Study on the Effect of Rock Blasting

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1. INTRODUCTION

The charge calculating methods to design rock blasting have been published earlier, and yet it has not been elucidated satisfactorily. It is found that the available theoretical basis for charge calculations is empirical values have been arrived at by test blasting and the accumulation of practical experience. In the case of breakage objective of rock concerned where there are generally soft rocks like coal mines in Japan and Korea where people use low strength explosives in blasting, it is regarded that it is very apt to acquire a dangerous over-charge if they use the excessive charge of explosives.

In coal mining stopes various mine accidents caused by blasting occurred in some cases not only by the direct cause of blasting, but also by both direct and indirect casues of such as gas and coal-dust explosion, gas burst, fallen rock after drilling, and unusual earth pressure or soft wall-rock. This is because the correct charge calculations have not been carried out theoretically and practically. Calculation of the charge primarily covers calculation of the minimum charge required for loosening the rock.

In the case of breakage objective of rock concerned where there is an abundance of geological variations in the blasting characteristics of the rock like Japan and Korea, it is regarded to design a certain blasting standards. But it is regarded to design it more logically if we take into consideration about the relationship of the hardness of rock, kinds of explosives,

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tamping methods and scale of blast, etc.

In this paper the Hauser's formula and its adjustment function have been reviewed, and it is indicated that the blasting theory of concentrated charges can satisfactorily be applicable to mines and quarries by drawing clear line between crater adjustment factor f(n) and blast scale factor f(W), and by promting to understand the rock factor(g), and by distinguishing the blast factor(C) and quantity of the charge per unit volume (kg/m³).

It is indicated in this paper that the new blast equations for column charges, bench blasting, boulder blasting are lead by the Hauser's equation by applying simultaneous blasting theory broadly. And it is also indicated that by adding the tunnel cross-section factor and the blast scale factor and the calculation and illustration methods to calculate standard charges for various blasting designs simply and plainly as a result of research and theoretical investigations, that was relied upon the empirical methods in the past. This is indeed an aspect of blasting which has been neglected to some extent and, both development work and studies are needed in this field.

2. Study on the old blasting theory

As described before, in the equation

$$L = f(n) \cdot e \cdot g \cdot d \cdot W^3$$

The g values of Messrs, Aoyama, Lares and Suzuki are described together in many books. But they are all different values and it is hard for anyone to take one among them. For instance, the Aoyama's values which are mostly applied in industries are almost 2.5 times of

those of the Lares, and this is resulted from the cross-sectional area factors in case of tunnel drilling. Therefore the Lares' values are recommended to be applied in industries as we are concerned, Taking another example from "The Coal Mines Blasting Readers", L is defined in the book as

$$L = f(n) g. e. d. W^3 \qquad (1)$$

$$f(n) = (\sqrt{1+n^2} - 0.41)^3 \qquad (2)$$

$$\text{or } f(n) = \left(\sqrt{1+\frac{1}{W}} - 0.41\right)^3$$

(by Dambrun)

The former of the two $f(n)s$ is the adjusting value of the Hauser's formula in case of varying the blasting scale. However it is recognized that the increase of the charge varies by the latter. In order to calculate the specific charge by varying the burden W, the increase of the quantity of the charge should be adjusted at the ratio of the scale of blast as follows: We have

$$f(W) = \left(\sqrt{1+\frac{1}{W}} - 0.41\right)^3 \qquad (5)$$

and the equation L should be as

$$L = f(W) \cdot C \cdot W^3 \qquad (3)$$

$$\text{where } C = e \cdot g \cdot d \qquad (4)$$

It is pointed out that the equation (3) is the general equation of calculating the specific charge where the blast factor C is constant even if the burden varies. Explanation giving notes is given as follows in order to clear the relationship of the above equations: Suppose the test blast is carried out where the quantity of the charge is L_1 and the burden is W, and we get the crater index f (n) or the crater adjustment function $f(n)$. Then the equation is

$$L_1 = f(n) \cdot C_1 \cdot W^3 \qquad (1)'$$

$$f(n) = (\sqrt{1+n^2} - 0.41)^3 \qquad (2)'$$

thus, the quantity of the change L is

$$L = \frac{L_1}{f(n)} \qquad (6)$$

That is to say, L_1 of the equation (1)′ is the quantity of the charge for a test blast, which is quite different from L of the equation (3) which is the specific quantity of the charge. C_1 of the equation (1)′ is the value of kg/m³ and it is not constant at all.

In order to apply the Hauser's formula, the equation[1] we will apply should be as

$$L = f(W) \cdot C \cdot W^3$$

$$C = e \cdot g \cdot d$$

$$f(W) = \left(\sqrt{1+\frac{1}{W}} - 0.41\right)^3$$

where L : specific quantity of the charge

W : buren

$f(W)$: scale factor of blast

C : blast factor

e : explosive factor

g : rock factor

d : tamping factor

However it is indicated that the equation for column charges or bench cut can be derived from the Hauser' formula on the basis of this theory by expansively applying the simultaneous blasing theory to the Hauser's.

3. Test blast

Now an example shows a practical calculation in detail applying the Hauser's formula by a test blast: (Example) As an example take the case where the burden $W=70$cm, the quantity of the charge $L_1=600$ grams and $n=R/W =1.1$ by the test blast. Thus the crater radius $R=77$cm.

$$f(n) = (\sqrt{1+1+1^2} - 0.41)^3$$
$$= 1.25$$

Specific charge $L = \dfrac{0.6}{1.25} = 0.48$ (kg)

That is to say, the crater radius 70cm can be obtained by the charge 0.48kg and the burden 70cm.

Now the crater volume is experssed as V (m³)

and we have

$$V = \pi R^2 W/3 \fallingdotseq R^2 W = W^3$$

Thus the quantity of the charge per unit volume $C(kg/m^3)$ is

$$C = \frac{0.48}{0.73} = 1.399 (kg/m^3)$$

Next the calculation shows the value $C = e \cdot g \cdot d$ by the general blast equation (3) described above:

$$L = f(W) \cdot C \cdot W^3$$
$$C = e \cdot g \cdot d$$
$$C = \frac{0.48}{1.514 \times 0.7^3} = \underline{0.924}$$

or $e \cdot g \cdot d = 0.924$

That is to say, C value is 0.924 and it is not 1.399 kg/m^3, and this is meant by a test blast carried ont where $e=1$, $d=1$ and $g \fallingdotseq 1$ that is, the Sakura dynamite (60% NG) was used with complete tamping against the granite.

Now it is indicated that the kg/m^3 values vary in accordance with variance of the burden W as follows:

If $W=0.7m$, then $f(W)C=1.514 \times 0.924$
$\qquad = 1.399 (kg/m^3)$

If $W=1.0m$, then $f(W)C=1.00 \times 0.924$
$\qquad = 0.924 (kg/m^3)$

If $W=3.0m$, then $f(W)C=0.413 \times 0.924$
$\qquad = 0.382 (kg/m^3)$

If $W=5.0m$, then $f(W)C=0.322 \times 0.924$
$\qquad = 0.298 (kg/m^3)$

If $W=8.0m$, then $f(W)C=0.275 \times 0.924$
$\qquad = 0.254 (kg/m^3)$

The above results were calculated theoretically by a test blast and it can be regarded not to obtain them by such experiences of the long period of time.

4. Simultaneous blasting[1]

Simultaneous blasting classifies into

(1) Simultaneous blasting in the column charge.

(2) Simultaneous blasting of plural charges by instantaneous caps.

The column charges are regarded as a series of cartridge group of concentrated charges by dividing several cartridges and each cartridge is seemed as a simultaneous blasting when the hole spacing of the cartridges is zero. So each cartridge may have a resultant force and an effect as that of it's force existing in the center of the charge. In simultaneous blasting of the column charge the line of least resistance of each cartridge is related with the detonating power. And the effect of simultaneous blasting will not go high if the lines of least resistance are not same. Especially it is the reason why it is required more rapid detonating velocity of explosive in the case of hard rock consisting greater resistance of rock. In th simultaneous blasting by instantaneous electric caps, the time period of giving stress coincides with that of giving rupture phenomina, Thus the simultaneous blasting effect will be imporved in this case.

In the simultaneous blasting by millisecond caps, the time period of giving stress delays a little and the following blasting is carried out before the rupture phenomina of the preceding blasting carries out the resultant of rupture will occur between before and after blasting. When the time difference of millisecond caps increases more than it is required, it will be instantaneous blasting, in other words, it will lose the effectiveness of simultaneous blasting and occur cap damage problems for close-by blasting. Therefore it is necessary to improve simultaneity in order to save the charge of explosives.

The milliscond cap or detonating velocity is related to the simultaneous blasting. If second-difference of millisecond caps increases and the detonating velocity decreases the explosion may not become complete simultaneity gradually. If simultaneity increases there generally remains no footleg at all because rock fragmention is carried

out verry well among loaded holes.

Now in the equation $L=f(W)e \cdot g \cdot d \cdot W^3$ in loading a charge of explosive compounds L at the distance of W from the free face we can carry out a normal blasting operation with the volume of breakage $V=W^3$. If four charges of the explosive compounds $4L$ are loaded at a distance of hole spacing W from the free the free face and ignitedasa simultaneous detonation, we can carry out a normal blasting operation with the volume of breakage $V=4W^3$ shown as in figure. 1. If the quantity of n^2 charge of the explosive compounds n^2L is loaded at a distance of hole spacing W from the free face and in a parallel direction to the free face, we can carry out a normal blasting operation with the quantity of breakage $V=n^2W^3$. Now it is recognized that each L can carry out a breakage of W^3.

