Water Inrush and Failure Characteristics of Coal Seam Floor Over A Confined Aquifer

Min Cao (✉ caomin727@163.com)

China University of Mining and Technology - Beijing Campus <https://orcid.org/0000-0002-6437-9780>

Shangxian Yin

North China Institute of Science and Technology

Bin Xu

North China Institute of Science and Technology

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Water Inrush and Failure Characteristics of Coal Seam Floor over A Confined Aquifer

Min Cao¹ · Shangxian Yin² · Bin Xu² ·

Abstract Failure behaviors of the floor rocks under coal seam mining in the conditions of hard magma rock roof and confined aquifer are studied. Based on the theory of rock stresses and elasticity mechanics, the combined effects of the abutment pressure induced by the hard roof and by the water pressure under the thin aquicludes of the floor rocks were considered, and a mechanical model was constructed along the strike of the working face. An analytical solution of stress distribution was derived in the floor rocks, the distributions of vertical, horizontal and shear stresses were calculated. In combination with the in-situ measurement, the results show that: 1) when the periodic pressure caused by the roof collapse occurs on the working face, and the maximum stress concentration in the floor appears at the elastic-plastic junction in the direction of the strike of the working face. With the increase of the depth of the floor, the horizontal stress coefficient tends to decrease, and the corresponding shear stress coefficient isoline shows a “symmetric spiral” distribution and propagates downward to the floor at a certain angle with the vertical direction. This causes the floor rocks to generate compression and shear or tension and shear failure. 2) when the immediate roof of coal seam is the magma rock, the abutment pressure shows a trend of a slow increase initially and then a rapid increase later. The peak value of abutment pressure appears at the location of 4 - 6 meters from the coal wall of the working face, and the concentration coefficient of the abutment pressure is between 1.4 and 1.8. 3) according to the measurement and calculation of the failure depths of the floor at different positions under the same coal seam, it is found that the maximum failure depth appears near the coal wall of the working face. The failure depth reduces by 11.6% after the floor goes through “the roof caving and re-compaction”, which causes the fractures in the floor to close and the thickness of the effective aquiclude increases. In the un-mined area of the working face, the failure depth is 55% of the maximum failure depth. 4) both the theoretical calculation and the numerical simulation show that the failure depth of the floor increases obviously under the combined action of high vertical stress and the water pressure. Under the condition that the thickness of the aquiclude is relatively thin, the water pressure of the floor and pressure intensity of the roof are the sensitive factors to affect the maximum failure depth of the floor.

Keywords Confined aquifer · Abutment pressure · Failure depth of coal seam floor · Hard roof

Introduction and Background

Many coal seams in China are located under very stiff and thick rock strata. Because the roof is consisted of the heavy weight of thick and hard rock strata, this hard roof causes the roof hanging in a very large area without collapse after mining, which results in frequent disasters such as rock burst. When coal seam mining undergoes both high stress action from the hard roof and from high pressure of the confined aquifer, the stresses in coal seam floor caused by mining redistribute, and rocks in the floor deform, causing different degree of damage or failure of the floor rocks. This can induce mine water inrush accident (Liu et al. 2010). If the coal seam roof

✉ Shangxian Yin, yinshx03@126.com

¹ School of Energy and Mining Engineering, China University of Mining and Technology, Beijing 100083, China

² Hebei State Key Laboratory of Mine Disaster Prevention, North China Institute of Science and Technology, Beijing 101601, China

consists of thick and hard rock formation, the hard rock is usually the key stratum, and its collapse or fracturing after coal mining usually causes very large movement of the overlying rock strata. The movement of the roof strata is the source of high abutment pressure in the coal wall around the working face and the potential disaster of roof collapse. After the coal has been excavated, the thick alluvium overburden has a bearing capacity that is insufficient to support its own weight and thus exerts excessive load on the thin underlying bedrock. The resulting intense disturbance to the working face can easily cause rock bursts, water inrush from the floor, and other hazards. The advanced abutment pressure is also the indirect reason of water inrushes from the floor rocks. The existence of hard thick rock strata in the roof usually leads to the advanced abutment pressure having a high peak value and a large exerting range of the advanced abutment pressure which causes a large failure range of the floor rock under the working face. Deep coal mines face problems due to high ground pressures, temperatures, and water pressure. With increasing mining depth, water pressure in aquifers continuously rises, and the risk of water inrush from the floor also tends to increase. Therefore, the study of mining-induced stress distribution, rock deformation and failure characteristics should be emphasized for prevention and control of water inrushes from the aquifer (Adams and Younger 2010; Singh 2013; Guo et al. 2008). At present, many studies on this topic have been carried out including theoretical analysis, numerical simulation, physical similarity simulation and good results have been obtained. The field measurement is very difficult to be performed because of high cost, long measurement time and tough working conditions; however, the measured data are more reliable and accurate. Generally speaking, there are no many accomplishments in field measurements, and it has been found that some difference exists between the measured results and the theoretical prediction.

In terms of theoretical analysis, Wang et al. (2013) established a spatial semi-infinite rock model with comprehensive consideration of the abutment pressure characteristics along the strike and dip directions of the working face and derived an iterative equation for calculating the vertical stress in the floor. Using this equation, the stress distribution of the floor in different depth was calculated using the software MATHCAD. The layered rock mass of the floor as transversely isotropic continuum and derived an analytical solution of stress at any point in the floor after mining. Based on the theory of elastic mechanics, a calculation model of the floor stress considering the combined action of the distribution of the abutment pressure and the water pressure in the confined aquifer along the dip direction of the working face (Lu et al. 2014; Song et al. 2018).

