INTRODUCTION

The gas flow in coal seam was restricted by many factors, among which the distribution characteristics of mining stress have an important influence on gas seepage law. During the mining process, the stress was changed, that is, stress redistribution, and the coal body undergoes repeated load and unload process, which leads to coal body deformation, instability or coal and gas outburst accidents. It is necessary for us to study the permeability of coal body during mining.1

At present, domestic and foreign scholars have made a lot of research results on the deformation and permeability characteristics of coal-rock. Evans2 carried out the compression strength test of coal. Hobbs3 and Bieniawski4 studied the strength and the stress-strain law of coal under triaxial compression. Zhou5 and Lin6 studied the mechanics and permeability characteristics of gas-bearing coal and rock, and proposed the relationship between gas pressure, coal adsorption, coal metamorphism and porosity and coal deformation. However, the permeability of the coal-rock is not only affected by its own physical properties, but also related to many external factors such as in situ field, temperature field and pore pressure, and so on. In the aspect of bedding effect, Chen et al7 performed adsorption-desorption tests and full stress-strain-seepage tests on the bedding interface with different inclination angles. Chen8 conducted a field test of the coupling relationship between the bedding orientation and coal seam extraction effect, and established a mathematical model of gas drainage. Pan et al9 carried out the permeability test of vertical bedding under different unloading conditions.

Abstract

In this paper, the seepage tests of mining coal were conducted by servo-controlled seepage apparatus. These tests consist of conventional triaxial compression seepage tests, and load and unload seepage tests. It was observed that the peak strength and corresponding axial strain of raw coal samples gradually increase with the increase of confining pressure, which conforms to the Mogi-Coulomb strength criterion, and the internal friction angle was calculated as \( \phi = 42.65^\circ \), and the cohesion force was \( c = 3.56 \text{ MPa} \). The ultimate strength of coal samples after load and unload test was obviously lower than that of the triaxial compression test under the same confining stress conditions, and the deviatoric stress-permeability curves consistent with the exponential function under two stress paths. In load and unload test, the damage degree of raw coal was characterized by the permeability damage rate and the maximum permeability damage rate. The permeability of coal seam was closely related to the mining stress, it presents a nonlinearly declining as the mining stress increases, and the permeability increases nonlinearly when the mining stress was released.

KEYWORDS
gas-containing raw coal, load and unload test, mechanical behavior, permeability, strength criterion
Deng et al. discussed the difference of deformation and permeability between parallel and vertical bedding coal and rock samples by experiments. In terms of the influence of gas pressure, Li et al. studied the creep test of coal briquetting under low gas pressure under staged loading. Zhang et al. studied the permeability characteristics of raw coal in complete stress-strain process under different gas pressure. Zhu et al. found that the permeability of gas-containing coal increased with the increase of gas pressure. Liu found that the permeability of coal showed a monotonous decrease in exponential form with increasing gas pressure. Yin et al. found that the higher the gas pressure is, the shorter the time from the beginning of unloading confining pressure to stop failure, that is, the greater the strength of the coal sample. In terms of effective stress, Li et al. analyzed the effects of effective stress and slippage on the gas permeability of coal and rock. Meng and Chen carried out the study on stress sensitivity of permeability. In terms of the influence of stress paths such as addition and unloading, Zhang et al. studied the permeability characteristics of gas-containing raw coal under different stress conditions. Xie et al. investigated the stress-fracture-seepage field behavior of coal under different mining layouts. Yin and Jiang conducted experimental research on gas seepage characteristics in coal under different stress paths. Yang et al. carried out the uniaxial cyclic loading and unloading tests, and investigated the nonuniform deformation evolution processes of granite specimens.

In summary, a series of research have been done about the deformation and permeation properties of coal and rock during the load and unload process; however, there are few studies involving mining coal, and the mining stress has a great influence on the deformation and permeability of coal and rock. So, we studied the deformation and permeation properties of raw coal, so as to have a depth analysis of the deformation law and gas permeation characteristics of mining, with an expectation to provide a theoretical guidance for gas treatment in low permeability coal seam.

### 2 | TEST CONDITIONS AND METHODS

#### 2.1 | Survey of working face and test specimens

The test samples were taken from the position 100 m in front of the S3012 working face, 2+3# coal seam of Sha mushu colliery of Sichuan Coal Group, Southwest China. The buried depth of the coal seam is about 450 m, the coal seam belongs to coal and gas outburst coal seam. The original gas content is 17.37 m³/t and the gas pressure is 0.4-2.34 MPa, the average gas pressure is 1.32 MPa, the S3012 working face has a length of 752 m, a slope length of 138 m, a coal seam thickness of 0.8-4.4 m, an average thickness of 3.1 m and an average inclination of 4°. The industrial indicators of the collected coal are as follows: coal moisture content of 0.96%, dry ash basis of 26.67%, dry ash free basis of 18.71%, fix carbon content of 54.28%, gas adsorption constant $a = 23.9692$ m³/t, $b = 1.2697$ m³/t. The mining method is longwall retreating mining on strike, the ground control is total caving method in U type ventilation. The No.S3012 working face near the strata is indicated in Table 1.

