penetration, and formation of macroscopic damage surfaces.

Hu [165] firstly used a similar simulation device in the laboratory to simulate the unloading damage of the surrounding rock around the borehole in the peripressure environment, and the borehole was changed from a round shape to a rugby-ball-like shape, the top and bottom of the borehole collapsed, and the left and right sides were damaged to form a crushed zone. A statistical analysis of the fracture of the borehole was carried

out by using a borehole peeper, and three kinds of macro- and microscopic fractures in the borehole were found, namely, straight fractures, curved fractures, and bifurcation fractures.

Some scholars [166–168] tested the acoustic emission of coal samples with different degrees of fissure development and non-uniform distribution of fissures during the loading process, and the studies showed that the acoustic emission energy of the coal samples with uniform distribution of fissures occurred at the beginning of the elastic deformation, and then incremented to the vicinity of a stable value when the coal samples were loaded.

Yang et al. [169] estimated the fracture density of rock cores by measuring the P-wave and S-wave velocities. The results showed that there is a negative correlation between elastic wave velocity and fracture density. Additionally, the wave velocity of the rock cores varied significantly at different measurement frequencies, exhibiting strong dispersion characteristics. Yu et al. [170] investigated the fracture damage of coal samples under repeated water immersion and showed that with the increase in the number of immersion times, the closure stress threshold of the coal sample fracture, the threshold of the initial development of the fracture, and the threshold of the fracture damage are linearly reduced.

3.1.3. Current Status of Research on Perimeter Rock Fissures of Coal Rock Drill Holes Considering Fissure Roughness

The above studies mainly focus on theoretical analysis, numerical simulation, on-site monitoring, etc., but all of them consider the roadway or coal rock drilling holes in accordance with the regular shape, while in engineering practice, due to the blasting vibration, drilling rig vibration, and various factors of excavation and the rock body's own anisotropic, non-homogeneous, uneven distribution of pore space, joint, and other characteristics, which lead to the unevenness of the surface of the coal rock drilling holes and the roadway, the unevenness of the inner surface of the roadway is even higher. As we all know, considering engineering examples, the rock body damage and fracture generation preferentially occurs in a weak surface or local stress concentration point due to the contact surface unevenness and expands and develops in the weak surface or local stress concentration point. The roughness of the roadway or coal rock drill hole has an important influence on the shear strength of rock body damage. Therefore, the unevenness of the surface of the roadway or coal rock drill hole, i.e., the surface roughness, has a crucial influence on the damage of the surrounding rock and the development of the fracture distribution of the coal rock drill hole [119,124,151–153,171–177].

Yang et al. [178] prepared rock-like specimens of non-persistent jointed rock bodies containing circular holes and tested them under uniaxial compression by modelling with discrete element software PFC 4.0, which suggested that cracks were generated at the joint locations and concentrated at larger bulges, and the number of cracks was positively correlated with the JRC and normal stress. The results showed that the peak strength and modulus of elasticity of the non-persistent jointed rock samples varied in a “U” shape with respect to the joint inclination. Noori et al. [179] investigated the fracture behaviour and microcracking process of anisotropic sandstones by means of experimental tests and particle flow code (PFC2D). The fracture behaviour of anisotropic sandstones was modelled by bonding particles' discrete element model (DEM) and embedding weak surfaces (smooth contact model) in intact rocks and compared with laboratory results to verify the accuracy of the model.

Some experts [180,181] quantitatively characterised the roughness of rock discontinuity surfaces by employing fractal geometry methods, but the existing methods suffer from computational complexity, inaccuracy, and difficulty in characterising undulations and anisotropy. Through 3D laser scanning and ArcGIS data processing, the discontinuous surface morphology of artificial granite samples is quantitatively investigated, and an extended 3D fractal dimension containing three parameters is proposed, which is able to comprehensively characterise the roughness, undulation, and anisotropy and has the advantages of simplicity, high efficiency, and high accuracy. Fan et al. [182], in order to study the three-dimensional joint roughness coefficients (JRCs) by laser scanning technology, a

surface morphology test, and direct shear test, proposed a method to convert 3D surface morphology to a 2D mean height profile and calculated JRC values for 15 nodules based on statistical parameters, used the improved Barton empirical formula to calculate peak nodule shear strength, and verified the reliability of the method in laboratory studies.

Chen Shijiang et al. [183] systematically summarised the development of JRC and concluded that the fractal dimension is still an effective tool for quantitative description of JRC and the application of 3D printing technology can effectively promote the study of JRC. Liu Xige et al. [184] used 3D printing technology and PFC to carry out double rough structural surface shear test research and concluded that the reason why the shear strength of a double structural surface is lower than the shear strength of either structural surface is that the rough morphology of the structural surface leads to the destruction of the intermediate interlayer of rock in the first place, which then leads to the overall shear strength of the double structural surface being greatly reduced. Tang et al. [185] proposed a new peak shear strength formula based on Grasselli structural surface roughness coefficients, and the new shear formula conforms to the Mohr–Coulomb criterion and has a clear physical significance. Cai et al. [186] proposed a three-dimensional roughness research index I_{pap} that can reflect anisotropy and compared it with Grasselli roughness evaluation method in engineering applications. The validity of I_{pap} was verified.

3.2. Rational Sealing Parameters for Coalbed Methane Extraction

3.2.1. Coalbed Methane Extraction Borehole Leakage

In the case that the coal rock body is relatively intact and not affected by tectonic or external forces to form large joints through the coal rock body, the reasonable sealing parameters of the coalbed methane extraction borehole directly depend on the distribution of the fracture field around the borehole. A schematic diagram of the air leakage fissure field of a coalbed methane extraction borehole is shown in Figure 8 [119].

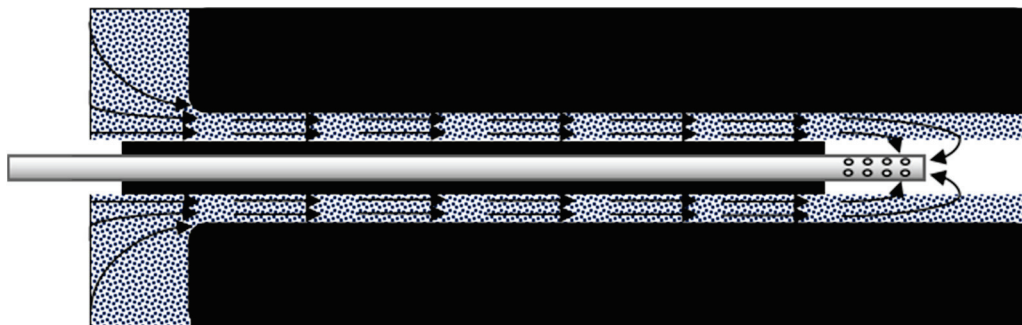


Figure 8. Schematic diagram of coalbed methane extraction borehole leakage fissure field [119].

The degree of air leakage from the borehole is an important indicator of the quality of the sealing hole, and there are three main ways of air leakage, and the actual air leakage is a combination of these three forms:

- (1) High permeability of the sealing material or the existence of cracks, resulting in the flow of air from the sealing material into the borehole, as shown in Figure 9a;
- (2) The hole sealing material cannot seal the fissures developed in the surrounding rock of the borehole, resulting in air leakage through these fissures, as shown in Figure 9b;
- (3) The depth of the sealed hole does not exceed the “zone of penetration between the fissure of the surrounding rock and the free surface”, which leads to the connection of the fissure with the fissure inside the sealed hole section and, consequently, the air leakage, as shown in Figure 9c.

