Research Article

Study of Two-Step Parallel Cutting Technology for Deep-Hole Blasting in Shaft Excavation

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1. Introduction

Shallow mine resources around the world are dwindling, but the demand for minerals continues to increase. Driven by mineral resources demand, many of China’s mines are close to or exceed 1000 meters in depth. Moreover, the vertical shaft of some metal mines is greater than 1500 m. As shaft depth increases, the increase of in situ stress affects the stability of underground engineering and auxiliary operation time (e.g., shift time for construction organization, gangue discharging, equipment lifting, explosives, and personnel) increases continuously. Blasting is currently the primary mining method [1–4] and tunneling [5, 6]; therefore, mid-deep-hole blasting technology is an essential measure to improve excavation efficiency [7–9]. Some mines have adopted deep blasting technology using blasthole depths greater than 5 m.

One key problem with deep-hole blasting technology is low blasthole utilization, which cannot meet production demands [10]. The situation is especially worsened when encountering hard rock with a stability coefficient greater than 12 [11]. During deep-hole blasting of hard rock, the strong clamping effect of the bottom rock mass leads to a large number of residual holes [12]. This affects shaft excavation progress and restricts the development of deep-hole blasting technology. Improving blasting excavation efficiency depends on cutting; therefore, the improvement of cutting technology is significant for achieving rapid excavation.

There are two main categories of cutting technology for hard rock in shafts: parallel cutting and inclined-hole cutting. Each category has its advantages and disadvantages. Based on the characteristics of the two cutting methods, scholars have improved both in space and time. Taking parallel cutting as an example, some scholars created a free surface using the “inclined hole” or “empty borehole with a large diameter” methods in space. Reference [13] developed...
quasi-parallel cut blasting technology, and [14] used large diameter empty hole cutting technology. Inclined-hole cutting includes duplex wedge cutting and multistage cutting methods. Reference [15] developed a multistage cutting method, which analyzed cavitation theory in detail and was successfully applied in field experiments in the rock roadway of a coal mine. These cutting methods achieved a good blasting effect in different rock types, but blast efficiency was low in vertical shaft, hard rock deep-hole situations with a high rock coefficient. Moreover, the above cutting methods were affected by workers' operation habits and drilling technology and were not always applied correctly. However, the two-step parallel cutting technology provides a new method for improving the cutting efficiency of deep-hole blasting in vertical shafts with hard rock.

Based on previous cutting methods, a two-step parallel cutting technology is proposed in this study that alters the order of cut blasting inside the hole, creating more free surface. Using theoretical analysis and calculations, two-step parallel cutting technology theory is described, and the concept of the optimum ratio value per unit explosive consumption between the upper section and lower section is introduced for the first time. Model and field experiments are used to validate the theory and practice of two-step parallel cutting technology. The key problem in cut blasting, that is, creating free surface and space for rock fragmentation, is resolved. By dividing the rock mass into an upper section and lower section and detonating them in turn, the clamping effect of the rock at the hole bottom is effectively reduced. The lower section makes full use of the free surface created by the upper section. Moreover, the two-step cutting blasting technology reduces equipment and shaft wall damage caused by blast vibrations in one-step blasting [16–18]. In short, this new method results in more safe and efficient cutting.

2. Mechanism of Analysis

Unlike one-step blasting [19, 20], in two-step blasting, the cutting blasthole is divided into two sections (Figure 1). These are charged with explosives and separated using stemming. After successive detonations (Figure 2), the upper section is forced upward under the combined action of explosive gas and stress waves, forming a small groove cavity. This groove cavity creates a sufficient free surface for the lower section. Meanwhile, the upper preloaded blast is applied to the rock in the lower groove cavity to cause preloading damage or even fragmentation. This results in a large number of explosion-induced cracks or even destruction of the lower chamber, enhancing the wedge effect of the detonation gas and rock fracturing by explosion stress waves during the lower section detonation, creating a larger and deeper chamber.

