Stability characteristics of a fractured high roof under nonpillar mining with an automatically formed roadway by using a visualized discrimination approach

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Abstract
Nonpillar mining (NPM) with an automatically formed roadway is an emerging mining technology that does not use coal pillar retention and drivage. This technology can be used to actively control the caving height of the roof by directional cutting technology and can promote the rapid equilibrium of a fractured high roof under the support of the caved gangue. Therefore, when analyzing the stability of a high roof under NPM, not only the interaction force of the fractured rock blocks but also the effects of the support provided by the gangue, and the coal body should be considered. In addition, if necessary, the roof support in the roadway should also be considered. Based on this idea, in this paper, we analyzed the stability characteristics of a fractured high roof and suggested the stability conditions of the fractured rock of a high roof, including actual stability, pseudostability, and instability and provided the criteria for determining these three stability conditions. Furthermore, a visualized approach for the determination of roof stability was proposed. The core idea of the proposed approach is to transform the parameters that cannot be measured in the abovementioned conditions into visualized data that can be directly or indirectly obtained in the field, allowing for the detection and occasional assessment of the stability condition of the fractured rock of a high roof. This visualized determination approach can be used to assess the roof stability condition in a timely manner during the production process and to ensure production safety. The abovementioned research results were successfully used in the S1201-II mining face of the Ningtiaota Coal Mine, indicating that the proposed approach is feasible and can be used under similar conditions.

KEYWORDS
determination of roof stability, high roof, NPM, roof fracture structure, roof stability characteristics
INTRODUCTION

NPM is a new longwall mining method with significant advantages in improving coal recovery, reducing roadway excavation, controlling uneven ground subsidence, and protecting the ecosystem and the groundwater system. The core idea of this technology for roof control is to transform the roof pressure into dynamic power using directional cutting technology to cause the gangue to fall, thus allowing the energy accumulated in the roof to be released. Moreover, the caving height of the roof can be actively controlled by reasonably selecting the cutting parameters (including the cutting height, angle, and explosive parameters) to create a greater volume of fallen rocks after crack expansion and filling, supplementing the extracted coal. In this way, the fallen gangue can support the uncaved high roof and promote the stability of a fractured high roof.

In China, the number of research reports about this new technology is increasing. Wang et al provided an introduction to NPM in terms of its basic principles and key technologies. He et al performed a simulation analysis of the stress field distribution characteristics of NPM using a numerical simulation method and compared the differences between this technology and traditional mining technology. Wang et al analyzed the roof deformation using an elastoplastic mechanics method and found that the revolution and deformation of a high roof and the width of the retained entry are the most significant factors impacting the deformation of the pressure relief structure; thus, an idea (and countermeasures) for the control of roof deformation was put forward by reasonably selecting the cutting height and controlling the width of the retained entry. However, NPM is a new proposed research direction. Most of the present studies on this technology are based on the macroscopical conditions of the site, and the stability of a high roof structure under NPM has not yet been considered.

Currently, the more mature theories concerning the stability of a roof structure under traditional longwall mining conditions include the masonry beam theory and the transfer rock beam theory. In addition, many scholars have examined the stability of roof structures under certain specific engineering and geological conditions (eg, a large mining height, a large burial depth, a steep slope, a soft rock roof, or a fractured roof) through field experiments, model experiments, and numerical simulation studies. Whichever geological conditions are considered, a common understanding is that the stability of a fractured high roof is a critical factor impacting the safety of a mining roadway, especially those incorporating gob-side entry. As a result, the stability of a fractured high roof is a key parameter that we must control throughout the production process.

However, in an assessment of the studies concerning the roof structure stability during traditional longwall mining, the stability conditions of these structures were often expressed by the interactions of the fractured rocks in the high roofs. For example, when the revolution angle of a fractured rock is too small, the fracture structure is likely to slip and become unstable; when the revolution angle of fractured rock is too large, the fracture structure is likely to extrude material, deform material, and generally become unstable. When mining with NPM, the interaction between the fractured rock blocks is not sufficient to indicate the actual stability condition of the fractured rock in the high roof. The core idea of NPM is to control the deformation and movement of the high roof using rocks caved due to crack expansion, thereby significantly diminishing the mine pressure characteristics arising from the overlying strata. As a result, in the process of mining with NPM, the interaction between the fractured rocks is not the only factor that maintains a stable structure. Any attempt to perform a stability analysis of a high roof should consider the effects of the support provided by the gangue and the coal body. In addition, if necessary, it is also required to consider the roof support in the roadway.

Unfortunately, in an assessment of the past roof stability research results, no method for conveniently and quickly detecting and evaluating the stability condition of fractured high roof rocks at different time points was proposed. Consequently, it is impossible to predict the stability of the abovementioned structure in advance; thus, appropriate countermeasures to improve the roof stability cannot be implemented in a timely manner. The main contributions of this work are to specify the stability determination criteria and the stability characteristics of a fractured high roof under NPM by taking the gangue support force and the coal body support force into consideration and to propose a visualized approach for the determination of the roof stability condition. The proposed approach was tested and found to be effective. Our research results can provide important guidance for the prevention of roof disasters and guarantee safe production at an NPM site.

