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The numerical analysis of borehole blasting and application in coal mine roof-weaken

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Abstract

Systematically analyzed the rock breakage process in borehole blasting, put forward the “dynamic and static press” breakage principle of rock borehole blasting. Use AUTODYN software to make borehole blasting numerical simulation, educed the single and double borehole blasting of rock breakage process and analyzed the borehole non-coupling charge blasting effect. At last, comparing theoretical analysis with numerical simulation methods in roof-weaken of coal mine by blasting, we got that the scene blasting fracture and monitoring coal mine press regulation verified the conclusion of study. The method not only plays a positive role in perfecting borehole blasting theory, but also verifies that the deep hole blasting in coal mine roof-weaken method is effective and feasible.

Keywords: borehole blasting; AUTODYN; coal mine; roof-weaken; numerical analyse

In coal production process, the work face roof character and type directly impact mine pressure behave. The roof hard and difficult to collapse has characters such like strength, high modulus of elasticity, crack joint not developed, thick and integrity. It has strong bearing capacity to load, after coal mining it can hanging in mined-out area for large-scale. There was large area, strong pressure when it initial collapse. There was an obvious dynamic pressure impact during periodic weighting period, often caused support equipments damage and personal injury accidents. It poses a serious threat to production equipments and personal safety. Thus, how to control hard and difficult to collapse roof is an important part of theoretical and practical research in rock pressure in mine\textsuperscript{[1]}.

In order to make the hard and difficult to collapse roof collapse easily in mining process, we must change its effect factors for such a cause. Blasting can weaken rock as a whole, increase in bedding and fissures, so as to achieve a easily collapse of roof. The deep-hole blasting method can cut off the roof, reduced size of the roof caving, weaken impact force to supporting when roof falling\textsuperscript{[2]}.

1. The basic principle of borehole blasting

1.1. Analysis of rock blasting process

Blasting is a high-temperature, high pressure and high-speed transient process. Rock crushing process is a very fast process under blasting, so it’s extremely difficult to study rock blasting failure mechanism. So far, there have
not a complete and accurate rock blasting damage theory [3]. Blasting of rock, there are two effects: dynamic pressure (shock, stress wave), just like “shake shattered” effect; static pressure (explosive bulge gas stress), just like “gas wedge” effect.

1) Dynamic pressure: after blasting in coal-rock borehole, the explosive sources neighbouring coal-rock undergo blasting impact loading. Around blasting holes form a compressive stress field, the coal-rock brings radial displacement by compressive stress. Radial fracture mechanism is put forward by scholars D.F. Coates from the United States in 1970 [4]. He deem that around the hole because of radial displacement and impact force by blast, make the borehole tangential rock undergo tensile force, give rise to tensile deformation, there will be radial fracture when the tensile strength greater than its intrinsic strength. People initially thought that the radial fracture directly start at the hole wall, until the use of high-speed photography technology, observed that the starting point of fracture often at rock defects or particles interface, and then gradually run-through with the hole wall.

2) Static pressure: after the high pressure gas shocks hole wall by blasting, the explosion energy has not completely disappeared, some still remain in the gas, and it is to a certain degree of pressure (about hundreds of atmospheric pressure) continue to make pressure on broken rock and play main role in rock crush and throw [5]. The rock shattered by dynamic pressure, in which the fracture has not fully inter-connecting. The high pressure gas can seep into the radial and all kinds of cracks, make their apex stress intensity increased and extended forward, and ultimately enable fissures run-through by each other. British scholar H.K. Kutter earlier discussed the static pressure role of explosive bulge gas in addition to dynamic pressure [6]. He also pointed out that the bulge gas can also form “quasi-hydrostatic stress field”, T.N. Hagan called this phenomenon as “gas wedge effect” [7], just because of a gas continuous wedging make cracks extended and mutually run-through and can able to make rock crush, and finally by the excess pressure of the gas throws out the crash blocks, so to complete the whole process of blasting rock.

