A new theoretical method for calculating front abutment stress during coal mining

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Abstract
The calculation of abutment stress has always been the most important topic in safe underground mining and design. In this paper, based on the formation mechanism of abutment stress, combined with the mechanical properties of actual strata and the theory of elastic foundation beams, a structure model of overlying strata with an elastic foundation is established, and a new theoretical calculation method of abutment stress is proposed. The new calculation method takes into account the interaction of the overlying key strata and the effect on the transmission of the front abutment stress and then analyzes the stress patterns of multilayered key strata under the combined action of the bending moment, shear force, and nonuniform load and its transmission to the underlying strata. In addition, based on field observations of panel 10 305 in the Xinglongzhuang coalmine, the influence range, peak position, and variation trend of the front abutment stress before and after breaking of the overlying key strata are studied and verified. The results show that the influence range of the abutment stress remains unchanged after KS1 breaks (eg, from 0 to 180 m), and its peak position shifts forward from 8 to 19 m, while the influence range (0-180 m) and its peak position (6-8 m in front of coal wall) remain unchanged after KS2 breaks; the abutment stress decreases, and the maximum decrease is 3.9 MPa in the range of 0-22 m ahead of the coal wall after KS1 breaks, while the maximum decrease in the abutment stress is 2.1 MPa in the range of 0-71 m ahead of the coal wall after KS2 breaks. The results of the new calculation method are basically consistent with the field observations and the existing theoretical results, showing that the method has good adaptability and reliability and can provide a new theoretical basis for on-site design and construction practices.

KEYWORDS
abutment stress, disturbance stress, key stratum, nonuniform load, transmission effect
1 | INTRODUCTION

The mining disturbance of the working face causes abutment stress in the surrounding rocks of the stope and roadway, and induces deformation and destruction of the surrounding rocks, even inducing dynamic disasters such as coal burst and rock burst. This has an impact on underground equipment and production systems and poses a threat to the safety of underground personnel. Hence, it is necessary to elucidate the law of mining disturbance’s influence on the abutment stress of the stope and roadway, which can provide some guidance for the safety of underground activities.

Based on the mention above, out-in-depth and detailed numerical studies on the distribution and variation in abutment stress around a stope by incorporating field monitoring, numerical simulation, and theoretical analysis have been carried. For instance, based on the analysis of anchor cable stress monitoring, support resistance monitoring, borehole stress gauge measurement, and microseismic monitoring results, the evolution law of front abutment stress in a working face was studied and summarized. By establishing a two-dimensional nonlinear numerical model with nonuniform characteristics, a distribution law of front abutment stress was simulated and analyzed. The front abutment stress was deemed to be composed of in situ stress and mining concentrated stress, and the former is a constant pressure, while the latter is caused by a periodic failure of the main roof, which is related to the main roof failure. By analyzing the five stages of the formation of abutment stress, a structural mechanics model of abutment stress was established. Based on the theory of an elastic foundation beam, a method for calculating abutment stress in overlying strata with double load zones was proposed. Generally, the influence range of front abutment stress caused by mining disturbance is 30-150 m, the stress concentration coefficient is 1.5-3, and the peak stress position is approximately 5-40 m ahead of the coal wall.

Previous studies reflect the influence of mining disturbance on the distribution and evolution of abutment stress to a certain extent but do not consider the interaction of overlying strata and the transmission effect of by the overburden key strata on front abutment stress. Hence, in this paper, a mechanical model of the elastic foundation of overlying strata is established based on the mechanical properties of actual strata and elastic foundation beam theory. Then, considering the interaction of the overlying key strata and the transmission effect on the front abutment stress, a new theoretical method for calculating the abutment stress is proposed. After that, the stress patterns of multilayer key strata under the combined action of bending moment, shear force, and nonuniform load and its transmission to underlying strata are analyzed. Finally, the results from field monitoring and previous studies are used to verify the results from the new method. These study results are expected to provide a new theoretical basis for the design and construction of coal mining.

2 | CALCULATION OF ABUTMENT STRESS BEFORE AND AFTER KEY STRATUM 1 BREAKS

With an increasing area of suspension of the overlying strata affected by mining disturbance, the overburden strata are continuously fractured, the stress is redistributed in the surrounding rocks, and the overlying load is applied to the coal and rock masses around the goaf. This moment, the loads of the coal and rock masses around the goaf, can be divided into two parts: (a) the self-weight of the overlying strata, which is the in situ stress and (b) the transfer effect of the bending or fracturing of the overlying strata, which is the disturbance stress. When the above two factors are superimposed, the front abutment stress is formed.

