A THEORETICAL AND PRACTICAL INVESTIGATION
OF LARGE-SCALE ROCK BLASTING.

By

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</tbody>
</table>
INTRODUCTION

The breaking down of rock has been taking place for centuries and in this modern age the quantities of the different types required are very high indeed. The two main classes of rock breaking on the surface as distinct from mining work are public works excavation for roads, railways, dams, etc., and quarrying for the production of roadstone, limestone, sand, gravel and metalliferous ores. The principles of breaking down the rock by explosives have changed little through the years and are similar for all classes of work. This paper describes investigations carried out in quarries mainly in Great Britain but the results of these investigations and the conclusions reached apply to all surface rock breaking.

There are two main divisions in the methods employed to place explosives in position to break down rock:-

1. By shot holes drilled into the rock.
2. By tunnels driven into the rock.

The above methods are treated separately and the paper is therefore divided into two parts.
PART I

THE BREAKING OF ROCK BY EXPLOSIVE CHARGES IN SHOTHOLES

DEPTH OF SHOTHOLES

The depth of shothole depends firstly on the type of drilling equipment available and secondly on the height of the rock face. Jack hammers or wagon drills have usually a limiting depth of hole of about 20 feet, though in recent times certain of these machines have been adapted to drill with extension steel and holes up to 80 feet deep have been drilled successfully. With well drills, practically any depth of hole can be drilled and these machines are used when it is desired to blast the full height of face in one operation.

Where the depth of the hole is not limited by the machine and full face drilling is practiced, the main consideration is the depth to drill below quarry floor level. Where the strata are well bedded, a good bedding plane is usually taken as floor level and in such conditions, good breaking-out of the "toe" can be obtained by finishing the hole at this plane. Where the rock has to be sheared off without assistance from lines of weakness, however, it is necessary to drill below floor level. The amount will vary from about 2 feet where the rock is easily fractured to 8 feet where the rock is very hard.

Deep vertical drilling with jack hammers and wagon drills is usually augmented by horizontal holes and where these are used to break out the rock at quarry floor level, then the vertical holes need only be drilled to within 1 or 2 feet of floor level.

DETERMINATION OF BURDEN FOR SHOTHOLES

In order to obtain the maximum efficiency in blasting, one of the most important factors is the correct burden on the shothole. The/
The burden is the distance of the bottom of the shothole from the nearest free face, and this should always be less than the depth of the hole. The maximum amount of work has to be done at the bottom of the hole as the explosive must supply sufficient force to shear the rock off at quarry floor level in addition to breaking it off along the line of hole and displacing the mass of rock.

Assuming homogeneous rock, and taking an extreme case as shown in figure 1, where the burden B is greater than the depth D if a charge is detonated at point X, the line of least resistance would be upwards, and the rock would be broken approximately along the lines XY and XZ.

In figure 2 B = D and in this case a charge detonated at X would have two equal lines of least resistance, and if the explosive contained sufficient energy then it would break off the corner along the line YXZ. In practice, however, the charge in a borehole is a column of explosives and not a point.

![FIG. 1](image1.png)  ![FIG. 2](image2.png)

so that under such circumstances the blast would tend to blow a crater upwards XY_W_1. 

In figure 3, B is less than D and a charge at point X would blow/
blow out a crater XYZ in the direction of the burden. A column of explosives in the borehole would, therefore, be equivalent to a large number of point charges each breaking out towards the free face along the lines \( X_1Y_1, X_2Y_2 \text{ etc.} \), and so the block of rock LXZM would be broken down.

Figure 4 shows B less than D where the rock has to be broken off along the bottom and in this case a charge at X cannot break off the rock downwards and the break-out will be as XYZ. Again a column of explosives will break out along the lines \( X_1Y_1, X_2Y_2 \text{ etc.} \) and break down the block LXZM. Assuming a break-out angle of 45° when the charge reaches \( X_n \) the break-out point will be \( Y_n \) or \( M \), any increase of the charges above this will result in the rock being broken upwards and so thrown in the air.

