The interaction between face support and surrounding rock and its rib weakening mechanism in hard coal seam

***Jing Xuan Yang*[1], *Chang You Liu*[1] *and Bin Yu*[2]**

This study was conducted in a closed system comprising the roof, near-wall coal, support, and floor at a coal face in a hard coal seam, aimed to find a way to overcome the difficulty of cutting hard coal. Energy analysis was applied to analyze the interaction between face support and surrounding rock. Based on this, effective weakening of hard coal ribs was achieved by lowering the setting load of the face support to an appropriate level. Expressions for the strain energy stored in the support rock system and the work done by external forces in the system were derived from the deformation characteristics of the system. Moreover, a stability criterion for the support rock system was obtained according to the principle of minimum potential energy. It reveals the negative relationship between the length of plastic and the length of the elastic zone in the near-wall coal. In addition, the relationship between the support force and the length of the plastic zone was obtained, it reveals through properly reducing the setting load of face support can enlarge the plastic zone and the weakening degree of hard coal ribs. The actual production at the coal face No.8706 in a hard coal seam at Xinzhouyao Coal Mine, Datong mining area, showed that a proper reduction in the setting load of the face support led to enhance rib weakening effect, increases in shearer haulage speed and coal output, mitigation of hard roof weighting, and lower support resistance.

Key words: *hard roof; energy analysis; stability criterion; rib weakening; setting load*

Introduction

Hard coal seams are widely distributed in China, especially in some representative hard coal mining areas, such as Yushen and Binchang mining areas in Shaanxi Province, Datong and Jincheng mining areas in Shanxi Province. As the most characteristic one, Datong mining area is famous for its “two-hard (i.e. hard roof and hard coal) mining conditions. At the coal face No.8706 in Xinzhouyao Coal Mine, Datong mining area, for example, the production process is beset by a series of problems, such as the difficulty of cutting the wall, serious wear and tear on shearer picks, slow shearer haulage speed, below-target output, and high supporting intensity along the mining roadway. These problems significantly impair the output and efficiency of the mine. Therefore, research is needed into the interaction between face support and surrounding rock to provide a basis for improving the roof support technique so that hard coal ribs can be easily broken by rock pressure in the face without adding auxiliary measures. Research on this subject is important to achieve efficient coal production under similar conditions.

Many studies have looked at the relationship between face support and surrounding rock. For example, (Qian M.G. et al., 1996) has investigated the coupling between face support and surrounding rock, which were considered as an integrated system, based on an analysis of the load on the face support. The results showed that the support load depended on the overall mechanical properties of the immediate roof, but had no correlation with the preset deformation of the main roof. Verma, A. and Deb, D., 2013 focuses on the evaluation of strata behavior while the coal face is supported by hydraulic-powered supports of different capacities. Wang G.F., 2014 considered face supported the core of mine support systems and used a support rock coupling model to analyze the modes of support rock coupling in terms of strength, rigidity, and stability. Then an appropriate working resistance of face support, the critical support force required to support the coal wall, and a stability control strategy for the support system were determined. Wang G.F. and Pang Y.H., 2015 found that the stiffness of support rock system affected the location of fracture in the main roof, and the magnitude and duration of the force exerted by the main roof on the face support, and that proper supporting intensity can help retard roof subsidence and rib fall. Cao S.G. et al., 1998 has applied the voussoir beam theory to investigate the boundary conditions and the displacement distribution of immediate roof as well as the relationship between support load and roof deflection (i.e. P-Δl curve) using, and found that the conventional P-Δl curve does not apply to the immediate roof in thick seam mining. Zhang K.B. et al., 1999 divided the overlying strata of a coal face into controllable strata and uncontrollable strata and then subdivided the controllable strata into strata needing control (i.e. immediate roof) and program-controllable strata (i.e. main roof). He believed that the controllable strata were the main object of rock pressure control while the abutment pressure in the coal at the face was induced primarily by the movement of uncontrollable strata. Liu C.Y. et al., 1997 analyzed the deformation and failure characteristics of immediate roof in the support rock system and put forward the concept of immediate roof

[1] *Jing Xuan Yang, Chang You Liu*, China University of Mining & Technology, School of Mines, Key Laboratory of Deep Coal Resource Mining, Ministry of Education of China, Xuzhou, 221116, China, jxyangcumt@126.com
[2] *Bin Yu*, Datong Coal Mine Group Company, Datong, 037003, China

stiffness and a method for calculating it. The study demonstrated that the support resistance did not restrict the final attitude of the main roof due to the influence of immediate roof. Some other geomechanics models, such as the detached block theory (Wilson A.H., 1975), the empirical method (Peng S.S., 1987), the load cycle analysis (Trueman R. et al., 2005 and 2009), and various numerical models (Navid Hosseini et al., 2014; Singh G.S.P. and Singh U.K., 2009 and 2010), had also been developed that aim to relate how a longwall support interacts with the surrounding rock. However, these studies analyzed the relationship between support and surrounding rock in specific environments and the results provide important guidance for improving production rate of coal face. However, most of the previous studies have focused on the vertical system consisting of the roof, support, and floor, but neglected the interaction between the lateral coal mass in the near-wall region and face support.

When a constant load is applied to the roof of a coal face, the roof, near-wall coal, support, and floor constitute a closed system, and the face support supports the roof in coordination with surrounding rock during mining. If the setting load is too high, the stress in the face cannot be easily transferred towards the wall. As a result, the plastic zone in the near-wall coal tends to be small, and the slightly deformed coal mass is allowed to maintain structural integrity. Besides, a high setting load can prevent the roof strata from breaking and caving punctually and easily induce sudden roof collapse, a type of rock burst in mines. If the setting load of the face support is low, the stress in the face is allowed to be transferred to the wall, resulting in a relatively large plastic zone in the near-wall coal, and the coal mass tends to break rapidly. However, rib spalling is likely to occur in this case. Moreover, a low setting load may cause the main roof to be activated too early, which will possibly induce a rock pressure high enough to push down the support. Therefore, further research is needed to investigate the interaction between face support and surrounding rock during hard coal mining and determine an appropriate setting load for the purpose of crushing the wall and roof control. It is expected that hard coal will be effectively broken by rock pressure in the face without adding auxiliary measures such as wall blasting and hydraulic fracturing, thus streamlining the production process and raising working efficiency. In this paper, therefore, energy analysis was applied to analyze the coordination between the face support and surrounding rock during hard coal mining face and a mechanical model of the support rock system at a coal face was developed. The study reveals the pattern of support rock interaction and provides a reference for efficient cutting operation and determination of proper setting load of the support.