According to previous studies, the key strata play a major role in controlling the overburden structure and movement under mining conditions, and their self-gravity and breakage movement have a significant impact on the abutment stress. Moreover, because multiple layers of key strata may occur in a stope, this paper assumes that there are three layers of key strata in the overburden strata and establishes the corresponding mechanical model to obtain the distribution and transmission law of abutment stress in the multilayered key strata. It should be noted that the in situ stress is neglected temporarily when discussing the variation in abutment stress because it is static stress, and only the disturbance stress caused by key strata is discussed.

The disturbance stress before and after breaking of the three key strata under mining conditions will be analyzed. Figure 1 is the structural mechanics model of the overburden before breaking of key layer 1 (main roof). The overburden strata are bent and deformed due to face mining, resulting in the gravity load of the key strata transferring to the rock masses at the boundary of the bed separation in the form of concentrated shear force $Q$ and concentrated bending moment $M$ transferring to the underlying strata. According to Winkler elastic foundation beam theory, the strata below the key layer are regarded as an elastic foundation, and Equation (1) is obtained.

$$P_k = -C_j y_k.$$  \hspace{1cm} (1)

2.1 | Influence analysis of the disturbance stress of key stratum 3 (KS3)

Combined with the model shown in Figure 1, the influence of disturbance stress before and after key stratum 3 (KS3) breaks is studied, and the force analysis model is established, as shown in Figure 2.
The elastic reaction force $P_3(x)$ is generated, and the total displacement $y_3$ is formed at the bed-separation boundary ($x = 0$) for KS3, which is affected by the shear force $Q_3$ and bending moment $M_3$, as shown in Equations (2) and (3). \(^{33}\)

$$y_3 = y_{3M} + y_{3Q}$$

The displacement produced by the bending moment $M_3$ and shear $Q_3$ is as follows:

$$y_{3M} = -\frac{2M_3\beta_3^2}{C_3}\phi(\beta_3 x), y_{3Q} = \frac{2Q_3\beta_3}{C_3}\theta(\beta_3 x). \quad (2)$$

According to mechanical equilibrium, the reaction force of elastic reaction $P_3(x)$ is defined as disturbance stress, as shown in Equation 3.

$$-P_3(x) = C_3(y_{3M} + y_{3Q}) = 2\beta_3 \left[Q_3\theta(\beta_3 x) + \beta_3M_3\phi(\beta_3 x)\right]. \quad (3)$$

where $\beta_3 = [C_3/(4E'K)]^{1/4}$ and $E' = E(1 - \nu)^2$. $\theta(\beta_3 x)$ and $\phi(\beta_3 x)$ are Krylov functions: $\theta(\beta_3 x) = e^{-\beta_3 x} \cos (\beta_3 x)$, and $\phi(\beta_3 x) = e^{-\beta_3 x} \left[\cos (\beta_3 x) - \sin (\beta_3 x)\right]$. Because of the small rotation angle of KS3, it is considered a fixed support state, and Equation 4 is obtained.

$$M_3 = \frac{1}{12} q_{KS3}L_3^2, Q_3 = \frac{1}{2} q_{KS3}L_3. \quad (4)$$

Generally, hard, thick strata usually break ahead of the coal wall. When the key strata break, assuming that the initial breaking position and form is shown in Figure 1, the breaking angle is $\theta$, and the relationship among $L_1$, $L_2$, and $L_3$ is shown in Equation 5.

$$L_3 = L_2 = L_1 + 2(h_1 + h_2)/\tan \theta. \quad (5)$$

### 2.2 Influence analysis of disturbance stress of key stratum 2 (KS2)

Similarly, the influence of disturbance stress before and after breaking of key layer 2 (KS2) is studied, and the force analysis model is established, as shown in Figure 3. At this time, KS2 is affected not only by the bending moment $M_2$ and shear force...
where \( y_{2M} \) and \( y_{2Q} \) are calculated in the same way as Equations (2) and (3).

The underlying strata of KS3 are regarded as a half-plane body (Figure 4A). According to elasticity theory, \(^3\) the stress of disturbance stress \(-P_3(x)\) transferred to arbitrary point \( M \) is established as Equation 7.

\[
f_3(x) = -\frac{2P_3(x)}{\pi} \int_0^b \frac{h^2 d\xi}{[h^2+(x-\xi)^2]^2} \quad 0 \leq x \leq b. \quad (7)
\]

