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Closure of Fracture Due to Cover Stress Re-establishment After Coal Mining

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Abstract

In situ measurements of deformations, stresses, and closure of fractures, affecting water inflow following coal mining, are challenging due to the inaccessibility of fractured rock. In this paper, the authors studied the closure process of the fractured rock mass with the cover stress re-establishment based on a theoretical analysis and a scale model testing. A quantitative analysis is used to study the fracture distribution in the fractured zone. A function to describe a fracture aperture distribution in the fractured zone is proposed, which takes into account the curvature and thickness of the fractured rock. The theoretical analysis and a scale model testing both indicate that the cover stress re-establishment with mining distance increasing and the relationship between the fracture closure and cover stress re-establishment both satisfy a logarithmic function. The scale model test also shows the following features: (1) the fracture ratio (which is the fracture area divided by the total area of fracture and intact rock with a unit width in the vertical or horizontal direction) in the lower part of the fractured rock mass is greater than that in the upper part; (2) the initially fast decreased of fracture ratios is then followed by a slower decrease during the cover stress re-establishment process; (3) in the upper part of the rock mass, the vertical directional fractures with small apertures are being closed with cover stress re-establishment, which indicates an increase in the water resistance reducing the seepage from these parts of the fractured zone. This study improves the general understanding of the fracture closure process and cover stress re-establishment in the fractured rock mass after coal mining ceased, and provides a theoretical basis for water resource protection in case of underground coal mining.

1 Introduction

The problem of maintaining the safety of underground coal mining operations is closely related to an issue of the water resource protection due to inrush accidents (Zhang and Shen [2004](#); Zhang et al. [2009a](#), [b](#), [2010](#), [2011](#)), which is especially challenging in arid mining areas. The water inrush accidents have a close relationship with the maximum scope of the fracture propagation, the fracture closure within the fractured rock mass following the coal excavation, which, in turn, impacts the

recovery of groundwater seepage from fractured rock. It is known that stress change may significantly impact rock permeability. When the coal seam is excavated, the disturbed overburden strata is divided into several parts—caving, fractured, and continuous deformation zones, given in the order from the immediate roof upward to the surface (Peng and Chiang 1984). When equilibrium conditions of the originally stressed rock strata are disturbed, the load previously supported by the extracted material transfers to the surrounding gates and pillars, creating the stress increased zones in front of the mining face, and the stress decreased zones in the goaf, as illustrated in Fig. 1. As the overburden collapses and gets in contact with the coalbed floor, the fractured rock mass in the goaf re-loads the weight of the overburden. In this process, the cover stress is being re-established to nearly original conditions (Yavuz 2004), and the fracture aperture is diminishing along the mining distance.

Fig. 1

Distribution of disturbed and stress zones around the mining face (adapted from Yavuz 2004): M is the mining thickness, h_c is the thickness of the caved zone, h_f is the thickness of the fractured zone, and h_b is the thickness of the continuous deformation zone

To understand the fracture propagation and permeability evolution after coal excavation, the researchers have applied various methods to detect the overburden failure (e.g., Palchik 2003; Zhang and Shen 2004; Karacan and Goodman 2009; Miao et al. 2011; Sui et al. 2015), and proposed many empirical formulae to calculate the height of fractured rock mass (Majdi et al. 2012). The changes in the effective porosity and permeability fields due to the redistribution of stresses were predicted using the initial hydraulic conductivity, Rock Quality Designation (RQD), and Rock Mass Rating (RMR) (Liu et al. 1999). Kim et al. (1997) proposed a fully coupled strata deformation and groundwater flow model resulting from underground longwall mining in variably saturated fractured geologic media, based on an equivalent porous elastic continuum media representation. Liu and Elsworth (1997) evaluated three-dimensional effects of hydraulic conductivity enhancement and overburden desaturation around a longwall mining

advancing face, using a three-dimensional coupled finite element model. However, investigations have mostly focused on the evaluation of the increase in fracture aperture and permeability, although the actual processes involve the fracture closure and permeability reduction with the cover stress re-establishment in the later period of mining.