Occurrence of the studied hard coal seam and mining problems

In Xinzhouyao coal mine, Datong mining area, the coal face No.8706 is situated in the coal seam occurring at a depth of 304～364 m, and it measures 726.8 m long along the strike and 142.5 m long along the dip. The coal seam ranges from 2.9 to 7.7 m in thickness, with an average of 3.6 m, and dips between 1° and 4°, 2° on average. The coal has a Protodyakonov coefficient of 4.8, the uniaxial compressive strength of 26.8 MPa, and an elastic modulus of 12.4 GPa. With a complex structure, the coal seam contains 0.1～0.2 m dirt band in the upper part. Fig. 1 shows the layout of the coal face.

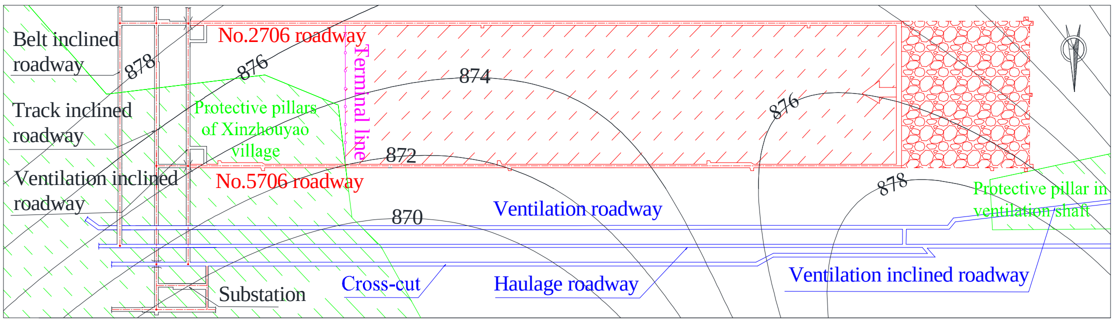

Fig. 1. The layout of the coal face No.8706.

The seam being mined at the coal face No.8706 is immediately overlain by the 0.35m thick false roof of arenaceous shale, which is structurally unstable and gradually falls as the face advances. Above the false roof is 1.38m thick immediate roof composed of interbedded medium-grained sandstone and sandy mudstone. This immediate roof has a high hardness and a Protodyakonov coefficient of 8.3. The main roof overlying the immediate roof is made up of arenaceous shale and fine-to-medium-grained sandstone. It exhibits structural stability, a high hardness and a Protodyakonov coefficient of 12.7. The floor of the face is composed of arenaceous shale and coarse-grained sandstone; it exhibits a high hardness and a Protodyakonov coefficient of 11.6. The stratigraphic column below (Fig. 2) shows the sequence of rocks on the face.

Strata	Thickness/m	Column	Lithological description
Main roof	10.26–10.52 / 10.39		Composed of arenaceous shale and fine-to medium-grained sandstone. It exhibits structural stability, a high hardness and a Protodyakonov coefficient of 12.7.
Immediate roof	1.30-1.45 / 1.38		Composed of interbedded medium-grained sandstone and sandy mudstone with a high hardness *f* = 8.3.
False roof	0.20-0.50 / 0.35		Composed of arenaceous shale, which is gradually falls as the face advances
Coal seam	2.90-7.70 / 3.60		Uniaxial compressive strength of 26.8MPa, elastic modulus of 12.4 GPa, with a complex structure, contains 0.1~0.2 m dirt band.
Floor	1.42-2.45 / 1.93		Composed of arenaceous shale and coarse-grained sandstone, exhibits a high hardness *f* = 11.6.

Fig. 2. Stratigraphic column of the face.

The coal face is supported by ZZ-8500-20/40 hydraulic shield support. The caving method is used in combination with forced caving by blasting for roof control. AMG300/700-WD shearer, which is capable of creating openings independently, is used to cut obliquely into the coal from both ends of the face; it has a maximum web depth of 0.63 m and advances 0.55 m per cutting cycle, and its haulage speed is between 0～12 m/min. The face plans to produce 3085.6 tons of coal through 8 cutting cycles per day.

The mining operations at the Xinzhouyao Coal Mine have demonstrated that the high coal hardness (Protodyakonov coefficient 4.8) is the key limiting factor on the advance rate and productivity of coal face. The mining problems caused by high coal hardness are detailed below:

As the hard coal is difficult to cut from the face, the shearer's haulage speed was relatively slow, with an average cutting rate of about 3.4 m/min, limiting the daily advance rate of the face. Consequently, the production at the face failed to reach the target amount in accordance with the production plan. Moreover, the wall of the face remained structurally stable after the coal had been extracted from the face by the shearer, as shown in Fig. 3.

Due to the high coal hardness, shearer picks needed a long time to cut coal from the face. During cutting, the huge amount of heat generated by friction between the picks and hard coal accelerated the wear and tear on the picks and then substantially shortened their service life, indirectly increasing the cost of production.

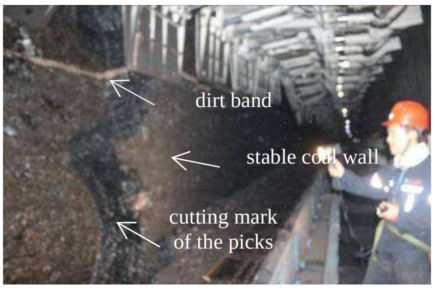

Fig. 3. Structurally stable wall after cutting of hard coal.

