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Technical Article

Study of Mine Water Inrush from Floor Strata through Faults

Longqing Shi[1] and R. N. Singh[2]

[1]Principal Visiting Fellow, Univ of Wollongong, from Shandong Univ of Science and Technology, China; [2]Professor of Mining Eng, Univ of Wollongong, Northfield Avenue, Wollongong, N. S. W. 2522, Australia; e-mail: raghu@uow.edu.au

Abstract. The mechanism of mine water inrushes in coal mines in China differs considerably from that in other countries. In China, most water inrushes occur from floor strata, the source of water inrush being karstic limestone aquifers. This paper describes the mechanism of mine water inrushes through a fault in the mine floor using principles of strata mechanics, and the path of water inrush from an aquifer to the working face. A criterion to judge whether the ground water inrush will occur through a fault or not is also described, together with a case history of water inflow in Feicheng Coalfield, China.

Key words: fault; Feicheng, China; inrush; mine water, strata mechanics

Introduction

There are very few cases of coal mine water inrushes from the floor in major mining countries such as the United States, Russia, Poland, Canada, Australia, Germany, India and the United Kingdom (Singh 1986). In these countries, the incidence of mine water inflow can generally be attributed to a sudden inflow of water of surface water, development workings accidentally contacting surface unconsolidated deposits, strata water entering working faces, a shaft encountering a confined aquifer during the sinking operation, clearing old shafts, contacting abandoned mine workings, failure of underground dams and seals, or leakage of water from an aquifer through a bore hole (Vutukuri and Singh 1995). The main method for the control of mine water inflow in these countries is the application of various mine dewatering techniques.

In contrast, more than 90% of water inrushes in Chinese coal mines are due to water inflow from karst aquifers through the coal seam floor. The main techniques for controlling water inrush through the floor are to leave a sufficient thickness of barrier against the aquifer and/or grouting of the floor (Shi 1999). Coal mine inundations in China can be divided into four categories, according to the source of the mine water inrush. Inundations caused by limestone aquifers account for 92.3% of the cases; flooding caused by surface water contributes to 4.9% of the cases; flooding caused by alluvial water accounts for 1.4% of the cases; and inflows caused by sandstone aquifers account for 1.4% of the cases. The principal limestone aquifer is Ordovician and is distributed extensively in Northern China. About 75% of the coal production in China comes from that region, where about 50% of the coal reserves are mined from coal seams of the Permian and Carboniferous Systems. These strata lie unconformably on the Ordovician limestone, which is about 800m thick; there is no Silurian and Devonian strata in Northern China. After nearly a century of mining, most Chinese mines have been extracting deeper coal seams, especially those coal seams located in the Permian and Carboniferous Systems. Because the distance between the current mining horizons and Ordovician limestone is only 20-40m, geological structures like faults play an important role in controlling the inflow of karst water into the mine workings.

Distribution of strata pressure on a coal seam as a consequence of mining

There are two patterns of vertical strata pressure distribution on a coal seam corresponding to different mining depths and coal seam strengths. One of the patterns is the single elastic distribution shown in Figure 1a (Song 1979). The main characteristics of this stress distribution are that peak vertical strata pressure is centralised on the fringe of the coal face. The pressure decreases in accordance with the negative exponential law towards the solid coal; the coal seam is in a state of elastic compression and the stress on the coal seam is proportional to the elastic deformation. Because the vertical strata pressure does not exceed the compressive strength of the coal seam, the strata pressure does not break the coal seam and associated floor strata. Therefore, the floor strata remain intact and water-resistant.

The second pattern of vertical strata stress distribution on a coal seam is based on both elastic and plastic stress distribution and the formation of a yield zone in the coal seam ahead of the face, as shown in Figure 1b (Wilson 1972).