The foregoing is on an approximately theoretical basis but practice approaches it very closely. The explosive column acts as a series of point charges, but all the energy from the explosive does not go into breaking out the rock. Some goes to pulverising the rock in the immediate vicinity of the charge, and some escapes up the shothole into the air. By filling the shothole above the explosive/
explosive charge with stemming material such as sand or sand-clay, the loss of energy up the shothole can be greatly reduced and the amount of stemming should not be less than the burden, otherwise the line of least resistance becomes upwards and not outwards. In practically every case also the shothole fractures and allows energy to escape before the full pressure possible from the explosive can be built up in the shothole.

It is difficult in practice to make the stemming as resistant
as the burden so that the length of stemming material should be greater than the burden, the amount varying according to the efficiency of the stemming used and the diameter of the shothole, small holes retaining stemming better than large ones.

Considering now the relationship between the burden on the shothole and the charge to be used.

If a charge is placed at $P$, when detonated it should blow out a piece of rock with an angle at the apex of approximately $90^\circ$. The volume of rock blasted is calculated by multiplying the area of triangle $EFG$ by the depth of the shothole.

$$ V = \text{Area of } \Delta EFG \times D $$

$$ = \frac{1}{2} \times 2B \times B \times D $$

$$ = B^2D $$

If it is assumed that the charge is proportional to the volume of the rock to be blasted and $k$ is a factor depending on the rock to be broken, then charge $C = kB^2D$.

Similarly for a volume of rock $V_1$, the charge required would be $C_1 = kB_1^2D_1$

$$ \therefore \frac{C}{C_1} = \frac{kB^2D}{kB_1^2D_1} \text{ or } C \propto B^2D \quad (1) $$

The charge is, therefore, proportional to the square of the burden on the shothole multiplied by the depth.

In blasting rock, therefore, a small increase in the burden on the shothole means that a greatly increased charge is required, e.g. where the same depth of hole is maintained but the burden is increased by 50% from say 6 ft. to 9 ft.

$$ \frac{C}{C_1} = \frac{B^2D}{B_1^2D_1} \quad (1) $$

$$ \frac{C}{C_1} = \frac{6^2}{9^2} $$

$$ \therefore \ C_1 = 2\frac{1}{4}C $$
An increase in burden of 50%, therefore means an increase in charge of 125%.

In the following example where a limestone quarry changed from a 6\(\frac{3}{8}\)" to a 9" hole the actual charges for the increased burden were very close to the theoretical:

<table>
<thead>
<tr>
<th>Bit Dia. in.</th>
<th>Explosive Dia. in.</th>
<th>Depth ft.</th>
<th>Burden ft.</th>
<th>Actual Theoretical Burden (B_1 = B + D_1) ft.</th>
<th>Spacing ft.</th>
<th>Actual Charge lb.</th>
<th>Theoretical Charge Calculated from (C = \frac{B^2}{\frac{1}{2}B_1^2})</th>
</tr>
</thead>
<tbody>
<tr>
<td>6(\frac{3}{8})</td>
<td>6</td>
<td>60</td>
<td>20</td>
<td>27</td>
<td>17.5</td>
<td>350</td>
<td>640</td>
</tr>
<tr>
<td>9</td>
<td>8</td>
<td>60</td>
<td>27</td>
<td>27</td>
<td>20</td>
<td>570</td>
<td></td>
</tr>
<tr>
<td>6(\frac{3}{8})</td>
<td>6</td>
<td>120</td>
<td>27</td>
<td>36</td>
<td>17.5</td>
<td>1000</td>
<td>1680</td>
</tr>
<tr>
<td>9</td>
<td>8</td>
<td>120</td>
<td>35</td>
<td>36</td>
<td>20</td>
<td>1600</td>
<td></td>
</tr>
</tbody>
</table>

The relationship between burden and cartridge diameter is discussed in the next section but it can be seen that the actual figures in the above table comply with the theoretical.

**DIAMETER OF SHOPTHOLE IN RELATION TO BURDEN**

The amount of charge which can be placed in a shothole depends upon its diameter and its depth and these are usually fixed by the equipment available. For good blasting results it is most important, therefore, to choose the correct burden for the equipment.