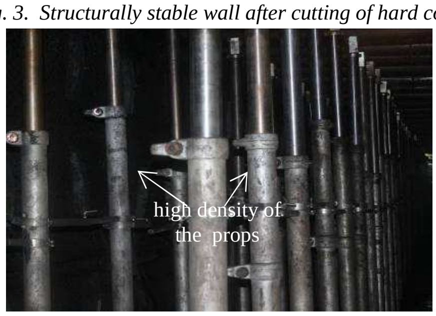

Fig. 4. Reinforced forepoling for the mining roadway.

As a stress-concentrated zone exists in the coal close to the wall, the near-wall coal mass tends to store relatively high strain energy. The superposition of mining-induced abutment pressure and lateral abutment pressure on the road side can intensify the concentration of stress and strain energy in the near-wall coal mass along the mining roadway in front of the face, increasing the risk of rock bursts. Therefore, DZ-4.0 hydraulic props and HDJA-600 metal bars are used to reinforce the forepoling for the mining roadway (Fig. 4). The coal face was initially supported by a relatively high density of props, with a prop spacing of 0.5 m and line spacing of 0.8 m. As the coal face progressed, workers needed to frequently disassemble, transport, and reassemble the props, resulting in high labor intensity.

To overcome the problems and cut the cost of hard coal mining at the coal face No.8706, it is necessary to study the interaction between face support and surrounding rock by modeling and analyzing the support rock system at the face and then to put forward effective measures to improve existing support techniques based on the analysis.

Mechanical analysis of the support rock system

At a coal face in a hard seam, the near-wall coal is hard to cut, and the stable and hard roof near the wall tends to deflect downward under the uniform load applied by overlying strata. The deflected roof and near-wall coal mass, in conjunction with face support and the floor, constitute a closed support rock system. In this paper, a mechanical model was constructed for the support rock system in a hard seam in Xinzhouyao Coal Mine based on its mechanical characteristics (Fig. 5).

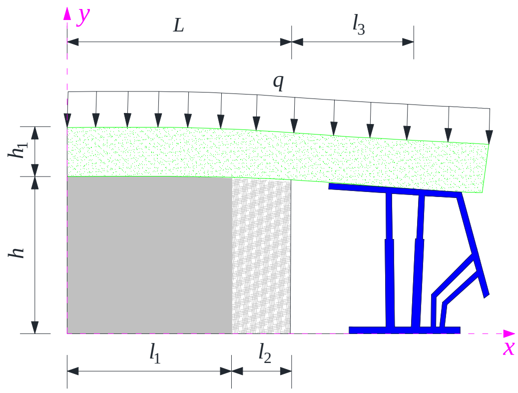

Fig. 5. A mechanical model of the support rock system.

Because the support rock system advances very slowly during mining, the kinetic energy conversion and heat loss within the system are negligible. Since the abutment pressure in the coal mass inside the closed system is a stress within the system that does not perform work on the system, there is no need to discuss the complex distribution pattern of abutment pressure. Then the total potential energy of the support rock system at the face can be expressed as:

$$\xi_z = \xi_d + \xi_w + \xi_{mt} + \xi_{ms} \tag{1}$$

where ξ_z is the total potential energy of the support rock system; ξ_d is the strain energy stored in the roof; ξ_w is the work done by external forces on the system; and ξ_{mt} and ξ_{ms} represent the strain energy stored in the elastic zone and the plastic zone in the near-wall coal, respectively.

As the face advances cyclically, the roof undergoes periodic destabilization and caving, and the support rock system changes slightly in structural morphology. Therefore, the forces acting on the system are considered to be similar between different cycles. The application point of the support force provided by the support is supposed to be fixed in the system. Then the stability criterion for the system under a preset load is derived according to the principle of minimum total potential energy:

$$\frac{\partial \xi_z}{\partial l_1} = 0\,,\ \frac{\partial \xi_z}{\partial l_2} = 0 \tag{2}$$

where l_1 is the length of the elastic zone in the coal and l_2 is the length of the plastic zone.

The analysis below was conducted to solve for the stored strain energy in the roof and coal and the work done by external forces on the system, in order to obtain the specific form of stability criterion for the support rock system in a hard coal seam.

Strain energy stored in the roof: Actual measurements demonstrate that roof deflection curve normally follows an exponential function (Pan Y.S., 1999). Let u_m denote the amount of roof subsidence above the wall. Then the formula describing the roof deflection curve can be obtained as follows:

$$u(x) = u_0\left(e^{\alpha x} - 1\right) \tag{3}$$

in which,

$$u_0 = \frac{u_m}{e^{\alpha L} - 1},\ L = l_1 + l_2$$

where $u(x)$ denotes the amount of roof subsidence; u_0 is the roof displacement parameter; u_m is the amount of roof subsidence above the wall; α is the roof deflection parameter; L is the distance from the wall position to the origin; and x is the position coordinate.

Since the roof maintains structural integrity and high hardness and exhibits slight deflection, the influence of shear stress within the roof strata is neglected herein. Then a coordinate system is established in the neutral plane within the roof, as shown in Fig. 6.

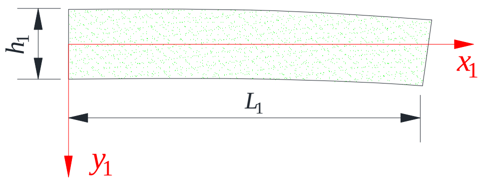

Fig. 6. The coordinate system within the roof.

According to the roof deflection curve (Eq. (3)) and the theories in mechanics of materials, the bending moment, stress, and strain at an arbitrary section of the roof obey the following relationships (Liu H.W., 2006):

$$\frac{d^2u_1}{dx_1^2} = \frac{M}{EI},\ \sigma_{x_1} = \frac{M}{I}y_1,\ \varepsilon_{x_1} = \frac{M}{EI}y_1 \tag{4}$$

where u_1 is the deflection of the roof; x_1 and y_1 are the horizontal and vertical coordinates in the roof; M is the bending moment at the section; E is the elastic modulus of the roof; I is the moment of inertia of the section; and σ_{x1} and ε_{x1} represent the stress and strain, respectively, at the section.