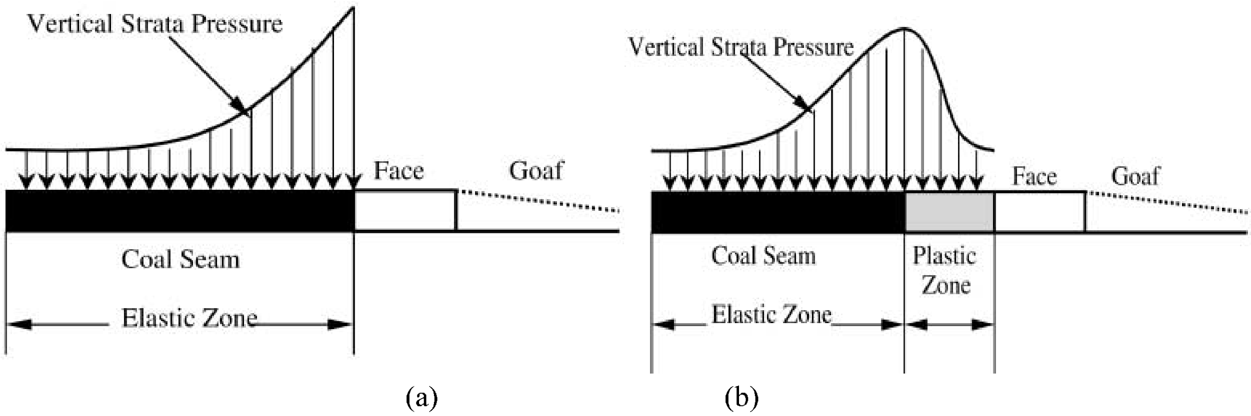


Figure 1. Distribution of vertical strata pressure on coal seam

In the elastic zone, the coal seam undergoes elastic deformation and the peak vertical strata pressure occurs on the interface of the elastic and plastic zones. The stress gradually decreases towards the solid coal until it attains the original virgin state of vertical stress. The stress in this zone is also proportional to the coal seam's elastic deformation. Therefore, the floor beneath this stressed zone does not undergo permanent deformation. In the plastic zone, in contrast, the coal seam suffers permanent deformation due to strain softening and/or plastic flow. In the absence of high horizontal stress, the coal seam would deform continuously. However, the horizontal stress increases towards the solid coal in advance of the working face and the force to prevent the plastic deformation also increases. Thus, the plastic deformation in the coal seam is limited. The stress distribution due to vertical pressure in the plastic zone is similar to that of the horizontal stress, namely the pressure presents a gradually rising curve from the coal face toward the interface of the plastic and elastic zones. Obviously, only the floor strata corresponding to the plastic zone has any possibility of being broken by the vertical strata pressure, creating a pathway of water inrush from the floor into the mine workings. The width of the plastic zone as shown in Figure 2 can be calculated in terms of the ideal elasto-plastic strain-softening model of the coal seam (Jiang, 1993). The deformation of the coal seam can be divided into three stages: elastic deformation, a plastic strain-softening stage and the flow stage. The degree of strain softening can be quantified by the softening angle θ_0 or softening modulus M_0, where M_0= Tan θ_0. For an ideal elasto-plastic body, θ_0=0°, M_0=0, and for an ideal elastic-brittle body, θ_0=90°, $M_0 \rightarrow \infty$. During the stage of elastic deformation when the deformation approaches the yield point of the coal seam, the Mohr-Coulomb criterion should be satisfied (Wilson 1980) as follows:

$$\sigma_1 = \sigma_c + k_p \sigma_3 \qquad (1)$$

Where,

$$k_p = \frac{1+\sin\varphi}{1-\sin\varphi}$$

σ_1 = ultimate strength of coal seam; σ_3 = minimum principal stress; φ = internal friction angle of the coal seam obtained from laboratory tests; and σ_c = uniaxial compressive strength of coal as obtained from the laboratory tests.

A large number of triaxial tests in the laboratory have shown that during the strain softening stage, the cohesive strength of a coal seam decreases more rapidly than the internal angle of friction of the coal. Thus, the main contributory factor to the deformation of coal is the loss of cohesive strength of the coal as expressed by the following relationship: $\sigma_1 = f(\sigma_3, \varepsilon_1^p)$. Thus, σ_1 can be expressed as

$$\sigma_1 = \overline{\sigma_c}(\varepsilon_1^p) + k_p \sigma_3 \qquad (2)$$

where, ε_1^p = plastic strain value, and

$$d\overline{\sigma}_c = -M_0 d\varepsilon_1^p . \qquad (3)$$

During the plastic stage of deformation, the coal seam strength decreases to the residual strength, denoted by:

$$\sigma_1 = \sigma_c^* + k_p \sigma_3 \quad (4)$$

where σ_c^* is the residual strength of the coal seam in uniaxial compression.