In-situ monitoring showed the drop in the groundwater level in the aquifer as the longwall face was retreated passing the monitoring wells; after the face passed for several weeks, the water table in the monitoring wells arose gradually (Zhang and Shen [2004](#); Zhang et al. [2010](#), [2011](#); Wang et al. [2013](#), [2014](#)). These studies indicate that the fractures that propagate into the overlying aquifer generally close with time, reducing seepage through the closed fractures into the collapsed mining space. The goals of this paper are to describe the fracture closure process of the fractured rock mass under a cover stress re-establishment due to underground coal mining, based on the results of a scale model study, and a theoretical analysis of the cover stress re-establishment process, leading to a fracture closure, as well as the evaluation of changes in the water inrush.

2 Engineering Background

The Taiping Coal Mine is located in the Southwestern part of the Yanzhou Coalfield, Shandong, China, and geologically situated on the margin of the Western Shandong Block of the North China Platform. The averaged thickness of the primary coal seam, No. 3, is approximately 8.85 m, and the overburden bedrock consists of clayey sandstone, siltstone and fine and medium sandstones. Panel S03 belongs to Southern part of the 6th district in the Taiping Coal mine. This area is located in the rising end of a syncline, the overburden bedrock thickness ranges from 20 to 32.6 m, and much area of the coal seam is directly covered by the Quaternary unconsolidated formations with an average thickness of about 160 m. A confined aquifer is located at the bottom of the Quaternary unconsolidated formations, with a specific capacity $q = 0.023\text{--}0.11$ L/s m, a water pressure $P = 0.597$ MPa, a hydraulic conductivity $K = 0.183\text{--}1.47$ m/day, and an average thickness of 30.0 m. In the mining process, the mining induced fractures propagate into this aquifer, and the water from this aquifer flows into Panel S03.

When the first slice of coal seam was excavated to a distance of 45 m, the water inflow in Panel S03 began to increase, and when excavation advanced to a distance of 50 m, the water inflow increased from 18 to 100–110 m³/h. At this stage, the aquifer water pressure decreased from 0.597 to 0.555 MPa. As the mining distance increasing, the water inflow decreased, while the water pressure increased: for instance, as the coal seam was excavated to a distance of 98 m, the water inflow reduced to 40–45 m³/h and the aquifer water pressure increased to 0.597 MPa, as shown in Fig. [2](#). These results indicated that initially opened fractures, which propagated into the aquifer, then became closed.

Fig. 2

In-situ monitoring of water inflow and pressure variations with time and mining distance in the water inrush process (Wang et al. [2016](#))

In the same coalfield, a borehole TV viewer system was used to monitor the fracture closure after the coal excavation 15 years ago. The results shown in Fig. [3](#) indicated that the fractures in the upper part of the fractured rock mass became closed. Based on multiple results of monitoring, using the color TV viewer inserted in the borehole, mud slurry leakage and RQD of cores, Wang et al. ([2013](#)) suggested that the height of the water flow fracture zone reduced by nearly 40 % compared to the height of the zone propagated into the aquifer. These in situ monitoring results have provided a basis for the scale model testing and a theoretical analysis of the fracture closure, which are described below.

Fig. 3

Images obtained from the color TV viewer lowered in a borehole, illustrating closed fractures (shown in yellow color). a 19.88 m above the coal seam, b 12.76 m above the coal seam

3 Materials and Methods

3.1 Scale Model Test

A scale model was developed to accurately study the relationships between important geometrical and physical features of the mining area. To provide a physical representation of the mining area, the dimensions of the scale model were decided to be 45 cm × 15 cm × 10 cm (length × height × thickness), as shown in Fig. 4, which correspond to the following scales 1:200 for geometry, 1:1.8 for gravity, and 1:360 for the uniaxial compressive strength. The material used to construct the model comprised a combination of quartz powder (particle size <0.075 mm) and gypsum, which reduced the influence of particle size on the fracture propagation. In order to study the fracture evolution of the bedrock due to coal mining, a uniformly distributed load was applied on the top of the model assembly to simulate the weight of the upper unconsolidated formation. Corresponding compensatory horizontal stresses, which were calculated according to the lateral pressure coefficient, were applied on both sides of the model. The simulated stratigraphic distribution of the bedrock was shown on the right hand side in Fig. 4. In the excavation process, a 15.0 m pillar was set on the right side, where the first cut was 2.0 m coal seam was excavated in every 15 min. The cover stress re-establishment at the bottom of the coal-bed model was monitored using 8 film Force-Sensitive Resistor (FSR™) sensors (made in China). The fracture propagation was monitored using a digital camera. The results of scale model testing are given below in Sect. 4.