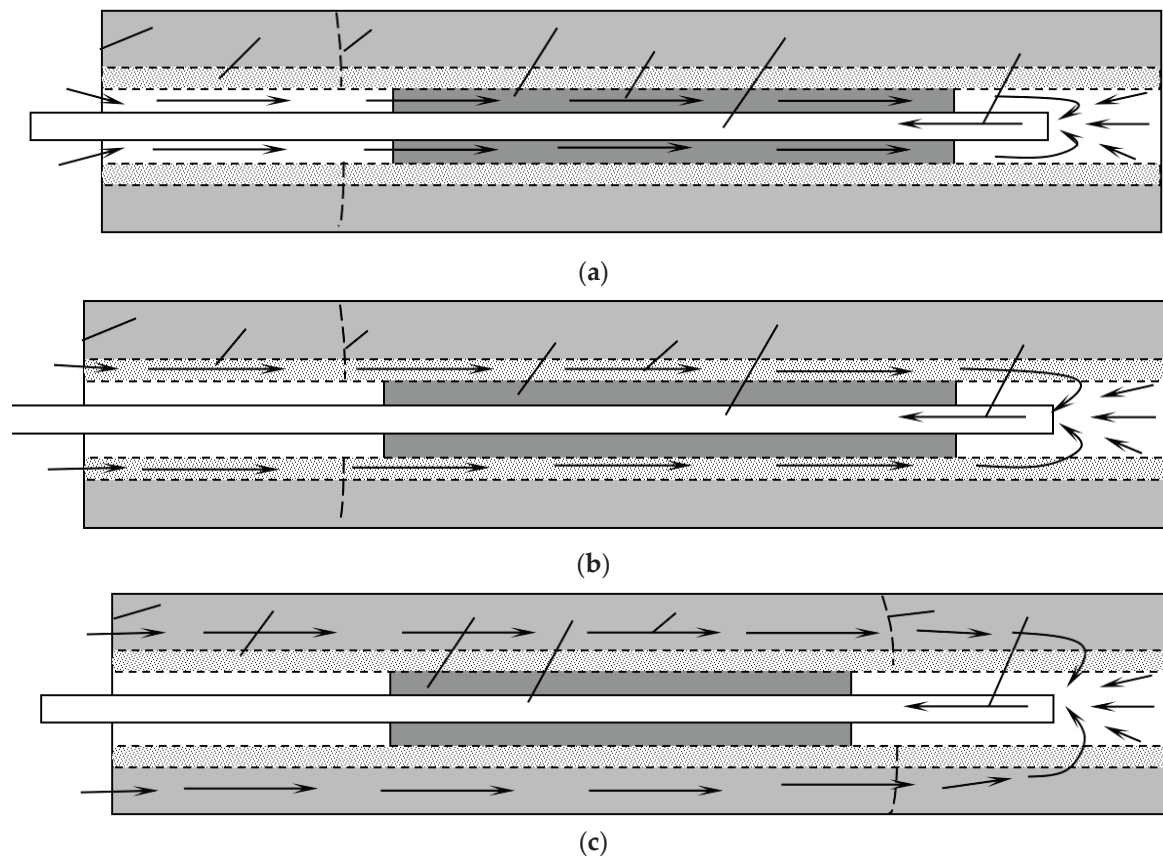


Figure 9. Schematic diagram of coalbed methane extraction borehole leakage mechanism [119]. (a) Leakage of air from sealing material. (b) Air leakage from drilled loosening rings. (c) Air leakage from the unloading zone of the coal body.

In order to simplify the problem of drilling “air leakage”, it is assumed that the surrounding rock around the drill hole is absolutely dense before excavation, i.e., the maximum pore and fissure scale is smaller than the diameter of gas molecules, and the permeability is zero. Then, the common point of the above three air leakage pathways is that the air leakage channel should eventually communicate with the free surface of the roadway. Even if the perimeter rock fissure of the drill hole has a high degree of development, if it is not ultimately connected with the free surface of the roadway wall, the drill hole will not leak air.

Therefore, it is possible to explore a classification based on whether fractures can connect with the free surface of the roadway, dividing the surrounding rock fractures into a “fracture-connected zone” and a “fracture-non-connected zone” with the free surface. Theoretically, it is difficult to judge whether the fissure is communicating with the free surface of the roadway, but in reality, there is permeability in the surrounding rock, and the gas inside the drill hole may seep into the roadway through the surrounding rock or the sealing material. Therefore, the problem of “air leakage” from the borehole is theoretically unavoidable.

Currently, research on permeability is relatively mature, and it is recommended to consider using permeability as a quantitative criterion for determining whether a borehole is “leaking” or whether the surrounding rock fractures are connected to the free surface. This can be specifically referenced against the critical permeability values used in standards for assessing the difficulty of coalbed methane extraction.

3.2.2. Study on Reasonable Sealing Depth of Coalbed Methane Extraction Boreholes

Reasonable sealing parameters for coalbed methane extraction mainly include sealing depth and sealing length. The fissures around the roadway and coal rock drilling holes are directly related to the sealing depth and sealing length of the underground coal seam gas drilling holes. Reasonable sealing depth and sealing length can block the leakage channel of the drilling holes, save costs, improve the efficiency of coal seam coalbed methane extraction, and ensure the normal development of the coal mine safety production work. Some scholars, based on the theory of loosening circles, unloading zone plasticity zone division, drilling destabilisation damage, and other bases, described the development of fissures in the peripheral rock of the roadway and coal rock drilling holes, and then determine the degree of reasonable sealing depth of coal rock drilling holes underground, and the related research results are abundant [7,18,119,167].

Some scholars think that the fissure system in the plastic zone of the roadway is very developed, that it is difficult to completely seal it, and it is better to choose the starting sealing depth of the drilling hole to avoid the plastic zone of the roadway [7,18,20,22], however, other scholars, on the basis of the distribution of the stress of the peripheral rock of the roadway as well as elasticity and plasticity characteristics of the coal rock body in the stress areas, concluded that the depth of sealing of drilling holes should be greater than the decompression area of the peripheral rock of the roadway but at the same time should be smaller than the depth of the peak stress point of the coal wall [187–191]. Some experts even conclude that air leakage can only be prevented when the sealing length of the drill hole exceeds the stress concentration area of the roadway [192].

When the depth of the sealing hole covers the peak stress point, the peak stress point and the sealing section correspond to the double peaks of the pressure distribution diagram of the surrounding rock of the drilling hole, which indicates that, in addition to the widely recognised rule that “the peak stress point has an adverse effect on coalbed methane extraction in the area, and there is a barrier effect”, there is also a similar effect in the vicinity of the sealing section. The main reason for the unsatisfactory coalbed methane extraction effect is that, near the peak stress point, the sealing section just covers the peak stress point, which is hindered by the blocking effect of the sealing section and the peak stress point on the flow of gas at the same time, and the superimposed effect of the two makes this place rely on the natural discharge of gas from the roadway, which ultimately leads to the formation of a gap in the coalbed methane extraction zone because very little gas can be discharged from the borehole; when the sealing depth of the coalbed methane extraction borehole is deeper than the peak stress point, there is a similar effect in the area. Although there is no superimposed obstruction effect of the sealing section and the peak stress point, the permeability of the coal seam near the peak stress point is low, and there is also a situation of high gas pressure and gradient when gas is discharged naturally only by the roadway.

3.2.3. Research on Reasonable Sealing Length of Coalbed Methane Extraction Boreholes

Based on the reliability of sealing materials, the impact of different sealing lengths on gas drainage effectiveness is minimal. In fact, longer sealing lengths may lead to a reduction in the effective drainage length, which in turn can result in a higher gas pressure gradient near the sealed section, thus adversely affecting the improvement of gas drainage efficacy.

At the same time, the increase in sealing length can be regarded as changing the reliability of the extraction system through the series connection of shorter sealing segments. Although the increase in sealing length can certainly improve the reliability of the extraction system, when the reliability of sealing can be ensured by shorter sealing segments, the significant increase in material and process costs is less effective. To sum up, under the condition of ensuring the reliability of the sealing section, reducing the length of the sealing hole appropriately can make the coalbed methane extraction effect improve.

It should be pointed out that in the current coalbed methane extraction work, it is difficult to ensure that the inner wall of the coalbed methane extraction borehole is completely well supported. As the coalbed methane extraction proceeds, the pore pressure of the coal body gradually decreases, and the unsupported area of the borehole will inevitably undergo damage, creating new fissures, which will affect the sealing effect of the sealed section of the coalbed methane extraction borehole. At the same time, due to the creep characteristics of the coal rock body, the stability of the coalbed methane extraction drilling and sealing section will inevitably deteriorate with the passage of time. The above two factors leading to instability or sealing deterioration of the borehole are difficult to solve by the existing sealing materials and techniques. If the sealing depth is in the unpressurised zone where plastic deformation has already occurred, the peak stress point will continue to move to the depth of the roadway as coalbed methane extraction proceeds, while the surrounding rock in the unpressurised zone of the roadway will be subjected to a stable state of stress and maintained at a small value. Therefore, under the premise of reliable sealing materials, trying to determine the sealing depth in the depressurisation zone may, on the contrary, have some reference value for the improvement of the sealing performance of the borehole.

3.3. Coalbed Methane Extraction Hole Sealing Process

3.3.1. Single-Plug Seal Drilling Technology

Single-plug seals are not common nowadays and can be divided into two forms: Non-grouted and grouted [9,193]. The non-grouting type is mainly seen in traditional clay (yellow mud) sealing or mechanical sealer and friction-type sealer used for shallow hole injection or shallow hole extraction. Polyurethane, cement rolls, capsules, etc. are mainly used as a single plug, and then the plug is placed in a suitable position according to the inclination of the borehole and then grouted. The single plug only relies on the self-weight of the plug and sealing material or the force with the inner wall of the borehole to achieve the sealing of the borehole. Mechanical sealers, single bladders, or polyurethane are used for temporary plugging of holes at the head of a coal mine or at the face of a rock portal for coalbed methane extraction. Mechanical sealers are relatively expensive compared to the bag sealers commonly used today and are difficult to use on a large scale to seal gas pre-extraction holes. The polyurethane foaming speed is faster, the strength is lower, and polyurethane material has a certain degree of toxicity. The “one plug, one injection” sealing method is mostly used for sealing drill holes with a large inclination angle. However, with the development of coalbed methane extraction sealing technology, there are more forms of sealing for inclined drill holes. The “one plug, one injection” sealing method does not allow for a pressure-retaining grouting process, which is not conducive to the penetration of the slurry into the fissure and does not allow for adequate sealing of the borehole.

In the early days, there was no special sealer for cement mortar sealing, and only the sealing cement was injected into the hole to seal it, so the length of the sealing hole was limited. The polyurethane sealing is used to install the chemical reaction “A” and “B” materials on the coil and then install the coil on the extraction pipe and send them into the hole together to seal the hole. Due to the special characteristics of the material, this sealer can only be made on site before use. Before the “two plugs and one injection” sealing process was widely used, most of the commonly used downhole sealers were mechanical sealers. The principle of this kind of hole sealer is to seal the hole opening section of the borehole through the deformation of the device itself and the hole wall to form a squeeze, friction, and other effects of fixing, and the corresponding hole sealers mainly include friction-type hole sealers and hydraulic expansion-type hole sealers. Mechanical hole sealers are simple in structure and easy to operate and can be reused, but the blocking part is mainly the drilled hole section, which cannot block the fissure inside the hole wall, so the blocking ability of the gas drilling hole is limited. In addition, the limitations of the principle and structure of the hole sealer lead to higher requirements for the drilled holes,

and the sealing effect is good for smooth and stable drilled holes. Therefore, this type of hole sealer is suitable for temporary sealing and not suitable for long-term work.

