Determination of the key parameters of high-position hard roofs for vertical-well stratified fracturing to release strong ground pressure behavior in extra-thick coal seam mining

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Abstract
Traditional methods of controlling hard roofs have a limited scope of action and cannot effectively release the strong ground pressure behavior (SGPB) induced by high-position hard roofs (HHRs) in extra-thick coal seam mining (ETCSM). Thus, an innovative control technology of fracturing HHRs by vertical-well stratified fracturing (VWSF) has been proposed. However, the key parameters of VWSF, namely fracturing horizon and fracturing thickness of HHRs, which have a significant influence on stope stability, remain uncertain. In this study, a mechanical model of the “coal wall–hydraulic support–gangue” support system is established by considering the effective loading acting on HHRs, through which modified expressions for the periodic breaking span and impact kinetic energy of the stope are deduced. Based on the self-bearing of bulking rocks, the stability principle of the surrounding rock, and energy dissipation theories, the criteria for determining the fracturing horizon and thickness of the HHR are obtained. Next, a numerical model of ETCSM, which accounts for the supporting effect of gangue, is constructed in FLAC3D. The support load and energy released by stratum breakage are determined through modeling under various hard roof parameters, thus verifying the correctness of the determination criteria. The results show that the energy released by a hard roof, average support load, and critical support load are positively correlated with thickness, and first increase before declining with respect to an increase in the fracturing horizon. The key parameters for a real coal mine are obtained by theoretical calculations and numerical simulations. A field application demonstrates that the support load and advance roadway deformation can be decreased using the proposed parameterization. This provides theoretical support for determining the key parameters of HHRs for VWSF and facilitates the widespread application of VWSF technology in HHR control.

Keywords
high-position hard roofs, microseismic monitoring, mine safety, strong ground pressure behavior, vertical-well stratified fracturing
1 | INTRODUCTION

The caving span of a hard roof is large, resulting in strong ground pressure (SGP) on the working face in extra-thick coal seam mining (ETCSM). Thus, the control of hard roofs is a challenging aspect of achieving safe and efficient coal mining. When the overburden contains hard roofs in ETCSM, breakages and instabilities in the multi-layer structures result in ground pressures of varying degrees.\(^1\)\(^2\) In particular, breakages and instabilities in high-position hard roofs (HHRs) result in strong ground pressure behavior (SGPB) and support failures in the working face and entry owing to their large caving span and hanging area (Figure 1).\(^3\)\(^5\) These hazards represent a severe danger to safe production and pose a new challenge to achieving hard roof control in ETCSM.\(^6\)\(^9\) According to incomplete statistics, up to 132 SGP activities (the shrinkage of the support pillar exceeding 600 mm) have been recorded in Tashan and Tongxin coal mines since the ETCSM in 2008, of which approximately 21 frame-pressing accidents occurred in stopes. These problems are not unique in China, and have been reported in India and the USA. Indeed, in the Chirimiri colliery and the Drift mine in Eastern Kentucky, the frequent occurrence of SGP led to cable bolt cutting, resulting in roof falls and at least four fatal accidents and two serious injuries during ETCSM.\(^10\)\(^11\) The high-position strata in large spaces have a complex structure and a wide range of ground pressure behavior in the presence of overlying hard roofs.\(^12\) The fracture and rotation of the high-position structure produces radial compression on the adjacent roadway, resulting in coal pillar stress of up to 3-4 times that of the original rock stress.\(^5\)\(^13\) Controlling HHRs and eliminating SGBP in ETCSM is a major problem that urgently requires a solution.

The problem of hard roof control has been hotly debated in research on mine disaster prevention.\(^14\)\(^16\) As hard roofs are notoriously difficult to collapse, control technologies have been studied since 1951, and a series of hard roof control theories and technological methods have been developed.\(^17\)\(^19\) At present, the most commonly used technologies are coal pillar supports, gob-side entry retention, goaf filling, a spatiotemporal coupling fracturing method using a nonexplosive expansion material, water injection to weaken the roof, presplitting blasting, deep hole blasting, energy-cavity blasting, and hydraulic slitting.\(^2\)\(^14\)\(^15\)\(^20\)\(^28\) The main ideas can be summarized as follows: (a) maintain the stability of the hard roof and support the stope space; (b) weaken the hard roof and reduce its rock mass strength; and (iii) transfer the stress or release the strain energy to eliminate the source of SGP.

To some extent, the existing hard roof control technologies inhibit the occurrence of SGP induced by hard roofs. However, there are obvious deficiencies in the prevention and control of SGP induced by HHRs, which are mainly reflected in the following aspects: (a) The locality of traditional methods cannot solve the problem of SGP induced by HHRs.\(^1\) Existing technologies are limited by the small underground working spaces and cannot apply high-performance equipment. Field construction is difficult, and the effect range is limited (mainly confined to strata in the range of 30-50 m from the coal seam). (b) The passivity of disaster prevention methods. Traditional methods are implemented after the formation of the working face system or roadway cross-section, lagging behind SGP. (c) The risks of the implementation process. Traditional methods are implemented in disaster risk areas, and SGP may appear in the process of implementation. (d) The timeliness of disaster prevention and control is poor. The functions of traditional methods have two aspects: Although they can inhibit the occurrence of SGP, they may also induce it. The timeliness of prevention and control cannot be guaranteed over a long period. (e) Contradiction with production. The construction and production of traditional methods cannot work in parallel, resulting in a contradiction between the implementation of suitable and the production process. (f) Labor-intensive nature of the work. Traditional methods are point-by-point construction techniques, which require considerable drilling, resulting in a large engineering effort.

