Evaluating the maximum rate of penetration for drilling borehole in soft coal seam

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Abstract
Most coal seams in China are characterized by high-gas pressure and high-gas content, which seriously threaten the safety of coal mines. Gas extraction is an effective measure for preventing gas disasters. However, the drilling process in soft coal seams is inefficient; it is difficult for the boreholes to reach the design length. The rate of penetration (ROP) plays an important role in determining drilling efficiency and borehole length in soft coal seams. In this study, we establish a simplified ROP model for soft coal seams and evaluate the maximum ROP of Hebi No. 6 coal mine. First, the drill cutting transport capacity of the drilling system and drill cutting quantity generated during the drilling process are analyzed. By simplifying drill cutting transport and coal seam properties, the ROP model is established. Second, the Rock Failure Process Analysis software is used to analyze the deformation region of boreholes, which has a great influence on the drill cutting transport space. Finally, a field test is implemented in Hebi No. 6 coal mine in Henan Province, China, to verify the model, and we propose a measure to control the ROP below the maximum value to increase the length of the borehole. The test results indicate that the model has high accuracy, with errors less than 15%, and the lengths of boreholes in air-return and transportation roadways are increased to 60 and 80 m, respectively, and the increase rate is greater than 50%.

Keywords
drill cutting transport, gas extraction, rate of penetration, RFPA2D, soft coal seam
INTRODUCTION

Coal is the primary energy source of China. Owing to the high-intensity exploitation of coal resources, the mining depth of some coal mines in China is over 800 m. The gas pressures and gas contents of these coal seams can reach 6 MPa and 22 m³/t. These factors, combined with uniaxial compressive strengths of less than 10 MPa, easily lead to gas disasters, especially coal and gas outburst.

To ensure the safety and efficiency of coal mines, effective measures must be taken to prevent gas disasters. Researchers have proposed many methods for preventing gas disasters. Among these, the most effective and commonly used in China are protective coal seam mining and gas extraction. Protective coal seam mining is a technology in which a coal seam without coal and gas outburst danger is selected as the initial mining seam during multi-seam mining. This method can significantly reduce the danger of coal and gas outbursts in adjacent coal seams. However, most coal mines in China have no protective coal seam. Gas extraction is a technology that extracts gas prior to mining from the boreholes that are drilled from the roadway into the coal seam to reduce gas pressure and gas content, which effectively reduces the risk of gas disasters. Therefore, gas extraction is the best method to prevent gas disasters for a high-gas single coal seam.

However, there are two main problems with gas extraction in a soft coal seam. The first problem is that it is difficult to expand the gas extraction area and increase the gas extraction rate. Many researchers have attempted to develop methods for high-efficiency gas extraction from high-gas coal seams. The second problem is the difficulty in drilling long boreholes in soft coal seams. Most of the high-gas coal seams in China are soft, with firmness coefficient (Protodyakonov's coefficient) less than 1, which means the uniaxial compressive strength is less than 10 MPa. They are accompanied by complicated geological conditions and high-gas content.

Boreholes drilled in these coal seams suffer from two primary problems. The first is that it is difficult to prevent borehole instability. The second is stuck pipe accidents, which reduce drilling efficiency. When drilling in a soft coal seam, the drilling process is less difficult due to the lower strength. However, the boreholes in the low-strength coal seams are more susceptible to deformation under the influence of stresses, and the drill cutting quantity produced in the drilling process is much larger than the ideal value. The deformation of the boreholes will reduce the drill cutting transport space and increase the transport resistance, which makes transportation difficult. When drill cutting quantity produced per unit time is greater than the drill cutting transport capacity of the drill system, the drill cuttings will accumulate. If they cannot be transported in time, it will cause a stuck pipe accident. Rate of penetration (ROP) measures the progress and the performance of the drilling bit, which is one of the most important parameters in drilling operations. To predict the ROP, many scholars have conducted extensive research. Chiranth et al. introduces a workflow that combines the ROP optimization process with a machine learning-based vibration model. Alireza et al. construct a drillability index (DI) base on drilling data, the ROP is normalized by drilling factors, making the DI more sensitive to coal rock strength. Gan et al. propose a novel model to predict the ROP considering the process characteristics. Melvin et al. utilizes drilling data obtained as the drilling progresses within a given well to present a feasibility assessment of predicting the ROP during wellbore drilling. However, these models are not suitable for ROP estimation of soft coal seams. This is because it is relatively easy-to-break coal bodies in soft coal seams due to the low strength of soft coal, but the drill cutting quantity is much larger than that of hard coal. The maximum ROP can only be achieved if the drill cutting quantity produced matches the transport capacity. Therefore, it is necessary to establish an ROP model related to the production and transportation of drill cuttings in order to maximize the efficiency of drilling in a soft coal seam.