3.3.2. “Two Plugs and One Injection” Seal Drilling Technique

The “two plugs and one injection” technique is widely used in the field of coalbed methane extraction and hole sealing and was developed based on the principle of “solid sealing liquid, liquid sealing gas” in the pressure measurement technology of Zhou Shining, an academician of the Chinese Academy of Engineering [194]. “Two plugs” refers to the use of two sealing plugs, which can be either packers or fast-curing sealing materials. “One injection” refers to a single grouting process, where sealing material is injected into the confined space formed between the two plugs and the borehole wall [9]. At this stage, based on building on the “two plugs and one injection” method, advancements have been made by improving aspects such as plugging heads, sealing materials, borehole stress conditions, and automation. By continuously combining related technologies, several techniques have been developed, including “packer-type sealing”, “strong–weak–strong sealing”, “three plugs and two injections”, “integrated packer sealing”, “casing sealing”, “integrated multi-channel sealing”, “solid–liquid coupled fluid–wall sealing”, and “drill–cut–seal” techniques. Additionally, a multi-injection technology has been developed based on the initial single-injection method [9,121,171,193,195].

However, as the extraction continues, the fissures around the drill holes develop, which in turn induce gas leakage from the drill holes. In view of the coal seam coalbed methane extraction leakage and the dynamic characteristics of the fissures around the drill hole, experts and scholars in related fields have proposed the concept of dynamic sealing of coalbed methane extraction boreholes in order to achieve the purpose of uninterrupted repair of the fissures and leakage channels in the boreholes. The difference is that the former section has a limited number of grouting times, generally two times, while in the latter sealing section of the hole, there is a dynamic grouting section. To this end, researchers have developed the “multiple plugs and multiple injections” sealing methods, as shown in Figures 10–12, including the “three plugs and one injection” sealing method, the dynamic sealing method, and the integrated sealing method.

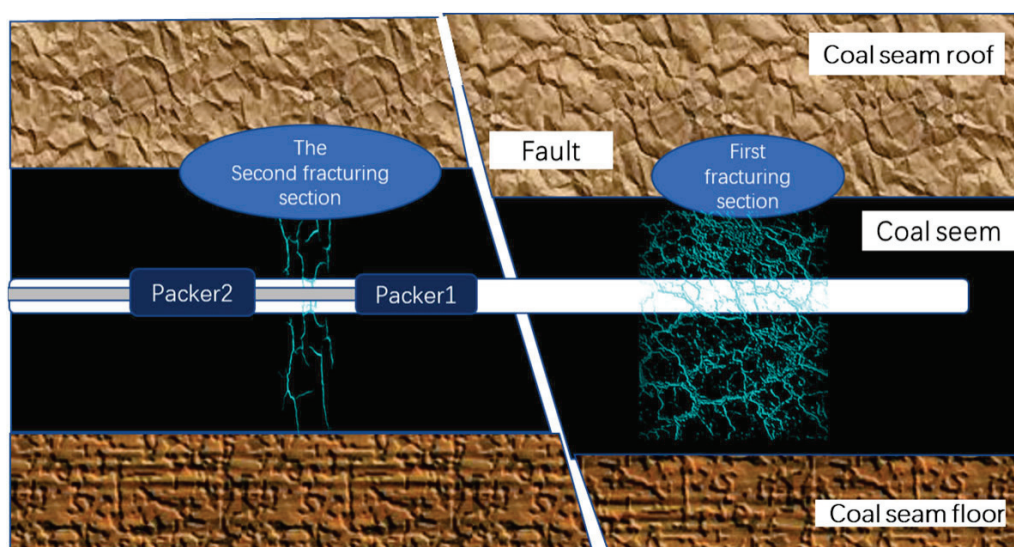


Figure 10. Dual-packer to avoid the structural stage fracturing [28].

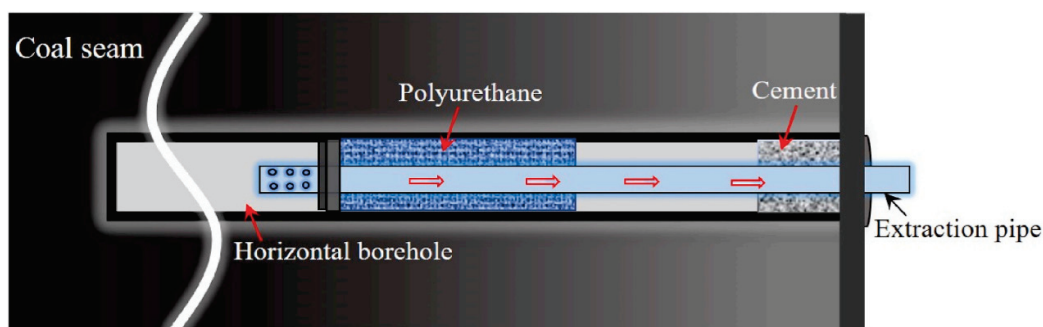


Figure 11. The process of the polyurethane plugging method [9] (reproduced with permission from Lou, Z. et al. *Gas Science and Engineering*; published by Elsevier; 2024).

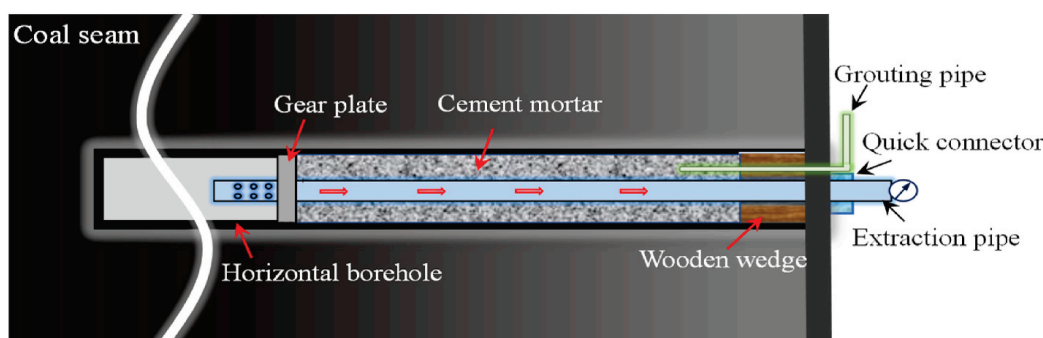


Figure 12. The process of the cement plugging method [9] (reproduced with permission from Lou, Z. et al. *Gas Science and Engineering*; published by Elsevier; 2024).

3.3.3. Other Seal Drilling Techniques

In addition to the above commonly used coalbed methane extraction borehole sealing techniques, a large number of scholars have attempted to study coalbed methane extraction borehole sealing techniques from other perspectives. Zhou Fubao et al. [8,21,196] proposed the concept of secondary sealing, which is used to plug the air leakage fissure by inputting ultrafine powder particles into the sealed drill holes, thus achieving the purpose of reducing air leakage and increasing gas concentration. Ji et al., by analysing the conventional negative-pressure pumping and nitrogen injection to strengthen the pumping technology, proposed the pulse-pressure nitrogen injection technology and carried out gas replacement experiments, which showed that pulse-pressure nitrogen injection is more efficient than uniform pressure and is more important for the efficient use of gas resources. It can improve coalbed methane extraction efficiency, reduce costs, and is of great significance to improve the prevention and control of gas disasters and the level of resource development [197]. Xiong [198] showed that, for the serious gas leakage of the coal seam borehole, the steps include the use of hydraulic slit cutting equipment to seal the hole section of the original material that is removed, cut slots in the boundary of the roadway stress elevation area, seal the borehole again, and perform grouting sealing again. After grouting and sealing, the sealing section of the borehole and the cement-filled groove around the borehole form an organic whole, which can effectively avoid the leakage of external air into the interior of the extraction borehole. Ren et al. [199], in order to solve the technical problem of low gas concentration and fast decay of the extraction borehole in the bottom extraction lane, analysed the mechanical characteristics of the lane and proposed the through pipe direct connection with pressure sealing technology, which eliminates the leakage of the extraction pipeline interfaces and then greatly improves the concentration of the coalbed methane extraction in the bottom extraction lane. The concentration of coalbed methane extraction in the bottom pumping tunnel has been greatly improved. For the sealing of downstream water-bearing drill holes, Rong et al. [200] proposed a complete set of technical solutions of

“sealing water in the area first, then drilling and sealing the holes”, and Zhai et al. [201,202] proposed a multi-hole parallel pressure–air drainage technology, and based on this, they developed a sealing process for downstream drill holes, which not only solved the difficulty of discharging water accumulated in the holes but also solved the problem of water leakage in the holes, which is difficult to drain. The technology not only solves the problem of water accumulation in the holes, which is difficult to discharge but also realises the integration of hole sealing, drainage, and extraction, which in turn improves the effectiveness of coalbed methane extraction.