In the field measurements, the mining symmetrical quadrupole resistivity method has been used to measure the failure depth of the floor at different mining thickness and the software Spass has been used to analyze the relationship between four influence factors of mining and the floor failure depth (Li et al. 2015). Xiao et al. (2001) combined thick seam stratified mining with the observation method of seam floor movement and obtained the movement and deformation law of seam floor rock in stratified mining (Liang et al. 2019). Zhang et al. (2006) used the seismic wave CT detection technology, combined with the detection section between the borehole and the roadway in the coal seam working face, and obtained the dynamic development and the characteristic of the floor failure in the coal seam mining process.

In the aspect of numerical simulation, Liu et al. (2017a) studied the delayed water inrush process of hidden faults in coal seam floor by means of the FLAC3D simulation analysis method, and put forward the formation law of potential water inrush passage under the influence of time effect. The results are in good agreement with the actual situation in the field (Meng et al. 2018; Sun et al. 2018; Zhang et al. 2017). Li et al. (2015) used FLAC3D analysis software to establish numerical calculation and analysis models under different working face width and coal seam burial depth, obtained the maximum floor failure depth corresponding to different parameters. They obtained the critical width fitting formula of No. 5 coal seam in Chenghe mining area under the condition of two factors by using the MATLAB software fitting analysis method. Using numerical simulation software FLAC3D, Yang (2018) studied the plastic failure of the floor under different mining thicknesses, mining lengths of the working face and widths of the coal pillar. Although many studies have been conducted as mentioned above, the circumstances of coal mining under a very hard-igneous rock roof and with a thin floor but having a high water pressure of confined aquifer below the floor are seldom encountered. Working face No. 29205 of the

Guoerzhuang Coal Mine in Hebei Province is under this condition. We use this coal mine as the background to conduct a detailed study. We establish the floor mechanical model along the strike of working face with consideration of the influence of the hard roof strata and confined aquifer of the Ordovician limestone under the floor. The stress distribution and failure characteristics of the floor under the thick and hard roof are theoretically analyzed. Using the field test results the correctness and reliability of the model are verified. Therefore, this paper may provide a theoretical basis for prevention and control of mine water inrush disasters under unexpected situations.

1 Research Site Overview

The burial depth of the working face No. 29205 is 251-280m with an average depth of 270m. The main mining seam is the 9th coal seam, and its thickness is 1.5-3.8m, with an average thickness of 2.98m. The dip angle of coal seam is 5-8°. The strike length of the working face is 680m and the dip length is 70m. There is a layer of shale in the middle and upper part of the coal seam with an average thickness of 0.84m. The immediate roof of the coal seam is the intrusive magma rock with an average thickness of 15.27m, and the immediate floor of the coal seam is mudstone and the Benxi limestone with an average thickness of 7.65m. There is no fault structure in the working face. The comprehensive histogram of strata exposed by drilling is shown in Fig. 1. The high pressure confined aquifer of the Ordovician limestone is located 27 - 40m below the coal seam, and the maximum water pressure can reach 2.8 MPa, which is a great threat to coal mining. Therefore, Study on the failure characteristics of the floor under this special condition is of critical significance for prevention and control of water inrush disasters in this area.

Figure 1 will be placed near here during the printing process

2 Mechanics Model of Floor Failure

Under the combined action of mining perturbation, roof pressure and water pressure of the aquifer, the maximum failure depth of the floor in the course of coal seam mining, as well as the stress concentration in different areas of the floor rock mass are studied. The mechanical failure model of the floor is established along the strike direction of the working face to obtain an analytical solution of the stress distribution in the floor.

With mining advance in the working face, the floor rock mass in front of the working face coal wall is compressed by the action of the advanced abutment pressure. When the magnitude of the advanced abutment pressure exceeds the ultimate strength of the floor rock mass, the floor rock mass will produce plastic deformation and cause rock failure. As the mining further advances, the floor rock mass with plastic deformation becomes the floor rock mass under the goaf because of the roof collapse, and then this part of floor rock mass changes from compression to expansion states because of the roof pressure relief in the goaf. Due to this process, the fracture zone or mining failure is formed in the floor rock mass.

2.1 Construction of Mechanical Model of the Floor Failure

According to practical rock pressure theory of mining (Tan et al. 2008) and without considering the influence of tectonic stress, the stress state of the floor rock mass can be simplified in Fig. 2 when the mining is carried out above the confined aquifer along the strike direction of the working face, as shown in Figure 2(a). Assuming that the effect of tectonic stress and horizontal principal stress on the floor is not considered (Fig. 2b) and according to the distribution characteristics of the abutment pressure and the effect of the water pressure on the floor, the mechanical calculation model in the floor is built when the first collapse of the roof occurs. The abutment pressure is simplified as shown in Fig. 3: the plastic zone (D) is simplified as a triangular linear load, the elastic zone (B

and C) is simplified as two trapezoidal loads, and Zone A is the initial in-situ stress zone. The water pressure in the Ordovician limestone aquifer is simplified to a uniformly distributed load exerted on the bottom of the floor.

Figure 2 will be placed near here during the printing process

Because the distribution pattern of the abutment pressure of the first and periodic pressures of the roof collapses on the working face have the similarity, the periodic pressure is here used as an example for derivation.