STABILITY PRINCIPLES OF A HIGH ROOF

After a coal seam is extracted, the soft rock of a low roof will be the first to cave under the action of mine pressure. Due to its high strength, a high roof is usually suspended. As the mining face is driven forward, the suspended area continuously increases. Research shows that if a high roof is considered a suspended “plate,” the maximum value of the bending moment is located in the middle section of the longer side of the plate (that is, the mining face and the open-off cut walls). Therefore, a high roof will first fracture at this point, and then, the middle section of the shorter side will fracture and form cracks one by one. Until the cracks surrounding the plate cut through and form a substantially O-shaped
circle of cracks, the boundaries surrounding the plate form simply supported conditions, and the maximum bending moment is transferred to the central section of the plate. When the bending moment of the central section exceeds the limit, an X-shaped failure will form. This process refers to the first loading of a high roof. During this period, the resulting high stress is likely to cause problems such as rock bursts.\(^1^8\) As the mining face continues to drive forward, whenever the suspended formation of a high roof reaches its limit length, there will be fracturing and movement. As a result, the roof structure will always go through a periodic development process of “stabilizing-movement-destabilizing,” that is, the high roof experiences periodic loading. The structural forms of the roof strata during this movement are shown in Figures 1 and 2.

As shown in Figure 2, one side of the roadway is a solid coal body, and the other side is the caved gangue in the goaf zone. Rock block A, above the coal body, rock block B, above the roadway, and rock block C, above the caved gangue, constitute a hinged structure. The right end of block B broke above the coal body, revolved, and sunk near the fracture. Then, the sunken left end came into contact with the caved gangue and gradually compacted it. This hinged structure functions as the main bearing body of the pressure of the overlying strata; the motion of block B plays a vital role in maintaining the stability of the roadway surrounding rocks.

Therefore, an important way to stabilize a roadway is to minimize the sinking motion of block B, thus reducing the dynamic pressure on the roadway. As shown in Figure 2, the most effective method to minimize the space available for the movement of fractured rock B is to control the height of the caving zone so that the caved gangue, after being cracked, can expand and fill the goaf space. In other words, the following equation of equilibrium should be satisfied:

\[
\Delta V_m = \Delta V_b, \quad (1)
\]

where \(\Delta V_m\) is the volume of the extracted coal body, and \(\Delta V_b\) is the increased volume after the caved gangue is cracked and expands.

To control the caving height and satisfy the above equation, Prof. He Manchao proposed the use of directional cutting technology.\(^1^9\) The increased volume of the caved gangue after being cracked and expanded meets the following equation:

\[
\Delta V_b = (K_0 - 1) H_s S_x, \quad (2)
\]

where \(K_0\) is the initial crack expansion coefficient of the roof rock within the cutting height range; \(H_s\) is the caving height that is controlled by the roof cutting, m; and \(S_x\) is the caving area controlled by the roof cutting.

Under the influence of the overburden pressure, the volume of the caved gangue will be further compressed until its bearing capacity is equal to the overburden pressure. Finally, the subsidence of the farthest end of the key block B can be expressed as

\[
H_s = H_m - H_c (K_1 - 1), \quad (3)
\]

where \(H_m\) is the mining height near the roadway, m, and \(K_1\) is the residual cracking-expansion coefficient of the roof rock. In the whole process, the surrounding rocks of the roadway will gradually deform when subjected to the settling compression action of rock block B. However, because rock block B is always supported by the gangue, there is not a very intense deformation of the strata in the roadway. After the compaction of the gob-side gangue, rock block B will form a stable structure under the supporting force of the gangue, the coal body, and the combined actions of the adjacent rock blocks, A and C.

3 | STABILITY CHARACTERISTICS OF A HIGH ROOF

3.1 | Establishment of the mechanical model

According to the roof rock strata structure shown in Figure 2, fractured rock block B above the roadway revolved and sunk along the fracture above the solid coal. As the gangue in the goaf was gradually compacted, fractured rock block B finally formed a stable structure under the combined actions of the caved gangue and the coal body; thus, the established stress model is shown in Figure 3.

For the convenience of analysis, the stress model with respect to fractured rock B is simplified as follows:

1. In the said model, \(F_{CB}\) and \(T_{CB}\) are the shearing force and horizontal compressive force generated by fractured rock C on rock B, the action point is located at H, \(L_{BH} = s/2\), and \(s\) is the contact length between the fractured rocks.
2. \(F_{AB}\) and \(T_{AB}\) are the shearing force and horizontal compressive force generated by stable rock A on fractured rock B, the action point is located at N, and \(L_{ON} = s/2\).
3. \(q_s\) is the load generated by the weight of the overlying strata, assuming that it is a uniformly distributed load; \(\sigma_x\) is the supporting load of the coal body in the lane; and \(q_{g1}\) is the supporting load generated by the caved gangue in the goaf zone of fractured rock B whose size depends on the compression of the gangue, assuming that it has a linear distribution.
4. \(x_0\) is the horizontal distance between the fracture position B and the coal wall, which is located at the elastoplastic boundary inside the roadway.
5. The revolving angle of B is \(\theta\), and the revolving base point is N.
6. Since $\theta$ is usually small under the conditions of NPM, it can be approximated as $L_B \approx x_0 + a + x_1$.

7. Since the mining process during NPM finally realizes the self-stability of the rock strata, the support force applied on the roof in the roadway will not be considered during the calculation process.

3.2 | Determination of the model parameters

(1) Overall dimensions of fractured rock B.