The complete form of the crater may not be formed by a blast if there is a surface restrained by a free face, although a charge is loaded at a distance of W from the free face. In the crater, the surface of rupture by tension will be formed by the restrained surface. In the neighbourhood of the intersection of the free face and restrained surfaces a crack forms partially and the crater becomes big and eccen-
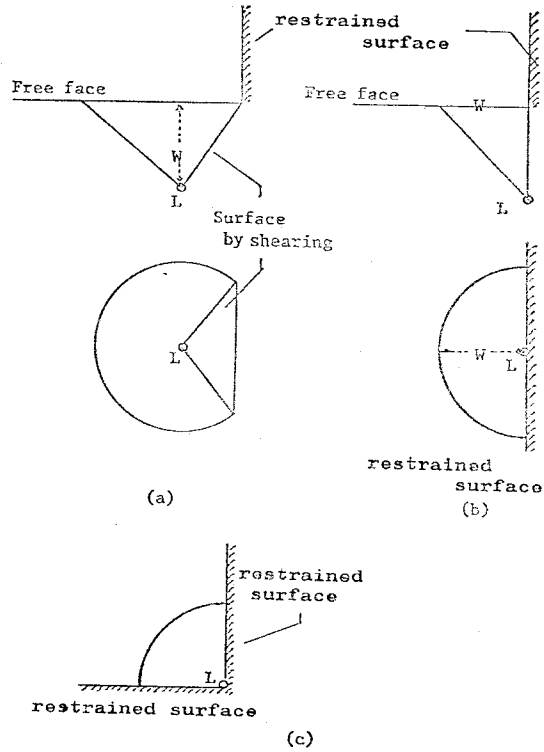

Fig. 2 Restrained blasting

tric and hence the crater may not be large enough to be calculated for as shown in figure 2(a). When the restrained surface comes near the loaded charge of explosive compounds, the crater becomes a semi-conical shape by the restrained surface; that is to say, $V=1/2\ W^3$ as shown in figure 2(b). When the charge is loaded in the neighbourhood of the intersection of two restrained surfaces intersecting at a right
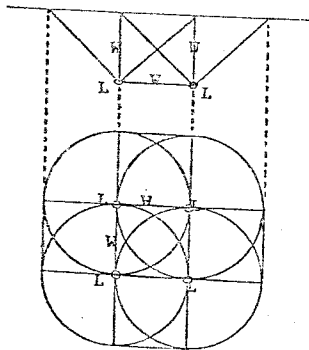

$$4L \longrightarrow 4W^3$$
$$L \longrightarrow W^3$$
$$n^2L \longrightarrow n^2W^3$$
$$V = n^2W^3$$
$$L \longrightarrow W^3$$
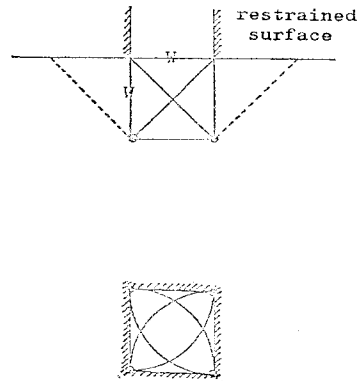
Fig, 1. Simultaneous blasting

Fig. 3 Simultaneous blasting (restrained free faces)

angle, the crater becomes a quarter cone; that is to say, $V=1/4\,W^3$ as shown in Figure 2(c).

Thus when four charges $4L$ are loaded at a distance of hole spacing W described above and at the distance of W from the free face and they are restrained the free face in the neighbourhood of the loaded holes, even if it is a sumltaneous blasting, it will repture the rock of W^3 by $4L$ as shown in figure 3. Next a charge n^2L will move a mass $(n-1)^2W^3$ when fired simultaneously; that is to say, $n^2L=(n-1)^2\,W^3$ as shown in figure 4.

$$3^2L \rightarrow 2^2W^3$$
$$n^2L \rightarrow (n-1)^2W^3$$
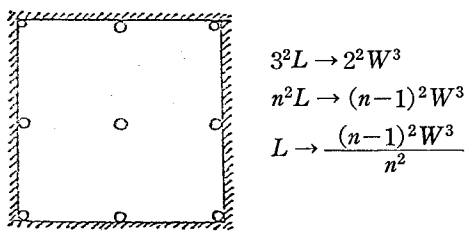$$L \rightarrow \frac{(n-1)^2W^3}{n^2}$$

Fig. 4 Simultaneous blasting

In obtaining the restraint factor a number of charges for a given length will be $n+1$ if the hole spacing is n. Thus we obtain

$$(n+1)^2L \longrightarrow n^2W^3$$
$$\frac{(n+1)^2}{n^2}L \longrightarrow W^3$$

In tunnel excavation $\dfrac{(n+1)^2}{n^2}$ is called a cross–section factor, which can be regarded as "The less cross–section areas are, the more the charges are required in the rock". The maximum of this value is 4 and it draws near 1 in accordance with the increase of the tunnel cross–section area.

5. Illustration and calculation methods[3]

(Example 1) As an example take the case where the specific charge is experssed as kg/m³ and $W=0.6$m, $L_T=500$ grams and the crater index $n=1.02$.

(Illustration method)

In figure 5 if we draw a vertical line down

form the point A of $n=1.02$ in the upper lateral axis, we have the intersecting point B of the $1/f(n)$ curve. From B draw a horizontal line to the point of intersection C of 0.5kg and draw a vertical line down from C to the lower lateral axis and we have the value of 480 grams. This is the specific calculated charge for the example problem.

(Calculation method)

$$L=\frac{L_T}{f(n)}$$

  L : specific charge (g/m³)

  L_T : the charge calculated from a test blast (g/m³)

$$\begin{aligned}f(n)&=(\sqrt{1+n^2}-0.41)^3\\&=(\sqrt{2.04}-0.41)^3\\&=1.054\end{aligned}$$

thus $L=\dfrac{500}{1.054}=\underline{474.38}(g)$

(Example 2) As an example take the case where the specific charge is expressed as kg/m³ and $L_T=0.4$kg and the crater index $n=1.2$.

(Illustration method)

In figure 5 we have the value 0.26kg in accordance with the dotted line according to the preceeding way. This is the specific calculated charge for the example problem.

(Calculation method)

  $1/f(n)=0.66$

thus $0.4\times0.66=\underline{0.264}(kg)$

6. Bench blasting[1]

In the case of bench blasting it is necessary to determine the respective bench height (H), hole depth (l), the line of least resistance (W) and drill hole diameter (d). Theoretically the above factors are related one another as shown in Figure 6.

Figure 6 (A) shows that it is possible to rupture the rock with a standard charge in each hole spacing when $H=3W$ in the location of the maximum burden W.
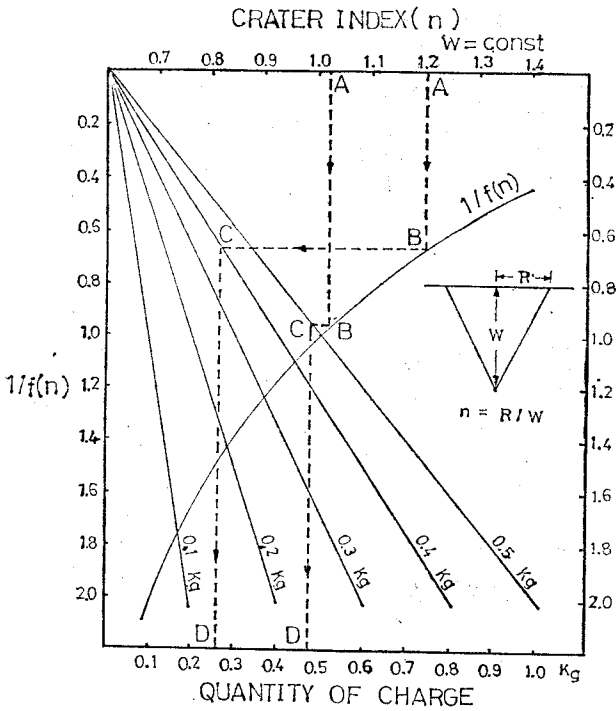
CRATER INDEX(n)

Fig. 5 Specific charge for test blast

length of W. In this case the length of the under-drillng becomes 0.5 W and the tamping length is 0.5 W.

Since each blasting method stated above produces a back break, the tamping length must be big enough. And if the length of the under-drilling is 0.5W it will be too long to waste the quantity of the charge.

The diamater d can be calculated by the relationship between a concentrated charge and a column charge when the concentrated charge L is calculated and then L is prolonged to the length of W by the Hauser's formula. This is proved the reason why the diameter (d) and the loading density have a close relationship each other.

(1) Example of a bench cut design

In a vertical bench, suppose

$$l=3W$$

where l : length of a drill hole

W : line of least resistance

then, the bench height H is,

$$H=3W-0.3W$$

and the tamping length$=W$, the length of the charge$=2W$ and the length of the under-drilling$=0.3W$.