The state of stress in a coal seam ahead of a longwall face can be represented in Figure 3 (Jiang and Han 1998) by considering the coal seam as a homogeneous continuous body, in plane strain condition and ignoring the effect of gravity. A section of the coal seam at a distance x ahead of a coal face is considered to be a plastic zone of width dx and height M. It is subject to the vertical stress σ_y and the horizontal stress σ_x. The sliding resistance force (T) on the interface between coal seam and roof or floor is given by:

$$T = \sigma_y Tan\varphi_1 = \sigma_y f_1 \quad (5)$$

where φ_1=friction angle between coal seam and roof or floor; and f_1=coefficient of friction between coal seam and roof or floor.

The stress on unit element in the y-direction is in a state of equilibrium while the stress on unit element in the x-direction is in limiting equilibrium as expressed by the following equation:

$$\begin{aligned} &M\sigma_x + 2\sigma_y Tan\varphi_1 dx - M(\sigma_x + d\sigma_x) = 0 \\ &Md\sigma_x = 2\sigma_y Tan\varphi_1 dx \end{aligned} \quad (6)$$

Based on the idealized strain softening elasto-plastic model, the strength in the plastic broken zone can be described in terms of equation (4). It may be noted that in equation (6), σ_x, σ_y are not principal stresses because there is a force of friction acting on the interface of the coal seam and the adjoining rock. Similarly, even when the strength test of coal is performed in the laboratory, there is also a friction force on the end plane, and σ_c, φ are actually obtained in the presence of friction. Therefore, the *in-situ* stress state of the unit element is basically the same as that obtained in the laboratory, and equation (4) can be substituted by following equation:

$$\sigma_y = \sigma_c + k_p \sigma_x \quad (7)$$

Applying boundary conditions and solving equation (6) and equation (7) the following solution is obtained:

$$\begin{aligned} \sigma_x &= \frac{\sigma_c^*}{k_p}(\ell^{D_1 x} - 1) \\ \sigma_y &= \sigma_c^* \ell^{D_1 x} \\ D_1 &= \frac{2k_p Tan\varphi_1}{M} \end{aligned} \quad (8)$$

Similarly, in the plastic softening zone, equation (3) and equation (4) can be approximated as:

$$\sigma_y = \sigma_c + k_p \sigma_x - M_0 \varepsilon_1^p \quad (9)$$

Applying the continuity conditions to the interface of the plastic strain-softening zone and the plastic yield zones and solving equation (6) and equation (9), the following solutions are obtained:

$$\begin{aligned} \sigma_x &= \frac{1}{k_p}\sigma_c^*(\ell^{D_1 x} - 1) \\ &+ \frac{D_2}{k_p}\left[\ell^{D_1(x-x_f)} - D_1(x-x_f)^{-1}\right] \\ \sigma_y &= \sigma_c^* \ell^{D_1 x} + D_2\left[\ell^{D_1(x-x_f)} - 1\right] \\ D_2 &= \frac{M_0 \varepsilon_b}{2k_p Tan\varphi_1} \end{aligned} \quad (10)$$

where ε_b= strain gradient in plastic zone; and x_f= the width of the yield (plastic broken) zone;

At the interface of the elastic and plastic zones, $x = x_p, \sigma_y = K\gamma H$, then according to Jiang (1993):

$$x_p = \frac{1}{D_2}\ln\left\{\frac{1}{\sigma_c^*}\left[K\gamma H - D_2(\ell^{(\sigma_c - \sigma_c^*)/D_2} - 1)\right]\right\} \quad (11)$$

where K= coefficient of stress centralisation at the interface of elastic and plastic zones; γ = volume weight of the overlying rock, and H = mining depth.

