Numerical Method Based on Large Deformation Damage Constitutive Model and Application

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ABSTRACT

For the high precision calculation of mechanics mechanism for surrounding rock with large deformation, numerical calculation and application of large deformation damage constitutive model have been researched. Based on large deformation finite element theory, large deformation calculation process was proposed and the statistical constitutive model and parameter estimation method of rock damage and evolution analyzed. Fully consider the region that structural unit change from elastic state to plastic state, the large deformation finite element analysis method been studied, and large deformation mechanics model of mining engineering roadway was proposed, which through weighted average method of around the node distance to calculated the excavation stress of boundary nodes. Based on engineering stable energy criterion, balance area scope and location evaluation method, as well as shear damage security evaluation method that the stability of the evaluation criteria for large deformation damage calculation was analyzed. The numerical method was used in the excavation simulation of Zheng jia-po iron mine, the results show that the calculation method can be qualitatively analyze the impact of different mining sequence and technology for the stope surrounding rock stability and get the best mining plan, but also provides important information for decision-making of deep deposit mining technology.

KEYWORDS: surrounding rock deformation; damage constitutive equation; numerical calculation; mining engineering; stability

INTRODUCTION

Due to various reasons, rock mass fissure of mine excavation is extremely development and poor stability. Moreover, each process has a certain time interval in mine engineering, most
surrounding rock show significant time effect in the process of deformation and failure, and stope and roadway surrounding rock have large deformation in virtue of continuous stress release. But for the surrounding rock deformation, numerical simulation methods are often used to reproduce the process of instability in the research work. Such as: Huang\textsuperscript{1} researched on numerical calculation method of the aging deformation in tunnel surrounding rock with characteristics of water degradation, and the viscous elastic-plastic model was inferred; Song\textsuperscript{2} used FLAC3D to study the surrounding rock mechanics status of chute reinforcement around, from the aspects of stress distribution of rock mass, plastic zone the movement of rock mass and so on; Yu\textsuperscript{3} considered the rock mass as viscoelastic medium, describing the rheological properties of rock mass with burgers model, and presenting a numerical method; Wu\textsuperscript{4} built a numerical analysis framework of elastic forecast-plasticity modification-damage modification, and gave stress update algorithm with unconditionally stable state and the corresponding algorithm consistent tangent modulus. For the damage constitutive model of surrounding rock, Li\textsuperscript{5} solved the equivalent strain of elastic damage and equation of damage evolution and established damage yield criterion by using the basic principle of energy dissipation; Zhang\textsuperscript{6} researched on cumulative damage range of large section tunnel by using double side heading method construction of pushing type reciprocating blasting operation.

The above studies have obtained great progress, generating positive role on engineering construction. However, at present most of the finite element numerical analysis cannot coincide with large deformation of surrounding rock in actual mine, cannot more accurately reflect the rock failure process. Therefore, to study on large deformation numerical calculation based on damage constitutive model has important significance in guiding mining engineering.

**BRIEF INTRODUCTION OF LARGE DEFORMATION FINITE ELEMENT THEORY AND METHOD**

In general, from the two aspects of macroscopic and microscopic, numerical calculation software can reveal the basic law of the rock deformation and failure. In the microscopic aspect, we should begin from the rock microscopic damage, introducing damage variable, establishing damage evolution constitutive equation, and then to establish the damage constitutive equation, providing theoretical basis of understanding failure mechanism of surrounding rock deformation and control of ground pressure; while in the macroscopic aspect, from the rock mass deformation relationship, introducing mechanics analysis method of material and geometrical nonlinear, fully revealed field range of surrounding rock stress and strain. On this basis, reasonable control measures were put forward.