The charge will depend on the diameter of the hole and its depth, then allowing for clearance for the explosive cartridges, the volume of the charge will be slightly less than the volume of the hole. The weight of charge will be directly proportional to the volume, therefore, the weight of charge will be:

\[ C = \frac{\pi}{4} d^2 x l x \Delta \] (2)

where

- \(C\) = charge
- \(d\) = dia. of cartridge
- \(l\) = length of charge
- \(\Delta\) = density of explosive
If the diameter of the cartridge is increased to \(d_1\) then

\[
C_1 = \frac{\pi}{4} d_1^2 \times 1 \times \Delta
\]

\[
\therefore \frac{C}{C_1} = \frac{\frac{\pi}{4} d^2 \times 1 \times \Delta}{\frac{\pi}{4} d_1^2 \times 1 \times \Delta}
\]

\[
\frac{C}{C_1} = \frac{d^2}{d_1^2}
\]

(3)

The charge in a shothole is, therefore, proportional to the square of the diameter.

It has already been shown that the charge is proportional to the square of the burden.

\[
\frac{C}{C_1} = \frac{B^2 D}{B_1^2 D_1}
\]

\[
\therefore \frac{C}{C_1} = \frac{d^2}{d_1^2} = \frac{B^2 D}{B_1^2 D_1}
\]

(4)

The total charge \(C\) in a shothole is a product of the area of base of cartridge multiplied by the length of charge so that the formula can be simplified if we take \(Cr\) as the charge of explosive per foot run of borehole, the formula therefore becomes:

\[
\frac{Cr}{Cr_1} = \frac{d^2}{d_1^2} = \frac{B^2}{B_1^2} \therefore \frac{d}{d_1} = \frac{B}{B_1} \text{ or } \frac{B_1}{B} = \frac{d}{d_1}
\]

(5)

Formula (5) can be applied to determine the burden for different diameters of shothole. To apply the formula in any particular quarry test shots are necessary to determine the burden of rock which can be broken by one diameter of explosive cartridge. The test holes can be drilled with a jackhammer and 6 to 8 feet deep would be sufficient. Such a machine would drill a hole to suit \(\frac{1}{2}\)" diameter explosive cartridges and the first hole should be drilled in a representative part of the face with a burden approximately equal/
equal to what could be blasted satisfactorily. Figure 6 shows an 8 ft. deep shot hole with a burden of 4 ft. at the bottom. The hole should be charged for 4 ft. with $\frac{1}{4}$" diameter cartridges, properly stemmed and fired. If the rock breaks out to the bottom of the hole then $\frac{1}{4}$" diameter explosive will take at least 4 feet of burden and it should be possible to form an opinion as to whether the burden could be increased slightly. If so a second shot should be tried and so on until the correct burden is established.
Where it is not possible to get a vertical face the correct burden can still be determined by a trial shot. Figure 7 shows a sloping face with the toe increasing with depth. In this case the burden at the bottom could be made greater, say 5 ft., and the hole charged and fired as before. If the rock is properly broken out to the bottom then 5 feet will be the minimum burden.

If the hole fails to break to the bottom the result will be as Figure 8 and the distance AB from the top of the hole socket to the old line of the free face will be the correct burden.

Having determined the burden for 1\(\frac{1}{2}\)" diameter cartridges (say 4 feet) by applying formula (5), the burden for any diameter of explosive can be found. For example, if it is desired to use larger bits allowing for 2" diameter explosive, then the burden which could be blasted by this size would be:

\[
B_1 = \frac{Bd_1}{d} = \frac{4 \times 2}{1\frac{1}{2}} = 6.4 \text{ ft.}
\]
Taking for another example a quarry which is using small drilling equipment and wishes to change to well drilling, then assuming it has been found that \( \frac{1}{2} \)" diameter cartridges will blast 4 feet if the well drills take 5" diameter explosive, then the burden for the well holes will be found from:

\[
B_1 = \frac{Bd_1}{d} = \frac{\frac{4}{5} \times 5}{\frac{1}{2}} = 16 \text{ ft.}
\]

If 9" well drills were available to take 8" diameter explosive then similarly the possible burden would be:

\[
B_1 = \frac{\frac{4}{5} \times 8}{\frac{1}{2}} = 25.3/5 \text{ ft.}
\]

In table 1 values of burden are given for various diameters of explosive. These are calculated from formula (5) for \( \frac{1}{2} \)" diameter cartridges where such cartridges have been found to break burdens of 4', 5' and 6'.