Select a coal sample with a block size greater than 200 mm × 200 mm × 200 mm, seal the lump coal with plastic wrap and transport it back to the laboratory, and process the lump coal into a standard cylindrical sample with a height of 100 mm and a diameter of 50 mm in the laboratory. The density of raw coal sample is about 1477.19 g/m³, and the initial parameters of the raw coal samples are shown in Table 2. The selected location and standard raw coal samples are shown in Figure 1.

#### 2.2 | Scheme design

After coal excavation, the coal and rock mass in front of the working face is in the stress reduction zone, the stress increase zone, and the original stress zone. With the advancement of the working face, the coal and rock mass undergoes complex loading and unloading process. At present, briquette samples are mostly used in the permeability test of coal, which is quite different from the actual situation on site. The raw
Coal samples were taken from the site, which is more in line with the actual working conditions. Therefore, it is of great significance to select the raw coal samples for mechanical and permeability tests in the loading and unloading process to prevent and control coal and gas disasters. Therefore, the design test scheme is as follows:

Conventional triaxial compression seepage tests: (a) gradually apply axial stress and confining pressure to 2 MPa hydrostatic pressure; (b) maintain axial stress and confining pressure were 2 MPa constant, adjust the gas pressure to 1.0 MPa till the end of the experiment; (c) gas is absorbed completely for 12 hour until reaching equilibrium; (d) maintain 2 MPa confining pressure constant, load axial stress till raw coal failure at displacement control loading rate of 0.1 mm/min; (e) at the end of the test, replaced the samples and changed the hydrostatic pressure to 3, 4, 5, 6, and 7 MPa, respectively. In this test, load axial stress at stress loading rate of 0.05 kN/s, load confining pressure at stress control loading rate of 0.02 MPa/s.

Load and unload seepage test: (a) gradually apply axial stress and confining stress to 8.6 MPa hydrostatic pressure; (b) maintain axial stress and confining pressure were 8.6 MPa constant, adjust the gas pressure to 1.0 MPa till the end of the experiment; (c) gas is absorbed completely for 12 hour until reaching equilibrium; (d) maintain 2 MPa confining pressure constant, load axial stress to 25.85 MPa, then unload axial stress to 8.6 MPa; (e) maintain 8.6 MPa axial stress constant, then unload confining pressure to 6.0 MPa; (f) maintain 6.0 MPa confining pressure constant, load axial stress to 25.85 MPa, then unload axial stress to 8.6 MPa; (g) maintain 8.6 MPa axial stress constant, unload confining pressure to 4.0 MPa; (h) maintain 4.0 MPa confining pressure constant, load axial stress till raw coal failure at displacement control loading rate of 0.1 mm/min, and the end of the test. In this test, load and unload axial stress at stress loading rate of 0.05 kN/s, unload confining pressure at stress control unloading rate of 0.02 MPa/s.

### Table 2: Main parameters of standard cylindrical raw coal specimens

| Test                          | No. | Height (mm) | Diameter (mm) | Mass (g) |
|-------------------------------|-----|-------------|---------------|---------|
| Uniaxial compression test     | a1  | 100.14      | 49.94         | 292.16  |
| Conventional triaxial compression seepage test | a2  | 100.12      | 49.92         | 291.10  |
|                               | a3  | 100.16      | 49.92         | 290.18  |
|                               | a4  | 99.98       | 49.92         | 288.12  |
|                               | a5  | 100.06      | 49.94         | 288.19  |
|                               | a6  | 100.08      | 49.94         | 288.14  |
|                               | a7  | 100.10      | 49.96         | 289.56  |
| Load and unload seepage test  | a8  | 100.04      | 49.94         | 288.86  |

FIGURE 1 Selected position of coal sample and standard cylindrical specimens

2.3 Test apparatus

The test equipment adopts the “Triaxial stress thermal-hydrological-mechanical coal gas permeameter” developed by Chongqing University, as shown in Figure 2, the equipment can conduct coal-rock gas permeability test under different stress and gas pressure. The maximum axial pressure is 200 kN, the confining pressure is 10 MPa, the maximum deformation displacement is 60 mm, and the maximum radial deformation is 6 mm. The temperature ranges from room temperature to 100°C. The stress measurement system has an accuracy of ±1%, a deformation accuracy of ±1%, and a temperature control accuracy of ±1%.
3 | TEST RESULTS AND DISCUSSION

According to the test results, the test data were analyzed, volumetric strain $\varepsilon_v$ of the coal sample was calculated by:

$$\varepsilon_v = \varepsilon_1 + 2\varepsilon_3$$  \hspace{1cm} (1)

which, $\varepsilon_1$ is the axial strain, $\varepsilon_3$ is the radial strain.