Here, large deformation analysis put forward the research methods are as follows: the finite element program development for large deformation damage analysis, program verification, to establish geological model and determine parameters of rock mass, to establish numerical analysis model, numerical calculation and result analysis, the optimal scheme decision. The specific technical route is shown in Figure 1.
In mining engineering, nonlinear analysis includes material nonlinearity and geometric nonlinearity two categories. One is the material nonlinearity, and the other is geometric nonlinearity, therefore, to modify the basic characteristics of material should be considered after the material occurs deformation. For the geometric nonlinear problems, equilibrium equations must be written that relative to the geometric position after deformation which previously predicted. Strictly speaking, all the problems should use already deformation position to give its equilibrium equation, but if the basic characteristics of the problem did not change with deformation, it can be described by the geometry position of deformation before. Namely: when the displacement of the object was small, the geometric displacement of deformation before can be approximate described. But for condition of large displacement, the impact of displacement on the basic characteristics must be considered, geometric position before deformation cannot be used to describe the basic characteristics of objects. Due to the geometrical position that had been deformed was unknown; it had brought certain degree of complexity to deal with geometrical nonlinear problem. Therefore, iterative method with moving coordinate can be adopted to solve it.

Figure 1: Flow chart of large deformation numerical calculation

Brief introduction of rock damage evolution statistical constitutive model

As we all know, rock mass is heterogeneity of void, pore, fracture, interface and other defects, the material under the action of load, the mechanical properties of its internal structure would continue to change. Krajcinovic\textsuperscript{7} from random distribution of defects in the rock material inside organically combined the continuum damage theory with statistical strength theory, presenting a statistical damage constitutive model. It had a simple form, parameters that were easily acquired, and can reflect the characteristics of rock evolution characteristic in a certain extent, opening up a new way for the study of rock damage evolution constitutive model.
(1) The basic concept of injury. In case of a simple drawing pillar, the pulling force with $P$, the rod cross section area with $A$, the total area of pore in cross section with $A_0$, due to the presence of porosity, and the effective bearing area $A$ was less than the cross section area in macro. The actual bearing area in the cross section was $A_{ef}$, therefore,

$$A_{ef} + A_0 = A$$

(1)

So, the stress acted on the uninjured material of the pillar that was called as effective stress $\sigma^*$,

$$\sigma^* = \frac{P}{A_{ef}}$$

(2)

The expression of effective stress that defined by damage variable $D$ was

$$\sigma^* = \frac{\sigma}{1-D}$$

(3)

In the formula, $D = A_0 / A$.

(2) The establishment of damage constitutive relation. Influence of damage on strain behavior through the effective stress to reflect, that is to say, the constitutive relation of damage material only to modify stress in the original no damage constitutive relationship to the effective stress. This was the Lemaitre strain equivalence hypothesis. Accordingly, the basic damage constitutive relation can be established as

$$[\sigma] = \frac{1}{1-D} [\sigma^*] = \frac{1}{1-D} [C][\varepsilon]$$

(4)

In the formula $[C]$ was Elastic matrix material, $D$ was damage variable, $[\sigma^*]$ was matrix of effective stress, $[\varepsilon]$ was matrix of strain.

(3) To establish the damage evolution equation of rock. Taking a cell from the rock, and the element size large enough to contain many micro fractures and micro void; but at the same time was small enough, small enough to be considered as a particle in continuum mechanics, on the assumption that the rock material was distributed evenly, and its defects was measured by their bearing capacity attached to the uniform medium, pure physical image of this defect would not exist. This would ensure that the people still can base on the basic laws of elastic mechanics at the microcosmic level. Due to the internal structure of the rock material was very uneven, there may be exist many weak links of different strength, each element had the different strength, considering the damage of rock material in the loading process was a continuous process. Therefore, hypotheses: ①The rock material properties of isotropic in macroscopic view; ②The rock micro unit before failure obeyed to Hooke's law, namely the element with wired elastic properties; ③The
micro unit strength obeyed Wei-bull distribution, the probability density function was shown in the follow

\[ P(F) = \frac{m}{F_0} \left( \frac{F}{F_0} \right)^{m-1} \exp \left[ -\frac{F}{F_0} \right] \]  

(5)

In the formula, \( F \) was distribution variable of micro unit strength random distribution; \( m \) and \( F_0 \) were the Waybill distribution parameters, which reflected the mechanical properties of rock material. Damage of rock materials was caused by the element continuous wreck. The number of element that had been destroyed under one level loads was \( N_f \), statistical damage variable was defined as the ratio of element number that had been damaged and number of total element.

\[ D = \frac{N_f}{N} \]  

(6)