<table>
<thead>
<tr>
<th>Dia. of Explosive</th>
<th>4'</th>
<th>5'</th>
<th>6'</th>
</tr>
</thead>
<tbody>
<tr>
<td>( \frac{1}{2} )&quot;</td>
<td>3.6</td>
<td>4.5</td>
<td>5.4</td>
</tr>
<tr>
<td>( \frac{1}{8} ) 7/16&quot;</td>
<td>4.6</td>
<td>5.75</td>
<td>6.9</td>
</tr>
<tr>
<td>( \frac{1}{8} )&quot;</td>
<td>5.6</td>
<td>7.0</td>
<td>8.4</td>
</tr>
<tr>
<td>2&quot;</td>
<td>6.4</td>
<td>8.0</td>
<td>9.6</td>
</tr>
<tr>
<td>2 1/4&quot;</td>
<td>7.2</td>
<td>9.0</td>
<td>10.8</td>
</tr>
<tr>
<td>2 1/4&quot;</td>
<td>8.0</td>
<td>10.0</td>
<td>12.0</td>
</tr>
<tr>
<td>2 1/4&quot;</td>
<td>10.4</td>
<td>13.0</td>
<td>15.6</td>
</tr>
<tr>
<td>6&quot;</td>
<td>16.0</td>
<td>20.0</td>
<td>24.0</td>
</tr>
<tr>
<td>8&quot;</td>
<td>19.2</td>
<td>24.0</td>
<td>28.8</td>
</tr>
<tr>
<td>11&quot;</td>
<td>25.6</td>
<td>32.0</td>
<td>38.4</td>
</tr>
<tr>
<td>35.2</td>
<td>44.0</td>
<td>52.8</td>
<td></td>
</tr>
</tbody>
</table>

The values given in Table 1 have been confirmed by actual practice.

In one whinstone quarry where holes using \( \frac{1}{2} \)" diameter cartridges had
to shear off the rock at floor level it was found that a 4 ft. burden gave the best results but where the holes were drilled up the face the burden could be increased to 6 ft. This increase was possible because of explosive being able to break downwards and gain advantage from the jointing off the rock.

In quarries where the bedding planes are well defined, such as limestone, 1½" diameter cartridges should be able to blast burdens of 6 ft.; this will not be affected by the position of the holes on the face as, whether at the base or up the face, the explosive will be blasting to a bedding plane.

A recent development in Great Britain is the long small diameter hole and it is interesting to apply the foregoing theory to this type of blasting. The diameter of hole drilled by one extension steel machine only permits of a 1½" diameter cartridge and from Table 1 this is only capable of blasting from 3.6 ft. to 5.4 ft. In practice with a high face (50 to 80 ft.) it is rarely possible to get the face straight enough to permit of such burdens at the bottom of the face and the holes have had to be drilled with greater burdens. In such cases the result has been that the explosive did not break out the rock to the bottom of the hole and "toe" was left. In one such quarry a burden of 10 ft. at the bottom of the hole was usual and it was found that the bottom 15 ft. of the hole was not broken out, even with holes spaced as close as 3'6". Such a result is to be expected as in hard rock 1½" diameter cartridges of even the strongest and densest explosive have not sufficient energy to shear the rock off at floor level, shear from hole to hole and displace the rock. In order to obtain sufficient energy at the bottom of the hole chambering with high/
high explosive was considered but ruled out because of the likelihood of the hole being closed through loose rock and so lost. The next suggestion was to treat the bottom rock as a separate blast and assume that the vertical holes would blast the rock off the face down to 10 to 15 ft. from the bottom. The bottom rock was then to be broken by horizontal or lifter holes which would be drilled to finish about 1 ft. behind the line of vertical holes. Both vertical and horizontal holes were fired simultaneously and such a scheme gave most successful results. The face was left almost vertical and it would then have been possible to have drilled with smaller burdens, but the combination of vertical and horizontal holes gave good fragmentation at a low cost and this method was continued. Similar schemes with this type of drilling equipment have been put into operation at other quarries with most successful results.