In view of the problems faced by current hard roof control methods and the advantages of vertical-well stratified fracturing (VWSF), an innovative method of weakening HHRs for VWSF is proposed to reform roof structure integrity. For VWSF technology, there has been significant research in the fields of petroleum and shale gas, and the design of hydraulic fracturing parameters for vertical wells has matured.\(^29\)\(^33\) The application of VWSF for petroleum and shale gas is different from that in the presence of HHRs. The main purpose of VWSF for HHRs in coal mines is to control the SGBP in the stopes. The weakening effect of VWSF on SGBP induced by HHRs is closely related to the fracturing horizon, fracturing thickness, fracturing technology parameters, drilling technology parameters, and hydraulic support parameters. Fortunately, the design of the fracturing process parameters, vertical well parameters, and hydraulic support parameters is relatively mature. Therefore, the design of fracturing horizons and fracturing thickness for VWSF is one of the core issues of this technology, having an important impact on HHR control, SGP weakening, and the surrounding rock stability in stopes. VWSF is a new technology for HHR control, and there has been little research on its key parameters, especially the fracturing horizon and fracturing thickness of HHRs. This study takes a fully mechanized top-coal caving face in an extra-thick coal seam in the Tashan coal mine, Shanxi Province, China, as the engineering background. The key parameters of HHR for VWSF are systematically studied.
by combining theoretical analysis, numerical simulations, and field tests.

2 | PROPOSITION OF HHRS FOR VWSF TO RELEASE SGPB IN THE STOPE

The excavation of a fully mechanized top-coal caving face in an extra-thick coal seam redistributes the stress around the stope, and the overlying strata movement will form large and small structures above the stope (Figure 2). After the HHRs in the large structure break, their weight and the follow-up rock load \( q \) are transmitted to the coal wall and the hydraulic support through the lower roof. The hydraulic support and the coal wall share the roof rock mass and transfer load \( F_L \) in the small structure of the stope. When the high stratum is an HHR, a large lateral cantilever (key block B) is formed. The movement process of key block B determines the stability of the small structure. The high-position structural instability causes the near-field roofs to break. The linkage instability effect is the essential reason for the SGPB of a fully mechanized top-coal caving face in ETCSM.

Reducing the thickness or hanging length of key block B is a significant means of improving the HHR control effect. Therefore, in view of the shortcomings of traditional control methods, this paper proposes an innovative method of weakening HHRs by VWSF based on petroleum fracturing technology (Figure 3). A fracturing fluid is injected into the HHRs. Large artificial cracks are formed in the target horizons, reducing the integrity and strength of the rock. Additionally, the overlying roofs may collapse, reducing the weighting strength of the roof. Moreover, this prevents the concentrated stress release from the sudden and large-scale collapse of the roofs. In general, the vertical

FIGURE 1 Damage caused by SGPB in an extra-thick coal seam in the Tashan coal mine: (A) map of China, (B) Tashan coal mine, (C) 8216 working face, (D) large shrinkage of supports, and (E) roadway damage

FIGURE 2 Large structure overlying strata
stress of HHRs is lower than their horizontal stress. The hydraulic fracturing cracks in HHRs extend along the horizontal direction.

The effect of fracturing HHRs by VWSF is to generate hydraulic fractures over a wide range, weakening the integrity of HHR, and produce a layered structure. The weakened part of the HHR will collapse with the mining, releasing the weight and energy through HHR breakage. The gangue of the bulking rock can support the overlying roof, further reducing the effective stress transferred to the small structure (Figure 4).

3 | THEORETICAL ANALYSIS OF KEY PARAMETERS OF HHR FOR VWSF

3.1 | Mechanical model of “coal wall–hydraulic support–gangue” support system for HHRs

3.1.1 | Construction of mechanical model of the “coal wall–hydraulic support–gangue” support system for HHR

According to the above analysis, a mechanical model of the “coal wall–hydraulic support–gangue” support system of HHR is constructed, as shown in Figure 5.

In Figure 5, \( h_c \) is the residual thickness of HHR, \( M \) is the coal seam thickness, \( H \) is the mining depth, \( h_0 \) is the height of HHR from the surface, \( H_L \) is the height of HHR from the coal seam, and \( q \) is the uniform load acting on the key block. \( F_L \) is the load transferred from the overlying roofs to the coal wall and hydraulic support. \( T_E \) is the extrusion pressure between key blocks A and B, \( T_H \) is the extrusion pressure between key blocks B and C, \( Q_E \) is the friction force between A and B, \( Q_H \) is the friction pressure between B and C, \( P_b \) is the support strength of coal wall, \( P_z \) is the support load, \( P_g \) is the pressure of gangue on key block B, and \( F_1 \) is the force exerted by the lower roofs on key block B.

FIGURE 3 Schematic diagram of VWSF system

FIGURE 4 Stress characteristics before fracturing (A) and after fracturing (B)
3.1.2 Effective load acting on the HHR

The subsidence of key block B \( s_x = S_0 + x \sin \theta \). The compression amount of gangue \( s_y \) is.

\[
s_y = S_y - \left\{ M - \left[ M (1-\delta) K_M + (H_L + h - h_c) (K_L - 1) \right] - S_0 \right\}
\]

(1)

where \( \delta \) is the recovery rate of working face; \( K_M \) and \( K_L \) are the crushing expansion coefficients of coal and rock, respectively; \( \theta \) is the rotation angle of key block B; and \( S_0 \) is the subsidence of key block B at the fracture. \( S_0 \) is small before the lower strata is fractured, while \( S_0 \) is large after the lower strata is fractured.

The supporting force produced by gangue per unit area is.

\[
f_1 = K_G s_y
\]

(2)

where \( K_G \) is the supporting coefficient of gangue.

When the gangue is touched at first, the rotation angle of the key block B is.

\[
\theta_0 = \arcsin \left( \frac{M - M (1-\delta) K_M - (H_L + h - h_c) (K_L - 1) - S_0}{x} \right)
\]

(3)

If \( \theta \leq \theta_0 \), key block B has no contact with gangue, and \( P_g = 0 \); if \( \theta > \theta_0 \), key block B has touched the gangue. If \( s_y = 0 \), let \( s_y = 0 \), and then the abscissa of key block B just touches the point.

\[
a_0 = \frac{M - \left[ M (1-\delta) K_M + (H_L + h - h_c) (K_L - 1) \right] - S_0}{\sin \theta}
\]

(4)

Therefore, the supporting stress of gangue is.

\[
P_g = \int_{a_0}^{L_1 \cos \theta} K_G s_y \left[ -\frac{2 (x-L_1)}{\tan \theta} \right] dx
\]

(5)

According to key strata theory, after fracturing, the load on HHR becomes,

\[
q_c = \frac{Eh_c^3 \left[ \gamma_1 h_1 + \gamma_2 h_2 + \cdots + \gamma_n h_n \right]}{Eh_l^3 + E_1 h_1^3 + \cdots + E_n h_n^3}
\]

(6)

where \( h_1, h_2, \ldots, h_n \) are the thickness of the overburden strata over the HHR, respectively, and \( \gamma_1, \gamma_2, \ldots, \gamma_n \) are unit weight of the overburden strata.