Take the position of the working face as the origin, the Y axis is parallel to the floor and points horizontally left along the coal wall and the working face boundary. The X axis is perpendicular to the floor, as shown in Fig. 3. If the initial vertical stress is γH , the maximum stress concentration along the strike direction of working face is $k_d \gamma H$ when the first collapse of the roof (the first roof pressure) occurs, and there is an inflection point where the stress increase reduces in the stress concentration area, and the stress value is $k_c \gamma H$. The effect of the abutment pressure on the floor is assumed to be 0 at the junction of the coal wall and the working face roof-control area. The width of the trapezoid load $F_a(y)$ and the triangular load $F_b(y)$ in the elastic area is g and f , respectively. The width of the triangular load $F_c(y)$ in the plastic area is e , and the width of the uniformly distributed load $F_d(y)$ in the confined aquifer is $g + f + e$.

Figure 3 will be placed near here during the printing process

According to the theory of elastic mechanics (Xu 2002), in the polar coordinate, the stress components generated from each linear load shown in Fig. 3 at an arbitrary point G (y, x) in the floor rock mass can be obtained in the following:

(1) The stress components under the trapezoidal linear load $F_a(y)$ are:

Equations (1) (2) (3) (4) (5) will be placed near here during the printing process

Where γ is the bulk density of overlying strata, H is the mining depth and k is the concentration factor of vertical stress.

(2) The stress components under the trapezoidal load $F_b(y)$ are:

Equations (6) (7) (8) (9) (10) (11) will be placed near here during the printing process

(3) The stress components under the triangular linear load $F_c(y)$ are:

Equations (12) (13) (14) (15) (16) (17) (18) will be placed near here during the printing process

Based on the principle of superposition of stresses, the stress components under each linear load are added correspondingly, the expression of vertical stress, horizontal stress and shear stress at point G (y, x) in the floor rock mass along the strike of working face for mining over the confined aquifer in the first roof pressure are as follows:

$$\sigma_x = \sigma_{xa} + \sigma_{xb} + \sigma_{xc} \quad (19)$$

$$\sigma_y = \sigma_{ya} + \sigma_{yb} + \sigma_{yc} \quad (20)$$

$$\tau_{xy} = \tau_{xya} + \tau_{xyb} + \tau_{xyc} \quad (21)$$

After replacing the corresponding parameters in the periodic roof pressure, the corresponding stress expression in the floor in the periodic roof collapse condition can also be obtained.

2.2 Calculation of Stress Distribution in the Floor

According to the geological data of working face No. 29205, the dip length of the working face is taken as $L = 80m$, the distance from the immediate coal seam floor to the upper boundary of the confined aquifer is $H_z = 50m$. Based on the borehole water level measurements, the highest water pressure of the confined aquifer in the last five years is $P = 2.8MPa$, and the rock bulk density is $\gamma = 25KN/m^3$. According to the field measurement results, along the strike of the working face, the width of the plastic zone is $e = 8m$, the elastic zone is $f = 25m$, K_d is taken as 1.8, and K_c is 1.4 in the direction of the coal wall under the periodic roof pressure condition.

Using the software MATHCAD and combined with the derived stress expressions in Eqs. 19, 20 and 21, the stress distribution in the floor along the mining advance direction of the working face under the periodic pressure condition is calculated, as shown in Figs. 4, 5 and 6.

Figure 4 will be placed near here during the printing process

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Figure 6 will be placed near here during the printing process

By comparing the contour lines of vertical stress, horizontal stress and shear stress coefficients in the floor rock mass along the strike direction of the working face under the periodic roof pressure, the following characteristics can be obtained:

(1) According to the vertical stress distribution, the vertical stress concentration and unloading phenomenon appear near the coal wall of the floor rock mass, and the maximum stress concentration is near the elastic-plastic deformation boundary, and the maximum stress concentration coefficient is 3.24, which is basically consistent to the measured result.

(2) According to the horizontal stress distribution, the stress concentration area appears in the middle part of the elastic zone when the initial roof pressure occurs, and gradually weakens to both sides. With the increase of mining depth, the horizontal stress contour line tends to relax.

(3) According to the distribution of shear stress, the maximum value of shear stress appears in the floor rock below the maximum abutment pressure, and the isoline of shear stress coefficient is bounded by the middle of elastic zone. The shapes of the shear stress distribution looks like a “symmetrical spiral”, and it propagates downward at a certain angle with the floor normal line, forming a shear stress transition zone. The existence of the shear stress transition zone can promote fracture extension and lead to the compression-shear or tension-shear failure. This is consistent with Zhang’s theory that shear deformation and shear failure are easy to occur in the floor rock mass under the lower edge of stope at the junction of compression zone and expansion zone, which is a perfect match for the actual floor failure (Zhang 1997).

3 Numerical Simulation of Floor Failure under Different Conditions

The failure characteristics of the disturbed floor was simulated using FLAC3D software. The embedded seepage analysis module allows for coupling of fluid and mechanical computations. In this study, we investigated the failure characteristics by observing the distribution of plastic zones in the model (Blöcher et al. 2010; Hamm and Sabet 2010).

In order to compare the difference of floor failure characteristics under the normal roof condition, hard roof condition and hard roof condition affected by the action of the confined water pressure, three kinds of numerical simulation models were established according to the mining conditions of the Guoerzhuang Coal Mine. In the

simulation model, the depth of coal seam is 270m, the mining height is 3m, the dip angle is 6° , and the length of working face is 80m. The numerical calculation model is simplified to contain 10 rock layers, in which the roof has 4 layers the floor has 5 layers and one for coal seam. The strike length of the model is 400m, the dip width is 400m. In the model, the strike direction of the working face is X axis, the dip direction is Y axis, and the vertical direction is Z axis. The four boundaries of the model are subject to normal constraints, and the bottom is subject to vertically fixed constraints. The parameters such as the overburden stress in each model remain unchanged.