3.4. Coalbed Methane Extraction Seals

3.4.1. Capsules

One of the sources of sealing capsule-type coalbed methane extraction sealing devices for coalbed methane extraction boreholes is the MWYZ-H I-, II-, III-type active coal seam gas pressure tester developed by the team of the North China Institute of Science and Technology led by academician Zhou Shining [194]. The basic principle of sealing holes in this device is that solid seals liquid and liquid seals gas. These instruments for measuring gas pressure have been widely used in coal mines. The connection of the solid capsules at the front and rear ends of the device and the connection of the external capsule use connecting rods. The main role of the front and rear capsule connecting rods is to connect the front and rear capsules into one and perform traction on the front capsule when recovering the manometer. The main role of the rear capsule orifice end connecting rod is to send the two sections of the capsule to a predetermined drilling depth and in the recovery of the pressure gauge traction rear capsule. The placement status of the existing capsule gas manometer in a coal rock body borehole is shown in Figure 13.

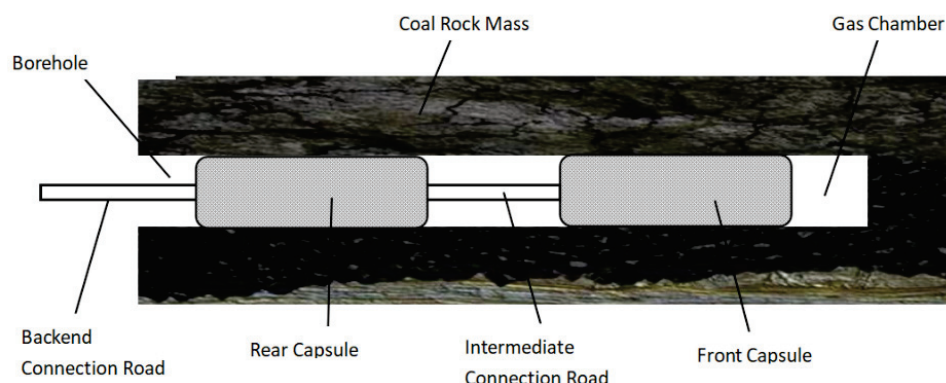


Figure 13. Schematic diagram of MWYZ-H active coal seam gas pressure tester.

The active sealing of the device on the pressure measurement borehole is mainly reflected in two aspects: Firstly, the solid hole sealing device at both ends will expand and deform under the action of the pressure of the internal expansion fluid to actively support and seal the borehole; secondly, the intermediate guard pipe will support the coal and rock body without large deformation, and the mucus (sealing material used for grouting) will actively seal the fissures around the borehole under pressure.

Bridge packers (Figure 14), i.e., special capsules, are the key components of hydraulic fracturing technology, which is widely used in the field of coalbed methane extraction (Figure 15). It has been found that the common type of failure of bridge packers is capsule rupture, which mostly occurs in sandstone and mudstone formations, with soft rock formations and capsule rupture (Figure 16) being the main factors. Preventing failures requires hard and stable rock formations, proper reaming, and strict quality control, and residual high-pressure water is a key factor leading to pressure relief failures and shot-rod events [203].

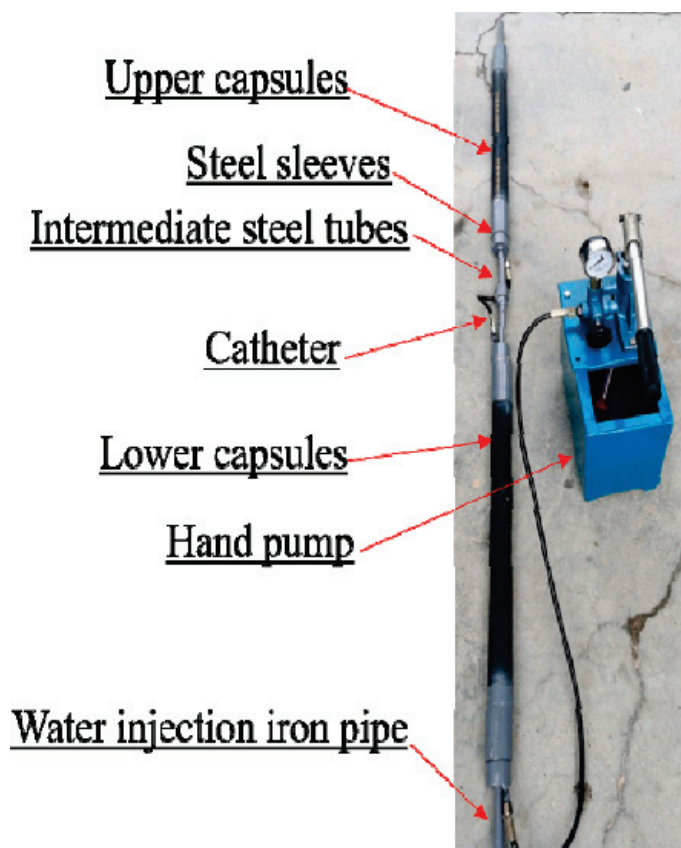


Figure 14. Structure of a coal mine hydraulic fracturing straddle packer [203] (reproduced with permission from Liu A., et al. Engineering Failure Analysis; published by Elsevier; 2023).

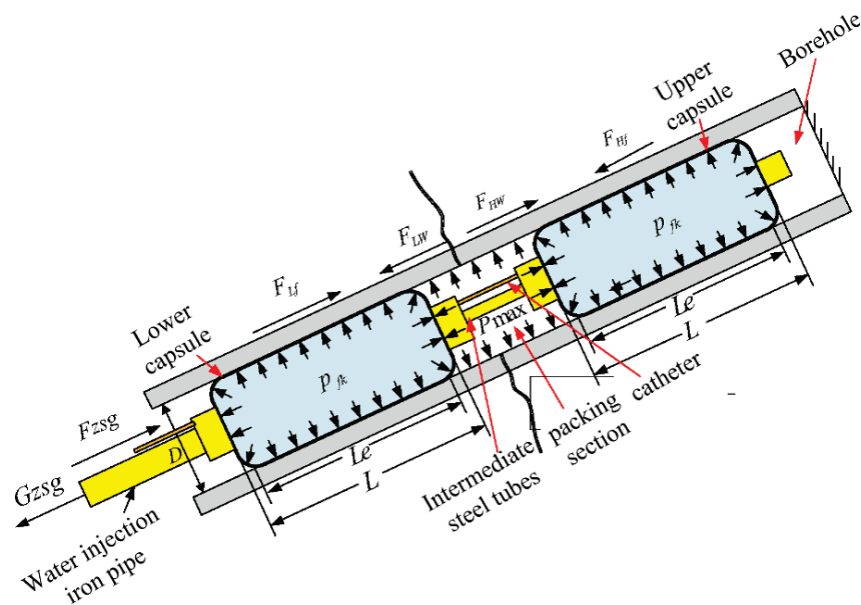


Figure 15. Schematic diagram of the sealing force experienced by the straddle packer [203] (reproduced with permission from Liu A., et al. Engineering Failure Analysis; published by Elsevier; 2023).

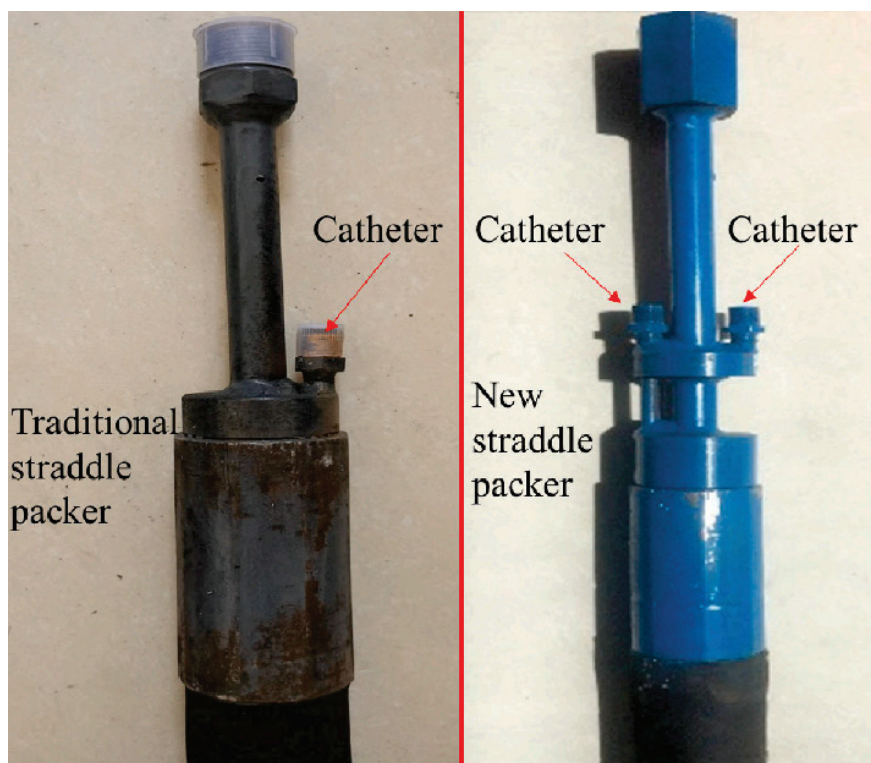


Figure 16. Schematic diagram of structure of the end of capsules [203] (reproduced with permission from Liu A, et al. Engineering Failure Analysis; published by Elsevier; 2023).