Some scholars have proposed the “three-stage theory” during blasting rock [8], that the first stage is caused by stress wave put rise to radial fracture, followed by is the reflected wave generated scale off phase, and finally is the explosive bulge gas bulge crack and throw out phase. In fact, as an artificial division of law is right, but to rock breaking real process, the “dynamic and static” two-stage theory seems more systematic. Because the reflected wave generated scale off cracks along with the extension of radial fractures, chip fracture and radial cracks is too close to differentiate which is first, they are interactive acting.

From the above we can see that the blasting dynamic pressure make rock shake shattered, reduced the resisting force of rock, while the follow-up static pressure of explosive bulge gas make fissures run-through, rock fragment thrown out. Dynamic pressure and static pressure are complementary to each other, conceive in hard rock blasting, if the role of dynamic shake shatter pressure insufficient, barely rely on static pressure of bulge gas to make rock damage is extremely difficult. The “pipeline effect” of non-coupling charge blasting proves the role of explosive bulge gas. Conversely, if there is only the dynamic pressure impact load, while the energy of gas is insufficient, then the fracture can not be run-through to large-scale, such as the use of high detonation velocity coupling charge, can only make crushed zone neighbouring holes wall [9].

1.2. Unlimited medium blasting mechanism

In order to analyze the broken mechanism of rock blasting, we often simplified charge as spherical in a free surface condition; spherical charge blasting is the basis principle of blasting. Rock destruct characteristics have significant change along with the distance to blasting source, according to the rock destruct characteristics, put the blasting rock into crush zone, crack zone and elastic vibration zone the three regions [10], as shown in Fig. 1.
(1) Crush zone (compressed zone)

When blasting, the explosion pressure quickly rose to several thousands MPa in a few microseconds, and impact on the surrounding rocks moment and sharply, its shock wave intensity far exceeds the dynamic compressive strength of rock, this time the majority of rock present brittle under impact load has been crushed, and then most of the shock wave energy consume into rock plastic deformation, crushing, heating and so on, resulting in a drastic deterioration of shock wave energy, and quickly dropped to insufficient to crush rock. So crush zone radius present smaller, usually is several times to explosives radius. Many scholars raise the crush zone radius calculate methods, due to their different starting point, and put forward different calculation methods [11], an estimation method described as below.

\[
R_c = (0.2 \rho_s \frac{c_p^2}{\sigma_c}) R_b
\]

\[
R_b = 4\sqrt{\frac{p_m}{\sigma_0 r_b}}
\]

\[
p_m = \rho_s D^2 / 8
\]

\[
\sigma_0 = \sigma_c 4^\frac{1}{2} \rho_s \frac{c_p}{\sigma_c}
\]

In formula: \( R_c \) is crush zone radius (m), \( \rho_s \) is the density of rock (kg/m\(^3\)), \( c_p \) is the rock wave velocity(m/s), \( R_b \) is the cavity radius formed after blast (m), \( \sigma_c \) is uniaxial compressive strength of rock (Pa), \( r_b \) is the borehole radius (m), \( p_m \) is the average explosive pressure (Pa), \( D \) is detonation velocity (m/s), \( \sigma_0 \) is rock strength in multi-direction stress condition (Pa).

(2) Crack zone (rupture zone)

When the shock wave spreads through the area of crush zone, with the spread of expanse scope, the rock units energy flux density will reduce, its impact load strength has been lower than the rock dynamic strength, can not be crushed rock, but it is still so strong to make outer layer of rock radial compression, resulting in the outer layer of rock have tangential tensile strain and radial expansion, when such tangential strain more than the rock beard, then the outer layer of rock will have radial fracture. When the tensile stress reduced to less than the tensile strength of rock, the crack would stop expansion.

After stress wave make crush zone and bring radial cracks, the energy of rock previously compressed release rapidly, rock deformation rebound, and then form into unload wave, unload wave generated centripetal tensile stress, when its beyond the rock dynamic tensile strength, there will be annulus (tangential) fracture. Radial and annulus fracture inter crossing each other, will split rock into blocks, this region is also called rupture zone.