After separated fracturing, the effective loading acting on HHR becomes.

\[
q' = \frac{q_c L_1 - P_g}{L}
\]

(7)

where \( L \) is the periodic weighting interval of HHR before fracturing and \( L_1 \) is the periodic weighting interval of HHR after fracturing.

The supporting stress of the gangue formed by fracturing is the direct reason for the reduction in the roof pressure of the working face, as shown in equation (7). The supporting effect of the gangue from the bulking rock on the roof decreases with distance from the hydraulic fractures. Therefore, the influence of hydraulic fractures has a certain range, and this influence is more obvious closer to the hydraulic fractures.

3.1.3 Periodic fracture span of HHR

The HHR satisfies the thin-plate theory requirement, and the roof breakage problem can be studied according to Kirchhoff’s hypothesis. Based on the theory of plates and shells, the HHR for a continuous mining face can be approximately regarded as an elastic rectangular thin plate with two clamped edges and two simply supported edges, as shown in Figure 6.

In Figure 6, the inclined overhang length is \( L_2 \). The perturbation expression of HHR with two clamped edges and two simply supported edges is.

\[
\text{FIGURE 6} \quad \text{Structural model of HHR}
\]
where \( D \) is the flexural rigidity of the hard roof.

As the HHR is far away from the coal seam, a fracture form of transverse “OX” type will be produced, and the upper surface of the solid supporting edge will reach the tensile limit \( t \) during the bending sinking process.\(^\text{40,41} \) The calculation formula for the broken span of the HHR\(^\text{38} \) is:

\[
L_1 = \sqrt{\frac{-B - \sqrt{B^2 - 4AC}}{2A}}
\]

(9)

where.

\[
\begin{align*}
A &= 12\pi^4 - 675 - \frac{8892q' L_2^2 (\pi^2 - 4)^2}{25\pi^3 h_0^2 \sigma_t} \\
B &= 2 \left( 2\pi^2 - 3 \right) L_2^2 \\
C &= 12\pi^4 - 675 \right) L_2^4
\end{align*}
\]

3.2.1 Determination principle for the fracturing horizon of HHR

A reasonable fracturing horizon allows the rock strata in the fracturing range to fully collapse and fill the whole goaf, providing a better supporting role for the upper strata. Moreover, the impact load of HHR breakage and the disturbance of HHR rotary subsidence on the working face are reduced by VWSF. Furthermore, the SGPB in the stope is effectively weakened.

Criterion for the HHR supported by caved rocks

Before fracturing, the criterion for the HHR supported by caved rocks is as follows:

\[
\Delta = \frac{121275q_{xy}}{32768L_1 L_2 D} \left( x_1^2 - L_1^2 \right)^2 \left( y_1^2 - L_2^2 \right)^2 - \left\{ M - \left[ M (1 - \delta) K_M + H_L \left( K_L - 1 \right) \right] \right\} \geq 0
\]

(15)

where \( \Delta \) is the free space height\(^\text{44} \), \( K_L \) is the average residual fragmentation coefficients of strata, which is usually 1.06-1.15; when \( \Delta \) reaches the critical value, the HHR will be supported by caved rocks.

Therefore, the highest level of fracturing horizon is

\[
H'_L = \frac{M \left\lfloor 1 - (1 - \delta) K_M - w \right\rfloor}{K_M - 1}
\]

(16)
Criterion for critical support force

The hydraulic supports are located under key block B. The longer cantilever causes the hydraulic supports to bear excessive additional load and damage, thus affecting the deformation and instability of the surrounding rock structure of the stope. Key block B is very important for the stability of the stope. Stress analysis of key block B is illustrated in Figure 7.

By \( \Sigma M_{EF} = 0 \), we obtain the following:

\[
\frac{qL^2L_2}{2} = F_1x_1 + P_sx_2 + T_H(h - L_1 \sin \theta - e) + Q_HL_H
\]

(17)

where \( x_1 \) is the moment of \( F_1x_1 \); \( x_2 \) is the moment of \( P_sx_2 = s + d + \tan aH_L; s \) is the limit equilibrium zone size on one side of the roof wall in the mining space; \( d \) is the roof control distance; \( L_H \) is the moment of \( Q_H, L_H = L_1 \cos \theta - (h - L_1 \sin \theta - e) \sin \theta ; \gamma \) is the average bulk density; \( e \) is the contact point of block A and C; and for block \( B, e = \frac{(h-L_1 \sin \theta)}{2} \).

According to the theory of stress balance in a loose medium, the following is obtained,

\[
s = \frac{M\psi}{2 \tan \theta_0} \ln \left( \frac{k\psi H + \frac{c_0}{\tan \theta_0} + P_s}{\psi} \right)
\]

(18)

where \( \psi \) is the lateral pressure coefficient, \( \psi = \frac{\mu}{1-\mu} \), \( \mu \) is Poisson's ratio; \( \theta_0 \) is the internal friction angle of the coal seam, \( c_0 \) is cohesive strength; \( P_s \) is the supporting strength of the solid coal side; \( k \) is the maximum stress concentration coefficient; and 2-3.5 is selected according to the following experience.

According to formula (1), the friction force between B and C is.

\[
Q_H = \frac{qL^2L_2}{2} - 2T_H(h - L_1 \sin \theta - e) - 2F_1x_1 - 2P_sx_2
\]

(19)

The thrust of key block C to key block B can be calculated according to formula (3).\(^{46}\)

\[
T_H = \frac{qL^2L_2}{2(h - L_1 \sin \theta)}
\]

(20)

Take \( \Sigma F_x = 0 \); then,

\[
T_H = \frac{qL^2L_2}{2(h - L_1 \sin \theta)}
\]

(21)

Take \( \Sigma F_y = 0 \); then,

\[
Q_E = \frac{qL^2L_2}{2} + Q_H - P_s - F_1
\]

(22)

The increase in thickness of one-time mining coal seams in the fully mechanized caving faces of extra-thick coal seams produces an increase in the mining disturbance range. Additionally, the surrounding rock of the working face changes from “rotary instability” to “sliding instability”.\(^{47}\) According to the “S-R” stability theory of voussoir beam structures, the stability of key block B is affected by key block A.\(^{48}\) To prevent key block B from sliding and losing stability along the fracture position, the following condition must be satisfied:

\[
T_E \tan (\varphi - \beta) > Q_E
\]

(23)

where \( \varphi \) is the friction angle, and \( \beta \) is the breaking angle.