FIGURE 1 Schematic of drilling and drill cutting transport in soft coal seam
Pneumatic transport is a commonly used, effective drill cutting transport method for the drilling in soft coal seams in underground field application.\textsuperscript{33} It involves the use of high-pressure air to transport drill cuttings out of the borehole through the annulus between the drill pipe and borehole.\textsuperscript{34} Compared to other transport methods, pneumatic transport has the advantages of high efficiency and reduced damage to the borehole wall.\textsuperscript{35}

In this study, we established an ROP model and proposed a measure that controls the ROP to improve borehole length and drilling efficiency in soft coal seams. The transport capacity and drill cutting quantity are obtained. The model has high accuracy, and the final borehole length was improved.

2 | ROP MODELING

In the drilling process, the coal mass is cut into particles by the drill bit. High-pressure air is injected through the inner pipe of the drill pipe into the borehole. The same air stream then transports the drill cuttings out of the borehole (Figure 1). Therefore, the ROP model is established from two aspects, that is, drill cutting transport capacity and drill cutting quantity.

2.1 | Main assumptions of the ROP model

Coal is a heterogeneous body containing many features, and its strength is influenced by geologic structure and crustal stresses. During the transportation process, the drill cuttings have an extremely complicated movement. It is difficult to determine the theoretical relationship between ROP and drill cutting quantity and transport capacity; thus, it is necessary to simplify the model as follows:

- During the transportation process, the drill cuttings are suspended uniformly;
- The drill cuttings in segment $L$ are treated in aggregate;
- The drill cuttings are uniform spheres;
- The airflow cannot be compressed;
- The rock surrounding the borehole is an isotropic elastic body, and coal is a homogeneous material with no creep property;
- The properties of coal are consistent along the radial direction of the borehole.

2.2 | Drill cutting transport capacity

A geometric model for drill cutting transport was established using simplified conditions for the field, as shown in Figure 2. The geometrical parameters mainly include the borehole diameter ($D$), drill pipe diameter ($d$), borehole angle ($\alpha$), pneumatic thrust ($F$), and frictional resistance ($f$).

Owing to the influence of high-pressure airflow, the static drill cuttings accelerate until resistance is equal to thrust. The pneumatic thrust is expressed as follows:\textsuperscript{36}

$$F = CA\rho q \left( \frac{v_q - v_f}{v_0} \right)^2,$$

where $C$ is the flow resistance coefficient, $A$ the annulus, $\rho_q$ the airflow density, $v_q$ the airflow velocity, and $v_f$ the drill cutting transport velocity.

The flow resistance coefficient $C$ is related to the Reynolds number $Re$:\textsuperscript{37,38}

$$C = \frac{2gqL}{Ap_0 v_q^2 v_f} \left( \frac{v_q - v_f}{v_0} \right)^K,$$

where $v_0$ is the free suspension velocity of the drill cuttings, and $K$ is a coefficient associated with the Reynolds number. During drill cutting transportation, the Reynolds number is greater than 500, placing it in the Newtonian inertial resistance zone.\textsuperscript{39} Therefore, $K = 0$.

The pneumatic thrust can be obtained by substituting Equation (2) into Equation (1):

$$F = g q_f L \left( \frac{v_q - v_f}{v_0} \right)^2.$$

Gravity also affects the motion of the drill cuttings, and the component of gravity in the flow direction is expressed as follows:

$$G_\alpha = G \sin \alpha = \rho_f Al g \sin \alpha,$$

where $G$ is the gravity of drill cuttings, and $\alpha$ the angle of the borehole.
The drill cuttings near the borehole wall are affected by frictional resistance, which can be expressed as follows:

\[ f = \lambda \rho_f \frac{v_f^2 L}{2D} \cos \alpha = \frac{\lambda q_f v_f L}{2D} \cos \alpha, \quad (5) \]

where \( \lambda \) is the resistance coefficient, \( \rho_f \) the density of suspended drill cuttings, \( q_f \) the mass flow rate of drill cuttings, \( L \) the length of segment \( L \), and \( D \) the borehole diameter.