The crack zone radius can calculate by thick-walled cylinder of elasticity theory [12] and the tensile strength of rock estimate criteria, of which explosive bulge gas calculated by quasi-static pressure, we have:

\[
R_p = \left( \frac{p_j \sigma_t}{\sigma_t} \right)^\frac{1}{2} r_b
\]

\[
p_j = \frac{1}{8} \rho_s D^2 \left( \frac{r_c}{r_b} \right)^6
\]

In formula: \( R_p \) is crack zone radius (m), \( p_j \) quasi-static pressure on hole wall (Pa), \( \sigma_t \) is the tensile strength of rock (Pa), \( r_b \) is hole radius (m), \( r_c \) is detonator charge radius (m), \( D \) is detonation velocity (m/s), \( \rho_s \) is rock density (kg/m\(^3\)).

(3) Elastic vibration zone
For areas outside of the crack zone, because of the stress wave and bulge gas can’t destroy it, can only make particle vibration by flexibility, the region is called elastic vibration zone. Its scope can be estimate by formula as follow [13]:

\[ R_s = (1.5 \sim 2.0)s \sqrt{q} \]  

In formula: \( R_s \) is elastic vibration zone radius (m), \( q \) is explosive charge per unit volume of rock (kg/m³).

2. Numerical simulation analyses

ANSYS-AUTODYN using coupled finite difference and finite element method to solve solid, fluid, gas dynamics such variety of highly nonlinear dynamics. It can simulate and analyse explosive effects and high-speed dynamic load characteristics of material. It can be used in deformation and fragmentation, impact, and response analysis of explosion and shock wave.

2.1. Material models and equation of state

Considering inhomogeneity and porosity presence in rock, use random distribution of P-alpha state equation as below to describe the characteristics [14].

\[ P = f(\rho, e) \xrightarrow{\text{porous}} P = f(\rho \alpha, e) \]  

In formula:

\[ P = A_1 u + A_2 u^2 + A_3 u^3 + (B_0 + B_1 u) \rho e \]

\[ \alpha = 1 + (\alpha_{\text{init}} - 1) \left[ \frac{P_{\text{lock}} - P}{P_{\text{lock}} - P_{\text{crush}}} \right]^n \]

\( P \) is pressure on material, \( A_1, A_2, A_3, B_0, B_1 \) is the material constants, \( u = \frac{\rho}{\rho_0} - 1 \), \( \alpha \in (0 \sim 1) \) is uneven density, when the material completely compaction, \( P = P_{\text{lock}} \), and \( \alpha = 1 \).

In order to simulate the entire process of blasting, at any time the pressure of blasting source can be expressed as [15]:

\[ p = F p_e (V, E) \]

\[ F = \begin{cases} 
\frac{2(t - t_1)DA_{\text{max}}}{3v_e} & (t > t_1) \\
0 & (t \leq t_1) 
\end{cases} \]

\[ p_e = A \left( 1 - \frac{\omega}{R_1 V} \right) e^{-R_1 V} + B \left( 1 - \frac{\omega}{R_2 V} \right) e^{-R_2 V} + \frac{\omega E}{V} \]

In formula: \( p \) is blasting pressure (Pa), \( F \) is energy release rate of blasting, \( p_e \) is blasting sources pressure at any time, it meets the JWL equation of state. \( D \) is detonation velocity (m/s), \( t, t_1 \) respectively for the current time and point initial blasting time(s). \( A_{\text{max}} \) is the maximum cross-sectional area for explosives unit, \( V_e \) for the explosives unit volume, \( E \) is internal energy per unit quality, \( V \) for the specific volume, \( A, B, R_1, R_2, \omega \) for the constants of explosives-related material.
JWL equation of state the right first item play a major role in the high-pressure stage, the second item play a major role in the middle pressure, the third item calculate to low pressure, in this article JWL equation of state parameters selected in Table 1 as below.