Distribution of Ground Pressure on Floor

The stress distribution characteristics on the floor of an excavation can be simulated by finite element analysis or obtained by back analysis using measured displacement as an input parameter. Figures 4a, 4b and 4c show the results of the finite element analysis (Song 1989).

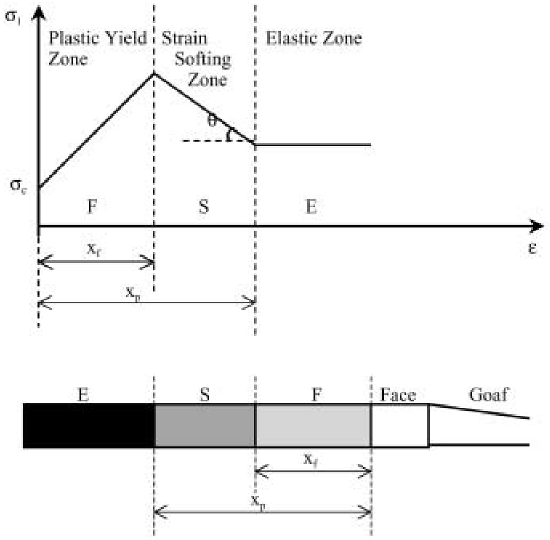


Figure 2. Ideal softening model of elastic-plastic strain and deformation areas of coal seam; F= plastic yield zone; S= strain softening zone; E= elastic zone

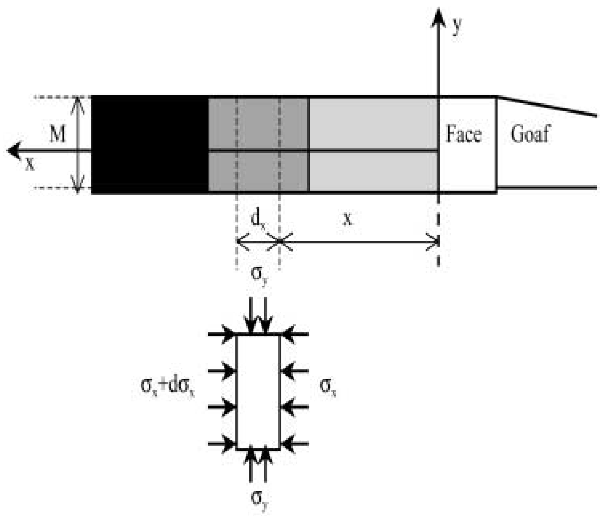
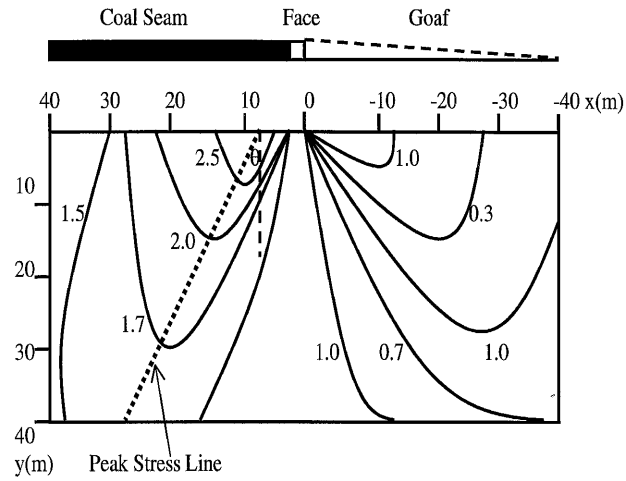


Figure 3. Stress state in the plastic zone

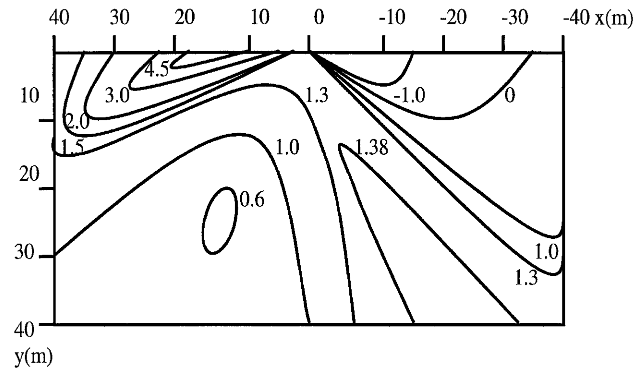
Figure 4a shows a typical example of the stress distribution on the floor of a coal seam in the immediate vicinity of a longwall face. The main observations are as follows:

The vertical strata pressure acting on the coal seam controls the vertical stress acting on the floor of the coal seam in the vicinity of a longwall face. Thus, the vertical stress distribution on the floor is relatively high in the area of the abutment zone while the floor stresses are relatively low in the destressed area underneath the goaf.