According to formulas (21), (22), and (23), the available supporting force \( (F_1) \) needs to meet the formula (24) so that key block B can keep its balance.

\[
F_1 \geq \frac{2qL_1(L_H + x_1)}{L_H + x_1} - \frac{qL_1(2 \tan (\varphi - \beta)h - L_1 \sin \theta)}{(L_H + x_1)(h - L_1 \sin \theta)} - \frac{(L_H + x_1)p_s}{L_H + x_1}
\]

(24)

According to the condition of mechanical equilibrium, \( P_z \) is as follows:

\[
P_1 = Q_E - T_E \tan (\varphi - \beta)
\]

(25)

The formula for calculating the minimum support load when the key block B does not slide and lose stability can be obtained as follows:

\[
P_Z = Q_E - T_E \tan (\varphi - \beta) + K_sF_1 + F_L - \int_0^L \sigma_s dx
\]

(26)

where \( K_s \) is the load transfer coefficient.

According to the stress equilibrium theory of loosened material, the width \( s \) of the stress limit equilibrium zone and seam interface stress \( \sigma_s \) are, \(^{14}\) respectively.

\[
s = \frac{M\lambda}{2 \tan \theta_0} \ln \left( \frac{k\psi H + \frac{c_0}{\tan \theta_0} + P_s}{\frac{c_0}{\tan \theta_0} + \frac{P_s}{\lambda}} \right)
\]

(27)
\[ \sigma_y = \left( \frac{c_0}{\tan \varphi_0} + \frac{P_x}{k} \right) e^{-\frac{2m \mu_0}{M}} - \frac{c_0}{\tan \varphi_0} \]  \hspace{1cm} (28) 

where \( \varphi_0 \) is the internal friction angle of the interface between the coal seam and the roof; \( k \) is the lateral pressure coefficient; \( \mu \) is the Poisson’s ratio; \( c_0 \) is the cohesion of the interface between the coal seam and the roof.

The load transferred to coal and rock is as follows:

\[ F_L = \sum_{j=1}^{m} K_{ij} (h_i + h_j) \gamma L_x \]  \hspace{1cm} (29) 

where \( F_{ij} \) is the load of the \( j \)th key strata block transferred onto the coal seam on one side, \( j = 1 \)–\( m \)–1, \( m \) is the number of key strata above the coal seam; \( K_{ij} \) is the load transfer coefficient of the \( j \)th key strata; \( h_i \) and \( h_j \) are the average thickness of the \( j \)th key strata and its controlled strata, respectively; \( l_j \) is the length of the \( j \)th key strata block; \( \gamma \) is the average volume weight of strata.

Therefore, the critical support stress of hydraulic supports for key block \( B \) to remain stable is:

\[ P_z = Q_x - T_x \tan(\varphi - \beta) + K_p F_1 + \sum_{j=1}^{m} K_{ij} (h_i + h_j) \gamma B - \int_0^L \sigma_y x \, dx \]  \hspace{1cm} (30) 

The design support strength of the hydraulic support is \([P]\). If \( P_z \) is greater than \([P]\) (satisfying equation (31)), the hydraulic support column will shrink too much during the SGPB, and key block \( B \) will slide and lose stability. In this scenario, stratified fracturing measures must be taken.

\[ P_z > [P] \]  \hspace{1cm} (31) 

**Criterion for the impact kinetic energy of the working face**

Too much energy in the HHR breakage will lead to rock burst in the working face. If equation (32) is satisfied, the HHR should be fractured by VWSF.

\[ \left[ \frac{187L^1_1L^2_1q^2}{97D (628L^1_1 + 134L^1_2 + 227L^2_2)} \right] \left( \frac{H_L}{\sin \alpha} \right)^{-w} > 10^7 \]  \hspace{1cm} (32) 

In summary, considering the construction efficiency, engineering effort, and cost input, it is not necessary to fracture all HHRs in practice. Once suitable parameters have been determined, the highest fracturing horizon of HHR is given by equation (16). The hard roofs near the highest horizon are then determined. And the relevant parameters are substituted into equations (31 and 32). As long as one of these expressions is satisfied, the HHRs should be fractured by VWSF.

### 3.2.2 Determination principle of fracturing thickness of HHR

**Criterion for HHRs supported by caved rocks**

We make full use of the characteristics of rock bulking, so that the strata in the fracturing fully collapse and fill the whole goaf, providing a better supporting role for the upper strata. Moreover, the impact load of HHR breakage and the disturbance of HHR rotary subsidence on the working face are reduced by VWSF. After fracturing, the criterion for the HHR supported by caved rocks is as follows:

\[ \Delta = \frac{121275 \gamma q x y \left( x^2 - L_1^1 \right)^2 \left( y^2 - L_2^1 \right)^2}{32768 L_1 L_2 D (45L_1^1 + 44L_2^1 + 45L_2^2)} - \left\{ M - \left( 1 - \delta \right) K_M + \left( H_L + h_c \right) (K_L - 1) \right\} \leq 0 \]  \hspace{1cm} (33)

When \( \Delta \) reaches a critical value, the HHR will be supported by caved rocks. Therefore, the minimum fracturing thickness of the HHR is:

\[ h_1 = h - h_c = h + H_L - \frac{M \left[ 1 - (1 - \delta) K_M \right] - w'}{K_p - 1} \]  \hspace{1cm} (34)

**Criterion for critical support force**

The purpose of fracturing the HHR is to ensure that the critical support force of the hydraulic supports \( (P'_z) \) after fracturing is not greater than \([P]\), that is.

\[ P'_z < [P] \]  \hspace{1cm} (35)