In the numerical simulation models, the normal and hard roof conditions, are modeled by only changing the physical and mechanical parameters of the immediate roof of the coal seam, as shown in Table 1. Three models were simulated: Model 1 with a normal immediate roof, Model 2 with a hard immediate roof, and Model 3 with a hard immediate roof and with a 2.8 MPa water pressure exerted to the floor. By comparing and analyzing the deformation and failure behaviors of the floor after coal seam mining under these three conditions, the following results are obtained.

Fig. 7, 8 and 9 are plane views of the displacement variations due to mining of the working face at depths in the floor of 1m, 10m, 20m and 30m under the conditions of the normal roof, hard roof and hard roof with confined water pressure, respectively. It can be seen that under the same floor depth, the floor displacement after mining under the hard roof condition is obviously larger than that under the normal roof condition, and the maximum displacement difference at the same depth is 15 cm. Compared the hard roof condition to the condition of the hard roof with the action of the water pressure, the displacement at the shallow depth of the floor is similar at the same depth of the floor after coal seam mining. However, it is different at the depths of 10m and 30m, and the displacement of the floor subjected to the water pressure is larger than that of the floor not subjected to it. This implies that the floor failure for the case with the water pressure is obvious in the deeper floor rock.

Table 1 will be placed near here during the printing process

Figure 7 will be placed near here during the printing process

Figure 8 will be placed near here during the printing process

Figure 9 will be placed near here during the printing process

Figure 10 will be placed near here during the printing process

Figure 11 will be placed near here during the printing process

Fig. 10 shows the stress distributions of 120m at the midpoint of the working face along the direction of the strike under three different conditions (three models). Fig. 11 plots the vertical stresses along the strike direction at different depths of the floor in three models. The following conclusions can be drawn from Fig. 11:

(1) The stress of the floor in 0-8m in front of the coal wall in the working face increases sharply at the shallow depth of the floor (within 20m), and the stress in 8-60m decreases slowly. The peak value of the stress under the hard roof condition is 21.7% higher than that of the normal roof, and under the condition of the Ordovician limestone water pressure, the vertical stress peak of the hard roof slightly increases with a small extent.

(2) The most obvious area affected by mining is at the depth of 0 and 25 m in the floor under the condition of normal roof, and the influence is weak in the depth of between 25 and 50 m, and the area where the depth is > 50 m is basically in the state of virgin rock stress. The area affected by mining is increased to the depth of 0 - 35 m in the floor under the condition of the hard roof, and the area is enlarged to the depth of 0 - 40 m in the floor with the action of water pressure of Ordovician limestone.

(3) At the shallow depth of the floor of less than 20 m, under the condition of the normal floor, the stress is released rapidly in the goaf of 0 - 20 m behind the coal wall of the working face, and the minimum stress value is near to 0. under the goaf farther away from the coal wall, the stress in the floor gradually increases and restores to 80% of the virgin rock stress at 60 m from the coal wall. In the condition of the hard roof, the stress in the floor

is released within 0-10 m in the goaf behind the coal wall of the working face; however, the minimum stress cannot reach zero, and the stress can only restore to 60% of the virgin rock stress in the floor under the goaf at the distance of 60 m from the coal wall.

4 Field Measurement

4.1 Field Measurement of the Advanced Abutment Pressure beneath the Coal Wall

4.1.1 Equipment of Measurement

The advanced abutment pressure in the floor caused by the roof stress on the coal wall is measured by means of a borehole stress testing device, which is composed of four parts: oil pump system, steel string force-measuring sensor, data collector and special power box. The hydraulic expansion pressure pillow and frequency meter are used to measure the relative stress in the rock around the borehole. The measurement combines the principle of steel string vibration with the characteristics of hydraulic technology. The stress change in the borehole is transmitted to the stress pillow through measuring the pressure change felt by the pressure pillow. The hydraulic pillow and the pressure-frequency converter are connected through an oil tube, and a collector collects the frequency change data caused by the pressure change.

4.1.2 Measurement Plan

As shown in Fig. 12, in order to make the measured data more representative and obtain the stress characteristics at different depths, the return airways of working face No. 29205 was taken as the observation roadway, and four horizontal boreholes were drilled from the roadway into the floor. The locations of the four boreholes (Boreholes No.1 to No.4) were 120m, 125m, 150m and 155m away from the coal wall of the working face, respectively. The drilling depths into the floor were different, which are 6 m, 9 m, 12 m and 15 m, respectively. and the drilling depths are different, which are 6 m, 9 m, 12 m and 15 m respectively. A borehole stress meter was installed in each borehole, and the height of the borehole is 2.5 m from the roof of the roadway. Real-time stress measurement was carried out by using the borehole stress meter in each borehole. According to the changes of the stresses measured from the borehole stress meter, the changes of the advanced abutment pressure on the coal wall was deduced to determine the location of the peak value of the maximum advanced abutment pressure, the range of the influence of the abutment pressure as well as the concentration coefficient of the abutment pressure.

Figure 12 will be placed near here during the printing process

4.1.3 Results and Analysis of the Measurement

The field observation data of the vertical stress in each borehole were collected as mining advance. Based on the measurement, the abutment stress curves in the four boreholes are drawn, as shown in Fig. 13, and the following results are obtained:

Figure 13 will be placed near here during the printing process

(1) Borehole No. 1 (Fig. 13a): after the stress gauge was installed into the bottom of the horizontal borehole, the stress on the borehole wall raised sharply and was in the stage of stress recovery on the borehole wall after the stress gauge installation. When the stress reached 7 MPa, the borehole stress tended to be stable, which could be considered that the stress was in the stable zone and reached the virgin rock stress (about 7 MPa). From the distance of 135 m to the coal wall, the borehole stress began to rise slowly and steadily. At the distance of 3.8 m to the coal wall, the stress reached its peak value, which was about 9.6 MPa. After the peak point, the stress decreased rapidly until the working face passed the monitoring borehole. According to the measured results in Borehole No. 1, the peak value of the advanced abutment pressure of the working face appeared 3.8 m from the

coal wall, and the peak value was 9.6 MPa. The peak concentration coefficient of the abutment pressure was about 1.37.