Fig. 4

Schematic of the scale model test: 1 # –8 # are (FSR™) sensors numbers. Dimensions on the figure are given in cm—scale 1:200

3.2 Theoretical Analysis

3.2.1 Basic Assumptions for Calculations

The strength and stiffness of the intact rock are much greater than those of the fractured rock mass. Most of the rock strata generate fractures instead of being stretched under tensile stress, thus we can assume that the fractures generate in the bending parts of the rock mass. As shown in Fig. 1, the cover stress re-establishment is partial in the bending parts in the broken roof (Wilson 1983; Qian et al. 1996; Yavuz 2004). In a bending rock stratum, the fractures extend generally vertical to the dip of broken rock strata with the similar deformation process in the cover stress re-establishment process. In order to describe the dynamic process of the fracture closure in the fractured zone, we make the following assumptions:

1. The rock mass in the fractured zone is stiff, and the fractures are generated in the bending parts.
2. The broken rock strata bending distance from the mining face into the goaf is the same as the distance (X_a) of the cover stress re-establishment in the goaf.
3. The fracture rock model includes a simplified representation of fractures extended vertically to the dip of broken rock strata with the same aperture along the length.
4. The closure process of single fractures is the same as that for the fractures in the bending fractured rock strata.
5. The cover stress re-establishment is only in consideration of gravity stress.

3.2.2 Sagging form of Fractured Zone

Based on the in situ measurements, Wilson (1983) proposed that the roof sagging curve could be described using a logarithmic or exponential equation, which is given by

$$w = w_0 \left(1 - e^{-\frac{x}{\eta}} \right) \quad (1)$$

where X is the distance from the mining face, w is the roof sagging, η is an empirical coefficient determined from the statistical approximation of the measured values, w_0 is the roof maximum sagging.

Qian et al. (1996) found that the sagging curve of several broken rock blocks approximately fit an exponential function (Eq. 1), and proposed a system of Eqs. (2)-(4):

$$W_X = W_0(1 - e^{-X/2L})$$

$$W_X = W_0 \left(1 - e^{-\frac{x}{2L}} \right) \quad (2)$$

$$W_0 = M - h_p(B_p - 1) \quad (3)$$

$$L = \frac{h}{1 - \nu^2} \sqrt{\frac{2R_T}{q}} \quad (4)$$

where W_X is the sagging displacement at a distance X from the mining face, W_0 is the maximum sagging displacement, X is the distance from the mining face, L is the periodic fracture length, h and R_T are the thickness and tensile strength of the key stratum respectively, q is the total weight of the key stratum and above strata, B_p is the remnant bulking factor of the broken strata below the key stratum, h_p is the distance from the key stratum to the coal-bed, ν is the Poisson ratio.

In this paper, based on the assumptions given in Sect. 3.1, Eq. (2) is chosen to describe the sagging form of the fractured zone.

4 Results and Analysis

4.1 Scale Model Test

4.1.1 Fracture Distribution

Image binaryzation is used to quantitatively analyze the fracture distribution. The fracture propagation in the fractured rock mass forms a trapezoid distribution as a whole. The fracture apertures in the lower parts are greater than those in the upper parts. When the coal seam is excavated at the distance of 34.0 m, the fractures propagate to the maximum height of 18.0 m, and the overburden failure is shown in Fig. 5. Through statistical analysis, the fracture ratio of the fractured rock mass decreases as the distance increases from the coal seam floor in the vertical direction, and basically can be described by a logarithmic function given by

$$(5) \quad y = a - b * \ln(c + x)$$

with $R^2 = 0.68$, as shown in Fig. 6a.

Fig. 5

Fracture distribution of the overburden failure at a mining distance of 34 m

Fig. 6

Fracture ratio distribution within a mining distance of 34.0 m: **a** graph of the vertical fracture ratio along the coal seam floor, and **b** graph of the horizontal fracture ratio along the mining

The fracture ratio distributes like an “M” shape in the horizontal direction, and it is greater at both sides and lower in the middle zone, as shown in Fig. 6b.

4.1.2 Fracture Closure

In order to investigate the fracture closure process, a typical fractured zone with a width of 4.0 m from 32.0 to 36.0 m is selected as the study object. The fracture distributions at different mining distances are shown in Fig. 7 that illustrates that fracture apertures decrease, while the mining distance increases, and the fracture apertures in lower parts are greater. To quantitatively analyze the fracture closure, the fracture ratio was determined as a ratio of the area of fracture pixels to the area of all pixels. The dynamic distribution of the horizontal fracture ratio along the vertical direction of the study area is shown in Fig. 8. This figure also indicates that the fracture ratio significantly decreases in most parts of the fractured rock mass while the mining distance increasing. There is an especially obvious decrease of the fracture ratio when the mining distance increases from 44 and 60 m.