The deviatoric stress $\sigma'$ of the raw coal sample was calculated by:

$$\sigma' = \sigma_1 - \sigma_3$$  \hspace{1cm} (2)

which, $\sigma_1$ is the axial stress, $\sigma_3$ is the confining pressure.

Assuming that the gas flow in the coal sample conforms to Darcy's law, the permeability of raw coal sample was continuously calculated by Jiang et al\textsuperscript{22}:

$$k = \frac{2v\mu p_2}{A(p_1^2 - p_2^2)}$$  \hspace{1cm} (3)

where, $k$ presents the permeability (mD); $v$ is gas seepage velocity (cm$^3$/s); $\mu$ refers to the gas kinematic viscosity (Pa s); $l$ is the length of raw coal sample (mm), $A$ is the cross-section area of raw coal sample (mm$^2$); $p_1$ is gas pressure at the sample inlet (MPa), $p_2$ is gas pressure at the sample outlet (MPa).

3.1 | Conventional triaxial compression tests

3.1.1 | Analysis of mechanical properties

The complete stress-strain curves of samples as shown in Figure 3, we have known that it was divided into four stages. (a) In compaction stage, the strain rate decreases slightly with increasing stress, the curve bends slightly upward, and the slope increases gradually. It is mainly caused by the unloading rebound caused by the stress release when the coal mass is separated from the parent coal and the closure of the joints or microcracks in the coal under the action of pressure. (b) In elastic phase, the stress-strain curve is substantially straight, the specimen is close to elasticity and has a slight hysteretic effect in this section, but no unrecoverable deformation occurs in the unloaded raw coal sample in this section. (c) In yield stage, the curve is bent downward until the peak intensity of the specimen, and the deformation is inelastic, the raw coal sample shows plastic deformation at this stage. The slope of stress-strain decreases gradually to zero with the increase of stress, and the specimen undergoes irreversible deformation. (d) In strain softening stage, the stress-strain curve suddenly drops due to the failure of the coal block, the slope of the shortening curve is negative, and the specimen exhibits a brittle stage, and the unloading in the section produces a large residual deformation. The sample forms a penetrating fracture surface in a direction parallel to the maximum principle stress, and the sample strength suddenly drops sharply. If the strength gradually decreases with strain growth, it is called progressive failure. The ultimate strength of the samples when the raw coal samples are destroyed was 26.3, 31.94, 38.15, 43.15, 47.31, and 54.27 MPa, respectively. And the axial strain was $1.259 \times 10^{-2}$, $1.408 \times 10^{-2}$, $1.568 \times 10^{-2}$, $1.753 \times 10^{-2}$, $1.824 \times 10^{-2}$, and $2.057 \times 10^{-2}$, respectively. Compared with the peak intensity of specimens at confining pressure of 2 MPa, the peak intensity of specimens increased by 21.44%, 45.06%, 64.07%, 79.89%, and 106.35%, respectively. Compared with the axial strain of samples at confining pressure of 2 MPa, the axial strain of samples increased by 11.83%, 24.54%, 39.24%, 44.88%,
and 63.38%, respectively. The test results were shown in Table 3.

### 3.1.2 Analysis of the strength property

#### Parabolic strength criterion

The parabolic strength criterion, that is, the two order polynomial strength criterion, as in formula (4):

\[(\sigma_1 - \sigma_3)^2 = E(\sigma_1 + \sigma_3) + F\]  \hspace{1cm} (4)

where, \(\sigma_1\) is strength of coal samples when \(\sigma_2 = \sigma_3\); \(E\), \(F\) are constant.

\[\sigma_s = \sigma_3 + \sigma_c - \sigma_D + \sqrt{\sigma_D^2 + m\sigma_3(\sigma_c - \sigma_D)}\]  \hspace{1cm} (5)

where, \(\sigma_c\) is uniaxial compressive strength, \(\sigma_D\) is strength parameter. The parameters \(\sigma_c\) and \(m\) can be determined by the regression of experimental data. When \(\sigma_D = 0\), formula (5) is a parabola, \(^{25}\) that is:

\[(\sigma_1 - \sigma_3)^2 = \sqrt{m\sigma_3(\sigma_1 + \sigma_3)} - \sigma_c^2\]  \hspace{1cm} (6)

The regression analysis of the data in Table 3, as shown in Figure 4A, can be obtained by the formula (6): \(n = 4\), \(\sigma_c = 16.88\) MPa, \(R^2 = 0.9969\).