Solving Eqs. (3) and (4) simultaneously yields the stress and strain at any section of the roof:

$$\sigma_{x_1} = Eu_0\alpha^2 e^{\alpha x_1} y_1,\ \varepsilon_{x_1} = u_0\alpha^2 e^{\alpha x_1} y_1 \tag{5}$$

Given that the roof strata above seams in Xinzhouyao Coal Mine are all sandy strata with structural integrity and high hardness, it is reasonable to assume that there exists a linear constitutive relation between the stress and strain at an arbitrary section of the roof. In this case, the strain energy stored in the roof can be calculated as follows:

$$\xi_d = \iint\left(\frac{1}{2}\sigma_{x_1}\varepsilon_{x_1}\right)dx_1 dy_1 = \frac{1}{2}Eu_0^2\alpha^4\int_0^{L_1} e^{2\alpha x_1}dx_1\int_{-h_1/2}^{h_1/2} y_1^2 dy_1 = \frac{E\alpha^3 u_0^2 h_1^3}{48}\left(e^{2\alpha L_1} - 1\right) \tag{6}$$

where L_1 is the length of the roof at the coal face; and h_1 is the roof strata thickness.

As Eq. (6) Indicates, the strain energy stored in the roof during subsiding depends only on its physicomechanical properties and physical dimensions. This, combined with the stability criterion for the system (Eq. 2) and the system's total potential energy (Eq. 1), demonstrates that the strain energy stored in the roof has no influence on the stability of the support rock system.

Work done by external forces on the system: Since the kinetic energy conversion and heat loss within the system are negligible, the work done by external forces on the system is negative and completely converted to potential energy of the system. It is clear that the main external forces acting on the system include the uniform load applied by the overlying strata, the support force provided by the support, and the support provided by the floor (Marcin Witek and Stanisław Prusek, 2016; Frith Russell C, 2015; G.S.P. Singh and U.K. Singh, 2010). The support provided by the floor does zero work on the system. Then the work done by external forces on the system is given by:

$$\xi_w = -\int_0^{L_1} qu(x)\,dx - P\,u(x)\Big|_{x=l_1+l_2+l_3} \tag{7}$$

where P represents the support force by the face support; q is the uniform load by the overlying strata; and l_3 is the distance from the support point position to the coal wall.

Substituting Eq. (3) into Eq. (7) and performing an integration gives:

$$\xi_w = -\left\{ q\left(\frac{1}{\alpha} e^{\alpha L_1} - \frac{1}{\alpha} - L_1 \right) + P\left[e^{\alpha(l_1+l_2+l_3)} - 1 \right] \right\} u_0 \tag{8}$$

Eqs. (1) and (2) indicate that the work of external forces affects the stability of the support rock system. In this case, the system's stability is constrained to some extent, primarily by the support force from the props.

Strain energy stored in coal mass: As the face length along the dip is much greater than the seam thickness; the coal mass is in a plane strain state at the section along the strike of the seam. Therefore, the z-axis of the rectangular coordinate system is determined to be parallel to the length of the face. Under compression by the deflected roof, the interior of the coal mass is subjected to triaxial stress in three orthogonal directions. Under this condition, the stress and strain in the coal mass in the direction of z-axis are given by:

$$\sigma_z = \mu\left(\sigma_x + \sigma_y\right),\ \varepsilon_z = 0 \tag{9}$$

where σ_x, σ_y, and σ_z are stress in the direction of x, y, and z-axis, respectively; μ is the Poisson's ratio of the coal seam; and ε_z is a strain in the direction of z-axis.

Given that the coal mass at the face in the hard coal seam is hard, strong and thus not easy to compress, the volumetric strain in the coal mass is assumed to be zero, i.e. $\varepsilon_x + \varepsilon_y + \varepsilon_z = 0$. In this case, the Poisson's ratio of the coal seam is 0.5 (Wang R. et al., 2009). Then the stress-strain relation of the coal seam can be obtained from Eq. (3) as follows:

$$\varepsilon_y = \frac{u_0}{h}\left(e^{\alpha x} - 1\right) = -\varepsilon_x \tag{10}$$

where ε_x and ε_y are the strain in the direction of x and y-axis, respectively; and h is the thickness of the coal seam.

Fig. 7 illustrates the stress-strain curve typical of a coal mass subjected to triaxial stress (Qian M.G. et al., 2015). At the No.8706 face in Xinzhouyao mine, the near-wall coal is insufficiently unloaded due to high coal hardness and thus still possesses relatively high load-bearing capacity. Therefore, the stress-strain curve of the hard coal seam can be approximated by a bilinear relation.

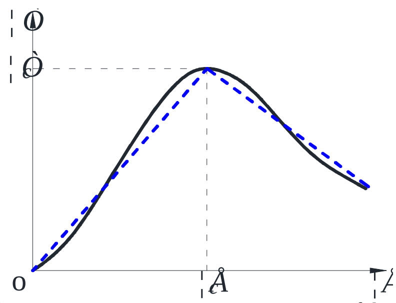

Fig. 7. Bilinear approximative stress-strain curve of the coal mass.

A linear constitutive relation (Dai J., 2013) is then derived from the linear approximation depicted in Fig. 7:

$$\sigma = \begin{cases} E_1\varepsilon & (\varepsilon < \varepsilon_c) \\ \sigma_c - E_2\left(\varepsilon - \varepsilon_c\right) & (\varepsilon \geqslant \varepsilon_c) \end{cases} \tag{11}$$

where σ is the stress in the coal mass; ε is the strain in coal mass; E_1 and E_2 are the elastic modulus and unloading modulus of coal mass, respectively; σ_c is the ultimate strength of coal mass; and ε_c is the ultimate strain of coal mass.