0.3W of the length of the under-drilling is more or less small compared with 2W of the length of the charge.

Figure 6 (B) is same as stated above and is shown a deck charge when the quantity of the charge L prolonged to 0.6W. In this case it requires 0.3W for the under-drilling charge and the tamping length will become 0.7W.

Figure 6(C) is same as stated above and is shown that it is a column charge when the the quantity of the charge L is prolonged to the
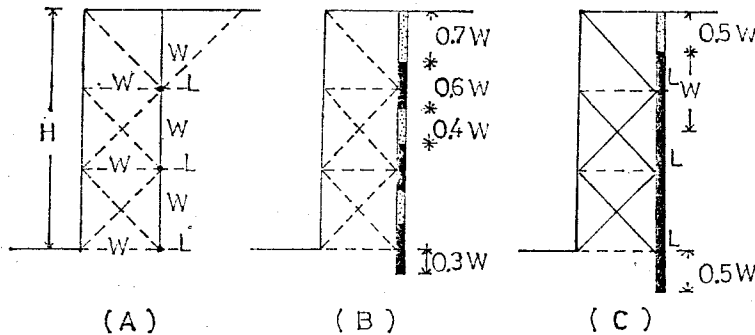

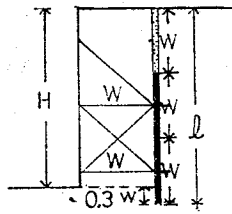(A)　　　　　(B)　　　　　(C)

Fig. 6

Fig. 7.

Accordingly the hole spacing D is,

$$D = W$$

For the specific charge L, we have

$$L = f(W) \cdot e \cdot g \cdot d \cdot W^3$$

The volume of rock breakage V,

$$V = 2.7 W^3$$

The quantity of the charge consumed per m³

$$q = \frac{2L}{2.7 W^3}$$
$$= \frac{2f(W) \cdot e \cdot g \cdot d \cdot W^3}{2.7 W^3}$$
$$= \frac{f(W) e \cdot g \cdot d}{1.35}$$

Now assume $W=3$m, $e=1$, $g=1$ and $d=1$. then

$$q = \frac{0.41}{1.35} = 0.304 \, (\text{kg/m}^3)$$

And if $W=3.5$m, $e=1$, $g=0.4$ and $d=1$, then

$$q = \frac{0.379 \times 0.4}{1.35} = 0.112 \, (\text{kg/m}^3)$$

(2) General equation for bench blasting

In a vertical bench, suppose

the line of least resistance $= W$

the length of a drill hole $= nW$

the length of tamping $= W$

the bench height $H = nW - 0.3W$ (vertical bench)

the length of the charge $= (n-1)W$

the length of the under-drilling $= 0.3W$

the hole spacing $d = W$ (normal)

then we have the quantity of the charge per hole L is

$$L = (n-1)f(W)e \cdot g \cdot d \cdot W^3 \qquad (7)$$

Breakage volume per hole V,

$$V = (n-0.3) W^3$$

Explosive consumption per m³ q

$$q = \frac{(n-1)f(W)e \cdot g \cdot d \cdot W^3}{(n-0.3) W^3}$$
$$= \frac{(n-1)f(W) \cdot e \cdot g \cdot d}{(n-0.3)} \qquad (8)$$

However provided that $n \geqslant 2.3$, q increases much more with the increase of n. And if $n < 2.3$, the blasting will be unstable.

(Example 1) Calculate $q(\text{kg/m}^3)$ if $W=3.2$ m, $n=2.3$, $e=1.1$, $g=0.4$ and $d=1$.

$$q = \frac{1.3 \times 0.398 \times 1.1 \times 0.4}{2}$$
$$= 0.114 \, (\text{kg/m}^3)$$

(Example 2) Calculate $q(\text{kg/m}^3)$ and the quantity of the charge per hole if $W=3.5$m, $n=3.5$, $e=1$, $g=0.5$ and $d=1$.

$$q = \frac{2.5 \times 0.379 \times 0.5}{3.2}$$
$$= 0.148 \, (\text{kg/m}^3)$$

And in the equation (7)

$$L = (n-1)f(W)e \cdot g \cdot d \cdot W^3$$
$$= 2.5 \times 0.379 \times 0.5 \times 3.5^3$$
$$= 20.3 \, (\text{kg/hole})$$

(3) **Result comparison**

The results of the comparative table in the follwing were calculated by varying the burden W where $e=1$ and $g=1$ by the equation of the specific charge L:

$$L = f(W) \cdot e \cdot g \cdot d \cdot W^3$$

W	L (This theory)[1]	L (Langefors)[2)9)]
0.6 m	0.39 kg	0.16 kg
0.8	0.66	0.35
1.0	1.00	0.68
1.2	1.43	1.2
2.0	4.24	5.2
3.0	11.07	17.0

(Example 2) Calculate the quantities of the

charges as the fixations at the bottom(The base rock exists.) vary by test blasts. But $g=0.8$ and $e=1$.

W	L(This theory)	L(Langefors)
0.6m	0.31kg	0.2kg
0.8	0.53	0.46
1.0	0.80	0.9
1.2	1.15	1.6
2.0	3.39	6.9
3.0	8.86	23.0

(Examle 3) Calculate the required quantity of the charge, the breakage volume and g/m^3 if $g=0.4$(soft sandstone) and $e=1$(explosives).

By the equations $L=f(W)\cdot e\cdot g\cdot d\cdot W^3$ and $V=W^3$, the following results were calculated as shown in the table:

By the calcation of this theory[1)3)10)]

W(m)	L(kg)	V(m³)	L/V(g/m³)
0.6	0.16	0.216	727
0.8	0.27	0.512	520
1.0	0.40	1.00	400
1.2	0.58	1.728	334
1.5	0.92	3.375	272
2.0	1.7	8	213
3.0	4.43	27	164

By the calculation of the Langefors[2)9)]

W(m)	L(kg)	L/V(g/m³)
0.6	0.11	486
0.8	0.24	469
1.0	0.44	440
1.2	0.71	411
1.5	1.5	444
2.0	3.4	425
3.0	12.0	444

As stated before, the quantity of the charge per m³ varies an entirely uniform enlargement by Langefors in spite of being grand blasts, but it is not favourable on the job sites. It is evident that the calculation of the grand blasts described above are on the contrary to the fact that g/m³ is resulted in the reduction of the
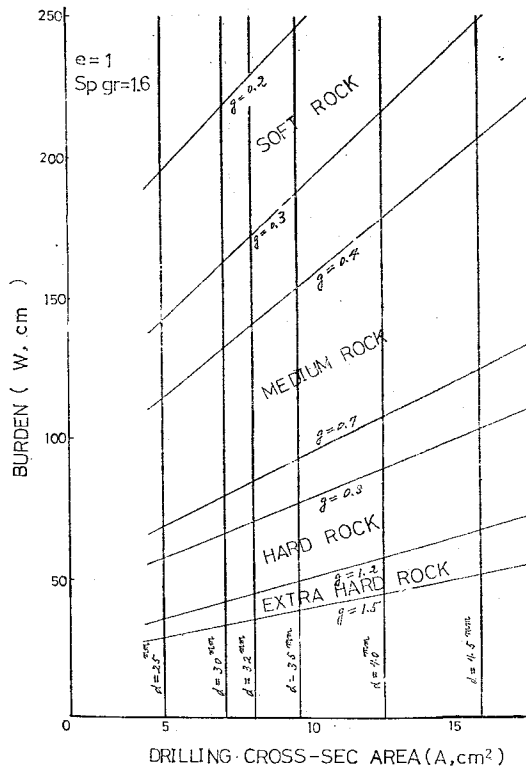

Fig. 8 Relationship between d & W

quantity of the charge. For the reference, the relationship between the hole diameter (d) and the maximum values of the burdens(W) in accordance with the rock factors shown as in figure 8.

(4) **The method to estimate the diameter of drill holes d (mm) by assuming the burden W, rock factor g and explosive factor e.**

(Problem 1) Decide the diameter of drill holes by assuming $W=3$m, $g=0.5$(Limestone), $e=1$(ANFO) and loading density $\triangle=0.8$.