**SPACING BETWEEN SHOTHOLES**

It is common practice in quarry blasting to fire several shotholes simultaneously or with very short intervals between each and the spacing between shotholes should be such that the rock is broken cleanly along the back of the face between shotholes. The jointing of the rock is the most important factor in determining the spacing. Good vertical jointing in line with the row of holes will permit of wide spacing whereas vertical jointing at right angles to the row of holes will require close spacing. With charges placed with a burden B(Fig.9) then/
then if the bursting angle of each charge forms a $90^\circ$ angle with a spacing between charges of $2B$, each charge would operate as a single charge even though the two charges were fired simultaneously.

If the charges were brought closer together until the spacing was equal to the burden, $S = B$, Fig. 10, then the bursting angles of the two charges would intersect at $\frac{B}{2}$ from the free face.

This relationship will be constant irrespective of the value of the burden $B$. The triangle of rock $MNP$, Fig. 10 is outside the theoretical bursting angle but in practice explosive charges do not break out from a point but break the rock around the charge, the extent depending upon the strength, velocity and diameter of the explosive cartridge. The area $MNP$, Fig. 10 will therefore be broken to a certain degree by the explosive charge and a successful break down of the rock in approximately the area $LMNO$, Fig. 10 will result.

It is found in practice that a spacing approximately equal to the/
the burden is satisfactory in homogeneous rock. Where jointing occurs, however, if the joints are in line with the holes, spacing can exceed the burden, and vice versa if the joints are at right angles to the line of holes.

In the case of 6" diameter well-holes with a burden of 16'-18' the spacings are found to vary from 14' to 20' and for 9" holes from 20' to 30'.

**CALCULATION OF CHARGES FOR SHOTHOLES**

Many different formulae have been propounded throughout the years for the calculation of explosive charges but in practice the charges are usually determined by the volume or weight of rock to be blasted. In order to compare the charges obtained by theoretical calculation with those used in practice three formulae have been selected as being representative and these have been applied to a range of shotholes such as occur in quarry practice.

**Lares Formula**

Of all the formulae which have been suggested for calculating explosive charges the German formula of Lares has the greatest number of factors. This formula is given in detail by Prof. Kegel in *Berghau* No.25, 4/12/41. The formula is as follows:

\[
L = f(n)w^3 e f s v d
\]

where

- \(L\) = charge in Kg per metre of hole
- \(f(n)\) = working factor of rock to be broken
- \(w\) = burden of line of least resistance
- \(e\) = brisance factor of explosive
- \(f\) = resistance factor of rock
- \(s\) = factor of structure of rock
- \(v\) = factor of tension of charge
- \(d\) = stemming factor (with proper stemming this is taken as 1)
The need for all these factors shows how complicated it is to calculate accurately a charge of explosive to break down rock.

Considering now the values given by Prof. Kegel for the different factors:

1) $f(n)$: The values for this factor are given for different burdens as follows:

<table>
<thead>
<tr>
<th>$w$</th>
<th>$w^3$</th>
<th>$f(n)$</th>
<th>$f(n)w^3$</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>1</td>
<td>1</td>
<td>1</td>
</tr>
<tr>
<td>2</td>
<td>8</td>
<td>0.49</td>
<td>3.92</td>
</tr>
<tr>
<td>3</td>
<td>27</td>
<td>0.33</td>
<td>8.91</td>
</tr>
<tr>
<td>4</td>
<td>64</td>
<td>0.28</td>
<td>17.92</td>
</tr>
<tr>
<td>5</td>
<td>125</td>
<td>0.24</td>
<td>30</td>
</tr>
<tr>
<td>6</td>
<td>216</td>
<td>0.22</td>
<td>47.52</td>
</tr>
<tr>
<td>7</td>
<td>343</td>
<td>0.21</td>
<td>72.03</td>
</tr>
<tr>
<td>8</td>
<td>512</td>
<td>0.19</td>
<td>97.28</td>
</tr>
<tr>
<td>9</td>
<td>729</td>
<td>0.18</td>
<td>129.78</td>
</tr>
<tr>
<td>10</td>
<td>10000</td>
<td>0.17</td>
<td>170</td>
</tr>
</tbody>
</table>

2) $e = \frac{6500}{\text{Vel. of detonation of explosive}}$

In Britain there are three velocities of explosive in common use - 2500, 3500 and 5000 metres per sec.