(a) Length of fractured rock B

According to the existing studies, the fracture length $L_B$ of rock B can be calculated based on the fracture line theory:\textsuperscript{20}

\[ L_B = L \left( \frac{L}{S} + \sqrt{\frac{L^2}{S^2} + \frac{3}{2}} \right), \]  
(4)

where $S$ is the length of the mining face, and $L$ is the periodic roof loading pace of the mining face.

(b) Fracture location

The fracture location of rock B generally extends a certain distance into the coal walls of the lane, which is given in Reference [\textsuperscript{21}]:

\[ x_0 = \frac{hA}{2 \tan \varphi} \ln \left( \frac{kyH + \frac{c}{\tan \varphi}}{\frac{c}{\tan \varphi} + \frac{E}{A}} \right), \]  
(5)
where \( h \) is the roadway height, \( A \) is the lateral pressure coefficient, \( p \) is the coal support strength, \( H \) is the roadway burial depth, \( c \) is the coal body cohesion, \( \varphi \) is the internal friction angle of the coal body, \( \gamma \) is the average unit weight of the overlying strata, and \( k \) is the stress concentration coefficient.

In turn, the support length of the caved gangue in the goaf zone of fractured rock B can be obtained:
\[
x_1 = L_B - x_0 - a
\]  
(6)

(c) Engagement contact length of fractured rock
According to Reference \(^5\), the engagement contact length of a fractured rock can be calculated:
\[
s = \frac{1}{2} (H_B - L_B \sin \theta).
\]  
(7)

(2) Stress on fractured rock B
(a) Load of the overlying strata
The load on the upper boundary of a fractured rock is mainly correlated with its burial depth and the average unit weight of the overlying strata, assuming that the subjected load is a uniformly distributed load; then, it can be obtained:
\[
q_z = \gamma H_d.
\]  
(8)

where \( H_d \) is the burial depth of fractured rock B.

For the convenience of calculation, the load on the upper boundary is simplified to
\[
F_z = \gamma H_d L_B.
\]  
(9)

\[
L_{MP} = \frac{1}{2} L_B.
\]  
(10)

(b) Support load of solid coal
Fractured rock B revolved and deformed around the fracture point; then, the solid coal was compressed. Near the fracture point, the rocks were subjected to the support force of the solid coal. According to the limit equilibrium theory,\(^22\) the support load of the coal body is
\[
\sigma_x = \left( \frac{c}{\tan \varphi} + \frac{p}{A} \right) e^{\frac{2mG}{\gamma A} (x_0 - x)} - \frac{c}{\tan \varphi},
\]  
(11)

where \( 0 < x < x_0 \).

For the convenience of calculation, \( \sigma_x \) can be transformed into a linearly distributed load:
\[
\sigma'_x = \frac{p - Ak \gamma H}{A x_0} x + k \gamma H.
\]  
(12)

On this basis, the load of the above linear function is further simplified to a concentrated resultant force \( F_m \) by integral method in this paper, and the application point of the resultant force is located at the centroid position of the triangle formed by the linear function. This simplification will have little effect on the stability analysis of rock B, but will make the whole calculation process very simple.

\[
F_m = \frac{1}{2} \left( \frac{p}{A} + k \gamma H \right) x_0
\]  
(13)

\[
L_{QO} = \frac{2 p + Ak \gamma H}{3 (p + Ak \gamma H)} x_0
\]  
(14)

where \( L_{QO} \) is viewed as the distance from the position of the force \( F_m \) (point Q) to the rotation point of block B (point O).

(c) Support load of the caved gangue
According to the design principle of the roof cutting height, after being cracking, the caved gangue can expand and completely fill the extracted space. The high roof will sink and come into contact with the gangue, and the continuous compaction of the gangue will occur until it became stable. In this process, the support load on the fractured rock B is correlated with the compression acting on the gangue. We assume that the compression of the gangue at the gob-side end of fractured rock B is
\[
W_1 = H_s = L_B \sin \theta.
\]  
(15)

According to the geometric relationship shown in Figure 3, the compression of the gangue at the edge P of the loose lane can be obtained:
\[
W_2 = (x_0 + a) \sin \theta.
\]  
(16)

Therefore, the compression of the gangue at any point within the range of \( x_0 + a < x < L_B \) can be obtained:
\[
W_x = x \sin \theta.
\]  
(17)

The support load generated per unit area of gangue can be further obtained:
\[
q_g = K_G W_x,
\]  
(18)

where \( K_G \) is the support coefficient of the caved gangue.\(^23\)

For the convenience of calculation, the support load of the caved gangue can be simplified to an equivalent resultant force \( F_g \) applied at R; then,
\[
F_g = \frac{(L_B + x_0 + a) K_G x_1}{2} \sin \theta,
\]  
(19)

\[
L_{OR} = \frac{3 L_B - 2 x_1}{3 (2 L_B - x_1)} x_1.
\]  
(20)
where $L_{OR}$ is viewed as the distance from the position of the force $F_B$ (point R) to the rotation point of block B (point O).

(d) Interaction between adjacent rocks

Since the established model assumes that the rock revolves around N, that is, the rock is considered to undergo no relative displacement with the adjacent rock at N; thus, it can be approximated that $F_{AB} = 0$.

According to the static equilibrium, $\sum F_x = 0$, $\sum F_y = 0$, and $\sum M_M = 0$.