(2) Borehole No. 2 (Fig. 13b): In this borehole, the virgin rock stress was about 6.5 MPa. Starting from the distance of 68.2 m from the working face, the borehole stress began to increase obviously and the borehole was in the pressurization zone, commonly known as the abutment pressure influence zone. The zone of stress increase started from the distance of 68.2 m from the working face. At 6.5 m in front of the coal wall, the borehole stress reached the peak value, which was about 9.3 MPa. After the peak value, it decreased rapidly in a straight line, and finally dropped to below 4 MPa, which is obviously lower than the virgin stress.

According to the measured result in Borehole No. 2, the concentration coefficient of the peak abutment pressure was 1.41. The range of stress increasing zone was 61.7 m, and the farthest distance of the abutment pressure influencing zone was 68.2 m from the coal wall.

(3) Borehole No. 3 (Fig. 13c): As soon as the stress gauge was installed in the borehole, the stress of the borehole wall appeared relaxation; that is, the total deformation (elastic deformation and plastic deformation) of the borehole remained unchanged. The creep made the plastic deformation increase constantly and the elastic deformation decrease correspondingly, and the stress decreased slowly with time. After a short period of time, the stress relaxation disappeared, the borehole stress was stable and then raised slowly. This period could be regarded as that the stress gauge was located in the virgin rock stress zone, and the stress value was about 4.5 MPa according to the vertical coordinate axis in Fig. 13c. From the distance of 41.2 m to the coal wall of the working face, the increase of the stress value was obviously accelerated until it reached the peak value at the distance of 5.5 m from the coal wall of the working face. The peak value was about 8.2 MPa, and this distance was the abutment pressure influence zone (the length was 35.7 m). The stress decreased linearly after the peak value until the working face advanced to pass the borehole and then the stress was lower than the original rock stress.

According to the stress measurement at the measuring borehole No. 3, the following results were obtained: the peak value of the advanced abutment pressure of the working face appeared 5.5 m from the coal wall, and the peak value was 8.2 MPa. The coefficient of the abutment pressure was 1.82, and the range of the pressurization zone was 35.7 m. The farthest distance of the abutment pressure influence area was 41.2 m from the coal wall of the working face.

(4) Borehole No. 4 (Fig. 13d): As soon as the stress gauge was installed in the borehole, the stress of the borehole wall appeared relaxation. After a short period of time, the stress relaxation disappeared, the borehole stress remained stable and then raised slowly in the step-like form. During this period, the stress measured from the stress gauge could be considered to be the original rock stress, and the stress value was about 3.5 MPa according to the vertical coordinate axis in Fig. 13d. At the distance of 46.5 m from the coal wall of the working face, the increase of the stress was obviously accelerated until it reached the peak value at the distance of 6.2 m from the coal wall of the working face, and the length of the abutment pressure was 40.3 m. The stress decreased linearly after the peak point and until the working face advanced through Borehole No. 4.

According to the stress gauge measurement at the measuring borehole No. 4, the following results were obtained. The peak pressure of the advanced abutment pressure of the working face appeared 6.2 m from the coal wall, and the peak value was 8 MPa. The coefficient of the abutment pressure was 1.78, and the range of the pressurization zone was 40.3 m. The farthest distance of the abutment pressure influence area was 46.5 m from the coal wall of the working face.

From the above-mentioned measurements in four boreholes, the following results are obtained:

(1) when the distance between the fully-mechanized caving face and the borehole is 112 ~ 137 m, the reading of the borehole stress gauge begins to be greater than the virgin rock stress. Therefore, it can be explained that the influence range of the advanced abutment pressure of the working face is greater than 112m.

(2) When the distance between the fully-mechanized caving face and the borehole is 40 ~ 68m, the reading of the borehole stress gauge increases obviously, and there is an obvious inflection point, which indicates that

the influence range of the advanced abutment pressure of the working face can be divided into a slow rising area and a rapid rising area.

(3) The peak value of the advanced abutment pressure appears at the distance of 4 ~ 6 m from the coal wall, and the interval of the concentration coefficient of the abutment pressure ranges from 1.4 to 1.8.

According to the measured results, the measured vertical stress diagram in the coal seam floor along the strike of the working face is obtained, as shown in Fig. 14.

Figure 14 will be placed near here during the printing process

4.2 In-situ Measurement of Failure Depth of the Floor

4.2.1 Equipment of Measurement

It is an effective observation method to measure the failure depth of the floor by using double-end water sealing and injection device to conduct the water pressure tests, as shown in Fig. 15. Observing boreholes need to be drilled in the roadway at different azimuth and inclination angles downward (descending), and then differential water injection (discharge) tests can be performed. According to the volumes of water injection (discharge) in the borehole tests, the failure depth of the floor stratum can be determined.