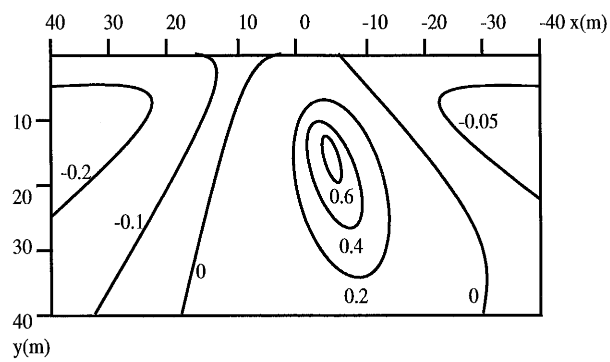
It can also be seen that the magnitude of the vertical stress on the floor is not proportional to the vertical strata pressure on the coal seam. Therefore, the contour of the vertical stress on the floor is not symmetrical with the vertical strata pressure on the coal seam. The vertical stress on the floor decreases with the floor depth following the negative exponential law.



(a) Distribution of vertical stress on the floor $(\sigma_r / \gamma H)$



(b) Distribution of horizontal stress on the floor $(\sigma_x / \lambda\gamma H, \lambda = 1/3)$



(c) Distribution of share stress on the floor $(\tau_{xy} / {}^{\gamma}H)$

Figure 4. Distribution of components of stress on the floor in the vicinity of a longwall face (after Song 1989)

The peak stress line, which is constructed by joining the peak flexure points of contours of vertical stress on the floor, is inclined in the direction of the working face, at an angle (θ) between 20° and 25° to the vertical line.

The distribution of the horizontal stress on the floor appears to increase or decrease due to the influence of the vertical stress concentration or destressing of the coal seam and the floor due to mining. Below the abutment zone, the horizontal stress is concentrated in a shallow zone on the floor and is relatively de-stressed in the deeper zone. However in the goaf area, the horizontal stress on the floor is relatively lower in the shallow area in comparison to the deep area (Figure 4b). The distribution of shear stress on the floor shows an area of high shear stress concentration as a consequence of mining (Figure 4c).

Figures 5 to 7 are practical examples showing the calculated stress obtained from the back analysis method for the floor of No. 2701 longwall face in Fengfeng Colliery, China (Gao and Shi 1999). The examples show that the stress distribution feature on the floor obtained by the back analysis is the same as that obtained by the finite element analysis.

Conditions for water inflow from the floor through a fault

Figure 8 shows the mechanism for water inrush from the floor through a fault (Shi 1999). The distance between the coalface and the fault surface is w, which is also the thickness of the barrier against water. The dip angle of the peak stress line on the floor is θ and the dip angle of a fault is α. Point A is the intersection of the fault surface with the peak stress line, z is the vertical distance from the bottom of the coal seam to a point A and x_e is the extent of the coal seam in the elastic zone. It may also be noted that x_p is the extent of the coal seam in the plastic zone. The thickness of broken strata on the floor is represented by h_l or h_2. CD is the horizontal projection corresponding to h_l, EF is that corresponding to h_2, and AB is that corresponding to point A.