(Illustration method)

In figure 9, if we draw a vertical line up from the point A($W=3$m) in the lateral axis, we have the intersecting point B of $e=1.0$ curve. From B draw a horizontal line to the point of intersection C of the $g=0.5$ line. From C draw a vertical line up to the intersecting
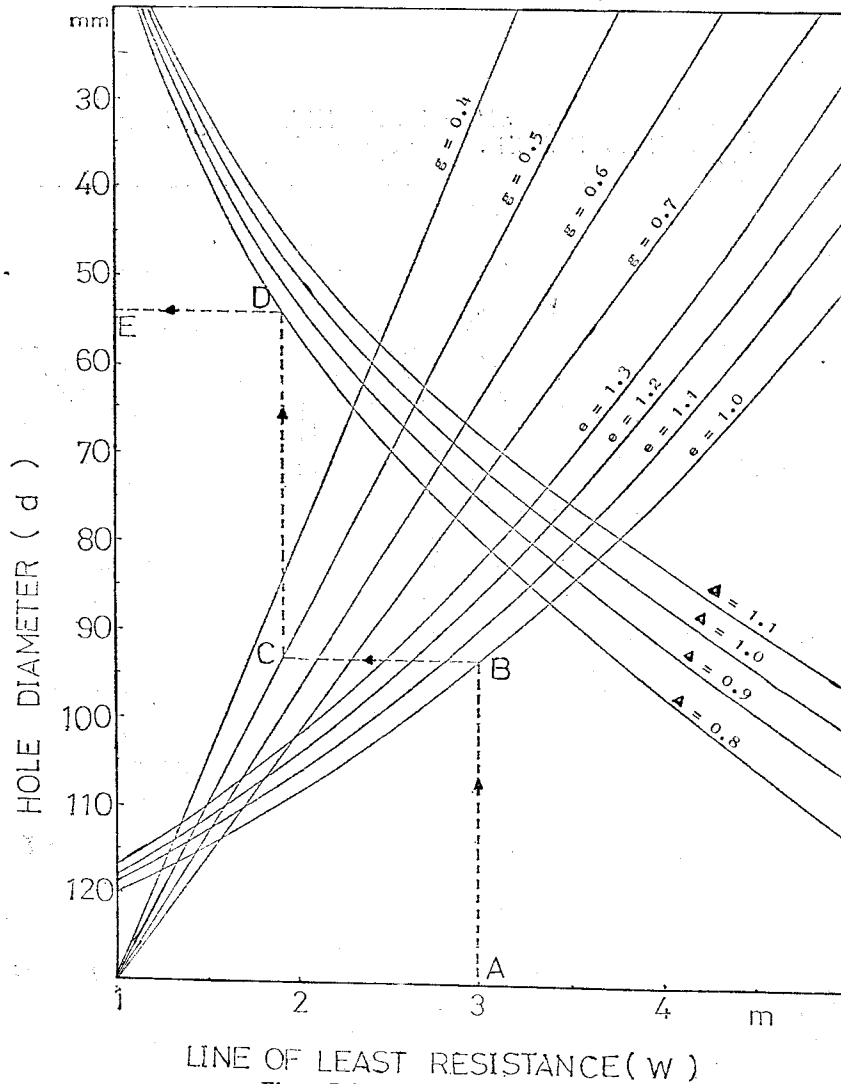
LINE OF LEAST RESISTANCE (W)

Fig. 9 Relationship among W. g. e. d.

point D of the △=0.8 curve, and draw a hori-zontal line to the intersection E of the vertical axis from D. The reading is 54mm, which is the required calculated diameter of drill holes.

(Calculation method)

$$L=f(W)\cdot C\cdot W^3$$
$$=0.41\times1\times0.5\times1\times3^3$$
$$=6.535(kg)$$

5.535÷0.8=6.918

6918÷300=23.06

23.06÷3.14=7.34

$\sqrt{734}$ =27.09

27.09×2=54.2(mm)

(5) **The method to estimate the quantity of the charge per hole by assuming the bench height, the bench slope, rock factor and explosive factor**

(Problem 2) Decide the quantity of the char-ge per hole where the bench height H=12m, slope=75°, W=4m, g=0.5(Sandstone) and e= 1(ANFO). But W(burden)=D(hole spacing) and the burden W is measured by the surface distance.

(Illustration method)

In Figure 10 if we draw a vertical line up from the point A of W=4m in the lateral axis,
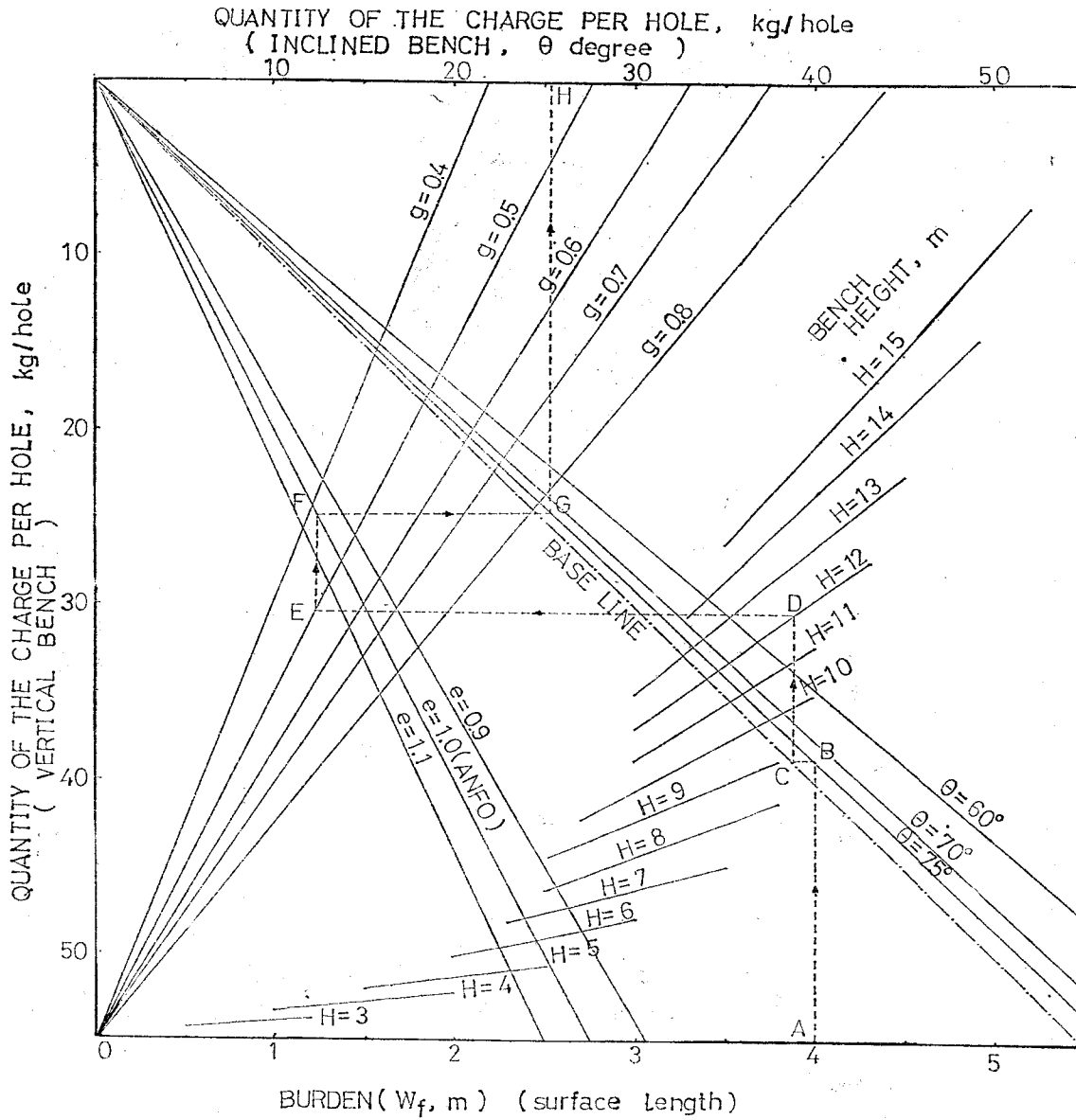
Fig. 10 Relationship among H. θ. g. e and kg/hole

we have the intersecting point B of the $\theta=75°$ line. From B draw a horizontal line to the point C of the base line and draw a vertical line up to the point of D of the $H=12$m line form B. From D draw a horizontal line to the intersecting point e of the $g=0.5$ line and we have

the intersecting point F of the $e=1.0$ line from C by drawing a vertical line. From F draw a horizontal line to the intersection G of the $\theta=75°$ line and we have the intersection H of the upper lateral axis. The reading is $H=25.5$kg/ hole, which is the required calculated charge

per hole for the problem.

(Calculation method)

    bench height $H=12$m

    cosec $75°=1.035$

    length of the slope$=HC$ osec $75°C$

        $=12.42$(m)

    length of the charge$=12.42-4+4\times0.3$

        $=9.62$(m)

    net length of the burden $W=4\times\sin 75°$

        $=3.864$(m)

$f(W)W^3=20.48$

The quantity of the charge per drill hole is,

$$L=\frac{9.62}{3.864}\times f(W)\cdot C\cdot W^3$$

$$=\frac{9.62\times0.5\times20.48}{3.864}$$

$$=25.49=\underline{25.5}\text{(kg/hole)}$$

*Note 1: In the case of a vertical bench we can estimate the quantity of the charge directly with the vertical axis, and not with the base line of the Figure.

*Note 2: This method was carried out by estimating both the actual burden (W_f) (used on the job sites.) and the theortical burden (W) separately. If not, it will become over-charges: that is to say,

    $W=4$m

    $f(W)W^3=22.72$

$$L=\frac{7.62\times22.72\times0.5}{4}$$

$$=27.32\text{(kg/hole)}$$

7. Boulder blasting

It is the blasting to rupture the lumps of the rock produced by tunnel blasts and bench cut blasting and boulders in the fields and roads into a transportable degree.

(1) **The previous rule:** The following equation has been applied for the charge calculation:

$$L=C\ D^2 \qquad (9)$$

    L: quantity of the charge(g)

    D: diameter of the boulder(cm)

    C: blast factor

Blasting method	C value
Boreholling	0.01—0.02
Snakeholling	0.05—0.07
Mudcapping	0.15—0.20

*According to the "Explosives and Blasting"[8], the above stated C values are derived in the case of using Katsura dynamites or Ammonium Explosives.