In this way, number of element that had been destroyed was \( N \cdot P(y)dy \) in arbitrary interval \([F, F + dF]\). When loading on arbitrary level \( F \), number of element that had been destroyed was

\[ N_f(F) = \int_0^F N \cdot P(y)dy = N \left[ 1 - \exp \left[ -\frac{F}{F_0} \right] \right] \]  

(7)

Taking formula (6) into formula(7), damage evolution variable \( D \) equation can be obtained,

\[ D = \frac{N_f}{N} = 1 - \exp \left[ -\left( \frac{F}{F_0} \right)^m \right] \]  

(8)

Statistical constitutive equation of rock damage evolution with parameter estimation and significance

(1) Estimation of parameters \( m \) and \( F_0 \) in the constitutive equation. Estimation of parameters \( m \) and \( F_0 \) in rock damage evolution statistical constitutive equation can resort to rock three - axis stress strain curve, and by fitting the three - axis stress strain curve to calculate the parameters \( m \) and \( F_0 \).

(2) Model validation. In order to verify the rock damage evolution statistical constitutive model proposed in the research, we also referred to the related data to verify. Here, the rock elastic modulus \( E = 90 \times 10^3 \text{ MPa} \), Poisson's ratio \( \mu = 0.25 \), internal friction angle \( \varphi = 31^\circ \sim 39^\circ \), comprehensive consideration of various factors, statistical constitutive equation for rock damage evolution was shown in the follow
(3) The physical significance of the parameters \( m \) and \( F_0 \). Parameter \( E \) represented the initial elastic modulus of nondestructive material in the formula (9), which reflected the elastic properties of rock material. When fixed \( E \) and \( m \), with the increase of \( F_0 \), the peak strength increased. \( F_0 \) reflected the rock macroscopic average strength; parameter \( m \) reflected the rock micro unit strength distribution concentration degree, namely the \( m \) was bigger, the micro unit strength distribution was more concentrated, the brittleness of the material was also higher. With the increase of \( m \) value, the curve after the peak more steep, brittleness increased; on the contrary, if the value of \( m \) was small, increasing the material ductility. This can explain parameter \( m \) that reflected the rock brittle degree.

**DESIGN OF LARGE DEFORMATION FINITE ELEMENT PROGRAM**

**Design of the finite element analysis method**

When carrying on finite element analysis of elastic-plasticity large deformation, analysis problem of the transition elements was considered. Generally speaking, in the process of stepwise load, the plastic zone expanded, some units were in the elastic region, but they were closed to plastic region, so in the process of adding load, they came into plastic zone. The units composed of the region known as the transition region. For the unit of transition region, because in the loading process from elastic state to the plastic state, if still simply by \([D]\) or \([D]_{ep}\) forming stiffness matrix that would lead great error, on the elastoplastic matrix must be modified, namely to calculate the weighted average elastic-plastic matrix

\[
[D]_{ep} = m[D] + (1 - m)[D]_{ep}
\] (10)

In order to determine the correction coefficient \( m \) in the formula, a generally applicable and convenient method was proposed, and applied in the calculation. Set before the load applied, the first stress invariant was \( I_{10} \) and the second stress partial invariant was \( J_{20} \). After loading respectively \( I_1 \) and \( J_2 \), then

\[
F = \alpha \left( \frac{I_1}{10} + \frac{J_{20}}{20} \right) - k_0
\] (11)

\[
F_i = \alpha \left( \frac{I_1}{10} + \frac{J_{20}}{20} \right) - k_0
\] (12)
For the joint unit, set before loading, shear stress was $\tau_{s0}$ and normal stress was $\tau_{n0}$; after loading respectively $\tau_{s}$ and $\tau_{n}$, then

$$F = |\tau_{s0}| - c + \sigma_{n0} \tan \varphi$$

(13)

$$F_{1} = |\tau_{s}| - c + \sigma_{n} \tan \varphi$$

(14)

In this way, calculation of the correction parameters as follows

$$m = \left| \frac{F}{F-F_{1}} \right|$$

(15)

If the transition units occurred, it must be repeated iteration, until convergence. In a computing incremental step may be occur unloading, at this time consider as elastic problem. To carry on finite element elastic-plasticity analysis of large deformation, a wide range of plastic zone may appear, then, convergence rate can be very slow. In order to improve the convergence speed of computation, to improve the computational efficiency, the introduction of calculation relaxation factor was introduced to change the convergence.