State of floor strata

Considering the state of the floor strata, the peak stress line is taken as a boundary line; all the floor strata on the left side of this line is unbroken by the strata pressure and has retained its original water resistance. The ground pressure may break the floor strata on the right side of the peak stress line. Thus, only the floor stratum that is below the plastic zone of

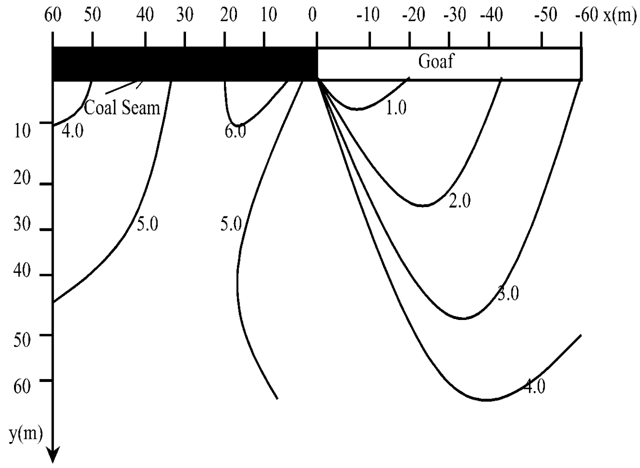


Figure 5. Vertical stress contours on the floor of No.2701 longwall face at Fengfeng colliery, China (after Gao and Shi 1999)

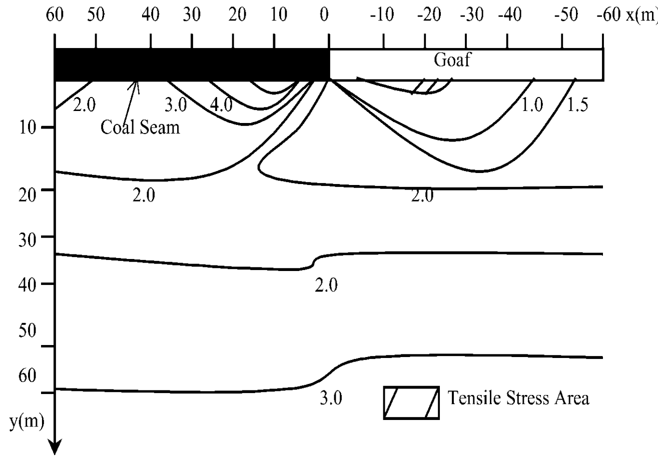


Figure 6. Horizontal stress contours on the floor of No. 2701 longwall face at Fengfeng Colliery, China (Gao and Shi 1999)

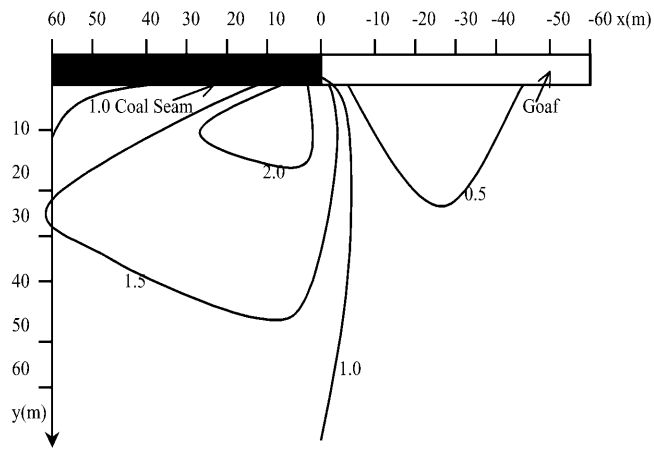


Figure 7. Shear stress contour on the floor of longwall face at Fengfeng Colliery, China (after Gao and Shi 1999)