It is noteworthy that the near-wall coal is subjected to various degrees of damage under the joint action of mining and abutment pressure, indicating that in the near-wall region, the unloading modulus is smaller than the elastic modulus, i.e. $E_1 > E_2$.

Strain energy stored in the elastic zone: The coal mass in the interior of the face is under triaxial stress in three orthogonal directions. The location of incipient roof deflection is chosen as the origin of coordinate axes, in light of the linear elastic constitutive relation. The strain energy stored in the elastic zone at the coal face is given by:

$$\xi_{mt} = \sum_{i=x,y,z} \iint \left(\frac{1}{2} \sigma_i \varepsilon_i \right) dxdy = E_1 \int_0^y dy \int_0^{l_1} \varepsilon_y^2 dx \tag{12}$$

where σ_i and ε_i represent the stress and strain in each of the three directions, respectively, and i is a symbolic variable, $i = x$, y, and z.

A rectangular coordinate system is established in the coal mass at the face (Fig. 5). The equation governing the deflection of the coal mass' upper boundary is determined as follows:

$$y = h - u_0 \left(e^{\alpha x} - 1 \right) \tag{13}$$

Solving Eqs. (12) and (13) simultaneously and performing an integration gives the strain energy stored in the elastic zone:

$$\xi_{mt} = E_1 \frac{u_0^2}{h} \left\{ F_1(l_1) - F_1(0) - \frac{u_0}{h} \left[F_2(l_1) - F_2(0) \right] \right\} \tag{14}$$

in which,

$$F_1(x) = \frac{1}{2\alpha} e^{2\alpha x} - \frac{2}{\alpha} e^{\alpha x} + x, \; F_2(x) = \frac{1}{3\alpha} e^{3\alpha x} - \frac{3}{2\alpha} e^{2\alpha x} + \frac{3}{\alpha} e^{\alpha x} - x$$

where $F_1(x)$ and $F_2(x)$ are variadic functions, and $F_1(l_1)$, $F_1(0)$, $F_2(l_1)$, and $F_2(0)$ are their function values under boundary conditions.

As can be seen from Eq. (14), the strain energy stored in the elastic zone at the coal face also has some influence on the stability of the support rock system; in the case of a coal seam of a certain thickness, the amount of elastic strain energy stored in the support rock system primarily depends on the length of the elastic zone.

Strain energy stored in the plastic zone: As the near-wall coal mass at the face is subject to small lateral force, the compression force exerted by the bending roof can increase the plastic deformation of coal and even induce failure of near-wall coal. Studies have found that rib fall at coal faces typically occurs in the form of shearing and sliding (Wang J.C., 2007; Yang P.J. et al., 2012). Therefore, equivalent stress and strain are used here to describe the strain energy stored in the plastically deformed coal mass in the near-wall region. In the same rectangular coordinate system, the volumetric strain in the plastic zone is also assumed to be zero. The equivalent stress and equivalent strain in the plastic zone (Wang R. et al., 2009) can be obtained from the unloading constitutive relation (Eq. (11)) as follows:

$$\sigma_d = \frac{\sqrt{3}}{2} \left| \sigma_x - \sigma_y \right| = \sqrt{3} E_2 \varepsilon_y, \; \varepsilon_d = \frac{2\sqrt{3}}{3} \varepsilon_y \tag{15}$$

where σ_d and ε_d are the equivalent stress and equivalent strain, respectively, in the plastic zone.

As can be seen from Eq. (15), there is a linear relationship between the equivalent stress and equivalent strain in the coal mass in the plastic zone. The stored strain energy in this zone is then given by:

$$\xi_{ms} = \iint \left(\frac{1}{2} \sigma_d \varepsilon_d \right) dxdy = 2E_2 \int_0^y dy \int_{l_1}^{l_2} \varepsilon_y^2 dx \tag{16}$$

Substituting Eqs. (10) and (13) into Eq. (16) separately and performing integration yields the stored strain energy in the plastic zone:

$$\xi_{ms} = 2E_2 \frac{u_0^2}{h} \left\{ F_1(l_2) - F_1(l_1) - \frac{u_0}{h} \left[F_2(l_2) - F_2(l_1) \right] \right\} \tag{17}$$

where $F_1(l_2)$, $F_2(l_1)$, and $F_2(l_2)$ are function values under boundary conditions.

Eq. (17) suggests that the stability of the support rock system is also affected by the strain energy stored in the plastic zone. In the case of a coal seam of a certain thickness, the amount of plastic strain energy stored in the support rock system depends on the lengths of both the plastic and elastic zones.

Substituting the expressions for the strain energy and work done by external forces (Eqs. (6), (8), (14), and (17)) into Eq. (2) separately yields the stability criterion for the support rock system at the face and the support force, given by:

$$\frac{\eta}{2} = \frac{hf_1(l_2) - u_0 f_2(l_2)}{hf_1(l_1) - u_0 f_2(l_1)} + 1, \ P = \frac{2u_0}{h\alpha} E_2 e^{-\alpha(l_1+l_2+l_3)} \left[f_1(l_2) - \frac{u_0}{h} f_2(l_2) \right] \tag{18}$$

in which,

$$f_1(x) = \left(e^{\alpha x} - 1\right)^2, \ f_2(x) = \left(e^{\alpha x} - 1\right)^3, \ \eta = E_1/E_2$$

where $f_1(x)$ and $f_2(x)$ are derivatives of the two variadic functions $F_1(x)$ and $F_2(x)$; $f_1(l_1)$, $f_1(l_2)$, $f_2(l_1)$, and $f_2(l_2)$ are values of the derivatives under boundary conditions; η is the ratio of the elastic modulus to the unloading modulus.

The stability criterion for the support rock system (Eq. (18)) reveals not only the negative relationship between the lengths of the plastic and elastic zones in the system under a set load, but also the relationship between support force and the length of the plastic zone in near-wall coal, which can provide a basis for improving existing rib weakening technique for hard coal mines and determining appropriate setting load of face support.