The energy distribution of two-step blasting is altered. The clamping effect of the upper rock mass is weak, the free surface of the upper rock mass is sufficient, and the energy demand for rock fragmentation is low. The lower rock mass has no free surface, the clamping effect is strong, and the energy demand for rock fragmentation is high. Two-step cut blasting technology enables the energy generated from the explosion to be used primarily for lower rock fragmentation and secondarily used for upper rock removal, which improves the utilization rate of cutting holes and reduces blast vibrations. Without changing the total explosive energy, the blasting effect can be improved using the two-step cut blasting technology due to better energy distribution.

In deep-hole blasting, the rock mass clamping effect increases nonlinearly with hole depth, which is abstracted into a curve OD. However, the explosive rock fragmentation effect does not change with depth. Therefore, this effect can be abstracted as the line EF (Figure 3). In one-step blasting, point P is the critical equilibrium point between the clamping effect and the rock fragmentation effect. B is the depth corresponding to point P. When depth is less than B, the rock fragmentation effect is greater than the clamping effect. When the depth is equal to B, the two effects are equal. We can assume that depth B is approximately the theoretical cutting depth under this charging condition. When depth is greater than B, the rock fragmentation effect is less than the clamping effect. The rock mass will be fragmented but cannot be ejected.

In the two-step blasting, cutting holes are divided into two sections without changing the total charge length. The upper section detonates first, creating a sufficient free surface for the lower section and reducing the clamping effect of the lower rock mass. Comparing Figures 3(a) and 3(b), the explosive AB is shifted upward to AB'. After the rock above B' is ejected by the explosion, a new free surface is created that reduces the clamping effect of the lower section from PD to ND', such that the rock below B' is also ejected to form a groove cavity. Without changing the total energy of the explosive, two-step blasting technology releases the free surface and improves the blasting effect.

3. Theoretical Formulas for Cutting Parameters

3.1. Cutting Hole Spacing. There is only one free surface for vertical shaft cut blasting, so each cutting hole should be in the blasting fissure area to ensure that rock in the cavity can be fully fragmented; that is,

\[ E \leq R_p, \]

\[ D_e \leq 2R_p, \]

where \( E \) is the cutting hole spacing, \( m; D_e \) is the cutting area diameter, \( m; \) and \( R_p \) is the blast-induced cracking zone radius, \( m. \)

For hard rock under high-stress conditions, the cracking zone is primarily formed by explosive stress waves:

\[ R_p = r_p \left( \frac{aP_t}{S_t} \right)^{1/t}, \]

where \( r_p \) is the blasthole radius, \( m; a \) is the lateral pressure coefficient; \( a = g/(g - 1); g \) is Poisson’s ratio; \( P_t \) is the initial pressure on the hole wall, MPa; \( N; S_t \) is the rock tensile strength, MPa; \( t \) is the stress wave attenuation coefficient, which is affected by the nature of the rock itself. The empirical value
of $t$ is $t = 2 - a$ or $t = 2.92-4.11 \times 10^{-2}\rho_m\sigma_p$, where $P$ is related to the charging form.

For decoupled charging [21],

$$P_r = \frac{n\rho_hv^2}{8} \left( \frac{r_c}{r_b} \right)^6,$$

Figure 1: Charge structure of one-step blasting and two-step blasting (1- stemming; 2- explosive detonated by 1#detonator; 3- explosive detonated by 2#detonator). (a) One-step blasting. (b) Two-step blasting.

Figure 2: Diagram of two-step cut blasting. (a) Profile diagram of two-step cut blasting in a vertical shaft. (b) Profile diagram after the upper section detonation. (c) Profile diagram after the lower section detonation.