The relaxation factor depended on the plasticity degree of the unit, which was calculated in each iteration, the bigger of $F$ value ($F > 0$), indicating that the stress far away from yield surface; when the Drucker-Prager Yield Criterion was used as the yield criterion, there existed

$$F_{1} = \alpha_{0}I_{1} + J_{2}^{1/2} - k_{0} > 0$$

(16)

Or

$$\alpha_{0}I_{1} + J_{2}^{1/2} > k_{0}$$

(17)

Definition of the ratio

$$\lambda = \frac{k_{0}}{\alpha_{0}I_{1} + J_{2}^{1/2}} = \frac{k_{0}}{F + k_{0}}$$

(18)

It can be used to characterize the plasticity degree of the unit, and taking

$$\xi = 1 - \lambda$$

(19)

Obviously, the bigger of $F$, the smaller of $\lambda$, and the bigger of $\xi$. When $F \rightarrow 0$, there exist $\lambda \rightarrow 1$ and $\xi \rightarrow 0$. Taking $\xi$ as calculation relaxation factor, then each iteration, equivalent node force distribution of nonlinear stress and initial stress were as follows:

$$[\sigma_{j}]_{j} = [\sigma_{j}]_{j-1} + (1+\xi)[\Delta \sigma_{j}]_{j}$$

(20)
\[ [\Delta F_j]_i = (1 + \xi) \int [B^j]_i \Delta \sigma_j \, dV \]  \hspace{1cm} (21)

where \( i \) represented the load increment step, \( j \) represented number of iterations. The effect of adopting calculation relaxation factor above-mentioned can obviously reduce the iterative time effect.

### Mechanical model

The underground mining engineering was different from the general mechanical structure, it was the load first, structure second. To the mining simulation of the mining process, it was actually to solve second stress field and displacement field. Therefore, we must firstly make clear that the mechanics analysis model of mining engineering, usually divided into external loading model A, reverse stress model B and the original rock stress model C. There were two methods to solve finite element analysis of mining engineering: the first method was to combine model B and model C. Displacement fields caused by actual excavation can be obtained by solving model B, but actual stress field caused by excavation can be obtained by stress superposition of model B and model C. The calculation results of stress and displacement can be expressed as

\[ [u] = [u]_B \]  \hspace{1cm} (22)

\[ [\sigma] = [\sigma]_B + [\sigma]_C \]  \hspace{1cm} (23)

Another method was to combine model A and model C, actual stress field can be obtained by solving model A, but actual displacement field can be obtained by displacement superposition of model A and model C. The calculation results of stress and displacement can be expressed as

\[ [u] = [u]_A - [u]_C \]  \hspace{1cm} (24)

\[ [\sigma] = [\sigma]_A \]  \hspace{1cm} (25)

### Simulation calculation of excavation

Calculation method of nodal force before excavation was interpolation method and around the node weighted average method. For some special cases, interpolation method was not feasible. The program used around the node distance weighted average method to calculate the boundary node stress. The stress calculation formula of boundary node A can be expressed as
$$\sigma^A = \frac{\sum_{k=1}^{m} \sigma_k L_k}{\sum_{k=1}^{m} L_k} \quad (26)$$

In the formula, $\sigma^A$ was the stress of node $A$; $m$ was the number of number units around the node; $\sigma_k$ and $L_k$ were respectively stress of $k$ units around node $A$ and distance of centre of $k$ unit and node $A$, $L_k$ can be expressed as

$$L_k = \sqrt{(x_d - x_k)^2 + (y_d - y_{k1})^2} \quad (27)$$

In the formula, $(x_k, y_k)$ and $(x_d, y_d)$ were respectively center of unit and coordinate of node $A$.