Fig. 7

Schematic illustration of the process of the fractures evolution due to coal mining at different coal mining distance

Fig. 8

Vertical distribution of fracture ratio for overall fractures with different coal mining distance

The height of the fractured rock mass is divided into nine 2-m segments. Figure 9 shows calculated averaged values of a fracture ratio of every segment along the mining distance. From Fig. 9, we can see that the fracture ratios of the lower parts in the fractured rock mass with large apertures are greater than those in the upper parts. The fracture ratio decreases greater in the lower parts than those in the upper parts with the mining distance increasing. The fracture ratios of all segments decrease faster in the stage, when the cover stress begins to re-establish, and as the mining distance increases the decrease rate becomes slower. These phenomena are related to the cover stress re-establishment and the stiffness of the fractures. Generally, the initial contact area, aperture distribution, small-scale roughness, strength and deformability of asperities, and thickness, type and physical properties of infilling material affect the normal stiffness of a rock fracture (Worthington and Lubbe 2007; Alireza and Lanru 2008; Philippe et al. 2011). Worthington and Lubbe (2007) found that the fracture aperture affected its stiffness, the stiffness of an air-filled (or 'drained') fracture diminished with an increase of the aperture and length, which indicated that the stiffness of fractures in the lower parts with greater aperture was less than the upper ones. Bandis et al. (1983) proposed a hyperbolic function to express the normal effective stress-closure relationship given by

$$(6) \quad d_n = \frac{\sigma_n}{k_n + (\sigma_n/d_{\max})}$$

where σ_n is the normal effective stress, d_n is the fracture closure, $d_{\max}$ is the maximum allowable closure, and k_n is the normal stiffness of the fracture at initial stress. From Eq. (6), we can see that the fracture closure rate generally decreases with normal effective stress increasing, which is the reason causing the fracture closure rates decrease in the later stage after the mining ceased.

Fig. 9

Graphs showing the decrease in fracture ratios at different heights (*color lines*) of fractured rock mass with the mining distance

Vertical fractures are main seepage channels for water inrush from the overburden. The vertical component of the rock mass permeability mainly controls the water inflow. As an example, Fig. 10 illustrates the closure process of the vertical fracture along the mining distance.

Fig. 10

Closure process with mining distance for one vertical fracture in fractured zone

The changes of the horizontal fracture ratio distribution in the vertical direction are shown in Fig. 11. When the mining distance was 44 m, the fracture ratios were all greater than zero, which meant that the vertical connectivity was significant, and water could flow through these seepage channels. As the mining distance increased and the fracture ratios decreased, when the mining distance reached 60 m, the fracture ratios mainly in the upper parts of the fractured zone (where the fracture aperture was small) decrease to zero. Because of poor hydraulic connectivity, water nearly could not flow through these rock masses. At the lower parts, though the fracture ratios were still greater than zero (although they greatly decreased), indicating that the fractures were still open.

Fig. 11

Vertical fracture ratio changes with mining distance for one vertical fracture in fractured zone

The average fracture ratios calculated as a function of the mining distance, using the data shown in Figs. 7, 11 are summarized in Table 1. The 2nd and 3rd columns of Table 1 include the calculated fracture ratios. To demonstrate how the fracture ratios changed along the mining distance, the 4th and 5th columns include normalized fractures ratios, obtained by dividing the calculated fracture ratios at different distances by the fracture ratios at the distance of 44 m. The changes in the normalized fracture ratio along the mining distance are shown graphically in Fig. 12. This figure demonstrates that the fracture ratios decrease as mining distance increases following a logarithmic function given by Eq. (5) with $R^2 = 0.98$ and 0.99 respectively.