#### Drucker-Prager strength criterion

The Drucker-Prager strength criterion considers the effect of second deviatoric stress and hydrostatic pressure on the failure of rock masses, the expression is:

\[\sqrt{J_2} = \beta l_1 + j\]  \hspace{1cm} (7)

where, \(\beta\), \(j\) are test constants; \(l_1\) is the stress first invariant, \(J_2\) is the second invariant of stress deviation, and there is:

\[l_1 = \sigma_1 + \sigma_2 + \sigma_3\]  \hspace{1cm} (8)

\[J_2 = \frac{1}{6}[(\sigma_1 - \sigma_2)^2 + (\sigma_2 - \sigma_3)^2 + (\sigma_3 - \sigma_1)^2]\]  \hspace{1cm} (9)

The regression analysis of the data in Table 3 is carried out according to formula (7), and the curve is shown in Figure 4B. According to the formula, (11,12) obtained the coal internal friction angle \(\varphi = 51.26^\circ\), cohesion \(c = 6.91\) MPa and \(R^2 = 0.9991\).

#### Hoek-Brown strength criterion

Hoek and Brown\(^{26}\) combined with the engineering practical experience and coal samples triaxial test data to carry out statistical analysis, the relationship between the principal stress of coal failure is proposed as follows:

\[\sigma_1 = \sigma_3 + \sqrt{a\sigma_c + b\sigma_c^2}\]  \hspace{1cm} (13)

where, \(\sigma_c\) is uniaxial compression intensity of intact coal; \(a, b\) is the empirical parameters describing the coal mass properties, \(b = 1\) for the intact coal mass, \(0 < b < 1\) for the broken coal mass.

In order to easily organize the data, the equation is mathematically transformed into:

\[\left(\frac{\sigma_1 - \sigma_3}{\sigma_c}\right)^2 = a\left(\frac{\sigma_3}{\sigma_c}\right) + b\]  \hspace{1cm} (14)

### Table 3 Test results in uniaxial and conventional triaxial compression

| Confining pressure (MPa) | Peak intensity (MPa) | Axial strain \(\varepsilon_1\) (10^{-2}) | Radial strain \(\varepsilon_2\) (10^{-2}) |
|--------------------------|----------------------|----------------------------------------|----------------------------------------|
| 0                        | 16.78                | 1.105                                  | —                                      |
| 2                        | 26.3                 | 1.259                                  | -0.772                                 |
| 3                        | 31.94                | 1.408                                  | -0.715                                 |
| 4                        | 38.15                | 1.568                                  | -0.514                                 |
| 5                        | 43.15                | 1.753                                  | -0.835                                 |
| 6                        | 47.31                | 1.824                                  | -0.423                                 |
| 7                        | 54.27                | 2.057                                  | -0.496                                 |

This paper triaxial test \(\sigma_2 = \sigma_3\), then the formula can be changed to:

\[J_2 = \frac{1}{3}(\sigma_1 - \sigma_3)^2\]  \hspace{1cm} (10)

For the plane strain problem, the relationship between Drucker-Prager criterion and Mohr-Coulomb criterion is as follows:

\[\beta = \frac{2\tan\varphi}{\sqrt{9 + 12\tan^2\varphi}}\]  \hspace{1cm} (11)

\[j = \frac{3c}{\sqrt{9 + 12\tan^2\varphi}}\]  \hspace{1cm} (12)
FIGURE 4  Regression analysis of each strength criterion

(A) Parabolic

(B) Drucker-Prager

(C) Hoek-Brown

(D) Mohr-Coulomb

(E) Mogi-Coulomb
The regression analysis of the data in Table 3 is carried out according to equation (13), and the curve is shown in Figure 4C. Thus, \( a = 15.624, b = 0.4586 < 1 \), indicating that the Hoek-Brown strength criterion is better for the regression of experimental data.

**Mohr-Coulomb strength criterion**

Mohr-Coulomb strength criterion\(^{27,28}\) considers that the failure occurs when a certain point of shear stress \( \tau \) in a coal sample reaches to the ultimate stress \( \tau_{\text{max}} \), and the expression is:

\[
\tau = c + \sigma \tan \varphi
\]  
(15)

where, \( \sigma \) is normal stress, \( c \) is cohesive, \( \varphi \) is the internal friction angle, the formula (15) is transformed into the formula (16) through the triangle relation transformation.

\[
\sigma_1 = M + N\sigma_3
\]  
(16)

where, \( \sigma_1 \) is the maximum principal stress, \( \sigma_3 \) is the minimum principal stress, and \( \sigma_2 \) are the maximum and minimum principal stress of coal failure, respectively; \( M \) is the strength parameter corresponding to the complete shear failure of uniaxial compression, \( N \) is the influence coefficient of confining pressure on axial-bearing capacity, which shows the influence of stress state on the bearing capacity of coal sample, equation (16) shows that the maximum axial stress \( \sigma_1 \) that can be carried by a given coal sample is linear with the confining pressure \( \sigma_3 \), denoted as:

\[
M = \tan^2 (45° + \varphi/2)
\]  
(17)

\[
N = 2c \cos \varphi (1 - \sin \varphi)
\]  
(18)

According to equation (16), the data in Table 3 are analyzed by regression, the curve is shown in Figure 4D. From this, the internal friction angle \( \varphi = 43.05° \), the cohesion \( c = 3.4 \text{ MPa} \).