As can be seen from Eq. (18), the coordination between the coal mass in the elastic and plastic zones and the support force determines the stability of the support rock system. Under conditions of a given roof load, changing the magnitude of support force will result in the redistribution of the plastic and elastic zones in near-wall coal. It was found that a proper reduction in the setting load of the face support can expand the plastic region in coal and thereby intensify the breakage of hard coal ribs.

According to the analysis above, accurate estimation of roof deflection parameter α by inversion is a precondition for obtaining an accurate expression for roof deflection. As the location of incipient roof deflection is not readily known, it is impossible to locate the coordinate system origin in the support rock system. This renders estimation of roof deflection parameter in an absolute coordinate system difficult. In view of this, difference operation was performed on the coordinates of adjacent points on the roof deflection curve, instead of calculating the derivative of roof deflection, thereby overcoming the difficulty of determining the absolute coordinates of the system. The difference formula for the derivative of roof deflection is obtained using Eq. (3) as follows:

$$\left.\frac{du}{dx}\right|_{x=x_k} = \lim_{x_k \to x_j} \frac{u_k - u_j}{x_k - x_j} = \alpha u_0 e^{\alpha x_k} \tag{19}$$

where x_k, u_k, x_j, and u_j are the coordinates of adjacent points on the roof deflection curve.

Arbitrarily choose three adjacent points on the roof deflection curve and randomly select one of them as the reference point, represented by (x_c, u_c), as shown in Fig. 8.

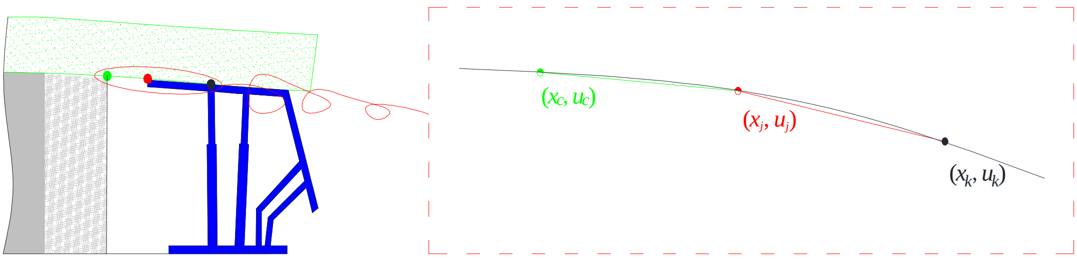

Fig. 8. The linear approximation of the tangent line to the roof deflection curve.

By performing difference operation on the coordinates of the three points to be measured, then an expression for deflection parameter α which uses relative coordinates was derived from Eq. (19):

$$\alpha = \ln\left[\frac{(u_k - u_j)(x_j - x_c)}{(x_k - x_j)(u_j - u_c)}\right] \Big/ (x_k - x_j) \tag{20}$$

where x_c and u_c denote coordinates of the reference point on the roof deflection curve.

Based on the technical conditions of the coal face No.8706 in Xinzhouyao Coal Mine, the roof's displacement relative to the end face of scraper conveyor rail was measured during a maintenance shift, avoiding the influence of mining. The position of the wall served as the reference point, where the relative roof displacement was measured at 34 mm. The average roof displacement above the front ends of the metal bars, which were at a horizontal distance of 0.4 m from the wall, was measured at about 36 mm. The average roof displacement above the front props, which were 2.7 m horizontally from the wall, was measured at about 51 mm. Plugging these values into Eq. (20) gives the roof deflection parameter: α = 0.11. The roof deflection curve at this face was then obtained (Fig. 9).

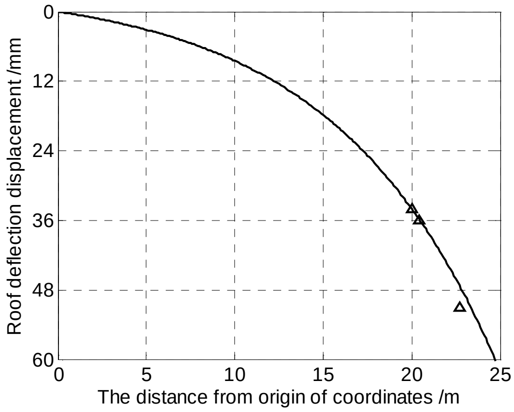

Fig. 9. The roof deflection curve and actual measurements.

As Fig. 9 shows, there is a relatively high correlation between the theoretical and measured values of roof deflection at the studied coal face, with the exception of deviations occurring in positions far from the face. This is because during the deflection parameter inversion, the large distances between the points of deflection measurement were bound to cause an error in the difference operation used instead of derivation. It follows that the points of roof deflection measurement should be as close as possible in actual measurement and inversion.

An analysis of the measurement results indicates that the mining-induced abutment pressure in the coal face No.8706 has a 20 m long scope of influence and the point of application of the support force is nearly 3.8 m from the wall. Therefore, the origin of the coordinate system in the coal was chosen to be at a point 20 m from the wall. The relationship of the length of the elastic zone l_1 to the length of the plastic zone l_2 for various η before and after unloading was obtained using the first formula in Eq. (18) (Fig. 10). The red, black, and blue lines in the figure below denote the relationship between the length of the two zones when η = 3, 5, and 10, respectively.

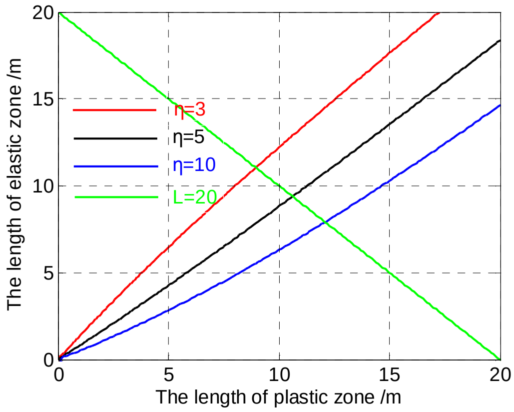

Fig. 10. l_1 versus l_2 before and after unloading.