Figure 3: Curves of the clamping effect and the rock fragmentation effect along with the depth. (a) One-step blasting. (b) Two-step blasting. ((a) and (b) show the difference between one-step cutting blast and two-step cutting blast).
where \( \rho_0 \) is the dynamite density, \( \text{kg/m}^3; v \) is the explosive velocity, \( \text{m/s; } c_\rho \) is the rock longitudinal wave velocity, \( \text{m/s}; r_c \) is the charging radius, \( \text{mm; and } n \) is the number of times pressure increases when explosive gas hits the blasthole wall, \( n = 8\sim10 \).

3.2. Interval Time. In two-step cut blasting, the cutting hole is divided into two sections. The detonation time interval between the two sections should be longer than the time it takes for the upper rock to completely separate from the rock mass [22]. The detonation interval between the two sections is \( t \):

\[
t \geq t_1 + t_2 + t_3,
\]

where \( t \) is the detonation interval between the two sections (suggested detonation time is greater than 50 ms); \( t_1 \) is the time that the initial stress field develops from the cutting hole to free surface; \( t_2 \) is the time from the beginning of crack production to crack expansion in the free surface; \( t_3 \) is the time from the beginning of rock movement to fissure formation.

\( t_1, t_2, \) and \( t_3 \) can be calculated according to the following formulas:

\[
t_1 = 0.05 r_0,
\]

\[
t_2 = \frac{K_c L_1}{Z_v v_s \cos (\theta/2)}
\]

\[
t_3 = \frac{K_q L_1}{v_s},
\]

where \( K_c \) and \( K_q \) are the correction coefficients for clamping effect, \( K_c > 1.0 \) and \( K_q > 1.0; L_1 \) is the upper section hole length; \( Z_v \) is the blasthole utilization ratio; \( v_s \) is the average expansion velocity of explosive cracks in the rock mass; \( \theta \) is the cut cavity angle; and \( v_s \) is the average rock movement velocity.

3.3. Blasting Model. The model shown in Figure 4 was established to analyze the effects of two-step cut blasting on unit explosive consumption. The cutting model is simplified to a cylinder with a base radius \( R \) and height \( H \). Cutting holes are evenly distributed on the side of the cylinder. The charging coefficient of the hole is \( a \). The hole depth is \( H \). The charging line density (mass of explosive per unit length) is \( a \). The upper charging length is \( L_u \). The lower charging length is \( L_d \). The total charging length is \( L \). The charging interval length between the upper and the lower sections is \( L_0 \). In conventional blasting theory, the amount of explosive is proportional to the rock crushing volume. The theoretical formulas below represent the unit explosive consumption of the full model and the unit explosive consumption of the upper and lower sections.

The unit explosive consumption of the whole model can be written as

\[
V = \pi R^2 H,
\]

\[
Q = Haab = abaH,
\]

\[
q = \frac{Q}{V} = \frac{Haab}{\pi R^2 H} = \frac{aab}{\pi R^2}.
\]

The unit explosive consumption of the upper section can be written as

\[
V_u = \pi R^2 (H - L_d - L_0),
\]

\[
Q_u = (L - L_d)ab = ab(L - L_d),
\]

\[
q_u = \frac{Q_u}{V_u} = \frac{ab(L - L_d)}{\pi R^2 (H - L_d - L_0)}
\]

The unit explosive consumption of the lower section can be written as

\[
V_d = \pi R^2 (L_d + L_0),
\]

\[
Q_d = abL_d,
\]

\[
q_d = \frac{Q_d}{V_d} = \frac{abL_d}{\pi R^2 (L_d + L_0)}.
\]

The unit consumption results obtained by equations (6)–(9) are compared as follows:

\[
q = \frac{Q}{V} = \frac{ab}{\pi R^2} \times a,
\]

\[
q_u = \frac{Q_u}{V_u} = \frac{ab}{\pi R^2} \times \frac{L_d}{(L_d + L_0)}
\]

\[
q_u = \frac{Q_u}{V_u} = \frac{ab}{\pi R^2} \times \frac{(L - L_u)}{(H - L_d - L_0)}.
\]

The ratio value per unit explosive consumption is defined as \( K_q^* \):

\[
K_q^* = \frac{q_u}{q_d}
\]

where \( q_u \) is the unit explosive consumption of the upper section, \( \text{kg/m}^3 \), and \( q_d \) is the unit explosive consumption of the lower section, \( \text{kg/m}^3 \).