The drill cuttings accelerate until the thrust and friction are equal, and we can obtain

\[ m_q \frac{dv_f}{dt} = F + G_a - f, \quad (6) \]

where \( m_q \) is the mass of drill cuttings.

The motion equation for the drill cuttings can be derived by substituting Equations (3) and (5) into Equation (6):

\[ m_q \frac{dv_f}{dt} = q_f \frac{v_f - v_0}{v_f} + G \sin \alpha - \frac{\lambda q_f v_f L}{2D} \cos \alpha. \quad (7) \]

When the thrust is equal to the friction, the motion of drill cuttings can be expressed as follows:

\[ q_f \frac{v_f - v_0}{v_f} + G L \sin \alpha - \frac{\lambda q_f v_f L}{2D} \cos \alpha = 0. \quad (8) \]

Assuming the drill cuttings are uniform spheres moving in unrestricted space, the suspension velocities of the drill cuttings can be expressed as follows:

\[ v_0 = \sqrt{\frac{4gq_f}{3C\rho_q}} \left( \rho_f - \rho_q \right). \quad (9) \]

In reality, the drill cuttings have irregular shapes, and their movements are limited by the borehole space. Therefore, the suspension velocity equation must be modified as follows:

\[ v_0' = 4.98 \sqrt{\frac{\rho_f - \rho_q}{\rho_q} df} \left[ 1 - \left( \frac{df}{D-d} \right)^2 \right]. \quad (10) \]

The density of the suspended drill cuttings is related to \( \theta \), which is the mass flow ratio of solid to gas. The relationship can be expressed as follows:

\[ \theta = \frac{G_f}{G_q} = \frac{\rho_f v_f At}{\rho_q v_q At} = \frac{\rho_f v_f}{\rho_q v_q}. \quad (11) \]

From Equation (11), we obtain

\[ \frac{\rho_f}{\rho_q} = \theta \frac{v_q}{v_f}. \quad (12) \]

Substituting Equations (10) and (12) into Equation (8) gives

\[ \sqrt{\left( \frac{\theta v_f^2}{v_f} - 1 \right) df} \left[ 1 - \left( \frac{df}{D-d} \right)^2 \right] = 4.98 \sqrt{\frac{\lambda v_f^2}{2gD} \cos \alpha - \sin \alpha}. \quad (13) \]

According to Equation (13), if we have the borehole diameter, drill pipe diameter, angle of borehole, and airflow velocity, we can obtain the drill cutting transport velocity.

The mass flow rate of drill cuttings can be obtained from Equation (11)

\[ Q_T = \frac{G_f}{t} = \frac{\rho_f v_f At}{t} = \theta \pi D^2 - d^2/4 \rho_q v_f. \quad (14) \]

During drilling in a soft coal seam, the borehole is gradually deformed under the action of crustal stress, gas pressure, and drill pipe disturbance, resulting in reductions in borehole diameter and associated transport space as well as increases in airflow resistance. Therefore, it is necessary to modify the diameter in Equation (14). The correction factor is \( K_a \) \((K_a = \frac{D - 2u_p}{D})\), where \( u_p \) is the displacement of the plastic zone). The modified mass flow rate is described by

\[ Q'_T = \theta \pi \left( \frac{K_a D}{K_a D - d} \right)^2 - d^2/4 \rho_q v_f. \quad (15) \]

Equation (15) shows that the transport capacity is positively correlated with the airflow velocity and the modified borehole diameter.

According to the Bernoulli equation, the airflow velocity can be described by

\[ v_q = \sqrt{\frac{2(p_0 - \eta p_f)}{\rho_q}}. \quad (16) \]

where \( p_0 \) is the initial wind pressure, \( p_f \) is the pressure drop, and \( \eta \) is the angle coefficient.

The airflow in the borehole can cause pressure drop due to friction and can be expressed as follows:

\[ p_f = \lambda \frac{L}{K_a D - \frac{d^2}{2}}. \quad (17) \]
Substituting Equations (16) and (17) into Equation (15) gives

\[ Q'_T = \frac{\theta \pi (K_o D)^2 - d^2}{4} \sqrt{\frac{2 \rho_o p_0}{K_o D - d + \lambda \eta L}}. \quad (18) \]

In Equation (18), \( Q'_T \) may be used to evaluate the drill cutting transport capacity of the drilling system.