Because the load increment \( f_3(x) \) is nonuniformly distributed, it can be considered a combination of multiple concentrated forces \( P \), as shown in Figure 4B. Hence, the displacement produced by a single concentrated force is shown in Equation 8.\(^3\)

\[
y_p = \frac{P\beta_2}{2C_2} \phi(x-t) + \frac{P\beta_2}{C_2} \left[ \theta(\beta_2 t)\theta(\beta_2 x) + \frac{1}{2} \psi(\beta_2 t)\psi(\beta_2 x) \right] \quad (x \geq t)
\]

\[
y_p = \frac{P\beta_2}{2C_2} \phi(t-x) + \frac{P\beta_2}{C_2} \left[ \theta(\beta_2 t)\theta(\beta_2 x) + \frac{1}{2} \psi(\beta_2 t)\psi(\beta_2 x) \right] \quad (x < t). \quad (8)
\]

Then, the displacement under a nonuniform load increment \( f_3(x) \) is as follows:

\[
y_{y3} = \int_0^b f_3(x)\beta_2 \phi(x-t)dt + \int_x^b f_3(x)\beta_2 \phi(t-x)dt
\]

\[
\quad + \int_0^b \frac{f_3(x)\beta_2}{C_2} \left[ \theta(\beta_2 t)\theta(\beta_2 x) + \frac{1}{2} \psi(\beta_2 t)\psi(\beta_2 x) \right] dt.
\]

\[
\psi(\beta_2 x) = e^{-\beta_2 x} \left[ \cos(\beta_2 x) + \sin(\beta_2 x) \right].
\]

Hence, it can be concluded that the disturbance stress formed by the interaction of \( Q_2, M_2, \) and \( f_3(x) \) before KS2 breaks is shown in Equation (9).

\[
-P_3(x) = -C_2(y_{2M} + y_{2Q} + y_{j}) = 2\beta_2 \left[ Q_2\theta(\beta_2 x) + M_2\beta_2 \phi(\beta_2 x) \right]
\]

\[
\quad + \int_0^b f_3(x)\beta_2 \phi(x-t)dt + \int_x^b f_3(x)\beta_2 \phi(t-x)dt
\]

\[
\quad + \int_0^b \frac{f_3(x)\beta_2}{C_2} \left[ \theta(\beta_2 t)\theta(\beta_2 x) + \frac{1}{2} \psi(\beta_2 t)\psi(\beta_2 x) \right] dt. \quad (9)
\]

2.3 Calculation of disturbance stress of KS1

2.3.1 Disturbance stress before KS1 breaks

In regard to KS1, the \( z-y \) rectangular coordinate system is established to analyze the stress (Figure 5). Before KS1 is
broken, the disturbance stress can be obtained under the combined action of $Q_1$, $M_1$, and $f_2(z)$ by referring to the calculation method described in section 1.2.

$$-P_1(z) = -C_1(y_1 + y_1) = 2\beta_1 [Q_1 \theta(\beta_1 z) + \beta_1 M_1 \phi(\beta_1 z)]$$

$$+ \int_{-\infty}^{z} f_2(z) \phi(z) dz + \frac{f_2(z)}{2C_1} \phi(z) + \frac{1}{2} \psi(\beta_1 z)$$

$$+ \int_{z}^{b} f_2(z) \phi(\beta_1 z) + \frac{1}{2} \psi(\beta_1 z)$$

$$\tag{10}$$

2.3.2 Disturbance stress after KS1 breaks

The overburden structure after KS1 breaks is shown in Figure 6, and the corresponding stress state of KS1 is shown in Figure 7. When KS1 breaks at $z = (h_1 + h_2)/\tan\theta$ ahead of the coal wall, the value of the load increment $f_2(z)$ ranges from $(h_1 + h_2)/\tan\theta$ to $b$, and the bending moment is 0. Hence, the shear force $Q_1'$ at the breaking position of KS1 is as follows:

$$Q_1' = \frac{(q_t) z}{2} \left( \frac{L_1}{2} + \frac{h_1 + h_2}{\tan\theta} \right).$$

Hence, when key stratum 1 is broken, the disturbance stress affected by the action of shear force $Q_1'$ and load increment $f_2(z)$ after KS1 breaks is as shown in Equation 12.

$$-P_1(z) = -C_1(y_1 + y_1) = 2\beta_1 Q_1' \theta(\beta_1 z)$$

$$+ \int_{-\infty}^{z} f_2(z) \phi(z) dz + \frac{f_2(z)}{2C_1} \phi(z) + \frac{1}{2} \psi(\beta_1 z)$$

$$\tag{12}$$

2.4 Calculation of front abutment stress

According to the above analysis, the disturbance stress $-P_1(z)$ and $-P_1'(z)$ before and after KS1 breaks are obtained, and the expression of the front abutment stress before and after KS1 breaks can be obtained by taking into account the self-weight of the overlying strata, as shown in Equations 13 and 14.