The following table is also used for the quantity of the standard charge in the case of using Sakura dynamite (50% NG) against the boulders of the medium hardness:

Specific charge for boulder blasting

Boulder diameter	Boreholling	Snakeholling	Mudcap-ping
40~ 50cm	30g	120g	350g
50~ 60	45	180	500
60~ 70	60	270	800
70~ 80	80	330	1000
80~ 90	110	450	—
90~100	140	540	—
100~110	170	660	—
110~120	200	780	—
120~130	240	960	—
130~140	270	—	—
140~150	310	—	—
150~160	360	—	—

According to the theory of this thesis, the equation for the calculation is

$$L=f(W)\cdot C\cdot W^3 \qquad (3)$$

    where L: specific charge(kg)

        W: burden(m)

        C: blast factor$=e\cdot g\cdot d$

    $f(W)$: adjustment function of the blast scale

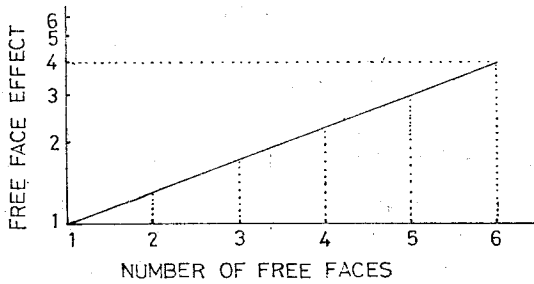As shown in figure 11 the rupturing degree becomes great from the carter radius (R) and

Fig. 11

the burden in the case of having one free face, and the boulders are ruptured all sides similarly in the boulde blasting. Besides the number of the free faces are a lot, so the quantity of breakage becomes great, in other words, it will become the quadruple in the case of a sphere which occurs the least breakage and it will become the octuple in the case of a cube. This is the reason why we take 4(which is the least quantity of breage) for the effect of the free faces(see figure 11).

Number of free faces	Quantity of breakage, V
1	W^3
2	$1.25W$
3	$1.75W$
4	$2.25W$
5	$3W$
6	$4W$

Now, assuming the number of the free faces is 6 and the free faces effect is 4 in boulder blasting, the above equation will become;

$$L = \frac{f(W)}{4} \cdot e \cdot g \cdot d \cdot W^3 \qquad (10)$$

That is to say, this equation is the same form with that of the crack zone.

(2) **Mudcapping blasting**: Here it is indicated that the existing equation is equal to the equation in the following:

$$L = \frac{f(W)W}{4} \cdot e \cdot g \cdot d \cdot W^2$$

If we let the burden W(m) be the diameter D(cm) and let the the quantity of the chage L (kg) be L(g),

then $$L = \frac{f(W)W}{4} \cdot e \cdot g \cdot d \cdot \frac{1000}{10000} \cdot D^2$$

$$= \frac{f(W)W}{40} \cdot e \cdot g \cdot d \cdot D^2 \qquad (11)$$

Since $f(W) = 2.311$ in the case of W = 0.5m,

$f(W)W = 0.5 + 2.311 = 1.155$

Likewise,

W	$f(W)$	$f(W)$	W	$f(W)$	$f(W)W$
0.6	×1.829	=1.097	1.1	×0.917	=1.009
0.7	×1.514	=1.060	1.2	×0.841	=1.009
0.8	×1.295	=1.036	1.3	×0.779	=1.013
0.9	×1.135	=1.022	1.4	×0.727	=1.018
1.0	×1	=1.0	1.5	×0.684	=1.026

From the above result, we have

$f(W)W \fallingdotseq 1$

Since the mudcapping blasting is an external charge, assume $d=3$, $e=1.0$ and $g=1.0$, then the equation(11) will become

$$L = \frac{3}{40}D^2 = 0.08D_2 \qquad (12)$$

(3) **Snakeholling blasting** : If we take 4 for the free faces effect, the equation (6) will become the equation (11), that is to say,

$$L = \frac{f(W)W}{4} \cdot e \cdot g \cdot d \cdot \frac{1000}{10000}D^2$$

$$= \frac{1}{40} \cdot e \cdot g \cdot d \cdot D^2$$

Since it is the Snakeholling method, assume $d=2$, $e=1.0$ and we have

$$L = \frac{2}{40}D^2 = 0.05\,D^2 \qquad (13)$$

(4) **Boreholling blasting**: Since the free faces effect is 4 in the equation(3), $W = D/2$ and we have

$$L = \frac{1}{40} \cdot e \cdot g \cdot d \left(\frac{D}{4}\right)^2$$

$$= 0.06\,D^2 \qquad (14)$$

(where $e=1$, $g=1$ and $d=1$.)

As stated above the existing equations for boulder blasting are able to be derived from the concentrated charge theory taking the free faces effect into account.

(5) **Some examples of the charge calculations**: It is indicated that some examples of the charge calculations are carried out by the equation(10) in the case of each boulder blasting method as follows:

1) Mudcapping method; (Problem) calculate the quantity of the charge where $W=1.2$m, $d=3$, $e=1$ and $g=1$.

From the table 8, $f(W)=0.841$

$$L=\frac{0.841}{4}\times 1 \times 1 \times 3 \times 1.2^3$$
$$=\underline{1.090}\text{(kg)}$$

2) Snakeholling method; (Problem) calculate the quantity of the charge where $W=0.7$m, $d=2$, $e=1$ and $g=1$.

$$L=\frac{1.514}{4}\times 1 \times 1 \times 2 \times 0.7^3$$
$$=\underline{0.260}\text{(kg)}$$

3) Boreholling method: (Problem) calculate the quantity of the charge where $W=1.2$m, $d=1$, $e=1$ and $g=1$.

$$L=\frac{0.841}{4}\times 1 \times 1 \times 1 \times 1.2^3$$
$$=\underline{0.363}\text{(kg)}$$

(6) **Illustation and calculation methods**; The method to estimate the quantity of the charge by assuming the boulder diameter, rock factor and explosive factor:

(Problem) Decide the quantity of the charge where the boulder diameter $D=2.2$m, $g=1$ (Granite) and the blasting is carried out by the boreholling method.

(Illustration method)

In figure 12 if we draw a vertical line from the point A of $D=2.2$m in the lateral axis,

we have the intersecting point B of $d=1.0$ line. Frm B draw a horizontal line to the intersection C of $g=1.0$ line by drawing a vertical line. From D draw a horizontal line to the intersection E of the vertical axis of which reading is 300 grams. This is the required calculated charge for the problem.

(Calculation method)

$$L=\frac{f(W)}{4}\cdot e\cdot g\cdot d\cdot W^3 \text{(Assume } W=1.1\text{m)}$$
$$=\frac{0.912}{4}\times 1.1^3$$
$$=0.303=\underline{300}\text{(g)}$$

8. Tunnel blasting

(1) **Previous rules in rock blasting**; It is found that the charge calculations in tunnel blasting have been empirical by the accumulation of practical experience. It has been said that the key to make a success in tunnel blasting is based on the result of blasting for the cut holes since the quantity of charge to be inserted in the cut and stoping holes are decided by both experience of operations and senses. In practice. a considerable quantity of excess charge is often consumed to compensate for faulty rocks and to provide the extra power needed for swelling, but it often turns to be unsuccessful in this case. It is also said that blasting the charged holes of stoping and walls are easily blasted out only if the cut holes are excavated completely, that is to say, it can be regarded very important to blast the center cut holes by the previons rules.

In blasting the following equations have been applied:

For the cut holes, $L=C\cdot W^3$

For the stoping holes, $L=C\cdot D\cdot W\cdot H$

where L: quantity of charge.
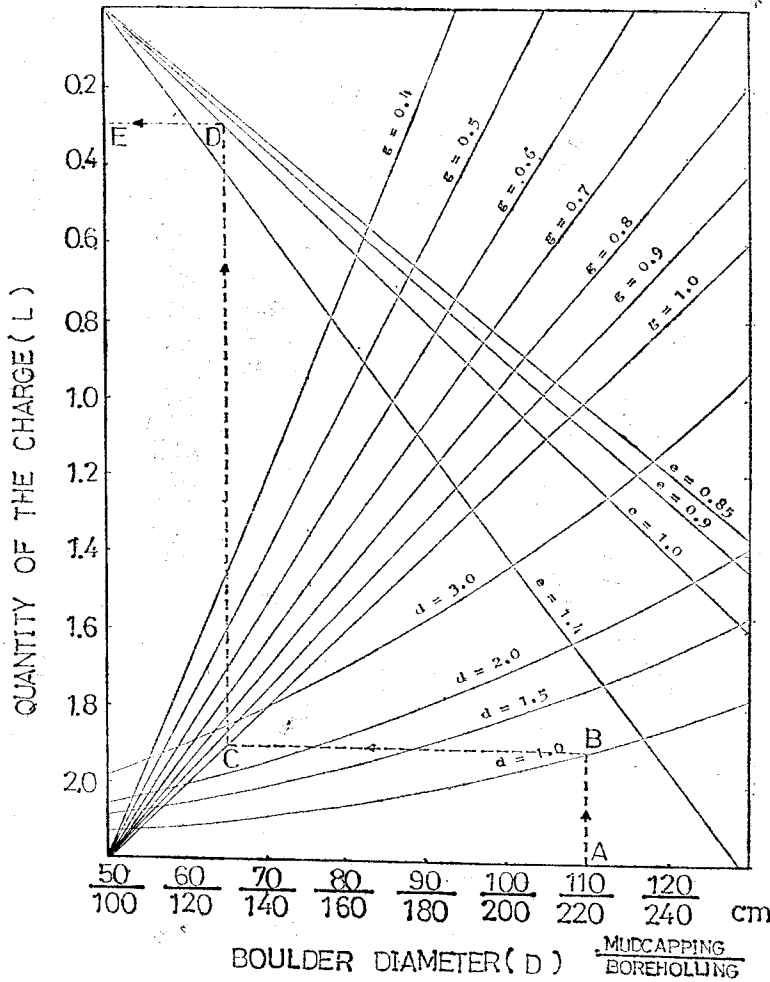
D: stoping holes spacing.