**Mogi-Coulomb strength criterion**

Mogi-Coulomb strength criterion considers that rock failure is due to the octahedral shear stress \( \tau_{\text{oct}} \) on the failure surface reaching its limit value, which is significantly different from the Mohr-Coulomb strength criterion, because the Mohr-Coulomb strength criterion for coal failure is that on the surface of the shear stress value \( \tau \) reached the limit value. The Mogi-Coulomb strength criterion considers the octahedral shear stress \( \tau_{\text{oct}} \) as an average function of the sum of the maximum and minimum principal stress when the coal specimen destroyed, there is:

\[
\tau_{\text{oct}} = \frac{1}{3} \sqrt{(\sigma_1 - \sigma_2)^2 + (\sigma_2 - \sigma_3)^2 + (\sigma_3 - \sigma_1)^2}
\]  
(19)

In this paper, \( \sigma_2 = \sigma_3 \), so there is:

\[
\tau_{\text{oct}} = \frac{\sqrt{2}}{3}(\sigma_1 - \sigma_3)
\]  
(20)

There is a linear mapping relationship between \( \tau_{\text{oct}} \) and \( \frac{\sigma_1 + \sigma_2}{2} \), and there is:

\[
\frac{\sqrt{2}}{3}(\sigma_1 - \sigma_3) = e + f(\sigma_1 + \sigma_3)
\]  
(21)

where, \( e \) and \( f \) are test parameters, and the relationship between the cohesive force \( c \) and the internal friction angle \( \varphi \) of the Mohr-Coulomb strength parameter is:

\[
e = \frac{2\sqrt{2}}{3}c \cos \varphi
\]  
(22)

\[
f = \frac{2\sqrt{2}}{3} \sin \varphi
\]  
(23)

The regression analysis of the data of Table 3 is carried out according to the formula (21), the curve is shown in Figure 4E. According to the formula, the internal friction angle is \( \varphi = 42.65° \), and the cohesive \( c = 3.56 \text{ MPa} \), \( R^2 = 0.9995 \).

**Strength criterion analysis**

With regard to the quantitative evaluation index of the fitting results of each strength criterion, the average deviation:

\[
mf = \frac{\sum |\sigma_{\text{calc}} - \sigma_{\text{test}}|}{N}
\]  
(24)

where, \( \sigma_{\text{calc}} \) is the calculated value of the different criterion (MPa); \( \sigma_{\text{test}} \) is the experimental value of maximum principal stress (MPa); \( N \) is the number of groups of test data.

As can be seen from the data regression analysis in Figure 4, the correlation coefficients of each strength criterion are greater than 0.96, the correlation is better. But compared to the uniaxial compressive strength, the Mohr-Coulomb strength criterion and Hoek-Brown strength criterion were 16.295 and 11.36 MPa, respectively, which were less than the measured uniaxial compressive strength 16.78 MPa. While the uniaxial compression strength of the parabolic strength criterion is 16.88 MPa, and the fitting result is larger. The correlation coefficient of Drucker-Prager strength criterion and Mogi-Coulomb strength criterion is largest, but the difference is very larger between the internal friction angle and the cohesion. The internal friction angle \( \varphi = 51.26° \), cohesion \( c = 6.91 \text{ MPa} \) of Drucker-Prager strength criterion, the internal friction angle \( \varphi = 42.65° \), cohesion \( c = 3.56 \text{ MPa} \) of Mogi-Coulomb strength criterion, and the internal friction angle \( \varphi = 43.05° \), cohesion \( c = 3.4 \text{ MPa} \) of Mohr-Coulomb strength criterion, which shows that the difference is relatively smaller between internal friction angle and cohesion of Mogi-Coulomb strength criterion and Mohr-Coulomb...
strength criterion. Combined with the evaluation index of strength criterion in Table 4, the evaluation index of analysis shows that the Mohr-Coulomb strength criterion and Mogi-Coulomb strength criterion are suitable for low confining pressure condition, but the mf value of the Mogi-Coulomb strength criterion is the smallest and the correlation coefficient is the largest. In summary, the Mogi-Coulomb strength criterion fits well to the test results under low confining pressure.