In order to carry out the finite element analysis of excavation engineering, release load caused by excavation must be equivalent moved into the boundary nodes. Suppose there were three consecutive nodes in boundary node, respectively $i - 1$, $i$ and $i + 1$, then equivalent nodal force $P_{xi}$ in $x$ direction and $P_{yi}$ in $y$ direction of node $i$ can be calculated in the following formula

$$P_{xi} = \frac{1}{6} \left[ 2\sigma_{x,i}(b_1 + b_2) + \sigma_{x,i-1}b_1 + \sigma_{x,i+1}b_2 + 2\tau_{xy,i}(a_1 + a_2) + \tau_{xy,i+1}a_1 + \tau_{xy,i-1}a_2 \right] \quad (28)$$

$$P_{yi} = \frac{1}{6} \left[ 2\sigma_{y,i}(a_1 + a_2) + \sigma_{y,i-1}a_1 + \sigma_{y,i+1}a_2 + 2\tau_{xy,i}(b_1 + b_2) + \tau_{xy,i+1}b_1 + \tau_{xy,i-1}b_2 \right] \quad (29)$$

In the formula, $a_1 = x_{i-1} - x_i$, $a_2 = x_{i-1} - x_i$, $b_1 = y_{i-1} - y_i$, $b_2 = y_{i-1} - y_i$, $(x, y)$ was coordinate of boundary node.

**Evaluation criteria of stability**

(1) Energy criterion of engineering stability. Large deformation elastoplastic finite element program for the stability evaluation of mining engineering, not only according to elastoplastic or crack zone appear or not to determine the stability of mining engineering structure. According to the range of elastic plastic zone size and position to evaluate the stability of mining engineering were also no sufficient basis. And imitating limit equilibrium method to calculate safety factor was not reasonable. Because, according to the stress distribution obtained by limit equilibrium method that significant different from stress distribution obtained by finite element analysis, it was more difficult to find the critical failure surface. Assessment method of shear damage safety degree only can evaluate structural local stability, no way to know whether the overall structure instability or not. From the view of system energy of mining engineering,
to evaluate the overall stability of structure, it did not need to care about the local stability problem, this was expected, and it had wide development prospects.

For a mining engineering system, suppose system displacement with $[U]$, strain with $[E]$, stress with $[S]$. Then the system energy (that was the elastic strain energy and plastic dissipation energy of the system) was

$$U_E = \int_V [E]^T [S] dV$$

(30)

Suppose applying virtual displacement $\delta[u]$ of an arbitrarily tiny and not contravention the geometric conditions in the system and new state can be obtained, so the external work was

$$\delta W_r = [u]^T [R]$$

(31)

In the formula, $[R]$ is external stress; if in the process of applying virtual displacement, load work $\delta W_r$ no more than internal energy $U_E$ increase, then, the equilibrium state of mining engineering system was stable. If a set of virtual displacements can be found, the load work greater than increase of internal energy, obviously, the excess energy would be converted into the kinetic energy of the system, namely the equilibrium state of the system was unstable. Therefore, the instability criterion can be expressed as

$$\delta U_E + \int_V [E]^T \delta[S] dV - \delta W_r < 0$$

(32)

Because the state was a state of equilibrium, it was satisfied with the equation of virtual work. Therefore, the criterion can be rewritten as

$$\int_V [E]^T \delta[S] dV < 0$$

(33)

Namely:

$$\int_V [u]^T [K] \delta[u] dV < 0$$

(34)

In the formula, $[K]$ was the system whole stiffness matrix of large deformation finite element analysis. Establishment of above formula completely depended on the system stiffness matrix whether positive definite or not. If $[K]$ positive definite, then the above formula was established, and the system overall was in the stable state. Otherwise, the system would be unstable. Therefore, positive definite of the large deformation finite element system whole stiffness matrix can be used to directly determine the system stability, performance on large deformation finite element solution procedure was to solve whether convergence or not. If not convergence, then the structure would be unstable. This was the energy criterion of structural stability in mining engineering. There was a need to explain, this was physically unstable. Another kind of instability was unstable in mathematical, because of the grid division irrational factors that caused the system not convergence.
(2) Evaluation method of equilibrium area range and position. The range size and position of equilibrium area (including plastic zone and fracture zone) can reflect wreck damage extent of engineering rock mass, and the wreck position can reflect the influence degree of destruction region on structural stability. The combination of these two can reflect the stable state of structure in a certain extent.