W: burden.

Fig. 12 Boulder Blasting

H: vertical distance from the top to the bottom of the stoping holes.

C: blast factor

For the blast factor C,

$$C = f(W) \cdot e \cdot g \cdot d, \quad (d=1)$$

$$f(W) = \left(\sqrt{1 + \frac{1}{W}} - 0.41 \right)^3$$

C is often defined as $C = e \cdot g \cdot d$. However it is a widely spread misunderstanding to assume that the above two Cs described are not the same factors and they have not yet explained in detail.

The quantity of charge per m³ of rock in tunnel blasting decreases by increase of the tunnel drilling cross-section areas. Therefore it is wrong that the quantity of charge per m³ calculated from the result of tunnel blasting is considered as a rock factor.

Figure 13 is drawn by the results of tunnel blasting at both Sakuma dam and the Kisei railroad line. It is found in the figure that the quantity of charge per m³ and the number of drill holes per m² dercease by increase of tunnel cross-section areas.

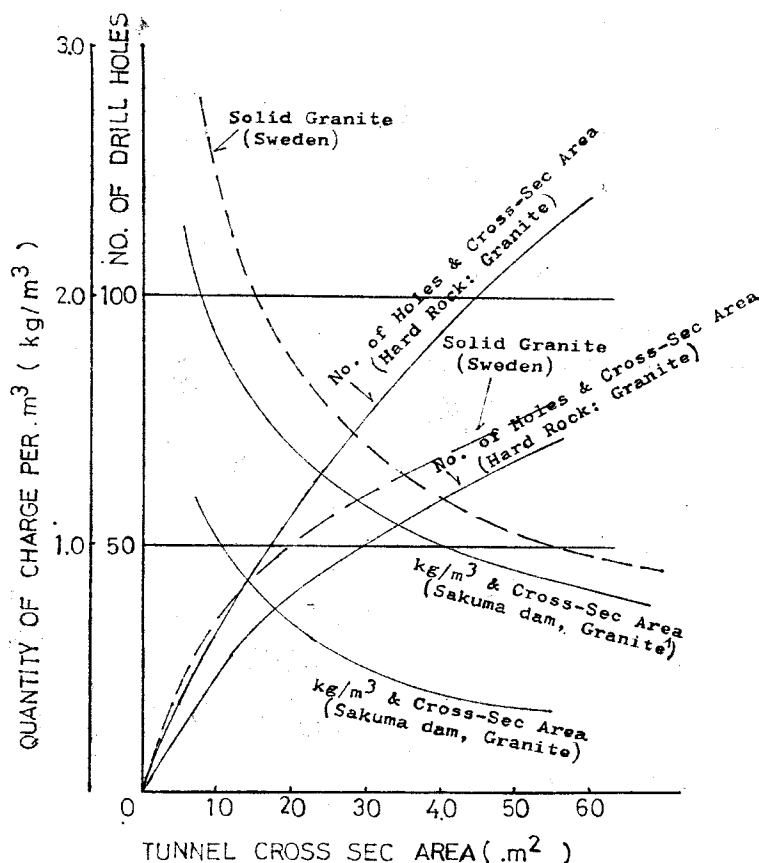1) Tunnel cross-section area and length of drill holes: The lenth of drill holes by V-cut is defined as,

Fgi. 13 Relationship among L, A, & n.

$l = a\sqrt{A}$

where l: length of drill holes(m)

A: tunnel cross-section area(m²)

a: 0. 6–0. 7

The length of stoping holes is around 10% shorter than that of the cut holes. The solpe of inclined holes by V-cut is 60–70 degrees. with this it is not fully explained about Burn cut or parallel cut drilling.

2) Length of drill holes and drilling length; The length of drill holes is greatly influenced upon the effect of blasting. If the length of drill holes is long, the rock walls are affected by this, and it is determined by the following conditions:

(a) the dimentions of tunnel cross-section area.

(b) desired drilliing length.

(c) drilling pattern and blasting method of cut holes.

(d) kinds of rock drills.

(e) kinds of rock wall.

3) Blasting cut holes; The blasting equation is defined as, $L = CW^3$. The role of the cut holes increase the number of free faces and they are classified as in the following by the drilling angle:

(a) Drilling inclined holes (old methods): V-cut, Pyramid-cut, Diamond-cut, etc.

(b) Drilling parallel holes (new methods):

Burn cut, Spiral cut, Center cut, etc. (The fundamental concept of group and simultaneous blasting is involved.)

4) Blasting stoping holes

(2) **This theory**; It is pointed out that the charge calculations in tunnel blasting were empirical and have not been attained theoretically calculated charge concentration yet. Thus the quantity of charge per unit volume(g/m³) or (kg/m³) is regarded as a charge factor. Since the C value is assumed and the quantity of charge is calculated by the formula $L=CW^3$, experience is given top priority in rock blasting at all times, and there was not involved the idea of a tunnel cross-section factor or a scale factor of blast in that calaulation.

It was common that the cross-section factor and a concept of the scale factor of blasting were not in the modern theory of tunnel blasting. Since new techniques of rock blasting developed in these circumstances, they were theoretically unsupported. Therefore some techniques came to an end to be failed such as Burn cut; that may not be said to be the best blasting method and that has not been fully explained the deficiencies of the method. It has been possible in this thesis to improve the compare and to examine carefully a blasting method to another in tunnel blasting.

New a theoretical explosive consumption per unit volume L_T(kg/m³) in tunnel drilling is

$$L_T = \frac{(n+1)^2}{n^2} f(W) \cdot C \qquad (18)$$

$$\text{or } L_T = \frac{(n+1)^2}{n^2} f(W) \cdot e \cdot g \cdot d \qquad (19)$$

Thus the quantity of charge consumed per round L(kg) is

$$L = \frac{(n+1)^2}{n^2} f(W) \cdot C \cdot W \cdot A \qquad (20)$$

$$\text{or } L = \frac{(n+1)^2}{n^2} f(W \cdot e \cdot g \cdot d \cdot W \cdot A \qquad (21)$$

$$\text{where } n = \frac{\sqrt{A}}{W} = \frac{\sqrt{A}}{l-0.1}$$

$$W = l - 0.1$$

$$f(W) = \left(\sqrt{1 + \frac{1}{W}} - 0.41 \right)^3$$

e : explosive factor

g : rock factor

d : tamping factor

A : drilling cross-section area(m²)

W : drilling length(m)

l : length of drill holes(m)

$\dfrac{(n+1)^2}{n^2}$: drilling cross-section factor

$f(W)$: blast scale factor

This relation gives that n means a number of spacing of a tunnel side and $(n+1)$ means a number of drilling holes, and $\dfrac{(n+1)^2}{n^2}$ is a tunnel drilling cross-section factor of which the maximum number is 4 and minimum is 1, so the quantity of charge should be added by such a figure in accordance with variance of tunnel cross-section areas.

A. **Tunnel drilling by V-cut**

(1) **Design conditions**

  (a) Rock: andesite

  (b) Drilling cross-section area: 5.58m²
    2.5m(width)×2.5m(height)

  (c) Drilling length per round: $W=1.2$m

  (d) Explosive: No.3 Kiri dynamite

  (e) Blasting cap: 25 millisecond cap

(2) **Design procedure**

  (a) $g=0.9$, $e=1.11$, $l=1.32$m($=1.2$m× 1.1)

  (b) specific charge per round:

$$L = \frac{(n+1)^2}{n^2} f(W) \cdot e \cdot g \cdot d \cdot W \cdot A$$

$$\frac{(n+1)^2}{n^2} = 2.274$$

$f(W) = 1$(DS blasting cap)

$f(W) = 0.92$(25ms blasting cap)

thus

$$L = 2.274 \times 0.92 \times 1.2 \times 5.58$$
$$= \underline{14.009}\,(\text{kg/round})$$

  (c) Number of drill holes per m²:

Assume 5 holes per m², then

$$5 \text{ holes} \times 5.58 = 27.9 \doteqdot \underline{28 \text{ holes}}$$

(d) Specific charge per hole:

$$14 \text{kg} \div 28 \text{ holes} = \underline{0.5 \text{ kg/hole}}$$

(e) Drilling pattern and data:

	No. of holes	Charge per hole, kg	Total charge, kg	25ms cap No.
Cut	4	0.5	2.0	No. 1
Cut Spreader	4	0.5	2.0	No. 2
Stoping	6	0.5	3.0	No. 3
Roof & Walls	7	0.5	3.5	No. 4
Floor	5	—	2.5	No. 5
Total	26		13.0	100ms

By the equation $L = f(W) \; e \cdot g \cdot d \cdot W^3$,

$$0.5 = f(W) \cdot W^3$$

$$\therefore \; W = \mathbf{0.69} \text{(cm)}$$
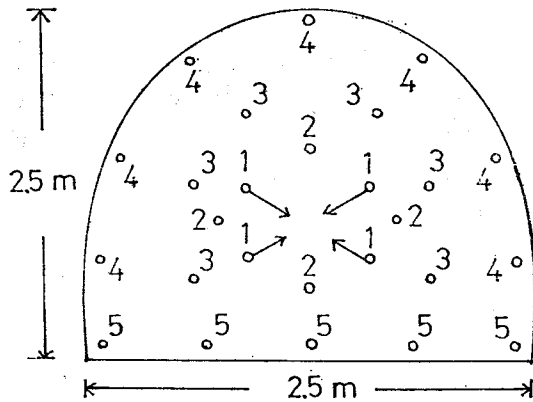
Fig. 14 Drilling pattern–Vcut

(diagram labeled 2.5 m height and 2.5 m width)

B. Tunnel drilling by Wedge–cut

(1) Design conditions

  (a) Rock: hard Rhyolite

  (b) Drilling cross–section area: 5.58m²

   2.5m (width) ×2.5m (height)

  (c) Drilling length per round: $W = 1.5$m

  (d) Explosive: No. 3 Kiri dynamite

  (e) Blasting cap: 25millisecond blasting cap

(2) Design procedure

  (a) $g = 0.9$, $e = 1.11$, $l = 1.65 (=1.5\text{m} \times 1.1)$
   $\Delta = 1.42 \text{g/cm}^3$

  (b) Specific charge per m³:

$$L_T = \frac{(n+1)^2}{n^2} f(W) \cdot e \cdot g \cdot d$$

$$\frac{(n+1)^2}{n^2} = 2.67$$

$f(W) = 1$ (DS blasting cap)

$f(W) = 0.84$ (25ms blasting cap)

$L_T = 2.67 \times 1 \times 1.11 \times 0.9$
   $= \underline{2.67} \text{ (kg/m}^3)$

$L_T = 2.67 \times 0.84 \times 1.11 \times 0.9$
   $= \underline{2.241} \text{ (kg/m}^3)$

  (c) Specific charge per round:

$L = 2.67 \times 1 \times 1.5 \times 5.58$
   $= 2.2348 \doteqdot \underline{22.4} \text{ (kg/round)}$ (DS cap)

$L = 2.67 \times 0.84 \times 1.5 \times 5.58$
   $= 18.8 \text{ (kg/round)}$ (25ms cap)

  (d) Number of drill holes per m²:

   Assume 5 holes (4~6holes) per m², then

$$5 \text{ holes} \times 5.58 = 27.9 \underline{= 28 \text{ hole}}$$

  (e) Specific charge per hole:

$$18.8 \text{kg} \div 28 \text{holes} = \underline{0.67 \text{ kg/hole}}$$

  (f) Drilling pattern data:

	No. of holes	kg/ hole	Total charge kg	25ms cap No.	30mmφ charge length	Tamping length cm
Cut (1)	6	0.7	4.2	No. 1	70cm	95
Gut Spreader (2)	4	0.7	2.8	No. 2	70cm	95
Stoping (3)	6	0.7	4.2	No. 3	70cm	95
Roof & Walls (4, 5)	7	0.6	4.2	4, 5	60cm	105
Floor (6)	5	0.7	3.5	No. 6	70cm	95
Total	28		18.7			

*In this tunnel drilling the hole length is considered long compare with the tunnel cross-section area.

  (g) Hole diameter

Hole diameter	$\overset{*}{l}$, kg/m	Cartridge (100g) length, cm
25mm ϕ	0.6967	14.4
39mm ϕ	1.0032	10.0
32mm ϕ	1.1415	8.8

* $l = 0.785 \times d^2 \times \Delta \times \dfrac{1}{1000}$ (kg/m)
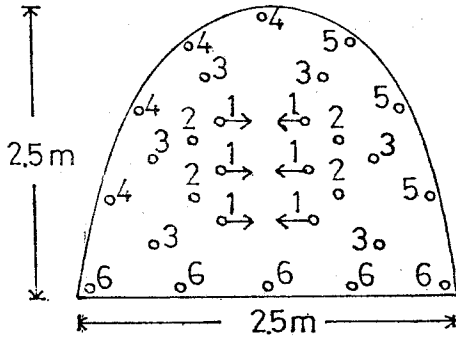
Fig. 15 Wedge cut

(h) Hole spacing

	No. of holes	Hole spacing, D	Line of least resistance, W
Cut	6	0.4m	) 0.5m
Cut Spreader	4		
Stoping	6	0.7m	0.5m
Roof & Walls	7	0.075m	0.5m
Floor	5	0.6m	0.5m

Cut: The quantity of charge is considered
much in this case.

$4.2 = f(W) \cdot e \cdot g \cdot d \cdot W^3$

$e = 1, \ g = 0.9, \ d = 1$

$4.2 = f(W) \cdot W^3 \qquad \therefore \ W = 1.98m$

Cut Spreader: (Cut + Cut Spreader)

$7.0 = f(W) \cdot W^3 \qquad \therefore \ W = 2.47m$

Stoping:

$0.6 = f(W) \cdot W^3 \qquad \therefore \ W = 0.76m$

$W = 0.5$ and $D = 0.75$

thus $D = 1.5W$

Floor: Do

(Conclusion) In the case of making 6 holes
of the center cut to 4 holes and making 7 holes
of the Roof and Walls to 8 or 9 holes, it is
considered that it will be the more stabilized
blasting. In DS blasting, it is also considered
that the simultaneity will be decreased and get
a worse blasting.

C. Tunnel drilling by the Burn Cut

(1) **Design conditions**

(a) Rock: hard rhyolite ($g = 0.9$)

(b) Tunnel cross-section area: $4m^2$

2m (width) × 2m (height)

(c) Drilling length per round: $W = 2m$

(d) Explosive: No. 3 kili dynamite,
LD Shin Katsula (for center cut)

(e) Hole length: 2.2m ($= 1.1W$)

(f) Hole diameter: 30mm or 32mm ϕ

(2) **Design procedure**

(a) Rock factor: $g = 0.9$

(b) Explosive: No. 3 Kili dynamite $\Delta = 1.42g/cm^3$, $s = 0.9$, $e = 1.11$
LD Shin Katsula $\Delta = 0.8g/cm^3$,
$s = 0.9$, $e = 1.19$

(c) Hole length: 2.2m

(d) Specific charge per m^3:

$$L_T = \frac{(n+1)^2}{n^2} f(W) \cdot e \cdot g \cdot d$$

$L_T = 4kg, \ f(W) = 1$ (DS cap)

$L_T = 4 \times 0.77 = 3.08kg, \ f(W) = 0.77$ (10ms cap)

$L_T = 4 \times 0.89 = 3.50kg, \ f(W) = 0.89$ (25ms cap)

(e) Specific charge per round:
$L = 3.08 \times 4 \times 2 = 24.64$ (kg/round)

(f) Number of drill holes:
Assume 6 holes/m^2, then $6 \times 4 = 24$ holes

(g) Specific charge per hole:
$24.6 \div 24 = 1.025$ (kg/hole)

(h) Drilling pattern data:

	Cu	Ctut Spreader	Stoping	Roof & Walls	Floor	Total
No. of holes	4	2	8	7	4	25
Charge per hole (kg)	1.0	1.0	1.0	1.0	1.0	
Total charge (kg)	4.0	2.0	8.0	7.0	4.0	25
10ms Cap No.	No. 1	No. 1	No. 2	No. 3	No. 4	30ms
Explosive	LD Ka.	No. 3 Ki.	Do	Do	Do	
Cartridge diameter, mm	30	Do	Do	Do	Do	
Charging density (kg/m)	0.565	0.565	0.565	0.565	⌣⌣	
Length of charge (m)	1.9	1.0	Do	Do	Do	
Tamping length (m)	0.3	1.2	Do	Do	Do	

(i) Drilling pattern and rupture zone:

CUT: $1.0(kg) \times 4(holes) = 4.0(kg)$

$$L = f(W) \cdot e \cdot g \cdot d \cdot W^3$$

$$4 = f(W) \times 1.19 \times 0.9 \times 1 \times W^3$$

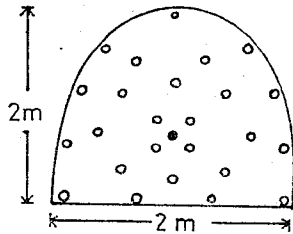$$f(W) W^3 = 3.74, \quad \therefore \quad W = 1.88m$$

Fig. 16 Burn cut

* void hole: 1 hole—low blasting efficiency

The distance between the center of charge and the free face is 1.25 meters.

The radius of the fragmentation zone in the center of the charge is 1.88 meters.

Consequently the center cut is possibly blasted

D. Smoth blasting in cuttings (1)

The blasting plan to blast out a trench in the granite of an open pit.