In the specific field, \( R, H, b, \alpha, H, a, L, \) and \( L_0 \) are constants. By altering \( L_d \) and \( L_u \), the curves for \( q_u - L_d, q_d - L_d \) and \( q_u - L_d \) can be obtained. Finally, the curve of \( K_q^* \) can be obtained through \( q_u/q_d \).

To more intuitively analyze the influence of changing \( L_d \) and \( L_u \), on \( q, q_u, q_d, \) and \( K_q, \) this paper used blast parameters from a shaft in Shandong, China: \( a = 2 \text{kg/m}, b = 8, H = 5.2, R = 0.8, \alpha = 0.78, L_0 = 0.5 \text{m.} \) Curves for \( q_u - L_d, q_d - L_d, q_u - L_d \) and \( K_q^* - L_d \) are plotted in Figures 5 and 6.

According to Figure 5, (1) as \( L_d \) increased, \( q_d \) gradually increased, but the rate slowed. As \( L_d \) increased, \( q_u \) decreased. 

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gradually, but the rate increased. Point C is the "critical point." (2) At point C, $q_a = q_d = q$; to the left of C, $K_q > 1$, $q_a > q_d$; to the right of C, $K_q < 1$, $q_a < q_d$. Theoretically, there is a value of $K_q$ that can match the optimum blasting effect or the optimum ratio value per unit explosive consumption. This value varies due to the influence of various factors such as rock hardness, explosive type, and blasting parameters. Considering that the clamping effect of the lower rock mass is relatively large, the ideal optimum ratio value was between 0.5 and 0.8.

4. Model Experiments

4.1. Similarity Scale. Based on similitude theory [23] and some model experiments [24, 25], the similarity criterion was deduced using the dimensional analysis method. Similarity ratio selection takes certain laboratory and field conditions into account. According to the parameters of a mine shaft in Shandong, the geometric similarity ratio was determined to be 15:1 and the bulk density similarity ratio was 1:1. Similarity ratios for other parameters were then calculated according to the similarity criterion. The size of the specimen is shown in Figure 7. The height was 700 mm, the upper diameter was 1200 mm, and the lower diameter was 1000 mm. The specimen size was large enough to ignore the boundary effect. The cutting area was located in the center of the upper section. The design depth of the cutting area was 300 mm, and the design diameter was 120 mm (Figure 7).

4.2. Specimen Materials. The material used in the specimen was gypsum, which was mixed evenly according to the design ratio (gypsum: water: retarder = 1:1:5:0.005) and then poured in layers. It was vibrated and compacted before the bubbles were discharged. Specimens were cured for 21 d at the appropriate temperature and humidity. The basic physical and mechanical parameters of gypsum were measured (Table 1).

4.3. Explosives. According to the cutting hole depth and the charging coefficient, the charging length was 210 mm. Explosives of different lengths were made according to the charge design structure shown in Figure 8. The design length included four specifications: 210 mm, 105 mm, 63 mm, and 147 mm. The explosive diameter was 6 mm, and the explosives consisted of black powder. The initiating probe was first fixed in the plasticine. Then, the plasticine was fixed in the bottom of the plastic straw using 502 glue. Next, black powder was loaded and compacted using a stick. Both ends of black powder were filled with cotton to avoid leakage. Finally, the explosives were blocked with plasticine and 502 glue (Figure 9).
Table 1: Measured physical and mechanical properties of gypsum.

| Density/(g/cm³) | Bulk density (g/cm³) | Compressivestrength/(MPa) | Elastic modulus/(GPa) | Poisson’s ratio | Ultrasonic velocity (m/s) |
|----------------|----------------------|---------------------------|-----------------------|----------------|--------------------------|
| 1.46           | 1.4                  | 6.5                       | 0.445                 | 0.3            | 3010                     |

The experimental group
S-4
S-3
S-2
5:5
S-1
3:7 7:3
Unsegmented

The control group
1
1
1
2 2

Figure 7: Parameters of the specimens and cutting area.