**Criterion for the impact kinetic energy of the working face**

To prevent SGP from occurring following a mine earthquake caused by HHR breakage, \( U_i \) should be less than \( 10^7 \) J.48 Thus,

\[ \left[ \frac{187L^1_1L^2_1q^2}{97D (628L^1_1 + 134L^1_2 + 227L^2_2)} \right] \left( \frac{H_L}{\sin \alpha} \right)^{-w} < 10^7 \]  \hspace{1cm} (36)

Theoretically, smaller values of \( h_c \) are preferable. However, considering the construction efficiency, engineering effort, and cost input, the reasonable HHR fracturing thickness can be obtained by simultaneous consideration of equations (34-36) once the relevant parameters have been determined.
4 | NUMERICAL SIMULATIONS OF KEY PARAMETERS OF HHR FOR VWSF

4.1 | Numerical simulation for ETCSM

To verify the theoretical analysis and obtain the optimal HHR parameters for VWSF, according to the on-site geological conditions of working face 8101 at Tashan coal mine (Figure 19C), a ETCSM numerical simulation method is used to establish a numerical calculation model by means of FLAC3D (Figure 8). This allows the key parameters of HHR for VWSF to be analyzed.

In the mechanical calculation, the rock strata was described by the continuum damage model. For the cave-in zone, a new consolidation model was developed based on a confined compression test on loose ceramic sand. The rock damage was used to determine the roof caving, meanwhile a local contact criterion was introduced to confirm the roof contact with the cave-in zone. The cave-in zone height is calculated as the product of the caved roof thickness and the bulk factor. In order to calculate the hydraulic support load numerically, a support element at the bottom corner of the working face is generated to approximately model the mechanical behavior of the hydraulic power support. According to the elasto-plasticity theory, a special module was developed to calculate the released energy. This model is 1,600 m long and 540 meters high, and it consists of units 52 890. On the upper part of the model, the free boundary is adopted, and the lower part of fixed boundary controlled in horizontal displacement. In order to eliminate the boundary effect, 200 m is reserved on each side of the model. 3 m of mining is simulated each time, and the mining range from cutting to 1200 m is simulated. The ground stress field is inverted according to the on-site measured geo-stress data. The rock mechanics parameters of the model are presented in Table 1.

The simulation results are shown in Figures 9 and 10. At the initial stage of mining (Figure 9A and 9), the low-position roof strata mainly suffer from shear failure. After the working face is advanced to 960 m, the high-position roof strata mainly suffer from tensile failure (the black rectangular range of Figure 9E and 9). The failure height of the roof strata is positively correlated with the goaf area. Finally, the caving zone height is maintained within 68.35 m and the fracture zone height is 117.46 m (Figure

**FIGURE 8** Geometric model

**TABLE 1** Physical and mechanical parameters of coal and rock mass

| Lithology          | Density (kg/m³) | Bulk Modulus (GPa) | Shear Modulus (GPa) | Cohesion (MPa) | Internal Friction Angle (°) | Tensile Strength (MPa) |
|--------------------|-----------------|--------------------|---------------------|----------------|---------------------------|------------------------|
| Cap rock           | 2330            | 6.7                | 5.0                 | 5.0            | 30                        | 4.5                    |
| Fine-grained sandstone | 2535        | 10.6               | 11.5                | 15.7           | 47                        | 7.8                    |
| Medium-grained sandstone | 2526       | 7.2                | 6.1                 | 6.8            | 31                        | 7.0                    |
| Coarse-grained sandstone | 2519        | 6.7                | 5.0                 | 6.0            | 31                        | 5.5                    |
| Sandy mudstone     | 2595            | 13.9               | 9.6                 | 5.5            | 33                        | 5.2                    |
| Mudstone           | 2654            | 14.3               | 8.6                 | 4.9            | 34                        | 4.8                    |
| Glutenite          | 2700            | 10.2               | 7.6                 | 5.2            | 37                        | 4.2                    |
| Coal               | 1426            | 2.6                | 1.1                 | 9.5            | 30                        | 2.6                    |
| Base rock          | 2590            | 12.3               | 10.0                | 8.5            | 31                        | 5.2                    |
10). The results in Figure 9A, 9, 9, and 9 indicate that shear sliding occurs on the roof directly above the working face, whereas the HHRs are mainly cracked through tensile failure in the nearly vertical direction.

Figure 9B, 9, 9 shows that stress-concentrated areas gradually appear in the surrounding rocks on both sides of the goaf. After the working face is advanced to 960 m, a stress-concentrated area forms in the HHR (layers #21 and #27; white rectangular range of Figure 9F and 9). As shown in Figure 9F and 9, the loose gangue densifies and begins to withstand the strata pressure from the HHRs. This is the result of the full collapse of the roof and the filling of the goaf, which transfers stress to the working face.

In the numerical modeling, the support load is also registered and compared with the measured results in Figure 11. The comparison reveals a similar tendency of the support load which is significantly violent with high peak load over 36.86 MPa (Figure 11).

Strata breakage is an energy release process. Figure 12 shows that the energy released by strata breakage declines with distance from the roof. The energy released by hard roofs is remarkably larger than that of the neighboring strata.
The energy released from the HHRs (layers #21, #27, and #38) is similar to that released by the near-field hard roofs (layers #7 and #12), revealing that lithology has a significant impact on energy release. Moreover, far away from the coal seam, the large thickness of layer #33 leads to a big energy release, which means that thickness also plays an important role in the energy release of rock strata.

In short, strata with characteristics of being close to the working face, strong rigidity, and large thickness produce an intense energy release. When any two of these characteristics are combined, the intense energy release will be followed by a shock, which will cause SGP to occur in stopes. Therefore, when fracturing HHRs, those with large energy releases should be prioritized. In working face 8101, all the HHRs with such characteristics are in layers #21, #27, and #38.

4.2 | Numerical simulation of hard roof horizon

To compare and analyze the distribution of energy released by stratum breakage and the support load under different conditions, hard roof horizons of 56 m (layer #12), 110 m (layer #21), 145 m (layer #27), 220 m (layer #38), and 270 m were
adopted in the simulations. The thickness of the hard roof was set to 15 m. The numerical simulation results are shown in Figures 13 and 14.