The total length of the elastic and plastic zones in the near-wall region, denoted by the green line in Fig. 10, is constant at 20 m. The points at which the green line intersects with the red, black, and blue lines represent the lengths of the two zones for the corresponding η. For example, when η = 3, the length of the elastic zone is about 11.3 m, and the plastic zone is 8.7 m long; when η = 10, the elastic and plastic zones are 8.2 m and 11.8 m long, respectively. This indicates that both η and l_2 increase as the modulus of coal mass in the plastic zone decreases.

Similarly, the variations in the support force provided by the face support for different modulus of coal mass in the plastic zone were derived using the second formula in Eq. (18) (Fig. 11). The red, black, and blue lines in the figure below illustrate the relationship between the length of the plastic zone and the support force when the modulus of coal mass in the plastic zone is 5.6 GPa, 3.4 GPa, and 1.3 GPa, respectively.

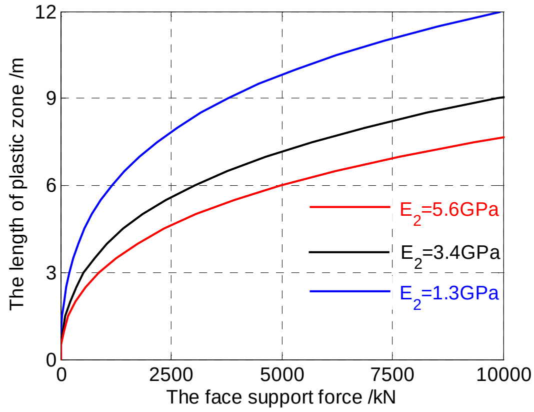

Fig. 11. The length of the plastic zone versus support force.

Fig. 11 demonstrates that as the plastic zone in the near-wall coal grows longer, the excess roof load from overlying strata tends to be transferred to the face support. For a particular support force, the smaller the modulus of coal mass in the plastic zone, the longer the plastic zone will be. Since the modulus of near-wall coal mass is associated with its structural damage and degree of breakage, an appropriate reduction in the support force in actual mining production can expand the plastic zone in near-wall coal, then deepen the breakage of near-wall coal and decrease the modulus of coal in the plastic zone, helping achieve softer coal at a coal face. Therefore, this method can be used to facilitate cutting of hard coal and provide a basis for determination of appropriate setting load of face support.

At the coal face No.8706 in Xinzhouyao Coal Mine, the hydraulic props should begin to advance under load in sequence when the props are 5～8 m behind the shearer. In order to increase roof stability by providing timely support to the newly exposed roof and retarding early subsidence of hard roof, to enable the support to reach a constant-resistance state as soon as possible, and to prevent violent strata behaviors caused by extensive collapse of hard and thick main roof (10.4 m thick on average), the support's setting load was initially set at 7755 kN, approximately 91.2% of its rated working resistance. However, the low shearer haulage speed (3.4 m/min on average) and below-target output of the face in actual mining production prove that a high setting load is unfavorable for the shearer to cut coal efficiently.

The theoretical analysis presented above suggests that properly lowering the setting load of face support can reduce coal strength and hardness by intensifying the breakage of near-wall coal and thus facilitate fast coal cutting. Therefore, the setting load was later lowered to 6400 kN, or 75.3% of the rated working resistance of the support, based on the results of related research. With this lower setting load, a better rib weakening effect was achieved, as shown in Fig. 12, and the shearer haulage speed was significantly increased to 5.7 m/min on average, ensuring high productivity of the face.

Fig. 12. Status of the wall after a reduction in the setting load.

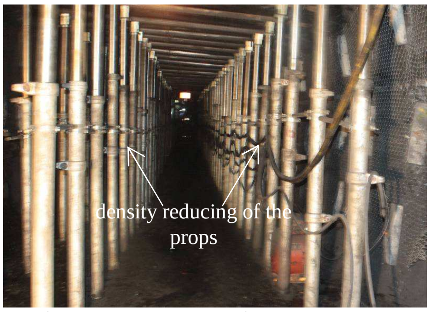

Fig. 13. Status of the mining roadway after a reduction in the setting load.

After the setting load of the face support was lowered, the density of props along the mining roadway in front of the face was reduced correspondingly, with the prop spacing set at 0.8 m and line spacing at 1.0 m (Fig. 13). This not only ensured the stability of ribs along the mining roadway, but also substantially cut the labor intensity.

By lowering the setting load while ensuring safe production, part of the strain energy stored in the roof in an initial deformation state can be released proactively, and then the intensity of roof weighting can be lessened. Moreover, as the coordination between the near-wall coal and face support in the support rock system was fully utilized to support the roof, the working resistance of the face support exhibited a decline after the setting load was lowered, as shown in Fig. 14.

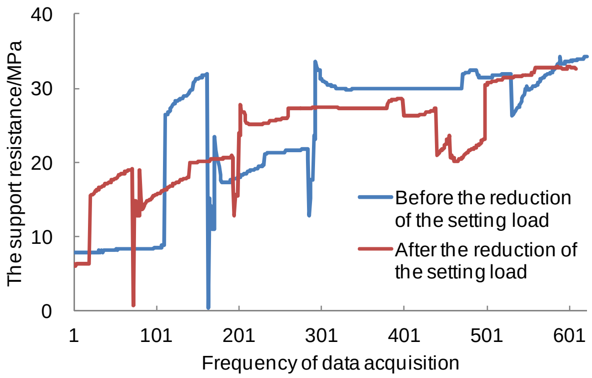

Fig. 14. Support resistance before and after a reduction in the setting load.

According to the analysis above, the proactive improvements to the hydraulic support technique, including a reduction in the setting load of the face support, have brought about an increase in the efficiency of hard coal cutting without auxiliary rib weakening measures, mitigation of roof weighting, and lower support resistance during the actual production at the coal face No.8706 in Xinzhouyao Coal Mine. This raises the productivity of the face and provides significant social and economic benefits.