2.3 | Drill cutting quantity

The drill cutting quantity is the mass flow rate of drill cuttings produced in the drilling process. Assuming that the coal mass around the borehole is not deformed or damaged, the mass flow rate can be expressed as follows:

\[ Q_D = A' \rho v = \frac{\pi D^2}{4} R \rho_c. \quad (19) \]

where \( A' \) is the borehole cross-sectional area, \( R \) the rate of penetration, and \( \rho_c \) the density of coal. This is a simplification. In reality, the stresses around the borehole are redistributed, and the coal mass around the borehole is damaged. The borehole wall is exfoliated easily under the action of gas pressure gradient and drill pipe disturbance, resulting in additional drilling cuttings.

Using the proportion \( K_d \), which represents the ratio of the exfoliated coal mass to that of the entire failure zone, the mass flow rate of additional drilling cuttings can be obtained as follows:

\[ Q_A = K_d \frac{D^2}{4} R \rho_c. \quad (20) \]

where \( D_c \) is the diameter of the failure zone in which the coal mass has been damaged by many cracks.

The modified mass flow rate can be obtained by summing Equations (19) and (20):

\[ Q'_D = \frac{\pi D^2}{4} R \rho_c + K_d \frac{D^2}{4} R \rho_c. \quad (21) \]

2.4 | ROP model

The drilling process consists of drilling and drill cutting transport. When the drill cuttings are produced in line with the transport capacity of the drill system, the maximum ROP can be achieved and can be described as follows:

\[ \theta \pi (K_o D)^2 - d^2 \sqrt{\frac{2 \rho_o p_0}{K_o D - d + \lambda \eta L}} = \frac{\pi D^2}{4} R \rho_c + K_d \frac{D^2}{4} R \rho_c. \quad (22) \]

In summary, the ROP model can be derived as follows:

\[ R = \frac{\theta \pi (K_o D)^2 - d^2}{\frac{D^2}{4} R \rho_c + K_d \left( \frac{D^2}{4} - D^2 \right) \rho_c} \sqrt{\frac{2 \rho_o p_0}{K_o D - d + \lambda \eta L}}. \quad (23) \]

3 | BOREHOLE DEFORMATION REGION OF SOFT COAL SEAM

The ROP model is established from drill cutting transport capacity and drill cutting quantity. The transport space has a great influence on the drill cutting transport capacity. However, the stress around the borehole is redistributed during drilling. Due to the low strength of soft coal, the borehole is gradually deformed under the action of crustal stress during drilling in a soft coal seam, resulting in reductions in transport space.\(^{41}\) In a typical high-gas soft coal seam, the deformation region around the borehole can be divided into three zones (the failure zone, plastic zone, and elastic zone), as shown in Figure 3.\(^{42,43}\) The damage first occurs in the failure zone. In Figure 3, the failure and plastic zones are the areas in which the coal mass is in the postpeak failure stage. The stress on the coal mass is reduced and transferred to areas further from the borehole.\(^{15}\) Many cracks form in the failure zone, and the internal friction angle decreases. The strength of the plastic zone decreases with increasing strain. The displacement gradually occurs under the influence of crustal stress and the gas pressure gradient, resulting in reductions in the available transport space. Therefore, the ranges of the failure and plastic zones have a strong influence on drill cutting transport.
To obtain the borehole deformation region and the changes of stress after drilling, the Rock Failure Process Analysis software RFPA\textsuperscript{2D} is used. This software consists of two-dimensional finite-element code designed to simulate the fracture and failure processes of quasi-brittle materials, its loading behaviors are in accordance with the elastic damage mechanics, and the element's damage threshold obeys both the maximum tensile strength criterion and the Mohr Coulomb failure envelope.\textsuperscript{44} The heterogeneity in the properties of coal masses is defined by the Weibull function.

### 3.1 Physical model

A geometric model was established using the geological and mechanical conditions for the 2145 air-return roadway of China’s Hebi No. 6 coal mine. Assuming that the rock properties are consistent along the radial direction of the borehole, the medium is inhomogeneous. To take this into account, the plane strain model and a Weibull distribution are adopted. The model size is 2 m × 2 m, divided into 40 000 units (200 × 200). Each unit is 10 mm × 10 mm in size (Figure 4).