Consequently,

\[
\begin{align*}
F_z &= F_g + F_m + F_{CB} \\
T_{AB} &= T_{CB} \\
F_{CB}L_B + F_mL_{OO} + F_gL_{OR} &= \frac{1}{2}F_zL_B - T_{CB}(H_B - s - L_B \sin \theta) \\
\end{align*}
\]

so

\[
\begin{align*}
F_{CB} &= F_z - F_g - F_m \\
T_{CB} &= \frac{F_zL_B - 2F_{CB}L_B - 2(F_mL_{OO} + F_gL_{OR})}{H_B - L_B \sin \theta}.
\end{align*}
\]  

(21)

3.3 Stability analysis and determination criteria

For traditional mining alternatives, according to the S-R stability theory of the masonry beam structure, as the fractured rock B of the high roof revolves and sinks, the engagement between the rocks will generate forces in the horizontal and vertical directions such that the equilibrium of the whole structure is largely influenced by the interaction between the fractured rocks and the rock strength at the engagement point. If the frictional resistance at the engagement point is insufficient to provide the shearing force required to maintain the equilibrium of the rocks, then the structure would slip and become unstable. If the horizontal squeezing force at the engagement point is greater than the compressive strength of the rock, then there is a local stress concentration at the squeezing location, which causes the rocks to enter the plastic state, resulting in a failure at the engagement point, causing fractured rock B to revolve and sink at a higher rate, ultimately leading to a large deformation and instability.

However, for the mining process with NPM, the said structural instability caused by the interaction between the fractured rocks is insufficient to indicate the actual stability condition of a fractured high roof. The core idea of NPM is to control the deformation and movement of the high roof by the caving and crack expansion of the rocks, thereby significantly diminishing the mine pressure arising from the overlying strata. As a result, in the process of mining with NPM, the interaction between the fractured rocks is not the only factor that maintains a stable structure. The attempt to perform the stability analysis of a high roof should consider the effects of the support provided by the gangue and the coal body. Therefore, during roof movement under NPM conditions, the stability conditions of fractured rock B of a high roof can be described by actual stability, pseudostability, and instability:

1) Actual stability

In the early stage of roof movement, fractured roof rock B begins to revolve and sink. In the case of a smaller revolving angle $\theta_1$, the degree of engagement between the fractured rocks is lower, and it is impossible to form the self-equilibrium state of the fracture structure by this interaction. The goal of controlling the surrounding rocks under NPM is to make full use of the support force of the gangue so that the fractured rocks are in equilibrium due to the support of the gangue and the coal body. If the support force provided by the caved gangue in this stage is sufficient, the fractured rocks can form an equilibrium structure, even if the roof under this state may be subjected to other external forces (eg, from the dynamic pressure disturbance that occurs when extracting coal on the adjacent mining face), because the caved gangue will not cause an unexpected compression. Therefore, fractured rock B will obviously not sink, and the surrounding rock of the roadway will always be under a very reliable stability condition. This stability is also referred to as actual stability, and the corresponding roof structure is shown in Figure 4.

According to the stress equilibrium characteristics of the roof structure under the actual stability condition, fractured rock B under this state meets the following stress conditions:

\[
\begin{align*}
T_{CB} \tan \varphi_B &\leq F_{CB} \\
T_{CB} &\leq \eta[s] \\
\end{align*}
\]

(23)

where $\tan \varphi_B$ is the coefficient of friction between the fractured rocks, which is generally 0.3, $\eta$ is the coefficient of contact between the rocks, which is generally 0.3, and $[s]$ is the compressive strength of the fractured rocks.
2) Pseudostability

If the caving height of the roof does not meet the design requirements, the caved gangue cannot support fractured rock B to form a stable state, and fractured rock B will continue to revolve and deform. As the revolving angle \( \theta_2 \) increases, the degree of engagement between the rocks will gradually increase. When the engagement reaches a certain degree, the friction generated from the engagement of the fractured rocks begins to contribute to the equilibrium of the fracture structure; thus, the fracture structure will enter a state of equilibrium under the combined action of the friction generated by the engagement and the support forces provided by the gangue and the coal body. However, this stability condition is fundamentally different from that of the actual stability condition. Under this state, the friction generated by the interaction between the fractured rocks will be crucial to the stability of the whole roof structure, and the support force of the caved gangue is insufficient to support fractured rock B to form a mechanical equilibrium. Therefore, even if the roadway is temporarily stable under this state, when the roof is subjected to another external dynamic pressure disturbance (e.g., when extracting coal on the adjacent mining face), the interaction between the rocks may be damaged. After the state of equilibrium is interrupted, the caved gangue will be further compressed, and the surrounding rocks of the roadway will be subjected to unexpected loading again. Therefore, this state is also referred to as pseudostability, and the corresponding roof structure is shown in Figure 5.

According to the stress balance characteristic of a roof fracture structure under the pseudostability condition, fractured rock B under this state meets the following stress conditions:

\[
\begin{align*}
T_{\text{CB}} \tan \varphi_B & \geq F_{\text{CB}} \\
T_{\text{CB}} & \leq s\eta[\sigma]
\end{align*}
\]  

(24)

3) Instability

If the caving height of a roof is very low, fractured rock B will continue to revolve and sink. When the degree of sinking reaches a certain threshold, the engagement squeezing load between the fractured rocks will be greater than the compressive strength of the rocks, thereby causing the fractured rocks to deform and become unstable. After entering this state, the interaction force between the fractured rocks along the vertical direction will substantially disappear, and because it is no longer possible to form an equilibrium structure between the fractured rocks by mutual engagement, all the loads from the overlying strata must be passively borne by the gangue and the coal body. Due to the large revolving angle of fractured rock B, the roof surface in the roadway under this state generally shows an obvious asymmetric tilting deformation, and the immediate roof rocks are likely to break along the coal walls of the lane subject to the squeezing effect of fractured rock B. Consequently, when the roof enters this state, certain reinforcement support should be provided to the roadway roof to prevent the roof rocks from immediately experiencing unexpected instability and caving. This state is also referred to as instability, and the corresponding roof structure is shown in Figure 6.