3.4.2. Capsular Bag

The capsule bag sealing device (Figure 17) was developed by Professor Sun Yuning's team at Henan University of Science and Technology, China, and it can be considered as a highly simplified version of the capsule sealing device, and the current “three plugs and two injections” or “new multi-capsule” devices can be considered as an upgraded and modified version of the traditional capsule type by increasing the number of capsules [9,23,204,205]. At present, the hole sealer is mainly researched on the basis of the “two plugs and one injection” sealing process, and researchers have developed hole sealers according to the analysis of the principle of the process, among which the most typical and widely used is the bag sealer. The hole sealer consists of a double-hole extraction pipe, composite bag, bursting valve, one-way valve, etc. The hole sealer has a compact structure and can efficiently seal the holes with pressure-retaining grouting, and the tests in many places have shown that this method has obvious advantages over the conventional polyurethane + cement “two plugs and one injection” hole sealing method.

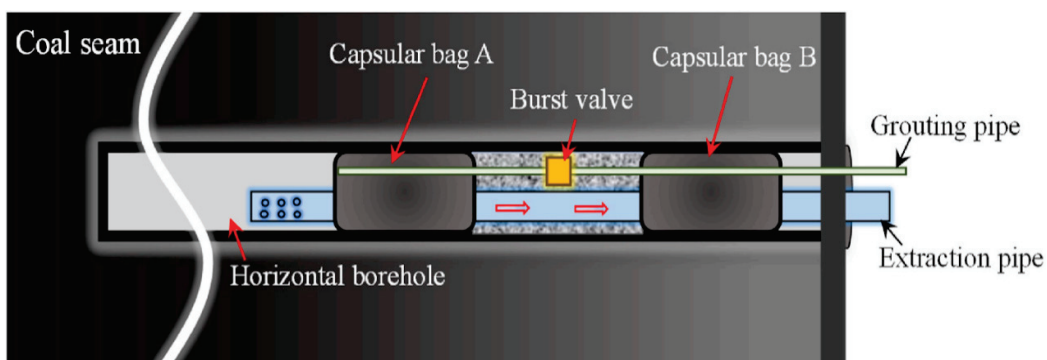


Figure 17. The process of the capsular bag plugging method [9] (reproduced with permission from Lou, Z. et al. Gas Science and Engineering; published by Elsevier; 2024).

Other scholars have developed a split hole sealer, which adopts a special pipeline to grout two separate bags and sets another pipeline to grout the intermediate section, which can guarantee the stable grouting of the bag to a certain extent, but due to more pipelines in the hole, the risk of leakage of the intermediate section of the high-pressure grouting between the two bags is increased, and it is not easy to realise the conditions of pressure-retaining grouting. For the problems of “two plugs and one injection” sealing, some scholars and enterprises have designed the “two plugs, one injection, and one row” grouting and sealing process. Compared with the traditional two plugs and one injection process, this technology adds one row of slurry pipe, and the industrial test results show that this technology can improve the concentration of coalbed methane extraction. The industrial test results show that this technology can improve the concentration of coalbed methane extraction. Some scholars have analysed the air leakage mechanism of the drilling holes of the bag sealer, obtained the sealing factors affecting the drilling holes, and thus designed a kind of split-type bag with pressure grouting sealer and the corresponding split-type bag with pressure grouting sealing technology, which improves the sealing effect, and the effect of gas control is more satisfactory. Some scholars have improved the material for filling the bag and developed a new type of bag sealer, which has better compactness and no obvious shrinkage after expansion of the bag, and the sealing effect is significantly improved compared with that before the improvement.

3.4.3. Other Hole Sealers

The principle of airbag sealing with pressure grouting sealing technology developed by Chen Team [206] is shown in Figure 18. In the figure, the two ends of the drilled hole sealing section are two sections of airtight airbags with good airtightness, and the expansion mode of the airbags is a chemical gas-producing reaction, so that the airbags will be expanded and the hole will be sealed by grouting in the airtight space in the middle of the two airbags.

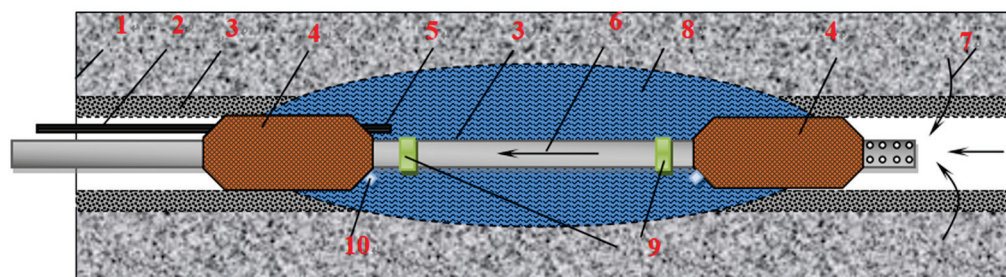


Figure 18. Schematic diagram of airbag sealing with pressure grouting to seal the hole. 1—Coal body; 2—Grouting pipe; 3—Loose ring around the borehole; 4—End airbag; 5—Slurry outlet; 6—Extraction pipe; 7—Gas flow; 8—Grouting material; 9—Injection one-way needle valve; 10—Through.

4. Sealing Materials for Coalbed Methane Extraction Boreholes

The coalbed methane extraction drilling sealing material is a material that plays a sealing role in coalbed methane extraction drilling, which can play the role of a single plug or can be used as a grouting material in “two plugs and one injection”. Under the most widely used “two plugs and one injection” sealing process, the slurry injected into the coalbed methane extraction borehole, under the action of reasonable injection pressure, spreads to the cracks of the coal body around the borehole and fills up the pores and concave–convex surfaces, and penetrates deep into the microcracks of the coal body, and the slurry, after solidification, solidifies with the coal particles, thus effectively blocking the air leakage channels around the borehole, thus improving the quality of the drilling process [7,9,113,120,204–207]. At the same time, as the coalbed methane extraction borehole sealing materials generally need to enter deeper into the borehole, the coalbed methane extraction borehole sealing materials are generally liquids, which need to enter into the borehole by means of a grouting pump or other similar means. Therefore, the coalbed

methane extraction borehole sealing materials can be classified into solidifiable and non-solidifiable materials.

4.1. Solidifying Sealing Materials

At present, solidified coalbed methane extraction borehole sealing materials can be broadly classified into two categories, inorganic and organic, according to their chemical compositions, where inorganic materials mainly refer to cement-based sealing materials and organic materials mainly refer to polymer materials [9,120].

4.1.1. Cementitious Sealing Materials

Cement-based hole sealing materials are not only widely used as grouting-type sealing materials but can also be used individually as plugs for drilling sealing, such as cement roll anchors. Cement-based materials are most widely used due to cement being widely available and low cost and cement-based materials exhibit a wide range of characteristics, including compatibility with fly ash, red mud, and other industrial by-products and solid wastes, making them the most widely used materials in construction.

Cementitious materials are the most widely used sealing materials, which cover ordinary cementitious composites and advanced cementitious materials, such as flexible paste materials, high-water materials, PD materials, ultrahigh-water materials, geopolymer materials, and so on.

Ordinary cement-based materials are the most adaptable to a variety of drilling conditions, and slurry viscosity and setting time can be controlled by adjusting the ratio, so that the length of the sealing hole can be guaranteed, and cement can be combined with various industrial by-products, such as fly ash, red mud, other industrial materials, and solid waste, with the formation of a variety of cement-based compliant materials, so as to seal the hole to further reduce the cost of materials. The obvious shortcoming of cement-based materials is that the existence of capillary pores in the late stage of cement hardening can lead to a certain degree of contraction of cement, resulting in a certain degree of crescent-shaped cracks between the cement and the upper part of the drilled hole, which affects the sealing quality of the drilled hole. At present, the main two ways to improve the late contraction of ordinary cementitious materials are to reduce the capillary pores that lead to the dry shrinkage of the hardened body in the late stage of the material and to develop new types of grouting and sealing materials, so that the late stage of the material can still be slowly expanded to compensate for the shrinkage of the hardened body. Some scholars [208,209] showed that when the dosage of fly ash is greater than 50%, the cement hardening body hazard hole capillary pores can be reduced more substantially. Some scholars [210,211] further analysed the various durability of cement slurries doped with fly ash from various perspectives. Lim et al. [212] showed that sand also has an important influence on the performance of cement slurries and that the strength and durability of the cured body of fine sand cement slurries are better than those of coarse sand cement slurries under the condition of high water–cement ratio. Microexpansion is a conventional method to improve cement-based grouting and sealing materials, but how to ensure that the expansion properties are not consumed completely in the early stage and continue to act in the late hardening stage to make up for the shrinkage of the material in the late stage is still a difficult problem, so how to control the expansion agent in the late stage of the hardening of the material to release the force slowly is the direction of future research [213–219].