Table 1

Average fracture ratios change with mining distance

Mining distance (m)	Fracture ratio I (%)	Fracture ratio II (%)	Normalized fracture ratio I	Normalized fracture ratio II
44	2.34	0.64	1.00	1.00
46	1.83	0.44	0.78	0.68

Mining distance (m)	Fracture ratio I (%)	Fracture ratio II (%)	Normalized fracture ratio I	Normalized fracture ratio II
50	1.41	0.28	0.60	0.44
56	1.16	0.22	0.50	0.35
60	0.88	0.17	0.37	0.27

Fracture ratio I represents the results shown in Fig. 7; fracture ratio II represents the results shown in Fig. 11

Fig. 12
Relationship between normalized fracture ratios and mining distance

4.1.3 Cover stress Re-establishment Monitoring Results

The stress monitoring results show that the coal-bed excavation causes the stress increase in front of the mining face. When the mining face passes the stress monitoring sites, the stress decreases to zero immediately. Then, as the overburden breaks and collapses on the coalbed floor, the stress increases, which is shown in Fig. 13. In this scale model testing, the first time and periodic roof weighting lengths are about 30 and 12 m respectively. The stress monitoring results also indicate that the cover stress re-establishment is spatially instable. In some stress monitoring sites, the cover stress re-establishes greater than the initial one, however, some monitoring sites show less re-establishment than the original one or even no re-establishment. The inner structures of the broken rock mass in the goaf are the main reasons for this phenomenon, and some film stress sensors do not or partly load the weight of the overburden, some film stress sensors load greater weight.

Fig. 13

Monitoring results of stress concentration coefficient variations on the coal seam floor in the mining process

Figure 14 depicts the monitoring data from three film stress sensors (3#, 4#, 5#), which were installed at the locations (see Fig. 1), where the cover stress re-establishment process was practically completed. The mining distance is set to zero when the cover stress begins to re-establish at the film stress sensor.

Figure 14 shows the cover stress (normalized) curves, which are obtained by dividing by the maximum cover stress re-establishment value. These curves are fitted well to a logarithmic function given by Eq. (5) with R^2 from 0.88 to 0.97.

Fig. 14

The relationship between cover stress re-establishment ratio and mining distance

Using the monitoring data from the film stress sensor 3#, shown in Figs. 7 and 11, we determined the relationship between the fracture ratio and the cover stress re-establishment, shown in Fig. 15. Note from Fig. 15 that the horizontal x-axis is the inverse of the ratio between the cover stress and the final value of the cover stress re-establishment, and the vertical y-axis is the decrease in the fracture ratio degree. A logarithmic function given by Eq. (5) can be used to fit these two curves.

Fig. 15

Relationship between the decrease in the degree of fracture ratio and the degree in the cover stress re-establishment

4.2 Theoretical Analysis

4.2.1 Fracture Aperture in the Fractured Zone

Different broken rock blocks compose the bending curve of the fractured rock strata. Here, an assumption is proposed that the bending curve is composed by n straight rock blocks, each one with a length of L and a dip angle of θ_i ($i = 1, 2, 3 \dots n$). The slope of each part can be determined by taking a derivative of Eq. (2) given by

$$\tan \theta_i = k_{X_i} = (W_{X_i})' = \frac{W_0 e^{-\frac{X_i}{2L}}}{2L} \quad (7)$$

$$\theta_i = \arctan \left(\frac{W_0 e^{-\frac{X_i}{2L}}}{2L} \right) \quad (i = 1, 2, 3 \dots n) \quad (8)$$

The dip angle difference between two adjacent rock blocks can be given by

$$\Delta \theta_i = \theta_i - \theta_{(i-1)} = \arctan \left(\frac{W_0 e^{-\frac{X_i}{2L}}}{2L} \right) - \arctan \left(\frac{W_0 e^{-\frac{X_{i-1}}{2L}}}{2L} \right) \quad (9)$$

Then, the aperture of a fracture can be calculated from the following equation:

$$(10) \quad (d_i)_{\max} = 2h_f \cdot \sin \frac{\Delta\theta_i}{2} = 2h_f \cdot \sin \frac{\arctan\left(\frac{W_0 e^{-\frac{x_i}{2L}}}{2L}\right) - \arctan\left(\frac{W_0 e^{-\frac{x_{i-1}}{2L}}}{2L}\right)}{2}$$

where h_f is the thickness of the fractured zone, and $(d_i)_{\max}$ is the maximum aperture of a fracture, which is located at the bottom of the fractured zone. At the top of the fractured zone, the fracture aperture is assumed to be zero, and the fracture aperture distributes linearly between the top and bottom of the fractured zone.