Before KS1 breaks: $\sigma = f_1(z) + \gamma H = \frac{-2P_1(z)}{\pi} \int_{0}^{b} \frac{h_1 \psi(z)}{\bar{h}_1^2 + (z-\zeta)^2} + \gamma H. \tag{13}$

After KS1 breaks: $\sigma = f_1(z) + \gamma H = \frac{-2P_1'(z)}{\pi} \int_{0}^{b} \frac{h_1 \psi(z)}{\bar{h}_1^2 + (z-\zeta)^2} + \gamma H. \tag{14}$

3 Calculation of Abutment Stress Before and After KS2 Breaks

With increasing mining area, a series of activities such as bending, breaking, and caving occur in KS1 and its underlying layer. At this time, KS2 is in a state of suspension, and the suspension area increases gradually, which makes it bend, and eventually triggers a similar movement of KS1.

Figure 8 depicts the structural state of the overburden before KS2 breaks. At this time, under the action of $M_2$ and $Q_3$, KS3 generates the disturbance stress $-P_3$, which is transferred downward, forming the nonuniform load increment $f_3$ acting on KS2. At this point, the following two situations are considered:

Before KS2 breaks, the disturbance stress $-P_2$ is formed under the action of $M_2$, $Q_2$, and $f_2$ and transfers downward, forming the nonuniform load increment $f_2$ acting on KS1; then, the disturbance stress $-P_1$ is generated affected by $M_1$, $Q_1$, and $f_2$ and continues to transfer downwards to the coal seam; hence, the front abutment stress is determined by superimposing these in situ stresses.

After KS2 breaks, the disturbance stress $-P'_2$ is formed under the action of $Q'_2$ and $f'_2$ (herein, $M_2 = 0$) and transfers downward, forming the nonuniform load increment $f'_2$ acting on KS1; then, the disturbance stress $-P'_1$ is generated by $M_1$, $Q_1$, and $f'_2$ and continues to transfer downwards to the coal seam; hence, the front abutment stress is determined by superimposing these in situ stresses.
When considering the above calculation and analysis, $M_1$ and $Q_1$ can be regarded as cantilever beam calculations because KS1 has been broken, and the other parameters are the same as those described in the previous section.

\[ M_1 = \frac{q_{KS1} L_1^2}{2}, \quad Q_1 = q_{KS1} L_1. \]  

(15)

4 | ENGINEERING EXAMPLES

The previous analysis concerns the front abutment stress before and after KS1 and KS2 breaks. To describe the variation in abutment stress more clearly, a numerical analysis of the above study results is carried out on panel 10 305 of the Xinglongzhuang coalmine.

4.1 | Engineering background

The Xinglongzhuang coalmine is located in Jining City, Shandong Province. The mining depth and length of panel 10 305 in the No. 10 coal seam marked are 372 and 220 m, respectively; the mining height is 6 m. Based on the theory of key strata, the position of the key strata and the physical—mechanical parameters of the overlying strata are listed in Table 1. Referring to a previous study, \(^2^6\) because the limit height of fracture development is generally approximately 1/2 of the width of the working face, the height of the fracture zone induced by panel 10 305 mining is 110 m, so the calculation used in the above new theoretical method only needs to consider the breaking of KS1 and KS2.

4.2 | Quantification of abutment stress before and after KS1 (main roof) breaks

Based on the above analysis, according to the field mining of panel 10 305, the weighting step of KS1 is $L_1 = 80$ m and its fracture angle is assumed to be $\theta = 80^{\circ}$; hence, $L_2$ and $L_3$ are obtained by Equation (6) as $L_1 = L_2 = 80 + 2 \times (6.9 + 34.8)/\tan 80^{\circ} = 94.7$ m. Moreover, the results can be calculated as $M_3 = 879$ MN m, $M_2 = 1032$ MN m, $Q_3 = 55.7$ MN, $Q_2 = 64.5$ MN, $\beta_3 = 0.019$ m$^{-1}$, $\beta_2 = 0.028$ m$^{-1}$, and the beam width of the key stratum is a unit width, $d = 1$ m. Hence, the loads exerted on KS2 and KS3 are shown as follows:\(^3^2\):

\[ q_{KS1} = \frac{E_6 h_6^2 (\gamma_6 h_6 + \gamma_7 h_7)}{E_6 h_6^2 + E_7 h_7^2} = 1176 \text{kPa} \quad q_{KS2} = \frac{E_4 h_4^2 (\gamma_4 h_4 + \gamma_5 h_5)}{E_4 h_4^2 + E_5 h_5^2} = 138 \text{kPa}. \]

The stiffness coefficient of the soft foundation is calculated by $C = E'/(3h)$; $C_3 = 0.15$ GPa, and $C_2 = 0.23$ GPa.