Table 1. JWL equation of state parameters

<table>
<thead>
<tr>
<th>A (GPa)</th>
<th>B(GPa)</th>
<th>R1</th>
<th>R2</th>
<th>Ω</th>
<th>E(GPa)</th>
<th>ρ(kg/m³)</th>
<th>D(m/s)</th>
</tr>
</thead>
<tbody>
<tr>
<td>54.09</td>
<td>0.937</td>
<td>4.5</td>
<td>1.10</td>
<td>0.35</td>
<td>2.1E9</td>
<td>1000</td>
<td>6000</td>
</tr>
</tbody>
</table>

At anaphase of detonation products, in formula (3) the front two items can be ignored, so the JWL equation of state can convert to an ideal gas equation:

$$p = (\gamma - 1)\rho E$$

In formula, $\rho$ is air density (kg/m³), $\gamma$ for the adiabatic index of air, $E$ for the air ratio internal energy (J/kg).

2.2. Mesh grid plot

In order to obtain more accurate detonation wave and shock wave peak pressure, the initial borehole pressure calculation model requires extremely fine grid. In this paper, the single borehole blasting simulation grid total of 4900, blast hole ring wall to mesh total number of 120. Taking into account if the ring rock to mesh remain unchanged, would lead to charge centre distance the more farther, rock ring mesh size the more larger, and the calculated fracture would gradually wider. This does not match the actual. The solution is: 100 mm away from the charge centre, the ring rock grid increases to total number 240 that is ring mesh refinement doubled. 500 mm from charge centres, the ring mesh refinement doubled again, encrypted to 480. Near the mesh refinement do appropriate transition in order to avoid isolated nodes, mesh as shown in Fig. 2.

Fig. 2. Numerical simulation grid

2.3. Boundary conditions

The boundary conditions for rock: for a single hole, set the radius of 10 m; for the two holes, the radius of 15 m. Rock external boundary impose transmission condition, so that the stress wave could transmit away from the border and would not reflect to interfere with results, in the external also imposed on the gas transmission (flow-out) boundary condition, blasting gas could transmit out from the border.

2.4. The calculate results analysis

In this paper, numerical simulation choiced three main options: ① Analysis of single-hole blasting; ② double hole blasting analysis; ③ non-coupling blasting analysis.

(1) Analysis of single-hole blasting

Single bore hole blasting fragment process analysis as shown in Fig. 3. Fig. 3 (a) is surrounding rock destruction at 0.1 milliseconds, and this time around the hole form initial crush zone, which is mainly caused by explosion shock wave; Fig. 3(b) is surrounding rock start be pulled off at 0.2 milliseconds, this stage is formation.
and expansion of radial cracks, from the Fig. can also see the ring cracks have begun to take shape; Fig. 3(c) is the state around hole at 0.4 milliseconds, can be seen from the Fig. 3, a large number of radial cracks continue to expand, while circumferential cracks also gradually expand, resulting in cracks mutual run-through and form a fracture network; Fig. 3(d) is the state of ultimate destruction, blasting cracks no longer continue to expand.

![Fig. 3. Single borehole rock blasting process](image1)

(2) Analysis of double borehole blasting

In order to study the two adjacent bore hole blasting effect and mutual influence, we simulated and compared blasting at the same time and delayed detonation process respectively. Fig. 4 for detonation at the same time, Fig. 4(a) is at 0.1 millisecond around rock damage state, this time around the hole are formed initial crush zone, which is mainly caused by the explosion shock wave; Fig. 4(b) for 0.3 millisecond the surrounding rock begun to be pulled off, this stage for the formation and expansion of radial cracks, circumferential cracks have begun to take shape and gradually expand; Fig. 4(c) for 1.1 millisecond the rock around fragmentation state. It can be seen from the Fig. 4(c), the cracks caused by blasting begun to run-through each other.