(3) Safety evaluation method of shear failure. Shear failure safety evaluation introduced into evaluation of plastic zone, the ratio value of unit material limit bearing of principal stress difference $\left(\sigma_1 - \sigma_3\right)_f$ and unit actual principal stress difference can be used to measure degree of unit shear damage; the coefficient was called as unit shear wreck safety degree $K_i$. To analysis of the plane problem, it can be expressed as

$$K_i = \frac{\left(\sigma_1 - \sigma_3\right)_f}{\left(\sigma_1 - \sigma_3\right)}$$

(35)

In the formula, $\left(\sigma_1 - \sigma_3\right)_f$ was two times of limit shear strength of unit material, which depended on cohesion and internal friction angle, namely

$$\left(\sigma_1 - \sigma_3\right)_f = \frac{2c_i \cdot \cos \varphi_i + 2\sigma_3 \sin \varphi_i}{1 - \sin \varphi_i}$$

(36)

In the formula, $c_i$ and $\varphi_i$ were respectively cohesion and internal friction angle of unit material, $\sigma_1$ and $\sigma_3$ were respectively maximum and minimum principal stress of unit actual bearing.

ENGINEERING APPLICATION IN MINING DESIGN

Model and calculation parameters of rock mass

Zheng Jia-Po iron deposit was composed of five ore bodies, ore bodies were layered, approximation layered, arranged in parallel or inclined to monoclinic form output, occurrence was consistent with the surrounding rock, with stable horizon. The north ore body was decisive boundaries with roof and floor of surrounding rock, and south ore body was gradient transition with roof and floor of surrounding rock. According to the geological and related information, the surrounding rock and backfill body filling using Mohr-Coulomb yield criterion, mechanical parameters of ore and main rock strata were listed in Table1,

The mechanics parameters of backfill body obtained by relevant information were shown in Table 2, according to hydrogeological and filling characteristics of the mining area, infiltration parameters were shown in Table 3. The calculation model was shown in Figure 2.
Table 1: Numerical calculation parameters

<table>
<thead>
<tr>
<th>Strata Title</th>
<th>Density /g.cm$^{-3}$</th>
<th>Uniaxial compressive strength /MPa</th>
<th>Uniaxial tensile strength /MPa</th>
<th>Cohesion /MPa</th>
<th>Internal friction angle /°</th>
<th>Modulus of elasticity /GPa</th>
<th>Poisson ratio</th>
</tr>
</thead>
<tbody>
<tr>
<td>Quaternary</td>
<td>2.1</td>
<td>35</td>
<td>1.0</td>
<td>0.12</td>
<td>35</td>
<td>20</td>
<td>21</td>
</tr>
<tr>
<td>Roof: containing garnet biotite diopside granulitite</td>
<td>2.6</td>
<td>50</td>
<td>6.1</td>
<td>10.48</td>
<td>37</td>
<td>30</td>
<td>26</td>
</tr>
<tr>
<td>Floor: biotite diopside granulitite</td>
<td>2.75</td>
<td>64.18</td>
<td>4.5</td>
<td>10.0</td>
<td>75</td>
<td>79</td>
<td>27.5</td>
</tr>
<tr>
<td>Floor: granite porphyry</td>
<td>2.7</td>
<td>64.18</td>
<td>20.0</td>
<td>15.5</td>
<td>60</td>
<td>100</td>
<td>27</td>
</tr>
<tr>
<td>Ore body</td>
<td>2.7</td>
<td>35</td>
<td>16.5</td>
<td>20.0</td>
<td>50</td>
<td>90</td>
<td>27</td>
</tr>
</tbody>
</table>

Table 2: Physical and mechanical parameters of tailing backfill body

<table>
<thead>
<tr>
<th>Title</th>
<th>Density /g.cm$^{-3}$</th>
<th>Compressive strength /MPa</th>
<th>Tensile strength /MPa</th>
<th>Cohesion/MPa</th>
<th>Internal friction angle/°</th>
<th>Modulus of elasticity/GPa</th>
<th>Poisson ratio</th>
</tr>
</thead>
<tbody>
<tr>
<td>Tailings</td>
<td>1.85</td>
<td>5</td>
<td>0.22</td>
<td>0.28</td>
<td>48</td>
<td>0.25</td>
<td>0.25</td>
</tr>
</tbody>
</table>