According to the stress equilibrium characteristics of the roof fracture structure under the instability condition, fractured rock B under this state meets the following stress conditions:

\[
T_{\text{CB}} \geq s\eta[\sigma].
\]  

(25)

4 | VISUALIZED CRITERIA FOR THE DETERMINATION OF HIGH ROOF STABILITY

The research on high roof stability should not be only theoretical. However, it is still impossible to determine roof stability during the field production process by using the abovementioned criteria because there is no way to monitor the mechanical parameters, such as \( T_{\text{CB}} \) and \( F_{\text{CB}} \), on site. As a result, we must transform the parameters in the discrimination criteria into visualized data that can be obtained directly or indirectly during a field project. In doing so, we can get visualized criteria for the determination of high roof stability. In other words, we can make the stable state of the roof structure accessible throughout the mining process. Once the roof is found to have become
unstable, it is then possible to take timely control measures and ensure the safety of the entire production process.

1) Visualized criteria for the determination of actual stability 

According to the stability determination criteria, during the revolving process of fractured rock B, except for the revolving angle always as a variable, the other parameters have fixed values. Therefore, the above theoretical criteria for the determination can be transformed into visualized criteria expressed by the revolving angle of fractured rock B, of which more details are given below:

\[
\begin{cases} 
A_1 \sin^2 \theta + B_1 \sin \theta + C_1 \leq 0 \\
A_2 \sin^2 \theta + B_2 \sin \theta + C_2 \leq 0 
\end{cases}
\] (26)

where

\[
\begin{align*}
A_1 &= -ML_B \\
B_1 &= F_s L_B - F_m L_B - M H_B + 2 M L_B \tan \varphi_B - 2 M L_{OR} \tan \varphi_B \\
C_1 &= F_s L_B \tan \varphi_B - 2 F_m L_{OQ} \tan \varphi_B - (F_s - F_m) (H_B + 2 L_B \tan \varphi_B) \\
A_2 &= -\eta \sigma L_B^2 \\
B_2 &= 4 M L_B - 4 M L_{OR} + 2 \eta \sigma H_B L_B \\
C_2 &= 2 F_s L_B - 4 F_m L_{OQ} - 4 L_B (F_s - F_m) - \eta \sigma H_B^2 \\
M &= \frac{1}{2} (2 L_B - x_1) K_G x_1
\end{align*}
\] (27)

\[
\begin{align*}
\theta &= \arctan \left( \frac{2a^2 (\mu^2 - 8 \mu + 4) + 4 H_e (1 - 2 \mu)}{12 a H_e^2 \mu (1 - \mu)^2 (1 - 2 \mu) - a \mu (2 a^2 (\mu^2 - 8 \mu + 4) + 4 H_e (1 - 2 \mu))} \right) \\
N &= \frac{12 \mu (1 + \mu) (1 - 2 \mu)}{E_i}
\end{align*}
\] (30)

Therefore, based on the field-measured roof deformation data, the revolving angle of fractured rock B can be inversely calculated by using Equation 30; then, this angle can be substituted into the determination equations for the three stability conditions, and the results of the stability condition of fractured rock B can finally be obtained. Since the deformation of the roof surface is easily measurable and is readily accessible at any time, the proposed approach can be considered to have attained the goal of “visualizing” the roof stability condition. The field measurement of the roof deformation allows for the real-time “visualization” of the roof stability condition so that an appropriate measure will be taken whenever a problem is found, which is crucial for guiding the roof support.

2) Visualized criteria for the determination of pseudostability

Similarly, the above criteria for the determination of pseudostability can be transformed into visualized criteria expressed by the revolving angle of fractured rock B, of which more details are given below:

\[
\begin{cases} 
A_1 \sin^2 \theta + B_1 \sin \theta + C_1 \geq 0 \\
A_2 \sin^2 \theta + B_2 \sin \theta + C_2 \leq 0.
\end{cases}
\] (28)

3) Visualized criteria for the determination of instability

Similarly, the above criteria for the determination of instability can be transformed into visualized criteria expressed by the revolving angle of fractured rock B, of which more details are given below:

\[
A_2 \sin^2 \theta + B_2 \sin \theta + C_2 \geq 0.
\] (29)

However, it is not enough to simply transform the theoretical criteria into visualized criteria as expressed by the revolving angle of fractured rock B. Although the roof stability condition can be determined by the revolving angle of fractured rock B, there is no convenient and fast method available to detect the revolving angle of fractured rock B over time, especially in real time during extraction on the mining face. If the abovementioned criteria for the determination of stability are used to guide field engineering, they must be transformed into metrics that are readily measurable on site. Based on the author’s research results concerning the roof deformation characteristics under NPM and its impact factors, the deformation of the roof surface is closely correlated with the revolving angle of fractured rock B. Therefore, we can further establish the relationship between the stability of the roof surface of the roadway and the revolving angle of fractured rock B as

\[ \nu_{\text{max}} \]
the S1201-II working face; the length of this face is 280 m, and the strike length is 2344 m. Meanwhile, the thickness of the coal seam is 3.81-4.35 m, its average thickness is 4.11 m, and its burial depth is 115-170 m. The existing conditions at the coal seam are stable, and the coalbed pitch is nearly horizontal. The roadway layout of the working face is shown in Figure 7. The headgate of S1201-II is tunneled in advance, and the air return way of S1201-II is formed by NPM without advance tunneling, producing a total length of 2344 m.