4.2.2 Cover Stress Re-establishment of Fractured Zone

The fractured zone is above the caved zone, its sagging displacement is taken place due to the compression of caved materials. According to the characteristics of the caved rocks, Salamon (1990) suggested the following equation to describe the backfill material and the effect of the stress-strain behavior of gob material:

$$(11) \quad \sigma = \frac{E_0 \varepsilon}{1 - \varepsilon / \varepsilon_m}$$

where

$$(12) \quad \varepsilon_m = \frac{B - 1}{B}$$

Yavuz (2004) proposed the following formula for E_0 :

$$(13) \quad E_0 = \frac{10.39 \sigma_c^{1.042}}{B^{7.7}}$$

where E_0 is the initial tangent modulus, σ_c is the uniaxial compressive strength of rock, ε is the strain occurring under the applied stress, σ is the applied stress, ε_m is the maximum strain of bulked rock material, h_c is the height of caved zone, m is the excavation thickness, B is the bulking factor of the caved zone.

The deformation of the caved material in the goaf can be calculated by the following equation:

$$(14) \quad \varepsilon = \frac{W_X}{h_c + M} = \frac{W_0 \left(1 - e^{-\frac{x}{2L}}\right)}{h_c + M}$$

By substituting Eqs. (12)–(14) into Eq. (11), we obtain the equation to describe the cover stress re-establishment function, given by:

$$\sigma = \frac{10.39\sigma_c^{1.042}(B-1)W_0\left(1 - e^{-\frac{x}{2L}}\right)}{B^{7.7}\left[(h_c + M)(B-1) - BW_0\left(1 - e^{-\frac{x}{2L}}\right)\right]} \quad (15)$$

4.2.3 Relationship Between the Closure of Fracture and Cover Stress Re-establishment in Fractured Zone

From Eqs. (10) and (15), we calculated the aperture and cover stress changes with the position of x . Calculated parameters, based on Panel S03 from Taiping Coal Mine in Shandong Province of China, are given in Table 2. From Eq. (4), the calculated value of L is 13.2 m. The closure process of a fracture and the cover stress re-establishment with the distance increase from the mining face, which were based on Eqs. (10) and (15), respectively, are shown in Fig. 16. A logarithmic function given by Eq. (5) was used to fit the experimental data with $R^2 = 0.997$ and 0.97 , respectively. The relationship between the closure ratio of fracture and cover stress re-establishment ratio is shown in Fig. 17, which also fit well the logarithmic function (Eq. 5) with $R^2 = 0.999$.

Table 2

Parameters for calculation

M (m)	h_c (m)	h_f (m)	σ_c (MPa)	B	R_T (MPa)	ν	q (MPa)	W_0 (m)
2.2	7.3	13.5	11.3	1.3	1.6	0.23	3.72	1.05

Fig. 16

The aperture closure and cover stress re-establishment with distance

Fig. 17

The relationship between closure ratio of fracture and cover stress re-establishment ratio

Thus, the analysis of experimental data confirms that the aperture of fracture decreases with the cover stress re-establishment, and can be described using the logarithmic function given by Eq. (5).

5 Discussion and Conclusions

The results of the theoretical analysis and a scale model testing were focused on the evaluation of the cover stress re-establishment and the fracture closure in the fractured rock mass after coal mining. These results indicate that the cover stress re-establishment with the increase in the mining distance and the relationship between the fracture closure and cover stress re-establishment both satisfy a logarithmic function given by Eq. (5). The scale model testing used in this study was based on the specific geological characteristics of the Taiping Coal Mine, and the evaluation of the cover stress re-establishment was based on only a vertical gravity stress.

Thus, based on the results the theoretical analysis and the scale model test, the cover stress re-establishment is increasing with the mining distance, and the relationship between the fracture closure and cover stress re-establishment both satisfy a logarithmic function given by Eq. (5).

The fracture ratio in the lower parts of the fractured rock mass is greater than that in the upper part, and fracture ratios decrease initially fast and then slower with time during the cover stress re-establishment process. The vertically directed fractures with small aperture in the upper part of the fractured rock mass are generally compressed and then become closed during the cover stress re-establishment, thus, reducing the seepage into the mining area.

Further studies should be directed on the evaluation of a horizontal stress re-establishment in the fractured rock mass. The developed approach can be used for the evaluation of the cover stress re-establishment and fracture propagation and closure form under different structures of overlying rock mass.

This study improved the understanding of closure process of fractures in the fractured rock mass, and provided a basis for further studying of the water-resisting capacity of the fractured rock mass after coal mining ceased.

Notes

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