Other conditions being the same, the energy released by hard roof breakage first increases and then decreases with distance from the working face, as shown in Figure 13. The energy released by the hard roof breakage at 110 m and 145 m horizons is much greater than that of the other hard roof horizons. As can be seen from Figure 13, the theoretical calculated energy is slightly larger than that given by numerical simulations, but the overall trend is similar. The impact kinetic energy of the working face caused by the breakage of the hard roof horizons from 56 to 220 m is greater than $10^5$ J, which indicates that measures should be taken in KS1-KS4.

Working face 8101 uses a ZF15000/27.5/42 hydraulic support. The rated support force is 15 000 kN, corresponding to the rated support load of 36.86 MPa. As shown in Figure 14A, the support load is less than 36.86 MPa and greater than 26.67 MPa, accounting for 0.82% of the breakage and instability of the hard roof at the 56 m horizon. Figure 14B shows that the support load is greater than 36.86 MPa (accounting for 2.41%) and greater than 37.37 MPa (accounting for 0.83%) of the breakage and instability of the hard roof at the 110 m horizon. Figure 14C shows that the support load is greater than 36.86 MPa (accounting for 8.84%) and greater than 40.69 MPa (accounting for 0.41%) of the breakage and instability of the hard roof at the 145 m horizon. Figure 14D shows that the support load is greater than 36.86 MPa (accounting for 2.81%) and greater than 37.81 MPa (accounting for 1.61%) of the breakage and instability of the hard roof at the 220 m horizon. As shown in Figure 14E, the support load is less than 36.86 MPa and greater than 35.81 MPa (accounting for about 5.22%) of the breakage and instability of the hard roof at the 270 m horizon. These results indicate that the theoretical analysis is consistent with the numerical simulations.
to the HHR thickness and that the critical support load obtained by theoretical analysis is consistent with the simulation analysis.

Figure 18 shows that the average support load and the critical support load are positively correlated with the hard roof thickness. The critical support load when the thickness is above 5 m is greater than the rated support load, which may result in overloading and SGPB in stopes. The results show that lower values of the hard roof thickness lead to smaller energy release by strata breakage, thus reducing the influence on the working face.

In summary, the hard roof horizon and thickness have a significant impact on the energy release and support load. When the hard roof thickness is less than 5 m and the horizon is 150 m, the energy released by hard roof breakage is less than $10^5$ J, satisfying the criterion for the impact kinetic energy of the working face. At the same time, the stress field of the surrounding rock is optimized.

5 | FIELD EXPERIMENT

5.1 | Engineering background

The site of the field test was selected as the 8101 working face in Tashan coal mine. The average coal seam thickness of the working face is 20.08 m. The coal seam dip is 2°, and the coal seam depth is 465.72 m. The length of the working face is 230 m, and the strike length is 1485 m. The working surface
is covered with 5 horizons of hard rock, which are mostly medium-fine sandstone and coarse gravel sandstone with compact and complete lithologies. For the 8101 working face, the KS3, KS4, and KS5 are HHRs within the range of 8-12 times the height of the No. 3-5 extra-thick coal seam (Figure 19C).

**5.2  Fracturing technology design**

**5.2.1  Determination of key parameters**

For these calculations, $M = 16.8$ m, $H = 467$ m, $b = 164.5$ m, $\mu = 0.29$, $\varphi = 30^\circ$, $k = 2.5$, $c_0 = 2.67$ MPa, $P_s = 0.73$ MPa, and $\gamma = 0.025$ MN/m$^3$. Considering the actual conditions of the 8101 working face, take $K_m = 1.2$, $K_g = 0.5$, $K_L = 1.15$, $\eta = 0.9$, $\tan (\varphi-\beta) = 0.35$, $\theta = 5^\circ$, $s_0 = 143.0$ mm, $\varphi_0 = 30^\circ$, $\lambda = 0.41$, $s = 12.1$ m, $x_1 = 8.9$ m, $\alpha = 31^\circ$, $H_L = 144.65$ m, $h = 9.75$ m, $q = 8.06$ MPa, and $\sigma_l = 10.8$ MPa, $[P] = 15,000$ kN. Use the calculated value $P_z = 16,558$ kN $> [P]$ and the fracturing heights $h_y = 9.6$ m for KS3 and $h_y = 10$ m for KS4.

**Determination of fracturing horizon for HHRs**

According to the on-site engineering geological conditions of working face 8101 in Tashan coal mine, $M = 16.8$ m and $K_p = 1.1$ according to equation (16).

$$H'_L = \frac{M \left[1 - (1 - \delta) K_M \right] - w}{K_p - 1} = 147.8 \text{ m}$$

The higher level of the fracturing horizon is layer #27, and so KS3 and KS4 are the fracturing ranges to be considered. Substituting the relevant parameters of KS3 and KS4 into equations (31 and 32), the calculation results show that $P_z = 16,558$ kN $> [P]$, $U_r > 1.20 \times 10^5$ J, satisfying the relevant conditions. Therefore, KS3 and KS4 need to be fractured by VWSF. At the same time, the numerical simulations verify the correctness of the theoretical results.

**Determination of fracturing thickness of HHR**

The minimum fracturing thickness of KS3 is.

$$h_{l_1} = h - h_c = h + H_L - \frac{M \left[1 - (1 - \delta) K_M \right] - w'}{K_p - 1} = 37.97 \text{ m}$$

Obviously, KS3 requires fracturing of the entire horizon, $h_{l_1} = 10.7$ m.

The minimum fracturing thickness of KS4 is.

$$h_{l_2} = h - h_c = h + H_L - \frac{M \left[1 - (1 - \delta) K_M \right] - w}{K_p - 1} = 4.9 \text{ m}$$
Substituting $h_{11}$ and $h_{12}$ into equations (35 and 36) gives $P_c = 14332 \text{ kN} > [P]$, $U_r > 7.4 \times 10^4 \text{ J}$, and the relevant conditions are satisfied. The numerical results also verify the rationality of the theoretical calculations.

According to the above results, when the fracturing thicknesses of KS3 and KS4 are 10.7 m and 4.9 m, respectively, the rock strata within the fracturing range can collapse in time and fill the whole goaf, providing better support for the upper strata. Moreover, the impact load of HHR breakage and the disturbance of HHR rotary subsidence on the working face are reduced by VWSF. Therefore, the fracturing thicknesses of KS3 and KS4 are determined to be 10.7 m and 4.9 m, respectively.

