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Research on the Grouting Thickness of Roof Purdah in the Mining of Karstic Water-filling Deposit

Shi-gen Fu\textsuperscript{1,2}, Kai-li Xu\textsuperscript{1} and Nai-bao Zhang\textsuperscript{3}

\textsuperscript{1}Northeastern University, Shenyang 110006, China. \textsuperscript{2}China Academy of Safety Science and Technology, Beijing 100012, China
\textsuperscript{3}Laiwu Mining Co., Ltd., Laiwu 271100, China

Abstract

The orebody roof grouting water protecting curtain technology is to drill hole in the orebody roof to enter into the limestone strong aquifer, forming grouting water protecting curtain. The reasonable thickness of ore deposit roof water protecting curtain is critical for the effective precautions against permeability of Karst flood deposits and the safety in production during the mine service period. Currently, there is no uniformly distributed water protecting thickness design specifications for the orebody roof curtain grouting. In allusion to the design conditions of the curtain grouting mining for Gujiatai iron ore, this paper utilized coal, tunnel specifications to determine the thickness of water protecting curtain. On this basis, this paper applied numerical analysis method to conduct simulation analysis and calculation for the stability of the water protecting curtain thickness. The results showed that the curtain thickness based on the empirical formula could not meet the watertight requirements, and it should properly increase the thickness of water protecting curtain in accordance with the sphere of stope room mining influence to ensure safety of exploitation, providing a basis and guidance for the design and construction of water protecting curtain, which had certain important theoretical and practical significance for the mining of Karst water ores.

Keywords: Karst water ore; thickness of water protecting curtain; empirical formula; numerical analysis

1. Introduction

Orebody roof curtain grouting technology plays a significant role in tackling the hydro-geological and engineering geological problems and the social and environmental issues caused by dewatering drainage technical methods (such as ground subsidence, groundwater depletion and village relocation and so forth caused by dewatering drainage)\textsuperscript{1\textendash}3. With respect to the orebody roof that is strong aquifer water-rich ore deposit, in order to prevent any water damage major accident in the production process, it usually grouts on a certain thickness of surrounding rock at the hanging direct roof of the orebody, to seal the cracks, forming a certain thickness of artificial water-resisting layer\textsuperscript{4\textendash}6. The theoretical safety thickness
of this water-resisting layer resistant to the hydrostatic pressure is called as the thickness of water protecting curtain.

Due to the presence of Karst water in the roof, if the thickness of grouting curtain is too small, it will increase the likelihood of loss of stability for the roof. During the service period of the mine, it may appear caving, collapse in the roof water protecting curtain, leading to the risk of roof water-inrush accidents and so on; if increase the thickness of water protecting curtain, the production costs will be increased accordingly, which is economic unreasonable. Therefore, the determination of a reasonable roof water protecting curtain thickness of the Karst flood ore deposit is crucial for the effective prevention of roof water permeability for Karst flood ore deposit as well as the safety production and stability during the service period of the mine. For the orebody roof featured with strong aquiferous Karst flood ore deposit, drilling directly into the aquifer in the mine, grouting to form water protecting curtain is a new technology for the mining of Karst flood ore deposits. However, non standard water protecting thickness design specifications have been put forward for the orebody roof curtain grouting at present. In this paper, the coal and tunnel design specifications was utilized to calculate and determine the thickness of water protecting curtain. On this basis, the numerical analysis \cite{7-9} method was applied to simulate and analyze the reasonableness of the thickness of water protecting curtain.

2. Project Overview

Gujiatai iron ore is a typical Karst flood ore deposit, which is featured with complex mining hydrogeological conditions, and rivers across the central orebody on the surface. The mining area consists of two aquifers, including the Quaternary alluvial proluvial sand gravel aquifer and Ordovician limestone Karst fissure aquifer, while the remaining strata are water-resisting layers. Specifically, the roof is limestone aquifer, orebody and orebody under stratum diorites are aquitard and water-resisting layers. It is able to form a watertight closure mining space to shape grouting curtain in the orebody roof aquifer and ore rock boundary zone.

According to the layout of geological control project, the sub-sectional height of mining area was 10m. In the orebody footwall rock mass of each 10m-height small section, taking the drilling drift as the grouting contact drift, deploy drilling chamber and grouting borehole according to the regulated layout spacing of $10 \times 10$m, while the roof grouting curtain design was shown in Figure 1.