$\therefore e = 2.6, 1.85 \text{ or } 1.3$

3) $f$: This is given as varying from 0.2 to 1.55 depending upon the hardness of the rock.

4) $s$: This factor depends on the structure of the rock and is given as the product of the following three coefficients -

$s_1$ depends on thickness of the beds and varies from 0.8 to 1.

$s_2$ depends on the jointing and varies from 1 when no jointing, to 1.4 when strongly jointed.

$s_3$ depends on the position of the hole with regard to the stratification when the hole is parallel the value is given as 1 and when at right angles 1.2.
5) \( v \): The factor of tension of the charge depends on three coefficients:

\( V_1 \) Angle of shothole. This varies from 1 - 1.45 for hard rock.

\( V_2 \) Depends on the assistance given to the charge by a joint or break. The influence of the help given by a break will vary according to the angle the shotholes make with the break. If the angle is obtuse or straight the help given will be at a minimum but as the angle increases so will the help increase. The value given for this factor varies from 0.5 to 1.

\( V_3 \) This is a coefficient of width and depends on the extent of rock face to be broken by the shothole. In hard rock it varies from 1.2 - 1.5.

The factor \( V \) therefore varies from 0.6 to 2.2.

Substituting the maximum and minimum values in the formula the charge \( L \) in Kg per metre of hole is calculated from the following formula:

\[
L = f(n)w^3e \ f \ s \ v \ d
\]

Minimum Value

\[
L = f(n)w^3 \times 1.3 \times 0.2 \times 0.8 \times 0.6 \times 1
\]

\[
= \ 0.125 \ f(n)w^3
\]

Maximum Value

\[
L = f(n)w^3 \times 2.6 \times 1.55 \times 1.68 \times 2.2 \times 1
\]

\[
= \ 14.8 \ f(n)w^3
\]

Mean Value approx.

\[
L = 7.5 \ f(n)w^3
\]

For a shothole with 1 metre burden \( f(n)w^3 \) is given as 1

\[
\therefore \text{charge per metre of hole} = 7.5 \text{ Kg which is equivalent to 16.3 lb. per metre or 5.06 lb./ft. which is absurd even in the hardest rock.}
\]
If the maximum value is taken, this figure is increased to 10 lb./ft. Taking the minimum value, however, the charge is .125 Kg/metre or .08 lb./ft. which is again absurd. It is most unlikely that all the minimum values or all the maximum values would occur together and especially that a slow explosive would be used in the hard rock and a fast explosive in soft rock. If the values for the explosive are interchanged then the minimum and maximum values become:

\[
L = f(n)w^3 \times 2.6 \times 0.2 \times 0.8 \times 0.6 \times 1
\]
\[
= 0.25 f(n)w^3
\]
\[
L = f(n)w^3 \times 1.3 \times 1.55 \times 1.68 \times 2.2 \times 1
\]
\[
= 7.5 f(n)w^3
\]
The mean value then becomes:

\[
L = 3.87 f(n)w^3
\]
For a 1 metre burden the charge is therefore 3.87 kg/metre or 2.62 lb./ft. which is still very high.

Applying the formula to well hole blasting and assuming a 6" well drive with a burden of 5 metres:

\[
L = 3.87 f(n)w^3
\]
\[
= 3.87 \times 30
\]
\[
= 116 \text{ Kg/metre}
\]
\[
= 77.5 \text{ lb./ft.}
\]
This figure is again absurd.

In discussing the relationship between charge and burden it was shown that experience and theory indicated that the charge per foot of hole was proportional to the square of the burden. In the Lares Formula the charge per metre is given as proportional to the cube of the burden which is the reason for the high values.

Modifying/
18.

Modifying the Lares formula by using the square instead of the cube of the burden, the charge in Kg/metre of hole for a well hole with 5 metres burden would be:-

\[ L = 3.87 \times f(n)w^2 \]
\[ = 3.87 \times 0.24 \times 5^2 \]
\[ = 23.22 \text{ Kg/metre} \]
\[ = 15.6 \text{ lb./ft.} \]

This would require a 6" diameter explosive cartridge which is higher than the 5" used in practice. (Appendix 1).