Fig. 5 is for the delayed detonation process. After the top bore hole firing 0.3 milliseconds, fire the below hole. Fig. 5(a) for 0.1 millisecond the top hole surrounding destruction state, this time around the top hole formed initial crush zone, and the below hole does not fired; Fig. 5(b) for 3 millisecond the top hole surrounding fracture state, at this time fire the below hole; Fig. 5(c) is 3.4 millisecond the two holes around rock crack state. From the Figure can be seen cracks begun to run-through each other. Compared with Fig. 4 can be seen, in delay blasting process the cracks run-through time later than fire at the same time.

![Fig. 4. The two holes fired at the same time](image2)
Fig. 5. The two holes one fired delayed the other

(3) Analysis of non-coupling charge

According to coal mine bit diameter, the diameter of standard explosives, deep-hole blasting operability, as well as blasting parameters on the effect, borehole and explosives diameters were chosen as shown in Fig. 6 of 10 kinds.

Fig. 6. Non-coupling charge conceptual drawing

After numerical simulation, analyze various precepts of non-coupling blasting stress and displacement near blasting source. Monitoring the distance from blasting sources 2 m, 4 m, 6 m, 8 m of rock particles Von Misses stress and radial velocity curve by time, see Table 2 as below. The table only lists the results of previous four precepts for length of article.

Table 2. Von Mises stress and radial velocity curve by time

<table>
<thead>
<tr>
<th>θ</th>
<th>φ⁰₅₀</th>
<th>φ⁰₅₀</th>
<th>φ⁰₅₀</th>
<th>φ⁰₅₀</th>
<th>φ⁰₅₀</th>
<th>φ⁰₅₀</th>
<th>φ⁰₅₀</th>
</tr>
</thead>
<tbody>
<tr>
<td>t</td>
<td>1.0</td>
<td>2.0</td>
<td>3.0</td>
<td>4.0</td>
<td>5.0</td>
<td>6.0</td>
<td>7.0</td>
</tr>
<tr>
<td>Stress</td>
<td>1.0</td>
<td>2.0</td>
<td>3.0</td>
<td>4.0</td>
<td>5.0</td>
<td>6.0</td>
<td>7.0</td>
</tr>
<tr>
<td>Velocity</td>
<td>1.0</td>
<td>2.0</td>
<td>3.0</td>
<td>4.0</td>
<td>5.0</td>
<td>6.0</td>
<td>7.0</td>
</tr>
</tbody>
</table>

From Table 1, from around borehole 2 m, 4 m, 6 m, 8 m rock Von Mises stress and particle radial velocity curve can be seen. The farther away from blasting sources, rock particles Von Mises stress and radial velocity decay
gradually. In the same amount of charge, with the charge non-coupling coefficient increases, the pressure peak has reduced. Which is due to the borehole diameter increase gradually, air consumed some energy of detonation products. From the Figure can be seen the coupling coefficient the greater, the increase in frequency of vibration peak more obvious. This also explains the process of explosive gas wedge effect repeatedly shake caused cracks.

3. Blasting test at scene

The thickness of coal seam in Lu Ning coal mine average is 4.43 m, the coal seam direct top lithology is sandstone, the compressive strength for 79.6 MPa, according to the roof and floor classification belonging to Category 2 (dense mudstone, silt shale, jointed aplasia). Basic top roof for fine grained-coarse grained-long grained quartz sandstone, feldspar sandstone composed, quartz content up to 60~70%, hardness greater, compressive strength greater, classification belonging to IVb roof, which would appeared strong pressure. During exploitation the basic top roof would not easy falling, its thick and some of its thickness can reached 16.75 m. Therefore in order to ensure safe and efficient production, must to be carried out weakening measure.

Use deep hole blasting to weaken the roof of that coal mine, using ZL-100 drilling rig hit diameter Φ60 mm, use coal mine 2° emulsion explosives, specification is Φ50 × 250 mm. First use the formula in subsection 1.2, estimated crush zone diameter about 430 mm, crack zone diameter is about 4221 mm. Yet again use AUTODYN software for numerical simulation analysis, take physical and mechanical parameters of the roof and 2° emulsion explosive parameters into the software parameters, calculate the borehole surrounded by 8 large cracks, length close to 4000 mm, one of the largest length of 4922 mm., and dozens of small cracks. Crush zone more or less in diameter to 450 mm.