Table 3: Permeability calculation parameters

<table>
<thead>
<tr>
<th>Types of filling body</th>
<th>Actual coefficient of permeability (m/d)</th>
<th>Permeability coefficient of calculation $(10^{-9} \text{m}^2/\text{pa.s})$</th>
<th>Void ratio</th>
</tr>
</thead>
<tbody>
<tr>
<td>Surrounding rock</td>
<td>203.47</td>
<td>235.498</td>
<td>0.25</td>
</tr>
<tr>
<td>Tailing backfill body</td>
<td>1600</td>
<td>1851.851</td>
<td>0.40</td>
</tr>
</tbody>
</table>

Figure 2 (a) continues
Here in order to optimize the sequence of stopping, the design mainly considered different middle stopping sequence and the same middle mining sequence of these two aspects to carry on the analysis. The specific calculation schemes were shown in the following:

1. For the range of -80m ~ -160m, in order to ensure the safety of mining, the design of -80m middle ore bodies for safety pillar (Design), therefore, the ore body mainly considered the same middle (horizontal) mining sequence, that was central to both ends; both ends to the center; from one end to the other end.

2. For the range of -160m ~ -280m, two kinds of mining sequence were considered. One was the different middle (vertical) stopping sequence, namely mining down from the bottom-up mining. The second was the same (horizontal) in the middle of the stopping sequence, which was central to both ends; two ends to the center; from one end to the other end.

The calculation results of ore body mining sequence for the range of -80m ~ -160m

Numerical analysis results of different stopping sequence of -160m were shown in Figure 3. If the stopping sequence of the central to the ends was adopted, the majority of the stope stress at relatively low status, vertical displacement of stope roof was smaller, especially after the end of mining, the stress values were smaller than other schemes, relatively favorable for ore mining; on the contrary, if taking from the two ends to the center of the mining and end toward the other end, then in mining of the central mining stope, the stope pillar and hanging rock in high stress state, more easily lead to falling of stope, security of roof and floor was affected, and when stop all mined out, the safety factor changed smaller. The mining sequence of central to the ends of the stopping was adopted, the final safety stability of whole stop was better.
The numerical analysis of orebody mining sequence for the range of -160m ~ -280m

The specific calculation results were shown in Figure 4. It can be known, mining from the bottom to the top, the maximum principal stress and maximum vertical displacement of stope roof were small, which was favorable for safe and efficient mining; and from up to down in the middle of the mining stope, roof maximum tensile was 13.12MPa which exceeded the maximum tensile strength of filling body, and the vertical displacement was larger, prone to fall. From the view of safety factor in the two schemes, from the bottom to the top in the early mining was smaller than mining from the top to the bottom, but with the whole ore body mining finished, safety factor was higher of mining from the bottom to the top. Therefore,
comprehensive consideration of these factors, stopping sequence of range of -160m ~ -280m ore body should be taken from the top down method, namely: the first step: mining -280m middle segment; the second step: filling -280m middle segment, at the same time, mining -240m middle segment; the third step: filling -240m middle segment, at the same time, mining -200m middle segment; the fourth step: filling -200m middle segment, at the same time, mining -160m middle segment; the fifth step: filling -160m middle segment.

(a) The relationship between maximum principal stress of roof and stopping middle segment

(b) The relationship between maximum vertical displacement of roof and stopping middle segment

(c) The relationship between subsidence of ground and stopping middle segment
CONCLUSIONS

According to the damage constitutive model, the numerical calculation method and process of large deformation were analyzed, and to carry on specific simulation in allusion to Zheng Jia Po iron, qualitative analysis on the effect of different mining sequence and extraction process coped with stability of surrounding rock in stope, revealing difference in field stabilization extent of different mining scheme, thus the optimal exploitation scheme was definite, at the same time providing important information on decision of exploitation process in the deep deposit, and laid the foundation for the optimization of stope system. Therefore, numerical calculation method based on large deformation damage constitutive model had important value to guide the mining production.

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REFERENCE