3.1.3 Analysis of the permeability properties

The permeability and deviatoric stress have a significant correlation with the volumetric strain, as shown in Figure 5. In the compaction and elastic phase, the permeability gradually decreases with the increase of the deviatoric stress, and the volumetric strain gradually increases in the positive direction as the deviatoric stress increases. At this stage, the coal sample is deformed under compression, and the axial deformation is larger than the radial deformation, resulting in a gradual increase in volumetric strain. During the loading process, the micropores and cracks in the sample gradually closed, which causes the gas seepage channel to be gradually destroyed, and the permeability of the sample gradually decreases. The initial permeability of coal samples under different confining pressure conditions were 1.039, 0.998, 0.804, 0.541, 0.436, and 0.341 mD, respectively, and the corresponding confining pressures were 2, 3, 4, 5, 6, and 7 MPa. It can be seen that the initial permeability of the coal sample gradually decreases with the increase of the confining pressure. Compared with the initial permeability at confining pressure of 2 MPa, the initial permeability of the samples decreases by 3.95%, 22.62%, 47.93%, 58.04%, and 67.18%, respectively. In the yield stage, the volumetric strain increases slowly and permeability continues to decrease with the increase of deviatoric stress. During this stage of loading, the material with lower internal strength first yields to failure, resulting in an increase in permeability of individual sample. In the failure stage, the axial stress reaches the ultimate strength of the sample, the deviatoric stress decreases rapidly, and the permeability of the sample shows a turning point and increases rapidly. The permeability of the sample increases to 1.391, 0.524, 0.679, 0.635, 0.69, and 0.391 mD, respectively. The volumetric strain appears a turning point and decreases rapidly, and when it decreases to zero, it begins to increase in a negative direction.

During the triaxial compression test, the permeability-deviatoric stress curve shows a trend of oblique “V” shape, that is, the permeability of the coal sample decreases first and then increases with the increase of the deviatoric stress, and the permeability increases obviously after failure, and the growth rate is greater than the decreasing speed of the initial stage. The fitted permeability and deviatoric stress are in accordance with negative exponential function distribution during the loading process, and the fitting formula is as shown in equation (25):

\[ k = c \exp(d\sigma') \]  

(25)

which \( c \), \( d \) are the fitting constants; \( \sigma' \) is deviatoric stress, MPa; the fitting curves were shown in Figure 5, the correlation coefficients of fitting curves are all greater than 0.92, thus displaying a relatively good correlation.

3.2 Load and unload test

3.2.1 Analysis of mechanical behavior

Figure 6 shows the deviatoric stress-strain curves of sample in load and unload test, the ultimate strength of the specimen is 29.7 MPa under load and unload test, while the ultimate strength is 38.15 MPa under conventional triaxial compression test at the same confining pressure. It is known that the ultimate strength of the specimen significantly reduces after two times of loading and unloading axial stress, and unloading confining pressure test.

3.2.2 Analysis of the permeability evolution

The load and unload process can be divided into three parts to analyze the permeability properties. Part 1: maintain 8.6 MPa confining pressure constant, load axial stress to 25.85 MPa, unload axial stress to 8.6 MPa, then unload confining pressure to 6.0 MPa, as shown in Figure 7A; Part 2: maintain 6.0 MPa confining pressure constant, load axial stress to 25.85 MPa, then unload axial stress to 8.6 MPa, then unload confining pressure to 4.0 MPa, as shown in Figure 7B; Part 3: maintain 4.0 MPa confining pressure constant, load axial stress till raw coal failure at displacement control loading rate of 0.1 mm/min, as shown in Figure 7C.

1. The first part: the permeability gradually decreases with the increase of axial strain, and the permeability decreases from 0.787 to 0.224 mD when the axial stress is loaded to 25.85 MPa. During the unloading phase, the

| Strength criterion       | mf   | \( R^2 \) |
|-------------------------|------|---------|
| Parabolic               | 19.534 | 0.9969 |
| Drucker-Prager          | 18.158 | 0.9991 |
| Hoek-Brown              | 4.427 | 0.9686 |
| Mohr-Coulomb            | 0.677 | 0.9966 |
| Mogi-Coulomb            | 0.676 | 0.9995 |
FIGURE 5 Volumetric strain-deviatoric stress-permeability curves of samples with different confining pressure

(A) $\sigma_3 = 2$ MPa

(B) $\sigma_3 = 3$ MPa

(C) $\sigma_3 = 4$ MPa

(D) $\sigma_3 = 5$ MPa

(E) $\sigma_3 = 6$ MPa

(D) $\sigma_3 = 7$ MPa
permeability gradually recovers with the decrease of axial strain. The permeability is restored from 0.244 to 0.479 mD, and the recovery degree is about 60.86%, which cannot form a closed loop curve, the load and unload curves do not coincide. During the unload confining pressure process, the permeability increases rapidly, the permeability of coal samples increases from 0.479 to 0.557 mD when the confining pressure is unloaded from 8.6 to 6.0 MPa, but failed to recover to the initial permeability.

2. The second part: the permeability gradually decreases with the increase of axial strain, and the permeability decreases from 0.787 to 0.224 mD when the axial stress is loaded to 25.85 MPa. During the unloading phase, the permeability gradually recovers with the decrease of axial stress. The permeability is restored from 0.557 to 0.252 mD, and the recovery degree is about 86.54%, which cannot form a closed loop curve, and the load and unload process curves do not coincide. During unloading confining pressure process, the permeability increases rapidly, the permeability of coal samples increases from 0.482 to 0.659 mD when the confining pressure is unloaded from 6.0 to 4.0 MPa, which is greater than the initial permeability at this stage.