5.2.2 | Fracturing technology

The overall structure of the fracturing borehole requires three drilling operations. One of these is into the stable bedrock, the second goes to 120 m, and the third reaches to the No. 3-5 coal seam. The design data of the well bore structure are presented in Table 2, and the structure of the fracturing well is shown in Figure 20. The site construction area is shown in Figure 21.

Fracturing was performed twice, the fracturing of the first horizon lasted 140 min, and the initiation pressure was 24.48 MPa. During the crack stability expansion stage, the tubing pressure was 7.08-9.47 MPa, and the casing pressure...
**FIGURE 18**  Peak value of support load under different hard roof horizons

![Graph showing peak value of support load under different hard roof horizons.](image)

**FIGURE 19**  Mine location (A), panel layout (B), and generalized stratigraphy column (C) of the test site

![Mine location map and test site layout with stratigraphy columns.](image)

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### Table: Generalized Stratigraphy Column (C)

| No. | Lithology       | Burial depth (m) | Thickness (m) | Hard rock | Columnar |
|-----|-----------------|------------------|---------------|-----------|----------|
| 40  | Medium sandstone| 227.81           | 5.1           | KS4       |          |
| 39  | Siltstone       | 232.78           | 4.7           |           |          |
| 38  | Coarse sandstone| 237.51           | 8.2           | KS5       |          |
| 37  | Coarse sandstone| 245.71           | 3.9           |           |          |
| 36  | Medium sand stone| 249.61         | 8.1           |           |          |
| 35  | Fine sandstone  | 257.61           | 9.4           |           |          |
| 34  | Coarse sandstone| 267.01           | 4.9           |           |          |
| 33  | Siltstone       | 271.91           | 11.1          |           |          |
| 32  | Siltstone       | 283.01           | 7.7           |           |          |
| 31  | Coarse sandstone| 290.71           | 2.8           |           |          |
| 30  | Medium sandstone| 293.51           | 6.5           |           |          |
| 29  | Coarse sandstone| 300.01           | 5.1           |           |          |
| 28  | Glutinite       | 305.01           | 6.3           |           |          |
| 27  | Fine sandstone  | 311.32           | 9.8           | KS4       |          |
| 26  | Mudstone        | 321.07           | 4.9           |           |          |
| 25  | Coarse sandstone| 325.91           | 5             |           |          |
| 24  | Medium sandstone| 330.91           | 4.8           |           |          |
| 23  | Glutinite       | 335.71           | 7.2           |           |          |
| 22  | Mudstone        | 342.91           | 2.3           |           |          |
| 21  | Fine sandstone  | 345.20           | 10.7          | KS3       |          |
| 20  | Siltstone       | 355.85           | 7.7           |           |          |
| 19  | Coarse sandstone| 363.51           | 5.7           |           |          |
| 18  | Glutinite       | 369.23           | 3.3           |           |          |
| 17  | Medium sandstone| 372.51           | 6.8           |           |          |
| 16  | Siltstone       | 379.34           | 15.1          |           |          |
| 15  | Fine sandstone  | 394.46           | 5.6           |           |          |
| 14  | Mudstone        | 400.06           | 5.8           |           |          |
| 13  | Glutinite       | 405.86           | 3.8           |           |          |
| 12  | Siltstone       | 405.86           | 9.1           | KS2       |          |
| 11  | Fine sandstone  | 409.66           | 6.4           |           |          |
| 10  | Fine sandstone  | 414.91           | 3.2           |           |          |
| 9   | Glutinite       | 430.41           | 4.1           |           |          |
| 8   | Medium sandstone| 433.61           | 5.1           |           |          |
| 7   | Medium sandstone| 437.61           | 9.5           | KS1       |          |
| 6   | Sandstone       | 442.66           | 5.7           |           |          |
| 5   | Sandy mudstone  | 452.10           | 3.2           |           |          |
| 4   | Glutinite       | 457.75           | 4.8           |           |          |
| 3   | Mudstone        | 460.95           | 3.2           |           |          |
| 2   | No.3-5 coal seam| 465.72           | 16.8          |           |          |
| 1   | Coarse sandstone| 485.22           | 3.1           |           |          |
was 9.27-11.86 MPa (Figure 22A). The fracturing of the second horizon lasted 131 min; the formation initiation pressure of the second horizon of fracturing was 7.03 MPa. During the crack expansion and expansion stage, the tubing pressure was 6.38-7.03 MPa, and the casing pressure was 6.82-7.46 MPa (Figure 22B).

### 5.3 Microseismic monitoring of the fracturing effect

The ground microseismic monitoring technology was used to monitor the expansion direction of the crack formed by hydraulic pressure. This technology used a geophone to monitor the microseismic wave elicited by the hydraulic fractured well to describe the geometrical shape and space distribution of the crack extension. The energy diagrams of microseismic events were shown in Figure 23A and 23, and the orientation of hydraulic fracture was shown in Figure 23B and 23.

- According to Figure 23, the half-lengths of the dynamic fractures in the hydraulic fracture of the first horizon in the long axis direction were between 93 and 125 m, with an average of 109 m (Figure 23A). The half-lengths of the dynamic fractures in the short axis direction were between 30 and 50 m, with an average of 40 m. The ratio of the long axis to short axis was 1:0.37. The half-lengths of the dynamic fractures in the hydraulic fracture of the second horizon were between 85 and 109 m, with an average of 97 m (Figure 23D). The half-lengths of the dynamic fractures in the short axis direction were between 20 and 30 m, with an average of 25 m. The ratio of the long axis to the short axis was 1:0.26. The hydraulic fracture parameters were shown in Table 3.
- Figure 23B and 23 showed that the hydraulic fracture of the first horizon extended along the NE65° direction. The hydraulic fracture of the second horizon extended along the NW68° direction. The two hydraulic fractures were distributed in strike on the east and west wings with the vertical well as the center. The hydraulic fractures in the two wings were spread evenly and are basically distributed symmetrically.
- The direction of the 8101 working face was NE22°, and the positional relationship between the 8101 working face and the two horizons of hydraulic fracture was shown in Figure 23C. It showed that the hydraulic fracture of the first horizon had the largest coverage area of the working surface. The hydraulic fracture of the second horizon was located in the coverage area of the hydraulic fracture of the first horizon. The hydraulic fracture area of the working face is 245 m along the working surface, and the boundary of the affected area is 301 m away from the working face (Table 3).
FIGURE 21 Construction site of fracturing operation

FIGURE 22 Fracturing curve: (A) fracturing curve of the first horizon; (B) fracturing curve of the second horizon
Influence of the stratified fracturing HHR on the ground pressure distribution in the working face

Taking the No.76 hydraulic support in the middle of the working face as an example, the support load and the ground pressure characteristics of the nonhydraulic fracture area (210-355 m) and the hydraulic fracture area (355-490 m) were analyzed. The results were shown in Figure 24. The displacement characteristics of the roadway were analyzed by placing measuring points on the top and side of the roadway with the working face at 210-355 m and 355-490 m, respectively. The results were shown in Figure 25.