Conclusions

At a face in a hard coal seam, the roof, near-wall coal, support, and floor constitute a closed support rock system. By means of mechanical modeling and analysis, the stability criterion for this system was obtained, and the coordination between the near-wall coal and face support in supporting the roof was revealed. It was found that a proper reduction in the support's setting load will enable the near-wall coal to play a proactive role in supporting the roof.

The study also uncovered a negative relationship between the length of plastic and the length of the elastic zone in the near-wall coal and the rib weakening mechanism of the proactive measure of lowering the support's setting load. The analysis shows that properly lower the support's setting load can enlarge the plastic zone in hard coal and intensify the breakage of near-wall coal, thus facilitating the weakening of hard ribs.

The actual production at the coal face No.8706 in Xinzhouyao Coal Mine demonstrates that a reduction in the support's setting load from 7755 kN to 6400 kN led to enhanced rib weakening effect, an increase in shearer haulage speed and coal output, and a reduction in the intensity of roof weighting and support resistance. As a result, the face is able to produce more and faster and thus provide greater social and economic benefits.

Acknowledgment: *This work was supported by National Natural Science Foundation of China (51604262), Foundation Research Project of Jiangsu Province (BK20160256), China Postdoctoral Science Foundation Project (2015M581896), Postdoctoral Science Foundation Project of Jiangsu Province (1601212C).*

References

Cao SG, Qian MG, Liu CY, Miao XX.: New research about support and surrounding rock relationship in working face, In*: Journal of China Coal Society, Vol.23, No.6, 1998, pp.575-579*.

Dai J.: Dynamic behaviors and blasting theory of rock, Beijing, *Metallurgical industry Press, 2013*.

Frith Russell C.: A holistic examination of the load rating design of longwall shields after more than half a century of mechanised longwall mining, In: *International Journal of Mining Science and Technology, Vol.25, No.5, 2015, pp.687-706.*

G.S.P. Singh, U.K. Singh.: Prediction of caving behavior of strata and optimum rating of hydraulic powered support for longwall workings, In: *International Journal of Rock Mechanics & Mining Sciences, Vol.47, No.1, 2010, pp.1-16.*

Liu CY, Qian MG, Cao SG, Miao XX.: Influencing mechanism of immediate roof on the relation between supports in the workings and surrounding rock, In: *Journal of China Coal Society, Vol.22, No.5, 1997, pp.471-476*.

Liu HW.: Mechanics of materials, *Beijing, Higher Education Press,2006*.

Marcin Witek, Stanisław Prusek.: Numerical calculations of shield support stress based on laboratory test results, In: *Computers and Geotechnics, Vol.72, No. 2, 2016, pp.74-88*.

Navid Hosseini, Kamran Goshtasbi, Behdeen Oraee-Mirzamani, Mehran Gholinejad.: Calculation of periodic roof weighting interval in longwall mining using finite element method, In: *Arabian Journal of Geosciences, Vol.7, No.5, 2014, pp.1951-1956.*

Pan YS.: Study on rock burst initiation and failure propagation, *Beijing, Tsinghua University, 1999*.

Peng SS.: Support capacity and roof behaviour at longwall faces with shield supports, In: *International Journal of Mining and Geological Engineering, Vol.5, No.1, 1987, pp.29-57.*

Qian MG, Miao XX, He FL, Liu CY.: Mechanism of coupling effect between supports in the workings and the rocks, In: *Journal of China Coal Society, Vol.21,No.1,1996, pp.40-44.*

Qian MG, Shi PW, Xu JL.: Mining pressure and strata control, Xuzhou, *China University of Mining and Technology Press, 2015.*

Singh GSP, Singh UK.: A numerical modeling approach for assessment of progressive caving of strata and performance of hydraulic powered support in longwall workings, In: *Computers and Geotechnics, Vol.36, No.9, 2009, pp.1142-1156.*

Singh GSP, Singh UK.: Numerical modeling study of the effect of some critical parameters on caving behavior of strata and support performance in a longwall working, In: *Rock Mech Rock Eng, Vol.43, No.4, 2010, pp.475-489.*

Trueman R, Lyman G, Callan M, Robertson B.: Assessing longwall support-roof interaction from shield leg pressure data, In: *Mining Technology, Vol.114, No.3, 2005, pp.176-184*.

Trueman R, Lyman G, Cocker A.: Longwall roof control through a fundamental understanding of shield-strata interaction, In: *International Journal of Rock Mechanics & Mining Sciences, Vol.46, No.2, 2009, pp.371-380.*

Verma A., Deb D.: Numerical analysis of an interaction between hydraulic-powered support and surrounding rock strata, In: *International Journal of Geomechanics, Vol.13, No.2, 2013, pp.181-192.*

Wang GF.: Theory system of working face support system and hydraulic roof support technology, In: *Journal of China Coal Society, Vol.39, No.8, 2014, pp.1593-1601.*

Wang GF, Pang YH.: Relationship between hydraulic support and surrounding rock coupling and its application, In*: Journal of China Coal Society, Vol.40, No.1, 2015, pp.30-34*.

Wang JC.: Mechanism of the rib spalling and the controlling in the very soft coal seam, In: *Journal of China Coal Society, Vol.32, No.8, 2007, pp.785-788.*

Wang R, Huang WB, Huang ZP.: Introduction to plasticity mechanics, *Beijing, Peking University Press, 2009*.

Wilson AH.: Support requirements on longwall faces, In: *GB Min. Eng, Vol.134, No.6, 1975, pp. 479-491*.

Yang PJ, Liu CY, Wu FF.: Breakage and falling of a high coal wall in a thick mined seam, In: *Journal of China University of Mining &Technology, Vol.41, No.3, 2012, pp.371-377.*

Zhang KB, Zhuang YL, Dai J.: The mechanism on supports working resistance and surrounding rock movement, *Xuzhou, China University of Mining and Technology Press, 1999.*