Advanced cementitious composites include flexible paste materials, high-water materials, PD materials, ultrahigh-water materials, etc. Xiang et al. [220,221] developed a flexible paste sealing material by using various polymers, admixtures, expansion components, and so on as cement additives. High-water material is a new type of material. A and B are two components in the water–cement ratio of 2.5:1 after mixing with water, the pumpability is good, the reaction time is controllable, and the compressive strength of the hardened body is higher in the later stage [222,223]. Zhai et al. [224] developed a new type of mining sealing material PD by using microencapsulation technology, which can make the expansion

force release slowly, but its setting time is long and the cost is high. China University of Mining and Technology developed higher water–cement ratio UHW materials and studied the setting time and compressive strength under various factors, the basic mechanical properties, micromorphology, and stability of the materials [225]. High-water materials and ultrahigh-water materials have low cost due to their high water–cement ratio, but there are certain defects in their strength and stability, which need to further improve. Some scholars have developed a high-strength geopolymer gel material based on fly ash and found that the appropriate amount of CaO in fly ash has a positive promotion effect on the development of the solid body of this material, and some scholars have prepared polymer grouting materials using fly ash, cement, and water glass as raw materials [226]. Although geopolymer materials have the advantages of being green and low raw material cost, geopolymer materials have more components, so it is difficult to control the stability of the materials. In addition, silicate cement, slag, fluorine gypsum, and quicklime were selected as raw materials to prepare grouting and sealing materials for coalbed methane extraction boreholes, and experimental studies have shown that the material has excellent expansion properties, but its compressive strength is only 3–5 MPa. Hao et al. [227] developed a method to prepare an environmentally friendly, mould-resistant, and long-lasting moisturising phase-change gel (EPCG) through a network of modified cellulosic gel polymer. The gel has an adjustable crosslinking time and an initial solidification time of 30–60 min, with excellent water retention and mould resistance to ensure a long-term sealing effect.

4.1.2. Polymer-Based Sealing Materials

Polymer material has the advantages of fast reaction speed, high adhesion, convenient operation, etc. and is not only used for a single plug in the inner wall of the borehole but also widely used as a spraying material for gas management. Common polymer sealing materials are polyurethane, phenolic resin, urea-formaldehyde resin, and so on. Polyurethane is the most commonly used organic polymer material, with the advantages of low density, large expansion coefficient, easy construction, etc. [228–232]. Polymer-based materials can be used to make plugs due to their quick-setting characteristics. Some scholars have studied the expansion and permeability of polyurethane and applied it in the construction process. Polymer-based sealing materials generally have excellent flexibility and abrasion resistance and can adapt to the deformation of the borehole wall. Meanwhile, their fast-curing properties make them excellent in emergency sealing needs, with high mechanical strength and good chemical resistance. However, polymer-based drill hole sealing materials are susceptible to ageing and have low strength, and their long-term stability still requires further research.

4.2. Non-Consolidating Sealing Materials

Non-consolidated sealing mucus material throughout the whole course of the coalbed methane extraction borehole is kept in a non-consolidated state and can effectively achieve the sealing of coalbed methane extraction boreholes [9,195,233]. The development history of mucus sealing materials is relatively short, still relying on the development of the pressure measurement method of Zhou Shining's rubber ring–mucus. In recent years, there have been mainly bentonite-based mucus materials and polymer mucus materials of two major categories. Cheng Jianwei's team [197,233] developed non-consolidated inorganic mucus-based materials and studied the grouting course and intelligent applications. Some scholars [198,199] created a new type of sealing slime based on sodium bentonite and studied the swelling, water retention, and sealing properties. Some scholars [200] developed sealing slime with superior performance based on polymer anionic cellulose ether and carried out the development of an adaptive dynamic sealing technology and system for coal seam drilling. Some scholars developed highly absorbent resin-based mucilage sealing materials and carried out performance improvement and field application.

A comparison of the performance of sealing materials for coalbed methane extraction boreholes is shown in Table 1.

Table 1. Comparison of sealing material properties for coalbed methane extraction boreholes.

Sealing Material	Labour Intensity	Process Complexity	Compressive Strength	Bonding Strength	(Manufacturing, Production, etc.) Costs	Hole Sealing Effect
Cementitious material	High	Moderate	High	Moderate	Relatively low	Good
Polymer	Low	Simple	Low	Good	High	Moderate
Non-consolidating sealing materials	High	Intricate	-	-	Moderate	Good

4.3. Study on the Transport Pattern of Sealing Materials in Coalbed Methane Extraction Boreholes

In order to improve the sealing effect of sealing materials in the fissures of the borehole perimeter rock, in the research on sealing materials for coalbed methane extraction boreholes, in addition to the research and development of sealing materials, a large number of researchers will further study and analyse the grouting diffusion law of sealing materials, especially the whole process of sealing slurry in the dynamic sealing of fissures, by means of theoretical analyses, laboratory experiments, and numerical simulations. A large number of slurry transport research results in the field of geotechnics can be lent to the field of sealing hole grouting for coalbed methane extraction, and some researchers will further investigate the slurry transport and diffusion law in the cracks in a situation closer to the actual situation, i.e., considering the roughness of the cracks.

4.3.1. Grout Diffusion Theory without Considering Fracture Roughness

Dupla et al. [234] concluded that the gradual accumulation of cement particles in porous media has a significant effect on the grouting process. Zied Saada et al. [235] presented a theoretical model for the seepage of cement under percolation in porous media. S. Maghous et al. [236] presented a numerical solution for the model equations of the percolation effect. Olivier Chupin et al. [237] investigated the quantitative relationship between grouting pressure and the percolation effect. Jong-Sun Kim et al. [238] concluded that the percolation effect plays a dominant role in the one-dimensional permeation of slurry and the quantitative relationship between grout pressure and the percolation effect. Chupin et al. [237] also studied the quantitative relationship between grouting pressure and the percolation effect and concluded that the percolation effect plays a dominant role in one-dimensional infiltration of slurry. Jong-Sun Kim et al. [238] established a test setup that takes into account the change in porosity and the time-varying viscosity due to the filtration of slurry particles.

B. Amadei and W.Z. Savage [239] obtained the analytical solution for the unsteady flow of Bingham slurry in a parallel planar fracture. Chun-I Chen et al. [240] obtained the pressure and velocity distributions for the unsteady flow. H.K. Moon [241] obtained that the flow of the slurry will be stopped when the slurry's pressure gradient meets certain conditions. M. Eriksson [242] investigated the flow pattern of slurry in a network of planar "channels". F. Bouclielaghem et al. [243] also investigated the flow patterns of slurry in a network of planar "channels" and the low-viscosity hydrodynamic properties of slurry during infiltration. These scholars derived theoretical formulas for the flow of different fluids as well as time-varying viscosity Bingham fluids in the fractured rock, determined the grouting pressure in line with the actual situation in the field, used Comsol Multiphysics to derive a reasonable grouting pressure, and investigated the flow of slurry within the rock fissures as well as the correlation between the grouting pressure and the opening of the slit by means of the UDEC. The effect of slurry diffusion under different grouting pressures was analysed, and it was concluded that the permeability of the coal seam was the key factor determining the effect of slurry penetration.

4.3.2. Diffusion Theory under Consideration of Fissure Roughness

The roughness of the tunnel or coal seam boreholes not only significantly affects the damage to the surrounding rock and the distribution and development of fractures but also results in fractures with relatively small widths and a certain degree of roughness. This roughness, in turn, exerts a considerable influence on the flow and diffusion of the grout during the grouting process.

Li et al. [244] used the discrete fracture network to establish the fractured rock model, deduced the correspondence between the fracture frequency and Hoek–Brown criterion parameter by calculation, and simulated the anisotropy degree of the rock strength after grouting reinforcement and the stress–strain curves in various directions and under various conditions of grouting cohesion with the help of PFC, which showed that the degree of the anisotropy of the rock body was obviously reduced after grouting, and the mechanical properties gradually changed from ductile to brittle based on the quantitative relationship between JCond and Hoek–Brown criterion parameters. Based on the quantitative relationship between the parameters of JCond and Hoek–Brown criterion, they established the relationship between grouting reinforcement and its impact on rock strength. Some scholars believe that joint roughness significantly affects the damage area and peak shear strength of rock joints. Through theoretical analysis and numerical simulation, it is found that the angle and height of roughness quantify the scope and degree of damage in the shear process, respectively, and through direct shear tests and 3D scanning tests, the morphology parameters for quantifying the damage area on the surface of the joints are proposed, and based on these parameters, a new model for the peak shear strength is constructed. The results show that the model performs well in predicting the shear strength of other nodal samples [245]. Yi et al. [246] investigated the fluid flow in rough microchannels based on the fractal theory, derived an analytical model of the friction factor and Poisson's number with respect to the relative roughness of an incompressible fluid without empirical constants when passing through the rough microchannels under laminar flow, and derived an analytical model of the relative roughness of the fluid flow in the rough microchannels based on the capillary bundle model of a porous medium. Based on the capillary bundle model, a fractal analytical expression for the permeability of the rough capillary bundle type is derived, and the effect of relative roughness on the relative change in permeability is discussed.

The flow law of fluid in rectangular and circular micron-scale channels was partially investigated by experiments, and the results showed that the shape of the microchannels had no significant effect on the flow resistance characteristics, the inlet effect of the microchannels was more obvious, and the local resistance of water flow in the mutant microchannels was in accordance with the conventional theoretical curves.