The drilling pattern and the contour holes are drilled with the pattern shown below. (see figure 17)
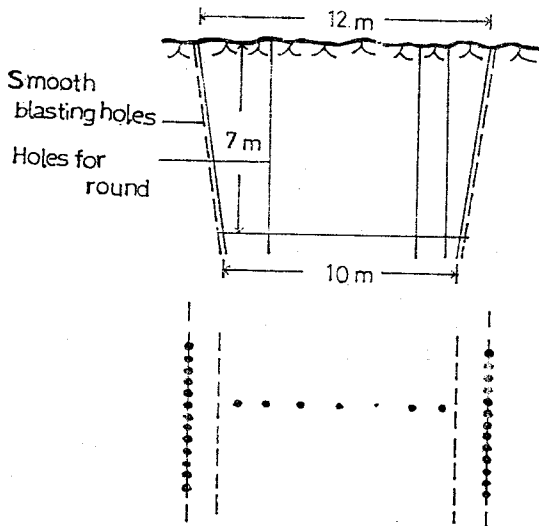

Fig. 17 Smooth blasting (1)

(1) Design conditions

  (a) Trench: top width   12m

                 bottom width 10m

                 depth   7m

   * Ruptured area $77m^2$

  (b) Explosive: ANFO mixtuees

  (c) Rock: granite $(g=1.0)$

(2) Design procedure

  (a) Rock factor: $g=1.0$

  (b) Explosive: $e=1.19$, loading density $\varDelta=0.6$

  (c) Burden: $W=1m$

It is required to decrease the quantity of charge per drill hole in order to protect the surface of the remaining and surrounding rock.

  (d) Hole spacing: $D=0.8$, $W=0.8m$

By this result we will have boulders of bigger than 0.8 meters, so the bottom hole spacing of both right and left flank holes will be decided as 0.3 meters.

  (e) Drilling pattern: see figure 17.

  (f) Quantity of charge per m:

$$L = f(W) \cdot e \cdot g \cdot d \cdot W^3$$

$$W = 1m, \quad f(W) = 1 \quad \therefore \quad L = 1.19(kg/m)$$

  (g) Loading volume:

$$1.19 \div 0.6 = 1.983(l/m)$$

Charging cross-section area: $1.983cm^2$

Drill hole diameter: $19.83 = \pi \left(\dfrac{d}{2}\right)^2$

$$\therefore \quad d = 5(cm\phi)$$

  (h) Quantity of charge per drill hole:

    Length of drill hole—7.3m

    Length of charge—6.3m

Quantity of charge per drill hole:

$$1.19 \times 6.3 = 7.497 = 7.5(kg/hole)$$

  (i) Number of drill holes:

    $12 \div 0.8 = 15(holes)$

    $15 + 1 = 16 holes$

    Both flank—2 holes

    Total **18holes**

  (j) Total quantity of charge per round:

    $7.5kg \times 16 = $**120kg**

(k) Ruptured quantity:

$77m^2 \times 1m = 77m^3$

(l) Quantity of charge per m³:

$120 \div 77 = 1.558 (kg/m^3)$

E. Smooth blasting in cuttings (2)

(1) Design conditions

Same as the preceding example (5∼6), but $W=2m$, $D=1.6m$ and the smooth blasting holes spacing is 0.5m.

(2) Design procedure

(a) Rock factor: $g=1.0$

(b) Explosive: $e=1.19$

(c) Burden: $W=2m$

(d) Hole spacing: $D=0.8W=1.6m$

    Wall side—$D=0.5m$

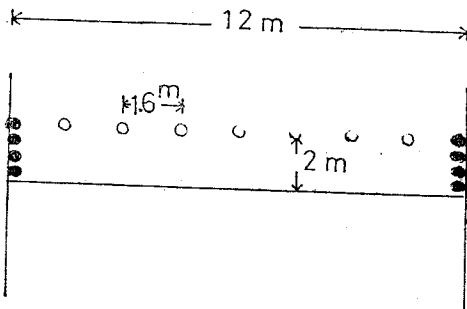   Bottom side—$D=0.2m$

(e) Drilling pattern.

    see figure. 18.

Fig. 18 Smooth blasting (2)

(f) Quantity of charge per m:

$L=f(W) \cdot e \cdot g \cdot d \cdot W^3$

$W=2m$, $f(W)=0.541$

$L=5.15 (kg/m)$

(g) Loading volume:

$5.15 \div 2 = 2.575 (kg/m)$

$2.575 \div 0.8 = 3.219 (l/m)$ $(\Delta=0.8)$

Charging cross-section area: $32.19cm^2$

Drill holes diameter: $r=\sqrt{32.19 \div 3.14}$

        $=3.202 (cm)$

      $\therefore d=6.4 (cm \phi)$

(h) Quantity of charge per drill hole:

    Length of charge=5.3m

$2.575 \times 5.3 = 13.648 (kg/hole)$

$13.648 (kg) \times 7 (holes) = 95.536 (kg)$

(i) Smooth blasting holes:

$W=1.2m$ (top), $W=0.2m$ (bottom)

average 0.7m, hole spacing 0.5m

$L=1.514 \times 1.19 \times 1 \times 0.7^3$

  $=0.618 (kg)$

$\dfrac{L}{W} = \dfrac{0.618}{0.7} = 0.883 (kg/m)$

Length of charge: $7.5m - 0.7m = 6.8m$

Quantity of charge per drill hole:

  $0.883 \times 6.8 = 5.563 (kg/hole)$

Total quantity of charge:

  $5.6 (kg/hole) \times 4 (holes (=22.4 (kg)$

  $22.4 (kg) \times 2 (faces) = 44.8 (kg)$

  $44.8kg \div 2 = 22.4kg$

thus $95.5kg + 22.4kg = 117.9kg$

Number of dril holes:

Usually 7 holes, but 8 smooth blasting holes (2.8kg/hole)

(j) Charging pattern of smooth blasting holes:

  $0.6-0.7kg/hole$

  $L=f(W) \cdot e \cdot g \cdot d \cdot W^3$

    $e=1.19 f(W) \cdot W^3$

  $f(W) W^3 = 0.504$ $\therefore W=0.69m$

(k) Blasting efficiency:

  $77m^2 \times 2m = 154m^3$

  $117.9kg \div 154m^3 = 0.766kg/m^3$

(l) Assuming no smooth blasting holes,

  $13.684 (kg) \times 9 (holes) = 122.832 (kg)$

  $122.832kg \div 154m^3 = 0.798kg/m^3$

(m) Drilling holes error:

Since the holes bottom spacing is 0.2m, special care should be taken to drill holes correct.

8. Comparison of the results

It is shown that the quantity of charge (which is the most important thing in blasting) was not calculated theoretically, but it was calculated by experience and that the mine prefers to take the Burn cut to the Pyramid cut because

the long drilling length and low cost. It is pointed out that the charge calculation is carried out by the Hauser's formula after deciding the blast factor constant by experience at the mine, or there are frequent cases in the mines that the blast factor or the rock factor is calculated by calculating it reversely from the proper quantity of charge determied by changing the charge through the test blasts. If this be the case, it will be acquired a very dangerous over-charge when they use the excessive charge of explosives, so that the cost of explosives goes high and harm the neighbouring soft wall-rock. However it will be no hope in the progres of blasting techniques if experience and the blasting result are given top priority and the study for theoretical pursuit is neglected.

9. Conclusions

Theoretical and practical methods for charge calculations are indicated that was relied upon the empirical methods in the past.

It is showed that the blasting theory of concentrated charges can satisfactorily be applicable to mines and quarrics by drawing clear line between the crater adjustment factor $f(n)$ and the blast scale factor $f(W)$.

It is also indicated in this paper that by adding the tunnel cross-section factor and the blast scale factor the calculation and illustration methods to determine specific charges for various blasting designs simply and plainly as a result of research and theoretical investigations.

This research also deals with experimental blatings at the mines, and the results were examined and new blasting designs resting on the basis of this theory have been practised at the mines in order to improve blasting efficiencies, that will be published in the next issue.

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岩石發破效率에 관한 硏究*

金雄秀·李根培

요약 : 1. 現場發破에 있어서 오늘날 充分히 實用할 수 있는 發破理論이 確立되어 있지 않다고 본다. 그 理由는 從來 사용해오던 Hauoser의 公式이 實用發破에 全혀 도움을 주지 못하기 때문이다.

即, i) 裝藥量修正에 關한 漏斗函數 $f(n)$와 發破規模修正項 $f(W)$와의 混用

　　 ii) 岩石坑力係數 g 와單位體積當爆藥消費量(kg/m³)과의 誤用

　　 iii) 爆破係數 C가 egd 인가, $f(W)$ egd 인가의 不明確性 등이다.

　　本研究에서는 이와 같은 諸問題點을 明確히 하고

2. 齊發發破理論을 擴大適用하여 bench 發破, smooth blasting 및 小割發破에 있어서의 裝藥量公式을 誘導할 수 있음을 證明하고

3. 坑道掘鑿斷面係數 및 發破規模에의 하여 修正한 單位體積當裝藥量(kg/m³)을 求하고 總裝藥量을 算出하여 發破設計를 할 수 있는 方法의 例를 들어 보였다.