As shown in Figure 8, the main roof of the test working face is a medium-grained sandstone that is 5.4-21.5 m thick and has thick cross-bedding. The immediate roof is a siltstone that is 0.78-4.05 m thick and presents a gray thin layer with horizontal and wavy bedding containing detrital plant fossils. The immediate floor is a siltstone that is 1.8-16.3 m thick. A thin layer of fine sandstone, showing cleavage, is interbedded in the immediate floor. The main floor is a fine-grained sandstone that is 3.2-19.6 m thick and has wavy bedding.

The cutting depth of the test working face is designed to be 9.0 m, and the cutting angle is 10°. The roof of the roadway is permanently supported using constant-resistance and large-deformation anchor cables whose designed length is 10.5 m, with 5 cables per row and a row spacing of 0.8 m. In the equipment design of NPM, cable drilling rigs for roof support are all fixed to hydraulic supports. When roof cable parameters are designed, the support strength and the passage space between 5 cable drilling rigs should be taken into account simultaneously. Since the installation position of cable drilling rigs depends on many factors, such as a safe distance and operating space, the spacing between the cables may not be uniform. In this test, the spacing of roof anchor cables was determined to be 1245 m, 1295 m, 1230 m, and 1230 m successively. Within 0-160 m behind the working face, temporary support using a row of gangue supports that are arranged on the goaf side is provided. The working resistance of each support is 4000 kN, and the center distance of adjacent supports is 2.4 m. When the surrounding rock of the roadway stabilizes, these supports are gradually removed. The specific support parameters are shown in Figure 9.

5.2 Analysis results

According to the physical condition of the test mining face in the Ningtiaota Coal Mine and The Supplementary Exploration Geological Report of the South Side of Ningtiaota Minefield (No. 185 Team of Shaanxi Coalfield Geology Bureau, 2008), the field parameters applied to the abovementioned equations were chosen as follows: $S = 280 \text{ m}$, $L = 18 \text{ m}$, $H_c = 8.86 \text{ m}$, $A = 0.6$, $\varphi = 33°$, $K = 3$, $\gamma = 26x10^3 \text{ N/m}^3$, $H = 170 \text{ m}$, $c = 0.85 \text{ MPa}$, $P = 0.048 \text{ MPa}$, $H_B = 10.34 \text{ m}$, $K_G = 2.5 \text{ MPa/m}$, $\eta = 0.3$, $[\sigma] = 19.7 \text{ MPa}$, and $\tan \varphi_B = 0.3$. After the above parameters are substituted into the transformed criteria for the determination of stability (see Equations 26-29), the critical revolving angles for the determination of the three stability conditions can be obtained. Then, according to the relationship between the revolving angle of fractured rock B and the deformation of the roof surface (see Equation 30), let $a = 6.738 \text{ m}$, $l = 1.50 \text{ m}$, $E = 10.16 \text{ GPa}$, $\mu = 0.24$, $p_g = 22.80 \text{ kN/m}^3$, $p_s = 0.32 \text{ MPa}$, and $p_l = 1.07 \text{ MPa}$; therefore, the critical values for determination of the deformation on the roof surface under the three stability conditions can be calculated. The calculation results are given in Table 1.

According to Reference [23] the residual coefficient $K_r$ of the crack expansion of caved gangue is 1.16-1.39. When calculated according to the designed cutting height, $H_c = 8.86 \text{ m}$, if the cutting effect can fully meet the design requirements, then the revolving angle of fractured rock B after the compaction of the gangue should be $0.81° - 6.41°$. In summary, the following analysis can be performed:

1. According to the measured data of the roof deformation on site, after the roadway becomes stable, if the deformation is less than 93.2 mm (namely, the revolving angle of fractured rock B is less than $6.41°$), this indicates that the fractured high roof was the actual stability. The support force of the caved gangue can meet the stability conditions of fractured rock B; this support is considered to provide the best control state of the surrounding rocks. If the roof deformation after the roadway becomes stable is 93.2 -109.3 mm, the control effect of the roof caving height is poor, and the caved gangue cannot fill the goaf; however, few voids remain within the suspended roof, leading to a slightly larger revolving space for the fractured rock. Due to the limited size of the voids, fractured rock B began to come into contact with the caved gangue after it had slightly sunk, ultimately forming a fractured high roof under actual stability conditions.

2. If the roof deformation after the roadway becomes stable is 109.4-148.2 mm, the control effect of the roof caving height may be poor, and there are larger voids between the caved gangue and the suspended roof. In this state, fractured rock B cannot fully leverage the support force of the caved gangue to form a structure in equilibrium; stability is also maintained by the friction force between the fractured rocks. While fractured rock B is under the mining influence of the adjacent mining faces, the roof of the roadway may be subjected to severe deformation. In this condition, the fore support of the mining face should be reinforced prior to mining the adjacent faces to prevent the occurrence of more severe deformation as a result of the loss of the interaction of the fractured rocks.