Some other studies have shown that, based on fractal geometry, the Weierstrass–Mandelbrot function can be used to describe the surface roughness of multiscale self-affine, a three-dimensional model for laminar flow in the channel was established, numerical simulations were carried out for the effect of surface roughness, and it is pointed out that the Poiseuille number of the rough microchannel increases approximately linearly with the Reynolds number. The higher the relative roughness, the more obvious the flow pressure drop caused by the backflow and separation of the flow. A molecular dynamics model of the coupled process of fluid flow and heat transfer in the rough nanochannel is established, and the influence of the solid–liquid interaction strength and wall stiffness on the velocity slip and temperature step at the interface is analysed, pointing out that under the action of the external force, the velocity distribution in the mainstream region of the nanochannel is parabolic, and the viscous dissipation of the nanochannel caused by the fluid flow makes it more difficult for the fluid to flow through the nanochannel. The viscous dissipation inside the nanochannel leads to a quadratic distribution of the temperature.

5. Future Directions

5.1. Gas Transport Theory

Over the past decades, gas workers have extensively studied the process of gas transport in coal seams, and many theoretical and empirical/semi-empirical models, such as the mathematical model of gas adsorption and desorption and the model of biporous media, have been established. In the theoretical models, most scholars have not reached a consensus on the mechanism of gas flow in coal matrix, and most of them adopt Fick's law to describe the process of gas diffusion, while some scholars believe that Darcy flow and Fick's diffusion coexist in the gas flow in coal matrix, and some scholars believe that the gas flow in coal matrix is more in line with Darcy's law. As for the empirical/semi-empirical models, these models are mostly based on the experimental results or measured results of coal samples from specific mining areas, and it is difficult to prove that they are suitable for coal samples from other mining areas, which makes it difficult to promote their application and has certain limitations. Moreover, some of these models can accurately describe the initial stage of the desorption process but cannot predict the whole desorption process. Some models can accurately predict the final desorption amount but cannot match the desorption trend during the desorption process. Therefore, there is still a lack of models that can be applied to the prediction of the entire desorption process.

In addition, coal is a porous medium composed of matrix and fissures, and scholars have mainly analysed the problem of the apparent phenomenon of desorption when studying the change in gas desorption characteristics during the crushing damage process, used empirical models, or used classical single-porous or double-porous models to fit certain results. When analysing the variation of the desorption curve, the influence of pore changes on the gas desorption curve is mostly analysed qualitatively, and the quantitative characterisation of the diffusion coefficient into a mathematical form containing the parameters of the pore structure is not included. In addition, the assumption of homogenisation of the diffusion coefficient does not fully consider the successive destruction relationship between matrix and fissure. Therefore, there is no clear mathematical model as a bridge between the change in the structural parameters of the pores of the coal body during the crushing process and the morphological change in the desorption curve.

Since the transport of gas in coal seams is affected by many factors, the following outlook is proposed for the research on the mechanism of gas transport:

- (1) Establishment of multi-scale and multi-physical field coupling model: In view of the different views on the gas flow mechanism in the coal matrix in the existing theoretical models, future research should be devoted to the establishment of a multi-scale and multi-physical field coupling model, which will comprehensively describe the theory of gas transport mechanisms in the coal seam by taking into account the microstructure of the coal matrix, the adsorption and desorption, diffusion, and seepage of the gas, and the process of the gas flow.
- (2) Application of advanced experimental techniques: Tracer technology, non-destructive testing technology, high-resolution imaging technology, microscopic observation technology, new sensing technology, and other advanced experimental means should be applied to carry out refined observation and measurement of the transport process of gas in the coal matrix, obtain more accurate experimental data, and provide a reliable basis for the improvement and validation of the theoretical model.
- (3) Unified description of Fickian diffusion and Darcy flow: In-depth study of the mechanism of gas diffusion and seepage in coal matrix, exploring the interrelationship between Fickian diffusion and Darcy flow and their scope of application under different conditions, should be carried out. Attempts will be made to construct a gas transport model that can unify the description of Fickian diffusion and Darcy flow to provide a more accurate theoretical basis.
- (4) Widely applicable empirical/semi-empirical models: In response to the problem that empirical/semi-empirical models are limited to specific mining areas, future research should be devoted to the development of empirical/semi-empirical models

with wider applicability. Through large-scale and multi-region experimental data accumulation, combined with machine learning and other data-driven methods, empirical/semi-empirical models that can be widely applied to different mining areas should be established, so as to improve the versatility and applicability of the models, or to respond to China's "one mine, one policy" and "one side, one policy" for gas management and to put forward proposals that can be adapted to specific mining areas. Or, in response to China's "one mine, one policy" and "one side, one policy" for gas management, we propose high-precision models adapted to specific coal mines or workings to solve the problem of inaccurate loss of raw gas content measurement.

- (5) Research and development of the whole desorption process prediction model: Currently, there is a lack of models that can accurately predict the whole desorption process. Future research should focus on developing a whole process prediction model that can cover the initial stage of desorption, the intermediate process, and the final amount of desorption to understand the dynamics of gas desorption process in depth and combine with advanced numerical simulation methods to optimise the existing model or propose a new model, which can accurately describe and predict the whole process of desorption dynamics and behaviour of gas desorption.
- (6) Quantitative characterisation of the whole process diffusion coefficient: Quantitative characterisation of the diffusion coefficient into a mathematical form containing the pore structure parameters, and in-depth study of the influence of the change in pore structure on the gas desorption curve, should be carried out. By combining experiments and numerical simulations, a clear mathematical model is established to quantitatively describe the relationship between pore structure parameters and gas diffusion coefficient.
- (7) Relationship between matrix and fissure destruction time effects: In-depth study of the destruction time effects of coal matrix and fissures and their influence on gas transport and construction of a mathematical model that reflects the relationship between matrix and fissure destruction should be carried out. Through experiments and numerical simulations, we will clarify the connection between changes in pore structure parameters of coal and changes in desorption curves and provide a more accurate description of gas transport.

5.2. Coalbed Methane Extraction Hole Sealing Technology

The decay of the coalbed methane extraction concentration is mainly due to the influence of the roadway and the unloading pressure of the borehole. The coal rock borehole under the action of the geostress, with the prolongation of the extraction time, gradually undergoes creep, which leads to the continuous development of the fissure channel between the peripheral rock fissure of the coal rock borehole and the free surface of the roadway, resulting in the inflow of air into the borehole, so that the effect of the coalbed methane extraction is poor [8].

At present, there is no uniform understanding about the reasonable depth of sealing holes, which mainly revolves around whether the depth of sealing holes should exceed the stress peak point in the stress concentration area or reach the original stress area. There is not enough research on the fissure zone under the coupling effect after the excavation of the roadway and the secondary excavation of the coalbed methane extraction borehole, and at present, the fissure zone is only divided into the concepts of loosening circle, unloading circle, plastic zone, crushing zone, the original rock stress zone, and so on. Therefore, there are more debates about the reasonable sealing hole depth determination, especially whether it exceeds the stress peak point of the stress concentration area. As we all know, for hydraulic cutting, hydraulic fracturing, and other means of penetration enhancement in the case of a fissure not connected with the free surface of the roadway, the development of a fissure is instead conducive to coalbed methane extraction, leading to the root cause of drilling leakage which is the coal rock drilling peripheral rock fissure channel communicating with the free surface of the roadway, and the existing research

on the fissure zone is more detailed and in line with the requirements of the pumping requirements, that is, regarding whether the fissure is connected to the roadway, there are fewer research studies.

As the main control factor for the quality of coalbed methane extraction drill hole sealing, the depth of the sealing hole directly affects the effect of coalbed methane extraction from the coal seam. The majority opinion is that, when sealing coalbed methane extraction drilling holes in the actual site, due to the excavation and unloading effect of the roadway and drilling holes successively, the fissures in the loose circle of the peripheral rock of the drilling holes have a high degree of development, and it is often difficult to realise that the fissures in the coal body around the sealing holes are completely filled by the sealing materials, which will reduce the concentration of the gas extracted from coalbed methane extraction drilling holes and affect the effect of the extracted gas. The authors of this paper believe that the above statement is debatable. The authors of this paper believe that the quality of fissure sealing is mainly related to the nature of the sealing material, the scale of the fissure, and a series of related factors, but it cannot be stated that the fissures of the loose circle cannot be sealed because of the high degree of fissure development. Otherwise, because there is a small loose circle in the borehole itself in the radial direction, hole sealing material inevitably moves first along the path of least resistance to penetrate into the loose circle in the radial direction of the borehole, causing the formation of a hole sealing material priority transport path. According to the above theory, the cracks in the loose ring cannot be sealed due to the development of cracks, and the sealing material will inevitably flow along the axial direction of the drill hole through the loose ring of the drill hole out of the roadway (upward to the drill hole) or flow into the bottom of the drill hole (downward to the drill hole), and it is impossible for the drill hole to be sealed. On the contrary, the current study shows that the drill hole can achieve a better sealing performance in the case of reliable sealing material, that is, it shows that sealing material with superior performance can complete the sealing of the loose ring to a certain extent.