3. The third part: the permeability gradually decreases with the increase of axial stress, the relationship between permeability and axial strain presents an inclined “V” shape. The permeability increases obviously in postpeak, and the growth rate is greater than the reduction rate of the initial stage.

In load and unload axial stress process, the permeability-deviatoric stress curves show a variation of negative exponential function. During the unload confining pressure process, the permeability-deviatoric stress curves show a variation of exponential function, satisfying the equation (22). The permeability was taken at every 1 MPa stress in load and unload process, the fitting curves between the permeability and deviatoric stress were shown in Figure 8.

### 3.2.3 Analysis of permeability damage rate

The deviatoric stress-permeability curves do not coincide in two times of load and unload axial stress, and the permeability is higher than that of unloading axial stress. That is to say, after one time of loading and unloading axial stress, the permeability was damaged to a certain degree. According to the research results of the reference, the permeability damage rate and the maximum permeability damage rate can be used to evaluate the recovery extent and decrease degree of coal sample permeability. When the permeability damage rate is larger, the recovery degree of permeability is worse; when the maximum permeability damage rate is larger, the reduction amplitude of permeability is greater.

The permeability damage rate $d_c$ can be calculated by formula (26). The maximum damage rate $d_{max}$ can be calculated by formula (27):

$$\begin{align*}
d_c &= \frac{k_0 - k_1}{k_0} \times 100\% \\
d_{max} &= \frac{k_0 - k_{min}}{k_0} \times 100\%
\end{align*}$$

where $k_1$ is the raw coal permeability at the last stress point of unload axial stress; $k_0$ is raw coal permeability at the first stress point of load axial stress; $k_{min}$ is the permeability at the maximum stress point. According to formula (26) and (27), the permeability damage rate is 39.24%, the maximum permeability damage rate is 72.15% at the first load and unload axial stress. The permeability damage rate is 14.29%, the maximum permeability damage rate is 55.36% at the second load and unload axial stress. It can be known that after the first load and unload axial stress, the permeability recovered to 60.86% of the original value, the permeability recovered to 86.54% after the second load and unload axial stress. Moreover, after the second load and unload axial stress, the permeability damage rate and the maximum permeability damage rate are all less than the first time, the recovery degree of the second permeability is larger, and the reducing amplitude of the first permeability is larger. Since the confining pressure is 8.6 MPa when the first unload axial stress, and the confining pressure has unloaded from 8.6 to 6.0 MPa at the second load and unload axial stress. It can be seen that the lower the confining pressure is, the higher the recovery degree of permeability will be.

Since the permeability always reflects the change of the effective porosity inside the coal sample, the increase of the permeability reflects the increase of the porosity, the decrease of the permeability reflects the decrease of the porosity.
Therefore, in load and unload test process, the permeability of sample has the same variation law with the effective porosity. The damage degree of permeability is also reflected in the porosity. After one times load and unload axial stress, the porosity of sample is also damaged, and the damage rate is equal to $d_k$. The sample showed secondary compaction, which is also the reason why the permeability of the sample could not be restored to the initial permeability in unload axial stress process under low confining pressure.

4 | ABUTMENT PRESSURE DISTRIBUTION LAW

4.1 | Distribution law of Abutment pressure

The abutment pressure of the coal seam was monitored by the drilling stress gauge installed in the return airway of the NO.S3012 working face. The drilling stress gauge was arranged as follows: the drilling holes were arranged along the coal seam at the return airway starting from 45 m at back of 2# coal exploration roadway, with the spacing between boreholes of 5 m, height 0.8 m, depth 8 m, a total of five gauges were arranged, the schematic diagram as shown in Figure 9.

The measurement method of coal seam abutment pressure: after the construction of the drilling operation, push the drilling stress gauge into the bottom of the hole with a coarse push rod and ensure that the stress gauge cylinder faces up, rotate the rod and make the drilling stress gauge cylinder extend to the hole wall, and then connect the measuring instrument to record the initial value of the drilling strain gauge. The drilling stress gauge employs the ZLG-40 type vibrating wire sensor, with the maximum initial support force up to 40 MPa. Before installing the drilling stress meter, the drilling stress

FIGURE 7 Deviatoric stress-axial strain-permeability curves
gauge should be connected with the detector. Then, please enter the initial frequency $f_0$ and the sensor A, B constant for each drilling stress gauge, encode the drilling stress gauges.