![Figure 1 The profile map of roof curtain grouting project](image-url)
3. Calculation of the Curtain Grouting Thickness

3.1. Coal Mine Empirical Formula

In accordance with the mechanical theory, the roadway impermeable understratum was considered as a beam fixed at both ends, and bore uniformly distributed hydrostatic pressure loads, and took account of the limit equilibrium conditions into consideration, the theoretical safety thickness formula of artificial curtain water-resisting layer against of hydrostatic pressure \[10\] is:

\[ M = \frac{L(\sqrt{\rho^2 L^2 + 8K_p H - \rho L})}{4K_p} \]  \tag{1}

Where,

- \( H \) —— The actual pressure value applied on the water-resisting layer on the understratum, kN/m²;
- \( M \) —— The critical thickness of water-resisting resistant to hydrostatic pressure (H);
- \( L \) —— The width of mining working profile space, m;
- \( K_p \) —— The average tensile strength of understratum water-resisting layer, kN/m²;
- \( \rho \) —— The rock density of understratum water-resisting layer of, t/m³.

The aforesaid calculated value was measured in the laboratory according to complete rocks, which was difficult to represent the actual rock strength. In accordance with the field hydrostatic pressure test data, due to the fissure development, the actual rock strength is lower than that of the rocks, and hence it should be multiplied by a certain safety factor in the use of the formula (1). Based on the physical and mechanical parameters of orebody roof surrounding rocks, the calculated thickness of orebody roof water protecting curtain, would be determined as the initial design value of water protecting curtain thickness.

The experimental determined \( K_p = 4.36 \text{MPa}, \rho = 2.7 \text{t/m}^3 \) for the mine, assuming the maximum width of mining working profile space was \( L = 40 \text{m} \). Thus, it could calculate the thickness of water protecting curtain depending on the hydrostatic pressure at different depths. For instance, if the mining was carried out in the level of - 120m, the hydrostatic pressure \( H = 2.92 \text{MPa} \), then calculate \( M = 23 \text{m} \).

3.2. Standard Methods of Highway and Tunnel

"Design Specifications of Highway and Tunnel" (JTG D70-2004)\[11\] provides that, the boundaries of shallow and deep tunnels, should be synthetically determined in accordance with the equivalent load height, geological conditions, construction methods and other factors. The determination formula based on the equivalent load height was:

\[ H_p = (2 \sim 2.5)h_q \]  \tag{2}

Where,

- \( H_p \)—— The depth of shallow buried tunnel boundary (m),
- \( h_q \)—— The equivalent load height (m) was calculated in line with the following formula:

\[ h_q = \frac{q}{\gamma} \]  \tag{3}

Where, \( \gamma \) is the bulk density of surrounding rock [kN/m³], \( q \) is the vertical uniformly distributed pressure of deep buried tunnel, which were calculated in accordance with the following formula:

\[ q = \gamma h \]  \tag{4}

\[ h = 0.45 \times 2^{h-1} \omega \]  \tag{5}

Where:

- \( q \)—— vertical uniformly distributed pressure (kN/m2);
\( \gamma \) —— weight of surrounding rock (kN/m²);
\( s \) —— surrounding rock level;
\( \omega \) —— width influence coefficient, \( \omega = 1 + i (B-5) \);
\( B \) —— tunnel width (m);
\( i \) —— The coefficient of increase and decrease of surrounding rock pressure for each 1m increase or decrease of \( B \), based on the surrounding rock vertical uniformly distributed pressure at \( B = 5m \), when \( B < 5m \), \( i = 0.2 \); while when \( B > 5m \), \( i = 0.1 \). The application of Equation (5) must meet the following conditions: 1) \( H/B < 0.7 \), \( H \) is the excavation height of tunnel; 2) general surrounding rock that would not produce a significant bias and expansive force.

Under the construction conditions of drilling and blasting method, for \( IV \sim VI \) grade surrounding rock: 
\[ h_p = 2.5h_q \]

For \( I \sim III \) grade rock surrounding rock: 
\[ h_p = 2.0h_q \]

References [12] provided the depth of shallow and deep buried tunnel boundaries under different tunnel widths, different surrounding rock levels according to the above-mentioned formula, as shown in Figure 1.