Figure 15 will be placed near here during the printing process

4.2.2 Observation Implementation

In order to observe the failure depth of the floor of the south side of the working face, three groups of boreholes were drilled in the roadway of working face No. 29205 (Fig. 16), and each group of boreholes consisted of 2 boreholes. The boreholes D01 and D02 were drilled at the azimuth angle of 90° to observe the failure depth of the floor in working face No. 29205. The boreholes D03 and D04 were drilled to observe the development of virgin fractures in working face No. 29205 oriented with the azimuth angle of 270°. The boreholes D05 and D06 were drilled to observe the failure of floor in working face No. 29205 with the azimuth angle of 200°, as shown in Fig. 16. The water pressure tests were carried out by using the double-end water sealing and injection device, and the degree of the fracture development in the floor rock mass was deduced from the tests for controlling the maximum failure depth of floor. In Fig. 17, the flow rate represents the volume of water injection per minute (equivalent to water leak off into the rock), and the blue line is the maximum failure depth of the floor at the corresponding location (vertical and inclined depths of each borehole).

Figure 16 will be placed near here during the printing process

4.2.3 Results and Analysis of the Measurements

All data observed in the field are collated, as shown in Fig. 16, and the following observation results are obtained:

(1) Boreholes D01 and D02: These two boreholes were drilled in the floor under a mined area. In order to ensure the accuracy of the test results, the injection water pressure in the borehole should be kept constant at 0.5 MPa. During the water injection test, the pressure was kept stable, and water was injected continuously. There was no water flowed out from the rock into the boreholes, and the sealing of the equipment was good. It is inferred that the injected water flowed along the mining-induced fracture zone of the floor to the natural fracture zone. This indicates that the floor was in a complete failure zone caused by mining and the mining-induced fractures were well developed. At the measured depth (borehole length) from 1m to 20m in Borehole D01 the leakage (leak off) of borehole to the rock was above 3L/min on average (Fig. 17a). This implies that the floor at this interval was in mining-induced fracture zone. Continuous leak off occurred in the section of 11 to 19m (measured depth) of the borehole. When the measured depth reaches 36 m (the vertical depth of 18.5 m), with water injection pressure 0.32 MPa the water injection rate (leak off rate to the rock) was 3.1 L/min. This indicates that the

borehole at this depth was still in the mining failure zone of the floor; however, from this depth on, the leak off rate dropped obviously, although there were some leakage at some locations, the floor was intact. At the measured depth (borehole length) from 1m to 11m in Borehole D02, the flow meter in the injection showed an ascending trend and reached more than 2 L/min on average (Fig. 17b). This implies that the rock in this section was obviously in the mining failure zone and the mining-induced fractures were well developed. At the borehole measured depth of 12-13m, the injection pressure increased obviously, the water injection into the rock was difficult and the reading of flow meter was close to 0. This implies that the rock in this section was intact. Passing this depth interval, the water injection rate increased obviously, indicating that the rock was still in the floor failure zone. At the borehole measured depth of 24 m (vertical depth of 15.36m), water injection rate decreased obviously, implying that the failure depth of the floor in this borehole was about 15.36m. It can be concluded from the water injection measurement in the two boreholes that the maximum failure depth of the floor was 18.5 m after the coal seam roof was fully caved and goaf was compacted for a long time.

(2) Boreholes D03 and D04: These two boreholes were used to observe the state of floor rock mass before coal extraction, which is used for comparison. In the same way to Boreholes D01 and D02, the injection pressure was kept constant at 0.5 MPa during the whole test. Each group was tested 10 times at each interval of 1 m with the water injection rate measured for 30 s in each test, and the borehole leak off rate at each interval was obtained by averaging the measured data of ten times. The measured data in Borehole D03 had obvious water leakage at the borehole measured depth (borehole length) from 1 to 19 m, and the maximum value was about 4.2 L/min (Fig. 17c). According to the water leakage and the analysis of the rock lithology, the water leakage might be caused by the borehole drilling or natural fractures, and it might be also possible that the floor was affected by mining of the adjacent working faces to generate fractures at a shallow depth. With the increase of borehole depth, the leakage was obviously decreasing, and many sections had no water losses, indicating that the floor rock did not develop fractures and the rock was relatively intact. The continuous leakage length of the borehole D04 was 11 m, and the maximum leakage (3.2 L/min) occurred at the borehole measured depth of 7 m (Fig. 17d). The average leakage at the shallow depth of the floor rock was 1-3 L/min, and the borehole had basically no leakage at the measured depth of 14-15 m, implying that the rock mass in this section had not been damaged and was intact. When the test was carried out between 16-18 m (borehole length), the water leakage of the borehole suddenly increased, and the leakage of this section was large, indicating mining-induced failure of the floor rock or the existence of primary natural fractures. From the borehole length of 20m to the bottom of the hole, the leakage in the borehole obviously reduced, and most sections had no water leakage, indicating that the lithology of the floor was intact and no fractures. Based on the observation and analysis of these two boreholes, the maximum failure vertical depth of floor was 11.5 m in the un-mined area.

(3) Boreholes D05 and D06: These two boreholes were used to observe the failure depth of the floor near the working face. The boreholes were affected by mining, so that they were easy to collapse, which brought great challenge to the measurement. During the injection tests, in order to prevent large-area collapse of the boreholes and avoid affecting the production of the working face, continuous tests and observation were conducted and ideal effects were achieved. Test results showed that the failure of floor in Borehole D05 was serious near the working face. In the shallow part of the borehole from 1 to 20 m of the measured depth, the water leakage to the floor rock was above 1.5 L/min, indicating that the floor was damaged sufficiently (Fig. 17e). The leakage of the borehole fluctuated greatly at the measured depth between 20 and 50 m in the middle of the borehole, and the leakage of the borehole at 30 m was very small. The leakage of the borehole reached its maximum value of 4.6 L/min at 38 m (borehole length), indicating that the mining-induced fracture distribution was not uniform in the section of 20-50 m, but in general, the rock in some depth was failed obviously. In the measured depth of 55 - 60m, the borehole leakage remained at a high level above 3 L/min, indicating that the floor was fully damaged by mining. When the borehole length just entered to 68m (vertical depth of 20.7 m), the leakage volume suddenly decreased, and it is presumed that no failure had taken place after this depth. Therefore, the maximum mining-induced failure depth was 20.7 m in this borehole.