Figure 10A presets the abutment stress curve of the field monitoring. It can be seen that as the coal mining face advances from the 2# coal exploration lane, the abutment stress of coal seam can be divided into the pressure relief zone, the stress increase zone and the original rock stress zone, reaching the peak value at 15-25 m in front of the working face. When the instability and collapse of the basic roof of the working face, there is a coal wall on the side of the overlying rock mass and a fallen rock layer on one side. Near the working face, the coal body is affected by mining, the strength is reduced, the stress releases and transfers to the deep part of the coal body, and the stress state of the coal body was changed from three-way load to two-way unloading pressure to form a pressure relief zone. With the advancement of the working face, the abutment stress of the coal seam gradually increases and is greater than the original rock stress, which is the stress increase zone. With the continuous advancement of the working face, the mining layer is gradually weakened by the mining effect, and the stress of the coal body gradually recovers to the original rock stress state.

Figure 10B presets the results of FLAC3D numerical simulation experiments. It can be seen that with the continuous advancement of the coal mining face, the stress redistribution of rock mass and the overlying strata around the working face. The distribution characteristics are as follows: most of the weight of the fracture zone above the coal face and the overlying strata on the side of the coal face, that is, the

**FIGURE 8** Change of permeability with deviatoric stress during loading and unloading
abutment stress in front of the coal face is much larger than the rear of the working face. Due to the advancement of the coal mining face, the coal wall and the caving zone of the goaf continuously move forward, so the abutment stress is the moving abutment stress supporting stress in front of the coal mining work is the moving supporting stress front and rear of working face. Because the fracture zone forms a semi-arched balance with the coal wall and the caving zone of the goaf as the front and rear support points, the coal mining face is in the pressure relief zone. The maximum value of the abutment stress formed in front of the coal mining face occurs in the middle of the working face. The peak value of abutment stress in front working face penetrates 5-10 m into the coal body, and its influence range is 90-100 m ahead of the coal face, this area is the stress increase zone, and the area is not affected by mining in front and rear of working face is the original stress.

4.2 | Analysis of the infiltration characteristics of raw coal under mining stress

According to mining stress distribution law in front of working face above field test and FLAC^3D numerical simulation, point A is in the original stress state, point B and G are located at 30% of the peak intensity, point C has 50% of the peak intensity, point D and F have 70% of the peak intensity, point E has the peak intensity, as shown in Figure 1A. According to the analysis of the laboratory results, the coal sample is not affected by mining stress at point A, the coal sample is in the original rock stress state, that is, \(\sigma_1 = \sigma_2 = \sigma_3 = 8.6\) MPa, and the original gas pressure is 1 MPa, the permeability is 0.79 mD at this point. As the working face carries forward, the coal seam was affected by mining stress, the mining stress (axial stress) is greater than the original stress, the coal seam is in the
stress increasing area, and the horizontal stress (confining pressure) of coal seam slowly decreases, $\sigma' = \sigma_1 - \sigma_3 > 0$, the permeability is 0.55 mD at point B, the permeability is 0.39 mD at point C, the permeability is 0.32 mD at point D, and the permeability is 0.251 mD at point E. With the continued progress of working face, the strength is reduced and the stress is released (ie, axial stress and confining pressure gradually decrease) near the working face. Correspondingly, the permeability gradually increases, the permeability is 0.498 mD at point F, the permeability is 0.654 mD at point G. It is known that when the coal is not affected by mining, the permeability is 0.79 mD. With the continuous advancement of mining face, the mining stress increases continuously and the permeability of coal seam decreases nonlinearly. When the mining stress reaches the maximum, the permeability is the minimum. With the continued advancement of the working face, the stress of coal is released and the permeability increases nonlinearly.

5 | CONCLUSIONS

In this paper, we investigated the permeability of raw coal samples under different stress paths by using a self-developed seepage apparatus. The conclusion is as follows:

1. With the increases of confining pressure, the ultimate strength and corresponding axial strain of raw coal specimens gradually increase.

2. Regression analysis of the ultimate strength of raw coal specimens under different confining pressure was carried out by using five strength criteria, the regression effect of Mogi-Coulomb strength criterion was better, and the internal friction angle was calculated as $\varphi = 42.65^\circ$, and the cohesion force was $c = 3.56$ MPa.

3. The ultimate strength of the sample after load and unload test was obviously lower than that of the triaxial compression test at the same confining stress conditions, and the deviatoric stress-permeability curves consistent with the exponential function under two stress conditions.

4. During the load and unload process, the permeability damage rate and the maximum permeability damage rate can be used to characterize the damage degree of raw coal.

5. The permeability of coal seam is closely related to mining stress, it presents a nonlinearly declining with the mining stress increases, and the permeability increases nonlinearly when the mining stress was released.

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CONFLICT OF INTEREST

The authors declare that they have no competing interests, and the original research has not been published previously. All authors gave final approval for publication.

AUTHOR CONTRIBUTIONS

Dong-ming Zhang and Yu-shun Yang conceived and designed the experiments; Bang-an Zhang, Yu-shun Yang performed the experiments; Bang-an Zhang, Yu-shun Yang analyzed the data; and Yu-shun Yang wrote the paper.

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