The above parameters are substituted into Equations (3), (7), and (9) so that the disturbance stresses $-P_3(x)$ and $-P_2(x)$, and incremental loads $f_3(x)$ and $f_2(z)$ can be depicted by MATLAB software, as shown in Figure 9.
As shown in Figure 9, the peak value of the disturbance stress \(-P_3(x)\) produced by the interaction of \(M_3\) and \(Q_3\) in KS3 is 3.1 MPa, and its influence range is from \(O_3\) to 110 m (\(O_3\) is the initial breaking point of KS3, which is a marker point in Figure 1); after transferring to KS2, the influence range of the load increment \(f_3(x)\) is basically unchanged, but the peak stress value decreases to 1.6 MPa, and the peak position moves forward 8 m. However, the peak value of the disturbance stress \(-P_2(x)\) produced by the interaction of \(M_2\), \(Q_2\) and \(f_3(x)\) in KS2 is 6.5 MPa, and its influence range is from \(O_2\) to 180 m (\(O_2\) is the initial breaking point of KS2, which is a marker point in Figure 1); after transferring to KS1, the influence range of the load increment \(f_2(z)\) changes to \(O_1\) to 180 m (\(O_1\) is 0), but the peak stress value drops to 2.6 MPa, and the peak position moves forward 9 m. Hence, when the disturbance stress is transmitted downward, the influence on the stress range of the underlying strata is not significant, but the stress will gradually attenuate, and the peak stress will shift forward.

The results of \(M_1 = 650\) MN m, \(Q_1 = 48.8\) MN, \(\beta_1 = 0.039\) m\(^{-1}\), and \(C_1 = 1.06\) GPa are obtained before KS1 breaks, while \(M'_1 = 0\) and \(Q'_1 = 28.9\) MN are calculated after KS1 breaks. Hence, the load exerted on KS1 is as follows:

\[
q_{KS1} = \frac{E_2 h_2^3 (\gamma_2 h_2 + \gamma_3 h_3)}{E_2 h_2^3 + E_3 h_3^3} \approx 1220\text{ KPa.}
\]

Hence, the variations in the disturbance stresses \(P_1(z)\) and \(P'_1(z)\) before and after KS1 breaks are obtained from Equations (10) and (12), as shown in Figure 10.

Figure 10 shows that the disturbance stresses \(P_1(z)\) and \(P'_1(z)\) are significantly different before and after KS1 breaks. Before KS1 breaks, the peak value of the disturbance stress is 8.5 MPa, which is located at the coal wall, and the influence range is 0-180 m; meanwhile, the peak value of the disturbance stress decreases to 6 MPa and is located at the fracture point, with the influence range remaining basically unchanged. Hence, it can be seen that the peak disturbance stress decreases by 2.5 MPa in the range of 0-10 m before and after KS1 breaks but increases in the range of 10-90 m and that the effect of KS1 breaking on its stress influence range is not significant.

From Equations (13) and (14), the distribution of the front abutment stress before and after KS1 breaks can be obtained, as shown in Figure 11. The peak value of the abutment stress is 15.2 MPa, 8 m ahead of the coal wall before KS1 breaks, while it decreases to 13 MPa and moves forward 19 m ahead of the coal wall after KS1 breaks. Hence, the peak value of abutment stress can be reduced by 2.2 MPa before and after KS1 breaks, and the position of the peak stress moves forward 11 m to the depth of the coal wall, which reduces the possibility of coal outburst and rock burst disasters.

| Strata sequence | Thickness /m | Lithology  | Density /kN m\(^{-3}\) | Elastic modulus/GPa | Poisson's ratio | Tensile strength/MPa | Note          |
|-----------------|--------------|------------|------------------------|---------------------|-----------------|----------------------|--------------|
| 1               | 6.9          | Siltstone  | 25.6                   | 19.5                | 0.31            | 1.9                  | Soft stratum 1 (SS1) |
| 2               | 34.8         | Gritstone  | 26.4                   | 31.1                | 0.21            | 3.8                  | Key stratum 1 (KS1)  |
| 3               | 13.1         | Claystone  | 24.3                   | 8.4                 | 0.26            | 1.1                  | Soft stratum 2 (SS2) |
| 4               | 39.7         | Sandy mudstone | 25.1           | 17.2                | 0.24            | 3.9                  | Key stratum 2 (KS2)  |
| 5               | 19.1         | Claystone  | 23.8                   | 7.8                 | 0.26            | 0.9                  | Soft stratum 3 (SS3) |
| 6               | 51.8         | Siltstone  | 21.4                   | 20.2                | 0.24            | 3.6                  | Key stratum 3 (KS3)  |
| 7               | 206.6        | Sandy layer| 20                     | 1.1                 | 0.32            | 1.9                  | Soft stratum 4 (SS4) |
Moreover, KS1 breaking has little effect on the influence range of the abutment stress (e.g., 0-180 m), and the area of significant change is 0-85 m. Additionally, the abutment stress decreases: the maximum decrease is 3.9 MPa in the range of 0-22 m ahead of the coal wall after KS1 breaks and the maximum increase is 0.56 MPa in the range of 22-180 m, which indicates that the stress relief effect near the coal wall is obvious after KS1 breaks.