Based on the research results in the field of coalbed methane extraction sealing technology, the following research outlook is proposed:

- (1) Deepen the research on the mechanism of fissure development: Due to the complex field conditions of mines and the non-homogeneous and anisotropic characteristics of the coal rock body, there is still a lack of comprehensive understanding of the mechanism of fissure development. In the future, we can combine on-site monitoring and laboratory simulation to further reveal the interaction mechanism between fissure development and the mechanical behaviour of coal rock body and gas transport through multi-scale and multi-physical field coupling methods, especially considering the characteristics of the coal rock body surroundings of the drilled hole under the conditions of pressure unloading, fissure roughness, and supporting or sealing section.
- (2) Optimise the sealing depth and length: Introduce advanced monitoring technology, such as 3D scanning, sensor network, etc., to monitor the fissure development and permeability change around the borehole in real time to provide accurate data support for the coalbed methane extraction process, to quantitatively characterise the permeability threshold value of the deep fissure of the borehole through the roadway from the measured data, to take the threshold value with a certain safety coefficient as the reasonable sealing depth, and to optimise the sealing length according to the support function of the sealing section. Then, according to the support function of the sealing section, the sealing length can be optimised.
- (3) Automation and intelligent sealing technology: With the construction of intelligent mines, automation and intelligent sealing technology will become a research hotspot. Through the introduction of sensor technology, artificial intelligence, and robot technology, and drawing on the high degree of automation in equipment such as synthesis excavators and shield machines, we will develop a complete set of multi-functional and integrated sealing technologies such as coalbed methane extraction sealing devices and sealing processes, which integrate sealing, drainage, and extraction and can

realise the whole process of sealing, automatic monitoring, real-time adjustments, and remote control of the intelligent sealing process, so as to improve the precision and efficiency of the sealing process.

5.3. Sealing Materials for Coalbed Methane Extraction

Scholars have performed a lot of research on sealing materials for coalbed methane extraction, and the existing widely used sealing methods are mainly “two plugs and one injection” sealing methods, which are characterised by one-time static sealing. The grouting and sealing materials in the middle of the two plugs are mostly solidification-type materials. However, due to the non-homogeneous nature of the rock surrounding the drill hole, the displacement generated by the creep of the drill hole is also non-homogeneous, and the solidified grouting sealing material cannot dynamically seal the cracks in the coal rock drill hole, resulting in the attenuation of coalbed methane extraction from the drill hole over time. Due to the superimposed effect of unloading pressure from roadway and borehole excavation, the coal rock body around the borehole can be regarded as composed of coal rock blocks cut by fissures, and the main channel for the flow of grouting material is the fissures due to the weak permeability of the coal rock blocks. These fissures are important factors affecting the stability of the extraction borehole because of their low strength and deformation parameters and poor physical properties. The geometrical characteristics and distribution form of the fractures in the coal rock body around the drill hole determine the seepage flow with large anisotropy.

In the process of grouting and sealing, the sealing material gradually fills the fissures in the surrounding rock, and after diffusion and solidification, it makes the densification of the coal and rock body around the borehole increase, and the sealing is improved. As the grouting pressure continues to increase, the splitting phenomenon at the tip of the fissure will be induced, and the fissure continues to expand, leading to an increase in the grouting volume. These splitting situations will not only cause a large amount of grouting and sealing materials to be lost but may also rupture part of the intact coal and rock body, going against the purpose of sealing the borehole and exacerbating the degree of air leakage in the borehole. Furthermore, because the seepage process of slurry during the process of grouting and sealing of drill holes is imperceptible, it is currently impossible to monitor the spreading range of slurry and sealing of fissures during the actual construction process. Therefore, how to strike a balance between grouting and sealing materials to seal the fissures and split fissures in the perimeter rock of the borehole becomes a scientific problem that needs to be solved urgently. Meanwhile, studying the impact of non-curing grouting sealing materials on coal seams of different metamorphic grades is essential for improving the sealing effectiveness of boreholes.

The inability to determine the range of slurry diffusion and the degree of sealing fissures not only leads to blindness and uncertainty of the grouting sealing parameters of the borehole but also means that the effect of grouting sealing cannot be judged, especially in complex geological conditions. The spacing of extraction boreholes, the size of the grouting pressure, the length of the grouting time, the ratio of slurry, etc. have a direct impact on the effect of grouting sealing and the stability of the drill hole [88]. Meanwhile, most grouting sealing materials are fluids with time-varying viscosity, but there are few studies that consider the time variation of viscosity when the grouting sealing material seals the fissures of coal and rock bodies around the drill holes, and the viscosity of the grouting sealing material at different locations in the diffusion area of coal and rock body fissures around the drill holes also changes with time.

Therefore, based on the existing research results, the following research outlook is proposed:

- (1) Intelligent sealing materials. The application of intelligent technology in sealing materials for coalbed methane extraction boreholes is promising. In the future, intelligent sealing materials can adapt to the complex and changing conditions of coal seams through self-perception, adjustment, and repair functions. Adopting intelligent

control systems and sensor technology, real-time monitoring and dynamic adjustment of sealing materials can be realised to ensure the continuous stability of hole sealing effects. In particular, non-consolidated or liquid sealing materials can facilitate intelligent pressure replenishment and other operations in the whole process of coalbed methane extraction through the grouting system.

- (2) In-depth study of sealing material transport laws. Future research will continue to analyse the grouting diffusion behaviour and sealing effect of sealing materials under different conditions by means of experiments and numerical simulations. In particular, considering the time-varying viscosity of grouting material, the complexity of fissure roughness, and pore structure, a finer transport model will be established to provide a theoretical basis for the optimisation of sealing material.

6. Conclusions

Scholars have conducted a lot of research on the theory of coal seam coalbed methane extraction and hole sealing, coalbed methane extraction and hole sealing devices, coalbed methane extraction drilling sealing material, and supporting coalbed methane extraction and hole sealing technology and have achieved a lot of innovative results and solved a large number of theoretical and technical problems. However, the gas disaster accidents that occur from time to time also show that there is still a great demand for the development of the theoretical technology and materials of coalbed methane extraction hole sealing. With the continuous advancement of coal mine intelligent work, coal mine coalbed methane extraction intelligence, platforms, and systems are also imminent.

- (1) Research on basic theories related to coalbed methane extraction. The theory of gas flow in coal matrix still needs to establish a variety of factors coupled in the porous medium gas transport model, and through gas diffusion and molecular dynamics simulation, lattice Boltzmann simulation and gas transport multi-scale simulation, and other molecular simulation methods, combined with laboratory experiments and field data, a new microscopic scale under the gas flow mechanism can be further put forward and then guide the reasonable selection of gas pumping hole sealing parameters.
- (2) Research on coalbed methane extraction hole sealing technology and devices. The existing coalbed methane extraction sealing technology and device research is relatively mature, but the degree of automation is not high, resulting in the quality of sealing to a large extent being dependent on the level of quality of the construction work, and the current sealing technology is only suitable for sealing shorter holes and cannot be a good match for the widely used and rapidly developing directional drilling technology. Therefore, against the background of intelligent development of coal mines, it is necessary to further develop coalbed methane extraction systems with a high degree of automation or even intelligent systems and to develop the drilling sealing technology for directional drilling.
- (3) Research on coalbed methane extraction drilling leakage detection technology and devices. There are many coalbed methane extraction borehole leakage detection technologies and related patents, mainly focusing on negative pressure detection and tracer gas monitoring, etc. However, due to the complex structure of the surrounding rock of underground boreholes, reference should be made to non-contact measurement methods and non-destructive monitoring methods, and fibre-optic measurements, ultrasonic measurements, magnetic particle detection, ultrasonic detection, liquid penetration detection, etc. should be introduced into the detection of gas leakage of coalbed methane extraction boreholes so as to realise the dynamic monitoring of and early warning for gas leakage. Relevant technologies should be integrated into the coalbed methane extraction system platform.
- (4) Development and application of sealing materials for coalbed methane extraction drilling. At present, there are many kinds of sealing materials for coalbed methane extraction drill holes, most of which are solidified sealing materials, but due to the

influence of the roadway and drilling unloading pressure, the creep of coal rock drill holes under the action of geostress is unavoidable with the prolongation of extraction time. Therefore, the development of liquid sealing materials adapted to the development of fissures in the surrounding rock of the drill hole creep has a wide range of prospects for application. At the same time, active support-type sealing materials with high strength, sealing materials with Poisson's ratio similar to that of the surrounding rock of the borehole, and gel-like sealing materials that can actively adapt to the deformation of the fissures of the surrounding rock of the borehole can all play a better role in specific scenarios.

Author Contributions: Conceptualisation, R.B.; methodology, R.B.; validation, M.G., R.B. and S.W.; investigation, S.W., Y.Z., X.S., X.C. and M.G.; resources, S.W., Y.Z., X.S., X.C. and L.Z.; writing—original draft preparation, R.B.; writing—review and editing, M.G. and L.Z.; supervision, S.W., Y.Z. and X.S.; project administration, R.B.; funding acquisition, R.B. and L.Z. All authors have read and agreed to the published version of the manuscript.

Funding: This research was funded by the National Natural Science Foundation of China (No. 52174181 and No. 52204212), S&T Program of Hebei (No. 22375401D), the Fundamental Research Funds for the Central Universities (No. 3142020003), Education and Teaching Reform Project (05020108020202), and Natural Science Foundation of Sichuan Province (2024NSFSC0971), and Open Research Fund of Hebei Key Laboratory of Mine Intelligent Unmanned Mining Technology (No. iium001).

Data Availability Statement: Data are contained within the article.

Conflicts of Interest: Author Shuai Wang was employed by the company Pingdingshan Tianan Coal Mining Co., Ltd., No. 5 Coalmine. Author Yunguang Zhang was employed by the company Zhengzhou Coal Industry (Group) Co., Ltd.; Author Xiaopeng Si was employed by the company Huiyang Coal Co., Ltd. of Shanxi Coal Transportation and Sales Group Co., Ltd.; The remaining authors declare that the research was conducted in the absence of any commercial or financial relationships that could be construed as a potential conflict of interest.

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ISBN 978-3-7258-8778-1