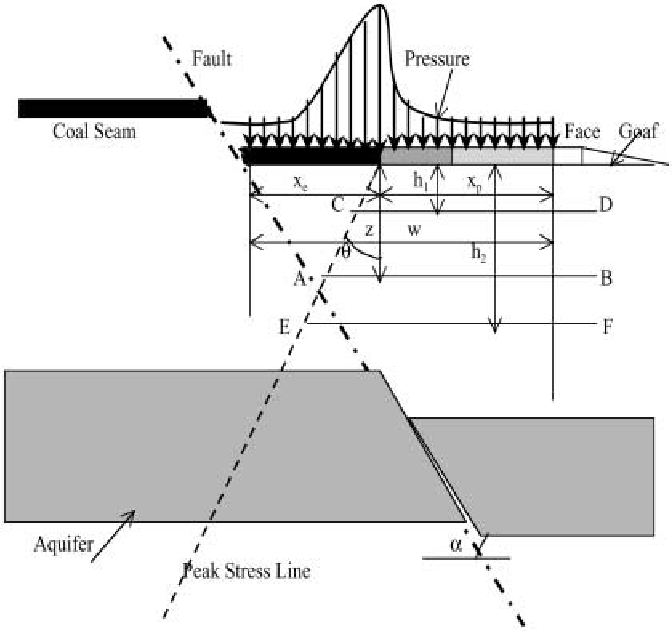


Figure 8. Mechanism of water inrush from the floor through a fault

the coal seam and behind the peak stress line has a possibility of being fractured by the ground pressure, provided that the compressive stress on the floor strata exceeds the compressive strength. It may be noted that the weaker the floor strata, the larger the thickness of barrier required to prevent water inflow.

Pathway of water inrush from the floor

The pathway of water inrush from the aquifer to the working face should be either from the aquifer through a fault and/or through the broken strata on the floor into the working face. The water will follow this path since there is no broken stratum on the floor on the left side of the peak stress line, and the water in the floor aquifer cannot flow through the strata on the left side. On the right side of the peak stress line, if $h=h_1$, the water inrush also cannot occur because there are intact strata between CD and AB and the intact strata intercepts the pathway between the fault and the broken strata in the floor. But if $h=h_2$, water inrush must occur since the pathway for water inflow has a continuity. Therefore,the condition for mine water inrush through the floor from a fault is met when the thickness (h) of broken strata in the floor is not less than the thickness (z) of the intersection point between the peak stress line and the fault surface; that is

$$h \geq z \quad (12)$$

Calculation of distance Z

It can be seen that the area (S) of the triangle formed by point A and x_e can be determined as follows:

$$S = \frac{1}{2} x_e z = \frac{x_e^2 \sin\alpha \sin(90-\theta)}{2\sin(\alpha + 90 - \theta)}$$

Since $x_e = w - x_p$, then

$$\begin{aligned} z &= \frac{x_e \sin\alpha \cos\theta}{\cos(\alpha-\theta)} \\ &= \frac{\sin\alpha\cos\theta}{\cos(\alpha-\theta)}(w - x_p) \end{aligned} \quad (13)$$

Equation (13) describes the situation where the fault's dip direction is against that of the peak stress line, opposite to the direction of mining advance. When the fault's dip direction is the same as that of mining advance, the equation is given below:

$$\begin{aligned} z &= -\frac{x_e \sin\alpha \cos\theta}{\cos(\alpha+\theta)} \\ &= -\frac{\sin\alpha\cos\theta}{\cos(\alpha+\theta)}(w - x_p) \end{aligned} \quad (14)$$

Let

$$A = \frac{\sin\alpha\cos\theta}{\cos(\alpha-\theta)}$$

$$B = Ax_p$$

then equation (13) can be rewritten as follows:

$$z = Aw - B \quad (15)$$

From equation (15), it can be seen that the distance z is proportional to w as shown in Figure 9. Therefore, the thickness of the barrier against water could be increased in order to increase the extent of the intersection between the fault surface and the peak stress line. The water-prevention barrier can be designed correctly in terms of the above equations. By substituting z = h in equation (13), we obtain $w = w_0$, and then the thickness of the safety barrier against

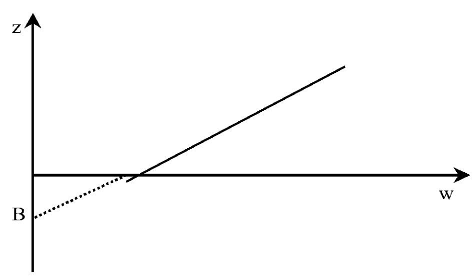


Figure 9. Relationship between z and w

the water should be greater than w_0 in order to prevent water inrush from the floor through a fault.