3. If the deformation of the roof after the roadway becomes stable is greater than 148.2 mm, which indicates that the
control effect of the caving height of the roof fails to meet the requirement, there will be large void between the caved gangue and the suspended roof; thus, fractured rock B will generate a large revolving angle. In addition, the large engagement between the fractured rocks may lead to deformation and instability, the interaction force between the rocks in the vertical direction will substantially decrease, and the roof structure will relies upon only the passive support provided by the caved gangue, the coal body, and the roadway bracket. This indicates that the roof has already entered the instability condition. Close attention should be paid to roof control under this state. The observation of the deformation of the surrounding rocks should be more carefully made, and the support for the roadway roof and the solid coal lane should be reinforced to protect fractured rock B from unexpected settlement and the roadway from instability.

To apply the above analysis results to field engineering, we selected the range of 0-1000 m of the S1201-II mining face with entry retainment in the Ningtiaota Coal Mine as the test zone. Measuring points were set up every 10 m within the whole test zone, and the deformation on the roof surface near the gob-side roadway was monitored and recorded. The measurements started from approximately 10 m behind the mining face and were recorded once a day. The measured results are shown in Figure 10.

According to the monitoring results shown in Figure 10, within the test range of 1000 m, the maximum roof deformation was 192 mm, and the average deformation was 127 mm. Seventeen monitoring points had roof deformations that were less than 109.4 mm (16.8%); 69 monitoring points had roof deformations of 109.4-148.2 mm (68.3%); and 15 monitoring points had roof deformations larger than 148.2 mm (14.9%). These results indicate that nearly 2/3 of the positions in the test range exhibited pseudostability, while only a few points exhibited actual stability or instability.

1) For the zone under actual stability conditions, as shown in Figure 10, the fractured high roof was considered to have formed a stable structure under the support of the caved gangue and the coal body. This stable structure is highly reliable and will not be damaged by external forces. As a result, we designed and provided the simplest means of support in this area. The specific parameters are as follows: the roof was supported with 5 constant-resistance and large-deformation anchor cables, and a row of concrete-filled steel tube pillars were arranged at the side near the goaf. The field support effect and deformation monitoring curves are shown in Figure 11. As shown in the field pictures and the monitoring curves, the roof and two roadsides in the zone under actual stability conditions were very flat, and there was basically no rib spalling on the solid coal side. The roof deformation tended to be stable, without significant growth over an extended period of time.

2) For the zone under pseudostability conditions, as shown in Figure 10, the field support effect and deformation monitoring curves are shown in Figure 12. As shown in the field pictures and monitoring curves, the roof and two roadsides in the zone under pseudostability were very flat, and there was basically no rib spalling on the solid coal side. The roof deformation tended to be stable, without significant growth over an extended period. However, based on the above results of the deformation, we considered that such a stable structure was not very reliable. If the roadway was subjected to an external force again, it might be easily damaged. Therefore, during mining on the adjacent faces, in addition to the use of the same support parameters as that for the zone under actual stability conditions, the temporary support shown in Figure 12 should also be provided within 50 m of the mining face for reinforcement of the support to prevent the high roof structure of the roadway from entering an instability condition due to the influence of mining on the adjacent faces.

3) For the zone under instability conditions, as shown in Figure 10, the whole fracture structure was already unstable due to the larger revolving angle of fractured rock B in the high roof. The roof under this state was indeed very dangerous. As shown in the monitoring curves presented in
The deformation of the roof surface at this point always tends to increase, unlike that of the roof under actual stability or pseudostability conditions, as mentioned earlier. The deformation of the roof at the two measuring points as described above increased and then tended to become stable. After the temporary support was removed, the deformation of the roof showed a continuous and significant increasing trend that quickly stabilized. This result indicates that the fractured high roof can maintain a mechanical equilibrium by itself and would not squeeze the immediate roof of the roadway into continuous deformation. However, in the instability zone, the deformation of the roof continuously increased but did not approach stability. Therefore, in the instability zone, two rows of densely distributed flexible formwork concrete pillars were used to provide reinforced support for the roadway to prevent the roadway roof from undergoing a large deformation.

6 | DISCUSSION AND CONCLUSIONS

This paper introduced the principles of high roof stability under NPM and high roof control and indicated that the interaction between the fractured rocks is not the only factor that maintains the stability of this structure. Any attempt to perform the stability analysis of a high roof should consider the effects of the support provided by the gangue and the coal body. In addition, if necessary, it is also required to consider
the roof support of a roadway. Given the combined actions of the said factors, we analyzed the stability characteristics of a fractured high roof, suggesting that the stability conditions of the fractured rock of a high roof include actual stability, pseudostability, and instability and provided the criteria for determining these three stability conditions.

To use the abovementioned theoretical criteria for determining field operations, the parameters that cannot be measured in these conditions were transformed into visualized data that can be directly or indirectly obtained in the field. Consequently, visualized criteria for the determination of the roof stability condition were further proposed. The proposed approach allowed
the detection and assessment of the stability condition of the fractured rock of a high roof over time. Consequently, the stability and safety of this structure can be predetermined so that an appropriate countermeasure could be applied in response to the changes in roof stability to ensure safety. The proposed approach was tested and found to be effective, and it can provide a reference for applications under similar conditions.