### 4.3 Calculation of abutment stress before and after KS2 breaks

From the analysis of the ultimate span of KS2 by Equation (15), it can be concluded that the ultimate span of KS2 is

\[ L_2 = h_4 \sqrt{\frac{2 \times q_{KS2}}{R_{T4}}} = 134 \text{ m.} \]  

(16)

Then, the results are obtained as follows: \( L_1 = 155 \) m, \( M_3 = 2355 \text{ MN m} \), \( Q_3 = 912 \text{ MN} \). Before KS2 breaks, \( M_2 = 2075 \text{ MN m} \) and \( Q_2 = 927 \text{ MN} \), whereas after KS2 breaks, \( M_2 = 0 \) and \( Q_2 = 1024 \text{ MN} \). Hence, the variation curves of the disturbance stress before and after KS2 breaks can be obtained with Equation (9), as shown in Figure 12.

The peak value of the disturbance stress decreases by 5.2 MPa after KS2 breaks, compared with the disturbance stress before KS1 breaks; moreover, the stress decreases in the range of 0-31 m and increases in the range of 31-180 m. Compared with Figure 9, the reduction in the disturbance stress before and after KS2 breaks is larger than that of KS1, which indicates that the release energy is larger.

When calculating the front abutment stress after KS1 breaks, KS1 presents the cantilever structure with KS2 in the limit state. Assuming that the cantilever lengths \( L_1 \) are 0, 10, 20, 30, and 40 m, respectively, the influence of different cantilever lengths on the front abutment stress in KS1 can be obtained, as shown in Figure 13. It should be noted that \( L_1 = 0 \text{ m} \) represents that KS1 has just broken, and the bending moment is 0; when calculating the shear force, the maximum length of the collapsed block is 40 m from Equation (10).

From Figure 13 and Table 2, we can see that the peak values of abutment stress range from 15.1 to 18.2 MPa with the variation in \( L_1 \) before KS2 breaks, while they range from 13 to 16.2 MPa after KS2 breaks; the influence range (0-180 m) of the abutment stress and the area of significant variation (0-120 m) in the abutment stress do not change after KS2 breaks. Moreover, regardless of whether KS2 breaks, the peak stress values increase with increasing cantilever length \( L_1 \); however, when \( L_1 = 0 \text{ m} \), the peak value is larger than that when \( L_1 = 10 \text{ m} \) because the fractured blocks are longer and the shear force at the breaking point is greater.

Meanwhile, the reduction in peak stress before and after KS2 breaks with the same cantilever length of \( L_1 \) remains the
same regardless of the value of \( L_1 \). Additionally, the abutment stress decreases; the maximum decrease is 2.1 MPa in the range of 0-71 m ahead of the coal wall after KS2 breaks and the maximum increase is 0.15 MPa in the range of 71-180 m, which indicates that the influence of KS2 breaking on the front abutment stress near the coal wall is less severe than that of KS1 breaking.

As shown in Table 2, before KS2 breaks, the position of the peak value is 9 m ahead of the coal wall with \( L_1 = 0 \) m and \( L_1 = 10 \) m, while it decreases to 8, 7, and 6 m when \( L_1 = 20, 30 \) and 40 m, respectively. This indicates that the position of the peak stress value gradually shifts toward the coal wall with the increase in the cantilever length of KS1. Meanwhile, the position basically remains unchanged with the increase in the KS1 cantilever length after KS2 breaks, which indicates that KS2 breaking has no influence on the position of the peak value of abutment stress. Combined with the results of section 3.2, these results show that the movement and structural state of KS1 will have a vital impact on the peak position of the abutment stress.