Belgian Formula

The Belgians have examined the Lares Formula and have concluded that it is wrong to adopt the cube rule. They have taken the square rule and compounded a formula from the German as follows:

\[ C = Kd^2 rle \quad (7) \]

- \( C \) = total charge in Kg
- \( d \) = burden
- \( r \) = coefficient of resistance of rock
- \( l \) = length of hole in metres
- \( e \) = factor relating to the strength of explosive

Factors for \( K \) are given for different burdens from 0m20 up to 1 m and these vary from 2.4 to 1.2. No factors are given for burdens above 1 metre but applying the formula to a shothole with a depth of 2 metres, a burden of 1 m and assuming the factor \( r \) and \( e \) to be 1, the charge would be:

\[ C = Kd^2 rle \]
\[ = 1.2 \times 1^2 \times 1 \times 2 \times 1 \]
\[ = 2.4 \text{ Kg} \]
\[ = 5.28 \text{ lb.} \]

Referring/
19.

Referring to Table 3 a shothole 6 ft. deep with a burden of 3 ft. would only require 0.9 lb., so the Belgian formula gives a value five times greater. The Belgian formula has been tried in underground mining with satisfactory results but with the value given to the factors it is obviously unsuitable for quarrying practice.

Military Engineering Formula

In Military Engineering, Vol. IV, the formula given for calculating charges in shotholes is:

\[ C = k v B^2 \]

where \( C \) = Charge in oz. per foot of shothole (8)

\( k \) = A constant depending on the nature of the rock and explosive used.

\( v \) = A constant depending upon the number of free faces. \( v = 1 \) for 1 free face

\( B \) = The burden or line of least resistance (LLR)

For average conditions of rock and using a standard gelatinous explosive \( k \) may be taken as 1 and in average quarry work there are two free faces so that the formula can be simplified to:

\[ C = \frac{B^2}{2} \]  

(9)

In Military Engineering the burden or LLR is given as a function of the depth of the shothole as:

\( B = \frac{3}{4} D \) for short holes (4\( ^{\prime} \)) and

\( B = \frac{1}{2} D \) " long " (12\( ^{\prime} \))

The length of charge in the shothole is also given:

Length of charge = \( \frac{2}{3} D \) to \( \frac{2}{3} D \)

On the above, therefore, where the fixed relationship between Burden and Depth is maintained the total charge for a given shothole is:

Total charge/
Total Charge \[ \frac{B^2}{2} = \frac{4}{3}D \text{ to } \frac{B^2}{2} \times \frac{3}{4}D \]

For a short hole \( D = \frac{4}{3} B \)

\( \therefore \text{Total Charge} = \frac{B^2}{2} \times \frac{4}{3} \times \frac{4}{3}B \text{ to } \frac{B^2}{2} \times \frac{3}{4} \times \frac{4}{3}B \)

\[ = \frac{4}{3}B^3 \text{ to } \frac{B^3}{2} \] (10)

For a long shot hole \( D = 2B \)

\( \therefore \text{Total Charge} = \frac{B^2}{2} \times \frac{2}{3} \times 2B \text{ to } \frac{B^2}{2} \times \frac{3}{4} \times 2B \)

\[ = \frac{2}{3}B^3 \text{ to } \frac{3}{4}B^3 \] (11)

The Military Engineering formula for holes about 12 ft. deep, therefore, gives the total charge in oz. as equal to \( \frac{2}{3} \) to \( \frac{3}{4} \) of the cube of the burden or line of least resistance in feet, this relationship only holding when the burden on the shot hole is in correct relationship to the depth of hole, that is to say a properly placed shot hole with the burden about \( \frac{1}{2} \) of the depth.

Taking a 12 ft. deep shot hole as an example the burden or LLR \( \frac{1}{2}D = 6' \).