According to the statistics of the support load, the ground pressure strength and the span distance, the following rules were obtained:

- Hydraulic fracturing could reduce the ground pressure strength in the hydraulic fracture area. The maximum support load before the cutting seam was 41.3 MPa, and the maximum support load after the cutting seam was reduced to 35.5 MPa. The average support load decreased from 30.1 MPa to 24.5 MPa, which was reduced by approximately 18.6%.

- The hydraulic fracture increased the periodic roof weighting interval. After hydraulic fracturing, the maximum roof weighting interval and the periodic roof weighting interval increased. The periodic roof weighting interval increased by 32% at approximately 3.5 m.

- Before fracturing, SGP appears in the working face when the mining position was located near 280 m and 305 m, and the pressure lasted for 12 m and 23 m, respectively. The safety valve of the No. 45–No. 90 hydraulic support was opened in the large area, and the coal wall spalling

| Horizon | Fracture length (m) | Width of fracture area (m) | Orientation | Occurrence |
|---------|---------------------|---------------------------|-------------|------------|
|         | East | West | Total | East | West |                  |             |
| KS3     | 110  | 140  | 250   | 30-50| 20-30| NE65°            | Horizontal  |
| KS4     | 100  | 118  | 218   | 20-30| 20-30| NW68°            | Horizontal  |

5.4 Influence of the stratified fracturing HHR on the ground pressure distribution in the working face

Figure 23 Correspondence diagram between hydraulic fractures and the 8101 working face: (A) orientation diagram of hydraulic fracture in KS3; (B) energy diagram of microseismic event in KS3; (C) correspondence between hydraulic fractures and the working face; (D) orientation diagram of hydraulic fracture in KS4; and (E) energy diagram of microseismic event in KS4

Table 3 Hydraulic fracture parameters

- Hydraulic fracturing could reduce the ground pressure strength in the hydraulic fracture area. The maximum support load before the cutting seam was 41.3 MPa, and the maximum support load after the cutting seam was reduced to 35.5 MPa. The average support load decreased from 30.1 MPa to 24.5 MPa, which was reduced by approximately 18.6%.

- The hydraulic fracture increased the periodic roof weighting interval. After hydraulic fracturing, the maximum roof weighting interval and the periodic roof weighting interval increased. The periodic roof weighting interval increased by 32% at approximately 3.5 m.

- Before fracturing, SGP appears in the working face when the mining position was located near 280 m and 305 m, and the pressure lasted for 12 m and 23 m, respectively. The safety valve of the No. 45–No. 90 hydraulic support was opened in the large area, and the coal wall spalling
was serious in the working face. After fracturing, the ground pressure strength of the roadway was obviously reduced in the area affected by the hydraulic fracture. The SGP did not appear in the working face, and the opening rate of the hydraulic support safety valve was reduced by 37%.

5.5 Influence of the fracturing HHRs on advance roadway deformation

Figure 26 shows that the deformation of the advance roadway is obviously improved in the hydraulic fracture area. The height deformation of the roadway was reduced from an average of 990 mm to 486 mm. The deformation of the roadway width was determined by an average of 446 mm and was reduced to 213 mm. The single pillars in the roadway were not damaged, and the advanced support section of the mining roadway was highly controlled. The VWSF solved the problem that traditional underground measures cannot fracture HHR. This dynamic was highly important for controlling similar SGPB disasters caused by the instability of HHRs.

6 SUMMARY

- A mechanical model of a “coal wall–hydraulic support–gangue” support system for HHRs has been established. The effective load acting on the HHR was introduced, and...
modified expressions for the periodic breaking span ($L$) and impact kinetic energy ($U_r$) of the stope were deduced. The supporting effect of the gangue caused by fracturing is the direct reason for the reduction in the hydraulic support pressure. The reduction of the effective load on the HHR is the fundamental reason for the increase in $L$ and decrease in $U_r$.

- Based on the self-bearing of bulking rocks, the stability principle of the surrounding rock, and energy dissipation theories, the criteria for determining the fracturing horizon, and thickness of HHRs were obtained. These provide a theoretical basis for the selection of key parameters.

- A numerical model of ETCSM, which accounts for the supporting effect of gangue, was constructed using FLAC3D. This numerical model was used to perform numerical simulations of different hard roof horizons and thicknesses. The support load and energy released by strata breakage were determined, verifying the correctness of the determination criteria. The results show that the horizon and thickness of the hard roof have a significant impact on the energy release and support load. The results indicate that the main strata affecting the SGPB of working face 8101 are the HHRs located between 110 m and 220 m. Therefore, with regard to the working face 8101, the best fracturing horizons are KS3 and KS4. The optimal fracturing thickness of KS3 is 10.7 m, and that of KS4 is 4.9 m.

- The research results were applied to working face 8101 of Tashan coal mine. Microseismic monitoring showed that the KS3 hydraulic fracture extends 250 m with an azimuth of NE65° and that the KS4 hydraulic fracture extends 218 m with an azimuth of NW68°. The support load of typical measuring points in the hydraulic fracture zone was reduced by 18.6%, and the periodic roof weighting interval was increased by 32%. The average height deformation of the advance roadway in working face 8101 was only 0.53 m, and the average width deformation of the advance roadway was only 0.23 m. There was no SGP during the mining process. This provides a support for determining the key parameters of HHR for VWSF and facilitates the widespread application of VWSF technology in HHR control.

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CONFLICT OF INTEREST
None.

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