### 4.4 Variation law of front abutment stress

Through the calculation analysis results before and after the breaking of KS1 and KS2, the following can be concluded:

1. The influence range of the abutment stress is 0-180 m, the stress concentration coefficient is 1.6, and the position of the peak stress is 8 m ahead of the coal wall before KS1 breaks. After KS1 breaks, the influence range is unchanged, the stress concentration coefficient is reduced to 1.4, and the peak stress shifts forward 11 m, which is 19 m ahead of the coal wall. Moreover, both before and after KS1 breaks, the significant variation area of abutment stress is 0-85 m.

2. The influence range of abutment stress is 0-180 m, the stress concentration coefficient is 1.5-2, and the position of peak stress is at 6-8 m ahead of the coal wall before KS2 breaks. After KS2 breaks, the influence range and the position of peak stress are unchanged, and the stress concentration coefficient is reduced to 1.4-1.8. Moreover, before and after KS2 breaks, the area of the significant variation in abutment stress is 0-120 m.

| \( L_1 \) | 0  | 10 | 20 | 30 | 40 |
|---------|----|----|----|----|----|
| Peak value position of abutment stress before KS2 breaks/m | 9  | 9  | 8  | 7  | 6  |
| Peak value position of abutment stress after KS2 breaks/m | 9  | 9  | 8  | 7  | 6  |
| Peak value of abutment stress before KS2 breaks/MPa | 15.6 | 15.1 | 15.9 | 16.9 | 18.2 |
| Peak value of abutment stress after KS2 breaks/MPa | 13.5 | 13 | 13.8 | 14.9 | 16.2 |
| Reduction of peak stress before and after breaks/MPa | 2.1 | 2.1 | 2.1 | 2.0 | 2.0 |

**FIGURE 12** Variations curves of disturbance stress before and after KS2 breaks

**FIGURE 13** Influence of different cantilever lengths on the front abutment stress of KS1

**TABLE 2** Values of the abutment stress with different cantilever lengths of KS1
3. The abutment stress decreases, and the maximum decrease is 3.9 MPa 0-22 m ahead of the coal wall after KS1 breaks, while the maximum decrease is 2.1 MPa 0-71 m ahead of the coal wall after KS2 breaks. Hence, compared with KS2, the abutment stress decreases greatly, and the reduction range is small after KS1 breaks.

5 | FILED OBSERVATION

5.1 | Stress Monitoring Scheme

To obtain the variation in the front abutment stress in panel 10 305, nine stations are set up in front of the coal seam before mining activities begin. Each station is equipped with two GMC20 stress sensors. The distance between the sensors is 2 m, the installation depths are 8 m and 15 m, respectively, and the installation height is 1.5-2 m from the bottom of the coal seam. The measuring station marked No. 1 is 40 m ahead of the open-off cut, and the distance between the two stations is 30 m, as shown in Figure 14. The sensor shows an initial stress of 5 MPa, and the sensors should be removed one by one when the working face advances close to the station.

5.2 | Analysis of Monitoring Results

According to the above-mentioned monitoring method, the monitoring data of stations Nos. 3 and 6 are selected for analysis of the abutment stress, as shown in Figure 15.

Figure 15 describes the variations in the relative abutment stress of monitoring points Nos. 3 and 6. When the working face
advances 21 m on May 7, the abutment stress begins to increase with the distance between monitoring point No. 3 and the coal wall at 79 m; when the working face advances to 80 m on May 19, the distance between monitoring point No. 3 and the coal wall is 20 m, the initial weighting of the main roof occurs, and the abutment stress reaches its peak value; on May 20, the abutment stress decreases after the main roof breaks, while on May 21, when monitoring point No. 3 is 13 m away from the coal wall, the abutment stress increases again; after that, the abutment stress decreases obviously, indicating that station No. 3 is in the plastic zone. When the working face is pushed to 81 m, the abutment stress begins to increase with the distance between monitoring point No. 6 and the coal wall at 109 m; when the working face advances to 179 m, point No. 6 is 11 m away from the coal wall, and the abutment stress reaches its peak value and then decreases rapidly, and the monitoring point is in the plastic zone. The area of significant variation in the abutment stress is 0-109 m, and the position of the peak stress is 11-13 m, which is basically consistent with the theoretical calculation results (6-19 m).

Because the monitored abutment stress is a relative value, rather than directly measured from the real environment in the field, the maximum abutment stress of monitoring point No. 3 increases by 4.8 MPa compared with the initial stress of 5 MPa before the main roof (KS1) breaks and by 9.3 MPa before KS2 breaks. According to the theoretical calculation in section 4, the maximum abutment stress before KS1 breaks increases by 5.1 MPa compared with the in situ stress and that of increased by 5.8-9.1 MPa before KS2 breaks. This indicates that the monitoring results are in good agreement with the theoretical calculation and the existing theoretical results, which further explains why the new theoretical method has good applicability. Moreover, this result shows that it is scientific to consider the interaction of overlying key strata and the transfer effect on the front abutment stress.