Figure 8: Order of explosion and structure of charge (the experimental group and the control group).

Figure 9: Explosives.
### 4.4. Experimental Design

In the experimental model, four specimens were prepared: S-1, S-2, S-3, and S-4 (Table 2). In the cutting area, eight cutting holes with a 6 mm diameter and 300 mm depth were arranged. The charging coefficients were 0.7, and proportions of the upper and the lower charging sections were 5:5, 3:7, 7:3, and unsegregated. S-1, S-2, and S-3 were classified as the experimental group, and their detonation was carried out in reverse order according to the sequence of 1-explosive to 2-explosive. S-4 was classified as the control group and also followed reverse detonation. The layout of the cutting holes is shown in Figure 10. The detonation sequence and the charging structure of the four specimens are shown in Figure 8.

### 4.5. Experimental Results

#### 4.5.1. Cutting Hole Utilization Rate

After the blast model experiment, groove depths were measured and the cutting hole utilization rate was calculated. The procedure was as follows: first, broken gypsum in the groove cavity was removed to expose the bottom of the cutting hole. Then, the distances between the residual part of each cutting hole and the surface of the original model were measured, cutting depth was averaged, and the utilization rate was calculated (Figure 11).

#### 4.5.2. Cavity Volume

After the model experiment, cavity radius and volume were measured. The procedure for measuring the radius was as follows: broken gypsum was removed from the cut cavity, the boundary position of the groove cavity was defined, the maximum and minimum damage radii were measured, and their average represented the groove cavity radius (Figure 12). The procedure for measuring the cavity was as follows: broken gypsum was removed from the cut cavity, dried and screened sand was poured inside, and the volume of sand was measured using a cylinder. The sand volume represented the volume of the cut cavity (Figure 13).

#### 4.5.3. Blasting Fragmentation

After the model experiment, gypsum fragments were collected and screened. The national standard for grading stone screening was used to divide the gypsum fragments into eight grades. Gypsum fragments in each grade were weighed. Quality, cumulative quality, quality percentage, and cumulative quality percentage values are shown in Table 3. Figure 14 shows the distribution of gypsum fragmentations in the four specimens.

### 4.6. Experimental Discussion

Figure 11 shows that the cutting hole utilization rates were 3:7 > 5:5 > 7:3 = unsegmented. Because charging coefficients for all four specimens were 0.7, it was assumed that the total explosive energy was approximately the same, and the change in the upper and lower section ratio was actually the energy distribution change. Because the upper section had a free surface advantage, it could be matched with less energy.
Because the lower section had a strong clamping effect, it could be matched with more energy. The experimental results were in good agreement with the theoretical analysis. Figure 14 shows the distribution of gypsum fragmentation for four specimens. According to the similarity criterion, gypsum fragments larger than 37.5 mm in diameter were considered as large gangue and were not handled well in the vertical shaft. The boulder yields for $S_{-1}$, $S_{-2}$, $S_{-3}$, and $S_{-4}$ were 41.3%, 25.9%, 27.9%, and 56.3%, respectively. It can be seen that boulder yields for specimens with two-step cut blasting were lower than those without segmentation. This is because two-step cut blasting released the explosive energy more evenly into the gypsum model, allowing the rock to be more easily broken.

Figure 13 shows the average diameter and volume of the cavity. The average diameter and volume were the lowest at the 3:7 segment. At the same time, the energy produced by explosives was consumed more in gypsum fragmentation and less in kinetic energy, sound energy, and heat energy. Therefore, the cutting cavity volume of the 3:7 segment was the highest. Meanwhile, due to the concentrated energy in the lower section, it was...
fully broken, which reduced the damage to areas outside the cutting area and minimized the average groove cavity diameter.