Case history: water inrush through a fault in the floor of the Daifeng Coal Mine

The Daifeng colliery is situated in Shandong Province, China. In this mine, three steeply dipping coal seams, namely coal seams 8, 9 and 10, are being mined by the longwall method of mining. The coal seams are in the Taiyuan group, which belongs to the Upper Carboniferous System. The main underlying aquifer is an Ordovician Limestone formation. The distance from the bottom of number 9 coal seam to the top of the Ordovician limestone is 45m.

In June 1970, Face No. 9204 in the No. 9 Coal Seam was being mined by the longwall mining method. The face length was 215m, the extracted seam height was 1.45m, the dip angle of the coal seam was 10°, and mining depth was 100m. Water inrush took place on 29th June 1970 when the longwall face had advanced 93m from the starting line of the face. The maximum rate of inflow of water was $1628m^3$ per hour, measured by neighbouring hydrologic observation boreholes. As a consequence, the face was completely flooded. The main cause of this inundation was later found to be the presence of a normal fault (designated as fault number F206) in advance of the face line. The dip direction of the fault was against that of the face advance and the dip angle (α) of the fault was 65°. Although the existence of the fault was known in advance, the technical personnel did not pay attention to it, because the throw of the fault was only 3.2m and it was thought to be 'a minor fault' that could not lead to a water inrush.

Actually, the fault turned out to be a major conduit of water irrespective of the fact that the fault had a small throw. When the extent of the face advance reached 93m, the distance from the coal face to the fault plane was only 12m (viz. w=12m). The survey of the site of the inundation revealed that the thickness of broken strata on the floor *in-situ* was 17.8 m (i.e., h=17.8m). The dip angle of the peak stress line was 20°. The ratio of the plastic zone (x_p) of the coal seam to the mining thickness (M) was 2.15, that is:

$\frac{x_p}{M} = 2.15$. Then

$$x_p = 2.15M = 2.15 \times 1.45 = 3.12m$$

Substituting w=12m, x_p=3.12m, θ=20°, α=65° into equation (13), we have:

$$\begin{aligned} z &= \frac{x_e \sin\alpha\cos\theta}{\cos(\alpha-\theta)} \\ &= \frac{\sin\alpha\cos\theta}{\cos(\alpha-\theta)}(w-x_p) \qquad (13) \\ &= \frac{\sin 65^\circ \cos 20^\circ}{\cos(65^\circ - 20^\circ)}(12-3.12) \\ &= 10.7m \end{aligned}$$

Since h =17.8m > z =10.7m. The water inrush should take place in terms of equation (12).

On the other hand, from equation (13), we can obtain the following equation:

$$w = \frac{z\cos(\alpha-\theta)}{\sin\alpha\cos\theta} + x_p \quad (16)$$

Taking z =h =17.8m into equation (16) and calculating w_0= w

$$\begin{aligned} w_0 = w &= \frac{z\cos(\alpha-\theta)}{\sin\alpha\cos\theta} + x_p \\ &= \frac{17.8\cos(65^\circ - 20^\circ)}{\sin 65^\circ \cos 20^\circ} + 3.12 \\ &= 17.9m \end{aligned}$$

Therefore, a barrier of at least 17.9 m should have been retained to prevent water inrush from the floor through the fault. When the extent of the face advance was 93 m, the distance from the coal face to the fault plane was only 12 m. In order to retain 17.9 m thickness of barrier against water, the face should have been stopped at the distance (93+12)-17.9 = 87.1 m from the fault. Thus, when the extent of the face advance reached 87.1 m, the mining activities at the working face No.9204 should have been stopped in order to avoid the incident.

Conclusions

From the above study the following main conclusions are reached:

When the vertical strata pressure exceeds the coal seam compressive strength, a plastic zone known as the Yield zone is formed ahead of the longwall face in a coal seam. The peak stress line of stress distribution on floor strata is inclined at an angle of 20°~25° to the vertical ahead of the working face. Only the floor strata in the rear area of the peak stress line have a possibility of being broken by the ground pressure. The pathway for mine water inrush from the floor is from the aquifer through a fault to the broken strata on the floor, and then into the working face. A mine water inrush can occur from the floor through a fault when the thickness of broken strata on the floor is not less than the extent of the intersection between the peak stress line and the fault surface. If this condition is followed, an adequate thickness of barrier against water inrush can be designed effectively.

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