According to the estimation method and the conclusions of the numerical simulation, the parameters of the hole layout was designed. In the work face transport and air return tunnel, lay a set of borehole every 25 m, each set include 8 holes, 8 holes put upper and lower two floors, each floor for 4 holes, the upper holes and coal seam vertical distance of 15 m, the lower holes and seam vertical distance of 10 m. The borehole specific parameters show in Table 3 and layout shown in Fig. 7.

<table>
<thead>
<tr>
<th>Serial No.</th>
<th>Hole length (m)</th>
<th>Elevation (°)</th>
<th>Horizontal angle (°)</th>
<th>Charge depth (m)</th>
<th>Charge quantity (kg)</th>
<th>Seal lute (m)</th>
</tr>
</thead>
<tbody>
<tr>
<td>A1</td>
<td>30.2</td>
<td>38.9</td>
<td>22</td>
<td>20</td>
<td>40</td>
<td>10.2</td>
</tr>
<tr>
<td>A2</td>
<td>28.1</td>
<td>29.6</td>
<td>22</td>
<td>20</td>
<td>40</td>
<td>8.1</td>
</tr>
<tr>
<td>B1</td>
<td>32.3</td>
<td>36.6</td>
<td>34</td>
<td>22</td>
<td>44</td>
<td>10.3</td>
</tr>
<tr>
<td>B2</td>
<td>30.3</td>
<td>28.3</td>
<td>34</td>
<td>20</td>
<td>40</td>
<td>10.3</td>
</tr>
<tr>
<td>C1</td>
<td>49.5</td>
<td>30.2</td>
<td>71</td>
<td>36</td>
<td>72</td>
<td>13.5</td>
</tr>
<tr>
<td>C2</td>
<td>48.2</td>
<td>24.7</td>
<td>71</td>
<td>35</td>
<td>70</td>
<td>13.2</td>
</tr>
<tr>
<td>D1</td>
<td>49.5</td>
<td>31.1</td>
<td>78</td>
<td>36</td>
<td>72</td>
<td>13.5</td>
</tr>
<tr>
<td>D2</td>
<td>48.2</td>
<td>25.5</td>
<td>78</td>
<td>35</td>
<td>70</td>
<td>13.2</td>
</tr>
</tbody>
</table>

After blasting we used rock drill hole observation instrument to verify the effect, see Fig. 8. We have detected that significant large cracks occurred around blasting source 4 m position, as shown in Fig. 9. According to the
CCRI exploit institute take pressure monitoring and analysis to the roof [16]. The basic top roof first weighting stage is for 64 m before it weakens by blasting, for the roof classification is IVb, rock pressure showed strong; After work face and roof weaken treatment, the basic top roof first weighting stage falls down to 34 m, the basic top roof classification turn to I, the pressure is not strong, in the process of exploiting the mining equipments have not been destroyed or big impulse.

4. Conclusion and prospect

(1) Explained the rock borehole blasting process system both by theoretically and numerically, and put forward “static and dynamic pressure hybrid” breakage principle.

(2) Through numerical simulation by AUTODYN software, we got single hole and double borehole conditions rock breakage process by blasting, and analyzed non-coupling charge blasting laws.

(3) From the scene practice, it’s confirmed that the advanced deep-hole blasting to improve the hard top roof is effective measures. After blasting weaken, the work face direct roof fall with mining, the basic top roof first weighting and periodic weighting stage did not impulse on the supporting.

(4) To coal mine roof weaken, we take formula to estimate crush zone and crack zone scope firstly. And then use AUTODYN software do numerical simulation to analyze and adjust the blasting parameters. Finally use detects instrument results were used to check the blasting effect. It’s formed a complete set of technical means to weaken roof.

Prospect

In this paper, the numerical model does not take into account of larger primary cracks; the next step will study the borehole blasting in rock with fracture.

References