5. Field Test

5.1. Field Background. The net diameter of the vertical shaft in Shandong was 6.5 m, and the excavated diameter was 7.3 m. The excavated area was 41.85 m², and the depth of the experimental layer ranged from −733 to −743 m. The lithology was dominated by gabbro. The rock stability coefficient was 12. The parameters for No. 2 rock emulsion explosives are shown in Table 4. Mfb-200 was selected as the initiator. A YSJZ4.8 umbrella drill frame was used to drill holes and 4 HYD200 independent rotary rock drills were utilized [26]. A B32 × 5500 mm hollow hexagonal drill pipe with Φ55 mm cross alloy bits were used for drilling. The cutting hole depth was 5200 mm, while auxiliary and peripheral hole depths were both 5000 mm. The diameters of all three hole types were 55 mm.

5.2. Blasting Scheme

5.2.1. Original Blasting Scheme. The cutting form of the original cut blasting scheme was a one-step parallel cut blasting with eight holes arranged in a circular pattern (Figure 15). The blasting parameters for a basic rock section are shown in Table 5. In the original blasting scheme, the single-cycle blasting footage was 4.5 m, the blasthole utilization rate was 86.5%, and the unit explosive consumption was 2.6 kg/m³. The blasting parameters of the cut hole resulted in a poor blasting effect. The cutting hole depth of...
5200 mm was too deep and the rock mass with stability of greater than 12 was too hard. These factors caused a large clamping effect, so the cutting hole utilization rate was low and poor blasting effects were inevitable.

5.2.2. Optimized Blasting Scheme. Two-step cut blasting technology was adopted to improve the blasting effect of the original scheme. Without changing the blasthole layout and amount of explosives, two-step cut blasting technology and short-delay blasting technique were used to detonate cutting holes (Figure 16). According to the previous theoretical analysis and model experiments, three volumes of explosives detonated using a #1 detonator were placed in the upper section, and six volumes of explosives detonated using a #2 detonator were placed in the lower section. $K_q$ was equal to 0.75 under this condition. In addition, the two sections were separated by 500 mm of gravel. The interval length was controlled by measuring the exposed length of the tamping stick. Figure 17 shows a schematic of the cut blasting charging structure. Table 6 shows the optimized blasting scheme.

5.3. Results. In the two-step (3:7) cut blasting scheme, the single-cycle blasting footage was 5.0 m on average, and the blasthole utilization rate was 96.2%. Compared with the original blasting scheme, the footage increased by 0.8 m, and the utilization rate was 9.7% higher. The two-step cut blasting was verified using field experiments to
improve low blasting efficiency in the hard rock of vertical shafts.

6. Conclusion

To resolve the low blasthole utilization problem in deep-hole blasting of vertical shafts with hard rock, this paper proposed two-step cut blasting technology and verified the hypotheses using theoretical analysis, model experiments, and field experiments.

(1) Without changing the total explosive energy, two-step cut blasting effectively improved rock-fragmentation, providing an effective method for deep-hole blasting. Improved energy distribution was the fundamental reason behind these improvements.

(2) Based on the classical explosives and rock fragmentation theory, formulas for the unit explosive consumption of the whole model, the upper section, and the lower section were derived. The optimum ratio per unit explosive consumption concept was also put forward.

(3) An experimental cutting model with four types of segments was designed and completed based on two-step cut blasting theoretical analysis and formula development. It was found that the 3:7 segment produced the optimum blasting effect. Finally, two-step cut blasting technology was utilized in a vertical shaft in Shandong to resolve the low-efficiency problem in deep-hole blasting within the hard rock. Using 3:7 segment and a reasonable delay-time, it was verified that this technology improved cutting efficiency without changing the original blasthole layout.

Data Availability

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

Conflicts of Interest

The authors declare that they have no conflicts of interest.

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