(4) The length (measured depth) of Borehole D06 was 58m, the angle of dip was 23° , and the maximum vertical depth was 22.62 m. The observation of the borehole D06 showed that, under the influence of mining, all mining-induced fractures in the shallow part of the floor rock mass were basically connected, and the length of the continuous water leakage section was 17m (Fig. 17f). The amount of water leakage in the 20 - 42m (borehole length) section of the middle part of the borehole remained relatively high, indicating that the failure of this section was very serious. At the measured depth of 50m in the borehole, there was a sudden change from the water leakage peak. This was the maximum failure depth of floor (vertical depth of 19.1 m). During the whole test process, it was found that there was less water leakage in some sections of the floor, indicating that there were several layers of mudstone in the floor with good water-resisting property and good water-resisting effect on water inrush from the floor. Based on the measured results in Boreholes D05 and D06, the maximum failure depth of the floor near the working face was 20.7 m.

Figure 17 will be placed near here during the printing process

To sum up, by observing the leakage of 6 boreholes in the floor, the maximum vertical depth of failure zone of the floor is determined as 20.7 m with high credibility, as shown in Fig. 17e. Fig. 18 shows the measured maximum mining-induced failure area of the floor in the mined area based on the borehole measurements in Fig. 17. It can be seen that the floor rock is damaged by mining, and the maximum failure depth is 20.7 m. Considering the following parameter with safety factors: the water pressure of the Ordovician limestone aquifer is taken as 2.8 MPa, the thickness of the effective floor aquiclude (without geologic structures) is defined as 25 m. The water inrush coefficient can then be calculated as 0.11 MPa/m. This water inrush coefficient exceeds the safety regulation of China's Coal Mine Safety Regulation. Therefore, it is required to carry out the regional control measures prior to mining to ensure safe production of the working face and to prevent water inrushes from the Ordovician limestone aquifer in the floor. The water inrushes are great threat to mine safe production.

Figure 18 will be placed near here during the printing process

By comparison with the measured data of the failure depths of the floor rocks in the near-horizontal coal seam in other mining areas, it can be concluded that under the high stress of the hard roof, the mining-induced failure depth of the floor is obviously larger (Dong et al. 2019). Because the coal seam is very close to the confined Ordovician limestone aquifer under the floor, prevention and control of water inrushes from the floor face with severe challenges. In order to implement safe mining, both hydraulic fracturing the hard roof to release roof stress and advanced regional treatment and reinforcement of the floor can be applied to ensure safe and efficient mining of coal seam (Huang et al. 2017; Zhao 2014).

5 Conclusions

(1) Considering the combined action of the abutment pressure of the coal wall and the water pressure from the confined aquifer under the floor, an elastic mechanical model is established along the strike of the working face during the first collapse of the roof. An analytical solution of the stress distribution of the floor is derived, and the distributions of the vertical, horizontal and shear stress coefficients are calculated. It can be concluded that the extent of vertical stress concentration on the coal wall side of the working face is larger than that of the horizontal stress. In the strike direction of the working face, the maximum stress concentration of the floor appears at the elastic-plastic deformation junction. With the depth increasing, the horizontal stress coefficient isoline tends to relax; the corresponding shear stress coefficient isoline shows a "symmetric spiral" distribution, and it propagates downward at a certain angle with the normal line of the floor, causing the compression-shear or tension-shear failure of the floor.

(2) Theoretical analysis and numerical simulation show that under the combined action of high stress of the hard roof and water pressure, the peak point of vertical stress in the shallow part of floor is 21.7% higher than

that under normal roof condition. The high roof stress causes a deeper failure depth in the floor. the floor under the goaf is broken mainly by the action of tension and shear, and the maximum depth of failure is about 24.6 m and the maximum floor heaving deformation is 650 mm. The most obvious range of the depth in the floor affected by mining is between 0-25 m for the normal roof conditions, between 0-35m for the hard roof condition, and 0-40m for the hard roof with the Ordovician limestone water pressure. At the shallow depth of the floor less than 20 m, in the condition of normal roof, the vertical stress of the floor is released rapidly within 0-20 m in the goaf behind the coal wall of the working face, and the minimum stress is close to 0; at 60 m in the goaf behind the coal wall of the working face the stress can be restored to 80% of the virgin rock stress. Under the condition of the hard roof, the stress releases within 0-10 m in the goaf behind the coal wall of the working face, but the lowest stress is greater than 0. The stress at 60 m away from the coal wall can only be restored to 60% of the virgin rock stress.

(3) Based on the measurements of the floor leakage at different locations in the mining area, it is concluded that after the goaf is fully caved and re-compacted, the failure depth of floor decreases and the thickness of effective aquiclude increases. In this case, the water inrush risk is low compared to the case that the working face is near the coal wall. For the floor under the un-mined area, there are different degrees of failures due to natural fractures in the floor or the mining influence from adjacent working face, but the fractures are concentrated at the shallow depth. At a deeper depth fractures are not developed, and the floor rock is relatively intact. In the mined area the maximum failure depth of the floor is 20.7 m and occurs in the floor under the coal wall of the working face. The floor in this area is subject to the combined action of mining disturbance, coal wall abutment pressure and water pressure from the confined aquifer, which makes the floor is prone to generate shear-tensile failure, lead to mutual connected fractures. This would be a very dangerous case and might cause water inrush incident if the fractures connect to the confined aquifer of the Ordovician limestone.