6 | CONCLUSIONS

In this paper, based on the formation mechanism of abutment stress, a new theoretical calculation method of abutment stress is proposed considering the interaction of overlying key strata and the effect on the transmission of the front abutment stress, and then, the stress patterns of multilayered key strata are analyzed under the combined action of the bending moment, shear force and nonuniform load and its transmission to the underlying strata. We can obtain several conclusions, as follows:

- Before KS1 breaks, the concentration coefficient of the abutment stress is 1.6, and the position of the peak value is 8 m ahead of the coal wall, but after KS1 breaks, it decreases in the range of 0-22 m, the coefficient of the peak stress drops to 1.4, and the position of the peak value shifts forward from 8 to 19 m.

- KS1 breaking has little effect on the influence range of the abutment stress: the range is always 0-180 m, and the area of significant change is 0-85 m.

- Before KS2 breaks, the concentration coefficient of the abutment stress is 1.5-2, and the position of the peak value is 6-9 m ahead of the coal wall; however, after KS2 breaks, the concentration coefficient of the abutment stress decreases in the range of 0-71 m, the coefficient of the peak stress drops to 1.4-1.8, and the position of the peak value remains unchanged.

- Before and after KS2 breaks, the influence range of the abutment stress remains unchanged at 0-180 m, in which the zone of significant change extends to 0-120 m.

The calculation results of the new calculation method have good agreement with the existing theoretical results, and this method can also calculate the dynamic change in abutment stress throughout the working face advancing process, which is not achieved by the existing theoretical calculation. Furthermore, the stress monitoring of panel 10 305 in the Xinglongzhuang coalmine is used to test and verify the above results; it is found that the variation in abutment stress before and after key strata break is consistent with that of the new theoretical method, which further shows that the new theoretical calculation method has strong applicability and scientific feasibility and provides a new and more practical method for on-site design and construction practices.

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LIST OF SYMBOLS

| Symbol | Description |
|--------|-------------|
| H      | Mining depth, m |
| b      | Acting length of incremental load, m |
| θ      | Fracture angle of overlying strata, ° |
| σ      | Front abutment stress, MPa |
| Cj     | Stiffness coefficient of soft stratum marked as No. j, j = 1,2,3,4, kN/m³ |
| dj     | Width of soft stratum j, m |
| Ei     | Elastic modulus of overlying stratum marked as i, i = 1,2,3,4,5,6,7, GPa |
| E’i    | Elastic modulus of overlying stratum i under a plane strain state, GPa |
| hj     | Height of overlying stratum i, m |
\[ I_i \] Flexural modulus of overlying stratum \( i \), m4
\[ \gamma_i \] Density of overlying stratum \( i \), kN/m3
\[ \nu_i \] Poisson’s ratio of overlying stratum \( i \)
\[ RTi \] Tensile strength of overlying stratum \( i \), MPa
\[ f_k \] Load increment of key stratum marked as \( k - 1 \) before key stratum \( k \) breaks, \( k = 1,2,3 \), MPa
\[ f'_k \] Load increment of key stratum \( k - 1 \) after key stratum \( k \) breaks, MPa
\[ L_k \] Suspension length of key stratum \( k \), m
\[ O_k \] Breaking initiation point of key stratum
\[ M_k \] Bending moment of key stratum \( k \) at the boundary of bed separation, MN·m
\[ P_k \] Elastic reaction force, MPa
\[ Q_k \] Shear force of key stratum \( k \) at the boundary of bed separation before it breaks, \( N \)
\[ Q'_k \] Shear force of key stratum \( k \) after it breaks, \( N \)
\[ q_{KSI} \] Overlyring load of key stratum \( k \), kPa
\[ y_k \] Total displacement of key stratum \( k \), m
\[ y_{kM} \] Displacement of key stratum \( k \) affected by bending moment \( M \), m
\[ y_{kQ} \] Displacement of key stratum \( k \) affected by shear force \( Q_k \), m
\[ y_{k+1} \] Displacement of key stratum \( k \) affected by load increment \( f_k + 1 \), m
\[ y \] Displacement of key stratum \( k \) affected by concentrated load \( P \), m
\[ \beta_k \] Coefficient of elastic foundation in underlying strata of key strata, m⁻¹

CONFLICTS OF INTEREST
The authors declare no conflicts of interest.

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DATA AVAILABILITY STATEMENT
The data used to support the findings of this study are available from the corresponding author upon request.

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