![Figure 2](image)

Figure 2 The depth of shallow and deep buried tunnel boundaries

The calculation results revealed that: the higher level of the surrounding rock, namely, more serious breakage, the thickness and span of the corresponding minimal cover will be greater. Since the surrounding rock of Gujiatai iron ore is \( III \)-grade, it inferred that the thickness of water protecting curtain was approximately 18m.

4. Stability Analysis of the Thickness of Waterproof Grouting Curtain

Based on the abovementioned theoretical calculation, roof curtain grouting was firstly carried out for the pre-mining area, to form a thickness of 23m of water plugging obstruction zone, and applied point-pillar upward horizontal stratification filling mining method. The stope room was deployed along the strike, with the length of 27.5m, the width of 20m, stage height of 50m, sub-sectional height of 10m, leaving a 5m of rib pillar between every two stope rooms. The shallow-hole blasting stoping method was utilized, and reserved \( 5m \times 5m \) standard point-pillar, with the point-pillar center distance of 12m, in addition, the rib-pillar and point-pillar were permanent natural ore pillars.

4.1. Computation Model

The three-dimensional nonlinear solid-fluid coupling numerical simulation [13], was used for the computation. With respect to the computation model, the X-direction width was 600 m, the Y-direction length was 315 m, and the height was about 152 m. The excavating horizontal direction was -120m, comprised by two stope rooms in the computation model with the length of 40m, as shown in the stope room mining 3-D model map of Figure 2.
In accordance with the field sampling and rock mechanics test results, the rocks presented clear plastic deformation characteristics at different confining pressure conditions. In this paper, Mohr-Coulomb yield criterion was used to determine the damage of rock mass, and the computation formula was as follows:

\[ f_s = \sigma_1 - \sigma_3 \left( \frac{1 + \sin \varphi}{1 - \sin \varphi} \right) - 2c \sqrt{\frac{1 + \sin \varphi}{1 - \sin \varphi}} \] (6)

Where, \( \sigma_1, \sigma_3 \) was the maximum and minimum principal stress respectively; \( c, \varphi \) was cohesive force and friction angle respectively. When \( f_s > 0 \), the material will suffer shear failure. Under the normal stress conditions, the tensile strength of soil is low, and hence it is able to determine whether the rock has suffered tensile failure in accordance with the tensile strength criteria \( (\sigma_3 \geq \sigma_T) \). The rock mechanical parameters used in this paper were listed in Table 1.

<table>
<thead>
<tr>
<th>Name</th>
<th>Density ( d(\text{kg/m}^3) )</th>
<th>Modulus of Elasticity ( E(\text{MPa}) )</th>
<th>Poisson’s Ratio ( \nu )</th>
<th>Cohesion ( c(\text{MPa}) )</th>
<th>Internal Friction angle ( \varphi )</th>
<th>Permeability ( k(\text{Darcy}) )</th>
</tr>
</thead>
<tbody>
<tr>
<td>Quaternary</td>
<td>2000</td>
<td>300</td>
<td>0.3</td>
<td>0.15</td>
<td>22</td>
<td>( 3 \times 10^{-7} )</td>
</tr>
<tr>
<td>Tertiary</td>
<td>2300</td>
<td>3780</td>
<td>0.26</td>
<td>0.025</td>
<td>27</td>
<td>0</td>
</tr>
<tr>
<td>Limestone</td>
<td>2700</td>
<td>10000</td>
<td>0.22</td>
<td>0.3</td>
<td>30</td>
<td>2.5 \times 10^{-7}</td>
</tr>
<tr>
<td>Magnetite</td>
<td>3740</td>
<td>11300</td>
<td>0.24</td>
<td>0.3</td>
<td>37.8</td>
<td>0</td>
</tr>
<tr>
<td>Skarn</td>
<td>2800</td>
<td>14200</td>
<td>0.18</td>
<td>1.2</td>
<td>25</td>
<td>0</td>
</tr>
<tr>
<td>Grout</td>
<td>2760</td>
<td>11000</td>
<td>0.2</td>
<td>0.35</td>
<td>32</td>
<td>0</td>
</tr>
</tbody>
</table>

4.2. Stability Analysis of Grouting Curtain Zone

In order to probe into the situation that may arise in the orebody surrounding rocks due to mining operation, it should systematically simulate the entire mining history and mining process. The excavation process should be carried out in successive steps, once excavation of a section, in full-thickness.