Because of the restrictions on the model element size, the borehole diameter in the model is set to 120 mm. In accordance with the geological conditions in the Hebi No. 6 coal mine, a 13.5-MPa load is applied in both the horizontal and vertical directions. Boundary conditions are that the bottom of the model is a fixed constraint. The parameters of the model are presented in Table 1.

### 3.2 Numerical results and discussion

The numerical simulation lasted ten steps. Step 1 is the model stabilization stage after loading. In step 2, the borehole was excavated. Steps 3-9 were the deformation stages.

![Geological model of numerical simulation](image)

**FIGURE 4** Geological model of numerical simulation

| Material | Homogeneity | Media of compressive strength | Elasticity modulus | Poisson ratio | Cohesive strength | Internal friction angle |
|----------|-------------|-------------------------------|-------------------|--------------|------------------|------------------------|
| Coal     | 2           | 4                            | 13.5              | 0.3          | 0.5              | 28                     |

**TABLE 1** Model parameters
and step 10 is the end of the borehole deformation. Figure 5 shows the displacement of borehole wall, shear and principal stress distributions, and acoustic emission after drilling. Depending on the simulation result, the change in shear stress, maximum principal stress, and minimum principal stress of the borehole level units (red dotted lines in Figure 5) with the fracturing crack extension are calculated. The red circle marks the original position and size of the borehole wall after excavation. In step 2, the borehole was excavated. A small amount of acoustic emission was observed near the borehole, indicating that there was a small amount of rupture unit around the borehole. The peaks of the maximum principal stress and shear stress appear around the borehole, which indicates that the equilibrium state of the initial stress is broken up, and the stress is redistributed. A stress concentration phenomenon occurs around the borehole, the maximum principal stress of which is close to 30 MPa. In step 4, the borehole is not obviously deformed, the stress concentration shifts to both sides, and the peak of the principal stress decreases. The number of and the energy of the acoustic emissions experience an increase. Based on the stress change around the borehole, the stress distribution can be divided into the following three regions: I (fully unloading area), II (stress concentration area), and III (original stress area). In step 7, the borehole is deformed, the borehole diameter decreased, and the acoustic emission range continues to increase. During this step, regions I and II increase along with borehole deformation. In step 10, the borehole is significantly deformed, and the number of acoustic emissions is maximized. Correspondingly, regions I and II also reach their maximum. When the stress reached a relatively stable state, the maximum principal stress of region I drops to less than 5 MPa, and the radius of region I is 0.165 m. The peak of maximum principal stress appears at the 67th and 133rd units, indicating that the radius of region II is 0.33 mm. According to the simulation results, the radii of the plastic zone and the failure zone are 0.33 and 0.165 m, respectively. The displacement of the borehole wall is 0.01 m.
4 | FIELD TESTS

4.1 | Test site and problems

Hebi No. 6 coal mine belongs to the Hebi coal group, located in Hebi City, Henan Province, China, as shown in Figure 6. The test site was selected at the transportation roadway and air-return roadways of the 2145 working face. The mining area measures 108 m × 1000 m. The mining seam was 2#1 coal seam which burial depth is 488-592 m and has an average thickness of 8 m and an average dip angle of 29°. The coal seam is mainly fragmented coal, and the firmness coefficient is generally between 0.2 and 0.4. The maximum gas pressure and content are 2.5 MPa and 15.9 m³/t, respectively, indicating that the coal seam is at risk of coal and gas outbursts (both the pressure and content exceed the critical values specified in the Regulation of Coal and Gas Outburst Prevention and Control of China). The geological setting is shown in Figure 7.

The mining area is faced with boreholes that can only reach a length of 30-40 m. Since the borehole cannot cover all the coal of the mining area, this will put some areas at risk of coal and gas outburst.

4.2 | Boreholes arrangement

Four hundred boreholes along the coal seam were drilled on the air-return roadway and 420 boreholes on the transportation roadway. The diameter of each borehole is 113 mm, while the design length of each borehole in the air-return and transportation roadways is 55 and 75 m, respectively. Since the average thickness of the coal seam is 8 m, double-row boreholes are adopted, and the spacing between the adjacent boreholes is 1.4 m, as shown in Figure 8.