However, it cannot be denied that the proposed method for determination also has an obvious problem. There are too many parameters involved in the stability determination criteria, and some are empirical parameters, the uncertainty in which results in a large deviation of the calculated results. To solve this problem, we suggest that a mining face or a roadway section should be selected as a test zone in which the determination method and criteria can be tested for field engineering. The calculated results should be evaluated and corrected based on the monitoring data of the test zone before the results are extensively used on other mining faces or under similar ecological conditions. However, this limitation does not prevent the proposed approach from providing theoretical guidance for engineering practice.

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REFERENCES

1. Wang Q, Jiang B, Pan R, et al. Failure mechanism of surrounding rock with high stress and confined concrete support system. Int J Rock Mech Min Sci. 2018;102:89-100.
2. Wang Q, He M, Yang J, Gao H, Jiang B, Yu H. Study of a no-pillar mining technique with automatically formed gob-side entry retaining for longwall mining in coal mines. Int J Rock Mech Min Sci. 2018;110:1-8.
3. He M, Wang Y, Yang J, Zhou P, Gao Q, Gao Y. Comparative analysis on stress field distributions in roof cutting non-pillar mining method and conventional mining method. J China Coal Soc. 2018;03:626-637.
4. Wang Y, Gao Y, Wang E, He M, Yang J. Roof deformation characteristics and preventive techniques using a novel non-pillar mining method of gob-side entry retaining by roof. Cutting. Energies. 2018;11(3):627.
5. Qian M. Mining pressure and strata control. Xuzhou: China University of Mining and Technology Press; 2003.
6. Song Z. Practical ground pressure. Xuzhou: China University of Mining and Technology Press; 1988.
7. Wang C, Zhang C, Zhao X, Liao L, Zhang S. Dynamic structural evolution of overlying strata during shallow coal seam longwall mining. Int J Rock Mech Min Sci. 2018;103:20-32.
8. Huang B, Liu J, Zhang Q. The reasonable breaking location of overhanging hard roof for directional hydraulic fracturing to control strong strata behaviors of gob-side entry. Int J Rock Mech Min Sci. 2018;103:1-11.
9. Liu C, Li H, Mitri H, Jiang D, Li H, Feng J. Voussoir beam model for lower strong roof strata movement in longwall mining—case study. J Rock Mechanics Geotechnical Eng. 2017;9(6):1171-1176.
10. Li X, Ju M, Yao Q, Zhou J, Chong Z. Numerical investigation of the effect of the location of critical rock block fracture on crack evolution in a gob-side filling wall. Rock Mech Rock Eng. 2016;49(3):1041-1058.
11. Rezaei M, Hossaini MF, Majdi A. Development of a time-dependent energy model to calculate the mining-induced stress over gates and pillars. J Rock Mechanics Geotechnical Eng. 2015;7(3):306-317.
12. Fan K, Liang H, Ma C, Zang C. Non-harmonious deformation controlling of gob-side entry in thin coal seam under dynamic pressure. J Rock Mechanics Geotechnical Eng. 2014;6(3):269-274.
13. Zhu S, Feng Y, Jiang F. Determination of abutment pressure in coal mines with extremely thick alluvium stratum: a typical kind of rockburst mines in China. Rock Mech Rock Eng. 2016;49(5):1943-1952.
14. Feng F, Li X, Rostami J, Peng D, Li D, Du K. Numerical investigation of hard rock strength and fracturing under polyaxial compression based on Mogi-Coulomb failure criterion. Int J Geomech. 2019;19:040190054.
15. Wang P, Jiang L, Zheng P, Qin G, Zhang C. Inducing mode analysis of rock burst in fault-affected zone with a hard-thick stratum occurrence. Environmental Earth Sci. 2019;78(15):467.
16. Jiang L, Wang P, Zheng P, Luan H, Zhang C. Influence of different advancing directions on mining effect caused by a fault. Adv Civil Eng. 2019;2019:1-10.
17. Pu H, Huang Y, Chen R. Mechanical analysis for X-O type fracture morphology of stope roof. J China Univ Mining Technol. 2011;40:835-840.
18. Feng F, Chen S, Li D, Hu S, Huang W, Li B. Analysis of fractures of a hard rock specimen via unloading of central hole with different sectional shapes. Energy Sci Eng. 2019;7(6):2265-2286.
19. He M, Zhang X, Zhao S. Directional destress with tension blasting in coal mines. Procedia Eng. 2017;191:89-97.
20. Li Y, Hua X. Mechanical analysis of stability of key blocks of overlying strata for gob-side entry retaining and calculating width of roadside backfill. Rock Soil Mechanics. 2012;33(04):1134-1140.
21. Qu T. Study on the evolvement law and control technology of rock deformation around the gob-side entry of fully mechanized top-coal caving faces in deep mines. China University of Mining and Technology, 2008.
22. Hou C, Ma N. Stress in in-seam roadway sides and limit equilibrium zone. J China Coal Soc. 1989;04:21-29.
23. Li Y, Hua X. Mechanical analysis on the stability of surrounding rock structure of gob-side entry retaining. J China Coal Soc. 2017;09:2262-2269.
24. Su C, Gu M, Tang X, Guo W. Experiment study of compaction characteristics of crushed stones from coal seam roof. Chin J Rock Mech Eng. 2012;01:18-26.

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