4.2.1 Analysis of the Maximum Principal Stress State

Figure 4 shows the maximum principal stress distribution map of the surrounding rocks after the stope room mining. With the increase in stope room mining area, the distribution state of the maximum principal stress was basically not changed, complying with the gradually increase trend from top to bottom, and extreme value of the maximum principal stress was at the bottom of the mode, namely, 5.51MPa.
With the gradual mining process of the stope room, the surrounding rock stress unloading area of the stope room was gradually increased. Subjected to the impact of mining disturbance, the phenomenon of stress concentration was still existed in the iron ores around the stope room.

Figure 4  The maximum principal stress map of the surrounding rock after the mining of the stope room

4.2.2 Analysis of Overburden Strata Failure State

Figure 5 showed failure field map of the surrounding rock along the inclination after the mining of Stage 1, indicating that the stope room roof failure height was 10m, which was within the control range of grouting, and hence the stope room would not subject to the impact of flooding.

At the end of the Stage II mining, the stope room roof failure height was increased to 32m, exceeding the control range of grouting (selected grouting thickness of 23m in the computation). The stope room was affected by the roof aquifer, 12m-range of roof beneath the Tertiary water-resisting layer was damaged, existing potential hydraulic connection with the skylight area, which threatened the safety of the mine.

With the increase of substep extraction layers, the failure height of the stope room was constantly increased. At the end of the Stage III, the failure height of roof had been up to 44m, which had exceeded the control range of grouting, and the extent of failure tended to gradually increase along the orebody. The simulation results showed that in the subsequent two-stage extraction processes, the failure height of roof remained essentially unchanged. Figure 5 revealed that the roof suffered shear failure along the orebody inclination due to the influence of the stope room mining disturbances. Table 2 listed the extent of failure statistics status for substep mining.

Table 2  Statistics of the extent of failure

<table>
<thead>
<tr>
<th>Excavation Stage</th>
<th>Roof Failure Height (M)</th>
<th>Communication Range of Roof and Water-resisting Layer (M)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Stage I</td>
<td>10</td>
<td>—</td>
</tr>
<tr>
<td>Stage II</td>
<td>32</td>
<td>12</td>
</tr>
<tr>
<td>Stage III</td>
<td>44</td>
<td>58</td>
</tr>
<tr>
<td>Stage IV</td>
<td>44</td>
<td>58</td>
</tr>
<tr>
<td>Stage V</td>
<td>44</td>
<td>58</td>
</tr>
</tbody>
</table>
4.2.3 Displacement Status Analysis

As illustrated in Figure 7, it showed the displacement change of surrounding rock after stope room mining. In accordance with the distribution of the displacement vector field, it indicated that the displacement direction of surrounding rock mainly pointed of the stope room. Due to the application of point-pillar upward horizontal stratification filling mining method, the mined iron ore will be backfilled immediately, resulting in lower displacement caused by the surrounding rock, with the maximum displacement of 1.8cm.

The maximum vertical displacement and X-direction horizontal displacement of surrounding rock were appeared inside the surrounding rock adjacent to the stope room. In the wake of the increase in mining sub-sections of stope room, the displacement of the surrounding rock was relatively increased, but the amplitude of variation was lower. In addition, the Y-direction horizontal displacement of the surrounding rock was negligible.
5. Conclusion

This paper, in allusion to the retained actual conditions of roof curtain grouting thickness in Gujiatai iron ore, applied the numerical analysis method to probe the design reasonableness of its water protecting curtain, indicating that:

(1) With the gradual exploration, the failure height of grouting curtain zone was continuously increased, exceeding the scope of grouting. When the mining was up to a certain height, the range of roof failure was remained essentially unchanged. Due to the impact of mining disturbance, the roof was suffered shear failure along the ore body inclination;

(2) The thickness of the water protecting curtain based on empirical formula should be multiplied by the coefficients of 1.5 to 2.5 in accordance with the hydrogeological conditions, which could meet the mining needs of Karst flood ore deposit;

The aforementioned results are being used in the production of Gujiatai iron ore curtain grouting practice, and it has indicated that the thickness of curtain grouting can properly safeguard the safety of mine exploitation in line with the monitoring results.

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References