4.3 | Drilling optimization

Since the drill cutting transport capacity has an upper limit, the ROP must be lower than the maximum value to be able to drill normally; otherwise, it will cause accidents such as stuck pipes and affect the drilling efficiency. To verify the accuracy of the model and increase the length of boreholes, the initial maximum ROP should be defined; the ROP data were obtained at a sampling rate of one sample every 5 m of drilled length. According to the ROP model and the drilling parameters (Table 2), the initial ROP should be controlled

FIGURE 7   Lithology of coal stratum in 2145 working face
below 0.92 m/min. Ease of drilling means that the drill cuttings generated by drilling can be transported in time, and the ROP should be increased appropriately. If difficulty is encountered in drill cutting transportation or drilling, it means that the drill cutting quantity exceeds the transport capacity, and the ROP should be slowed down.

4.4 Field-test results and discussion

The borehole lengths in the air-return and transportation roadways generally reached 60 and 80 m, respectively, indicating that the borehole lengths had increased by more than 50%. This showed that a reasonable ROP can effectively improve the drilling efficiency. However, a small number of boreholes still collapsed after controlling the ROP and is probably due to the small structure at the collapse, resulting in lower coal strength. These data are not common and are therefore excluded.

The ROP data for the field test is shown in Figure 9, which shows the ROP for boreholes at four angles compared to the ROP model. The average and floating ranges of the ROP are included in the figure. The ROP is inversely proportional to the length of the boreholes. The trend has a high similarity to the ROP model. Figure 9A,B show the ROP in the transportation roadway at an inclination of 33° and 29°, respectively. It can be seen from the comparison between Figure 9A,B that the borehole with large inclination has higher ROP. Since the boreholes are inclined upward, the drill cutting transport capacity increased under the action of gravity during drilling, thereby maintaining the ROP at a relatively high level. Figure 9C,D shows the ROP in the air-return roadway at an inclination of 24° and 29°, respectively. Unlike the inclined upward boreholes, the gravity in the inclined downward boreholes becomes a source of resistance to drill cutting transport, thus reducing the ROP.

In addition, the range of the ROP is proportional to the length of boreholes. This is because unlike the ideal state, the coal in the field is not static. As the length of the borehole increases, the influence of different coal properties on the borehole gradually accumulates, thus affecting the ROP. To obtain the accuracy of the model, the errors are normalized (Equation 24).

\[
\text{Normalized Error} = \left| \frac{\text{Actual Maximum Value} - \text{Model Value}}{\text{Actual Value}} \right|
\] (24)
Figure 10 shows the ROP model errors for each kind of borehole plotted as a line plot. Most of the errors are below 15%, indicating that the model has high accuracy. In addition, the average ROP of the field data is mostly below the model value, which also shows that the model has high applicability to those coal seams with similar properties.

However, these errors also command attention. It is well known that coal seams, especially soft coal seams, vary widely in their properties. The model assumes that the properties of coal are consistent along the radial direction, and the drill cutting transport process is also simplified, which inevitably causes errors. In future research, we plan to reduce the hypothesis of the model, make it closer to the reality, and thus reduce the errors. In addition, the model is suitable for coal seams with a firmness coefficient between 0.2 and 1. Since the strength of coal having a firmness coefficient lower than 0.2 is too low, the borehole is difficult to form, and avoiding a collapse accident, in turn, is more difficult. In contrast, for coal seams with a firmness coefficient higher than 1, because of the high strength, the difficulty of drilling increases, and the amount of additional drill cuttings is greatly reduced. This requires consideration of drilling rig parameters and coal seam properties and is more suitable for traditional ROP models.

5 | CONCLUSIONS

1. A new ROP model for soft coal seams with a firmness coefficient between 0.2 and 1 was established, and the maximum ROP was evaluated. The ROP model simplified the drill cutting transport process and coal seam properties. Field tests of boreholes with four angles were carried out in China’s Hebi No. 6 coal mine. The results show that the ROP model has high accuracy, and the normalized errors are below 15%.

2. The equations for drill cutting transport capacity and drill cutting quantity were established. The equations state that (a) the transport capacity is proportional to the wind pressure and inversely proportional to the borehole length, and (b) the drill cutting quantity is proportional to the stress and deformation region.

3. The failure and plastic zones of the soft coal seam were studied by numerical simulation. According to the parameters of Hebi No. 6 coal mine, the radii of the failure and the plastic zones are 0.165 and 0.33 m, respectively. Owing to crustal stress and gas pressure, the coal mass
in the failure zone falls off the borehole wall and is cut into particles by the drill bit, which is the reason for the increase in the drill cutting quantity.

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