(4) Through observing the advanced abutment stress of the coal wall and comparing to the change of the advanced abutment pressure under the normal roof condition, it is found that under the stress of the hard roof, the influence area of the abutment pressure in front of the coal wall is obviously enlarged, the influence zone of the abutment pressure can be greater than 110 m. The advanced abutment pressure shows an increasing trend with “slow increase first, then rapid increase”. The advanced abutment stress drops sharply at the distance of 4-6m from the coal wall with a small plastic zone. The abutment pressure concentration coefficient is between 1.4 and 1.8.

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Figure captions

Fig. 1 Comprehensive geologic columnar diagram in the studied area.

Fig. 2 The stress state of the floor rock mass along the working face with consideration of the water pressure

Fig. 3 Mechanical calculation model of the floor along the strike of the working face and above the confined aquifer.

Fig. 4 Contour plot of the vertical stress coefficient ($\sigma_x / (\gamma H)$) in the floor rock mass along the strike of the working face above the confined aquifer with water pressure.

Fig. 5 Contour plot of the horizontal stress coefficient ($\sigma_y / (\gamma H)$) in the floor rock mass along the strike of the working face above the confined aquifer with water pressure.

Fig. 6 Contour plot of the shear stress coefficient ($\tau_{xy} / (\gamma H)$) in the floor rock mass along the strike of the working face above the confined aquifer with water pressure.

Fig. 7 Displacement distributions in the floor at different depths under normal roof condition

Fig. 8 Displacement distributions in the floor at different depths under hard roof condition

Fig. 9 Displacement distributions in the floor at different depths under hard roof condition and confined water pressure.

Fig. 10 The vertical stress distributions along the strike direction in the three models

Fig. 11 Vertical stress changing curves of different depths of the floor

Fig. 12 Layout of four boreholes for stress measurement in the working face floor

Fig. 13 Field measurement of the advanced abutment stress at different depths of the floor along the strike of working face for mining above the confined aquifer

Fig. 14 Schematic diagram of stress distribution of the advanced abutment pressure from the borehole measurements.

Fig. 15 Double-end water sealing and injection device

Fig. 16 Borehole construction layout of the water injection tests

Fig. 17 Field measurement results of borehole leakage along the strike of working face above confined aquifer

Fig. 18 Borehole-measured results of mining-induced failure areas in the floor under the mined area

Table 1. Physical and mechanical parameters of the immediate roof of coal seam in the numerical simulation

Figures

Columnar Structure	Name of Rock Stratum (Coal Seam)	Thickness /m	Description of Rock Stratum Characteristics
	Mudstone	5.18	Black, fine, dense, and containing plant fossils and nodules
	Anthraconite	4.87	Gray, massive, brittle, including animal fossils and calcite veins, cracks developed, local mud
	Dioritic Porphyrite	15.27	Gray, massive structure, hard, crack development, containing calcite veins and massive pyrite, feldspar amphibole-based, pay minerals rare.
	Coal	2.98	Black-gray, mostly powder-like structure, showing a glass luster, with tonstein
	Mudstone	4.67	Dark-gray, smooth, fine, under the influence of force, joint development, the upper part of the extrusion into a sticky mud
	Benxi Limestone	7.95	Gray, massive structure, containing animal fossils and lumps of pyrite dense, brittle, calcite vein longitudinal
	Mudstone	21.23	Gray, purple-gray alternate, local variegated, horizontal bedding, in the broken, the lower part, fine
	Ordovician Limestone	14.20	Grayish, breccia-like, dense, brittle, partly dolomitic, with lumps of pyrite, crosscut with calcite

Figure 1

See the Supplemental Files section for the complete figure caption.

(a) Diagrammatic sketch

(b) Diagrammatic sketch

Figure 2

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Figure 3

See the Supplemental Files section for the complete figure caption.

Figure 4

See the Supplemental Files section for the complete figure caption.

Figure 5

See the Supplemental Files section for the complete figure caption.

Figure 6

See the Supplemental Files section for the complete figure caption.

(a) Depth of the floor 1m

(c) Depth of the floor 20m

(b) Depth of the floor 10m

(d) Depth of the floor 30m

Figure 7

See the Supplemental Files section for the complete figure caption.

(a) Depth of the floor 1m

(c) Depth of the floor 20m

(b) Depth of the floor 10m

(d) Depth of the floor 30m

Figure 8

See the Supplemental Files section for the complete figure caption.

Figure 9

See the Supplemental Files section for the complete figure caption.

(a) Under normal roof condition

(c) Under hard roof condition and confined water pressure

(b) Under hard roof condition

Figure 10

See the Supplemental Files section for the complete figure caption.

(a) Under normal roof condition

(b) Under hard roof condition

(c) Under hard roof condition and confined water pressure

Figure 11

See the Supplemental Files section for the complete figure caption.

Figure 12

See the Supplemental Files section for the complete figure caption.

Figure 13

See the Supplemental Files section for the complete figure caption.

Figure 14

See the Supplemental Files section for the complete figure caption.

Figure 15

See the Supplemental Files section for the complete figure caption.

(a) Borehole layout plan

(b) The vertical cross section of the borehole layout

Figure 16

See the Supplemental Files section for the complete figure caption.

(a) Borehole D01

(c) Borehole D03

(e) Borehole D05

(b) Borehole D02

(d) Borehole D04

(f) Borehole D06

Figure 17

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Figure 18

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