\( \therefore \text{Charge in oz.} = \frac{2}{3} \times 6^3 \text{ to } \frac{3}{4} \times 6^3 \)

\[ = 144 \text{ oz. to 162 oz.} \]

\[ = 9 \text{ lb. to 10 lb. 2 oz.} \]

Comparing these figures with the more common calculation derived from experience for 12 ft. deep shot holes in average rock with a burden and spacing of 6 ft., to assess the charge the volume of rock to be blasted would be calculated as follows:-

Figure 11/
Vol. = Depth x Area of base of rect. ABCD
  = 12' x 6' x 6'
  = 432 cu.ft.
  = 16 cu.yd.

Average rock weighs 2 tons/cu.yd.

\[ C = \frac{32}{5} \text{ lb.} = 6\frac{2}{5} \text{ lb.} \]

Experience has proved that for this type of blasting approximately 5 tons of rock should be obtained per pound of explosive so the charge would be \( \text{lb.} = 69 \text{ lb.} \)

In this example, therefore, the M.E. formula gives a charge about 50% greater than actual practice, average M.E. charge being 9\( \frac{1}{2} \) lb. compared with 6\( \frac{2}{5} \) lb. calculated as above.

Repeating for a 20 ft. deep shothole the M.E. formula would give:

\[ C = \frac{2}{3} \times 10^3 \text{ to } \frac{3}{4} \times 10^3 \]

= 41 lb. to 47 lb. approx.

say \( C = 44 \text{ lb.} \)

Calculated/
Calculated:

\[
\text{Vol.} = \frac{1}{2} \times 20 \times 10 \times \frac{20}{27} \text{ cu.yd.} \\
= 74 \text{ cu.yd.} \\
= 148 \text{ tons}
\]

5 tons/lb. explosive = 30 lb. approx.

In this case again the M.E. formula is about 50% greater than practice.

In looking for a reason for this the first point to note is that the M.E. formula gives a charge length of \(\frac{2}{3}\) to \(\frac{4}{3}\) the depth of the hole. This is a larger charge than would be used in practice and does not conform to the recommendation given previously (p.4) on the correct length of stemming. For a 12 ft. deep shot hole according to the M.E. formula the burden or line of least resistance would be 6 ft. and the length of charge 8 to 9 ft.

\[
B = \frac{1}{2}D \\
C = \frac{2}{3} \text{ to } \frac{4}{3} D
\]

The stemming would therefore be \(\frac{1}{2}\) to \(\frac{1}{4} D\) or 4 ft. to 3 ft.

It has already been stated that length of stemming should be equal to the burden, in this case 6 ft., and the charge should, therefore, occupy half the hole of 6 ft. Considering now the charge, this would give for a 12 ft. deep hole.

\[
C = \frac{B^2}{2} \text{ oz. per ft. run of hole} \\
= \frac{6^2}{2} \\
= 18 \text{ oz. per ft. run} \\
\text{Total Charge} = 18 \text{ oz. } \times 6 \text{ ft.} \\
= 63 \frac{1}{2} \text{ lb.}
\]

This compares with 6.4 lb. already calculated.

Again/
Again for a 20 ft. deep shothole:

\[ C = \frac{B^2}{2} = \frac{10^2}{2} \text{ oz. per ft. run} \]

\[ = 50 \text{ " " } \]

Total Charge \[ = \frac{50 \times 10}{16} \text{ lb.} \]

\[ = 31 \text{ lb.} \]

This compares with 30 lb. calculated.

Subject to this correction, therefore, the M.E. formula gives almost the same figures as for practice and a suggested alternative formula would be:

\[ C = \frac{B^2}{2} \text{ oz. per foot run of hole} \]

Total Charge \[ = \frac{B^2}{2} \times (\text{Depth of hole} - \text{Burden}) \]

So far all calculations have been based on shotholes having the burden in a definite relationship to depth, i.e. burden \( = \frac{1}{2}D \). This does not apply in the case of well holes but having established a formula for holes up to 20 ft. deep it is necessary to consider how this formula would apply to such holes.

Taking as a first example a limestone quarry where 6" well holes are drilled 60 ft. deep with 18 ft. burden and 18 ft. spacing:

\[ C = \frac{B^2}{2} \text{ oz./ft.} \]

\[ C = \frac{18^2}{2} \]

\[ = 10 \text{ lb./ft. approx.} \]

Length of charge \[ = D - B \]

\[ = 60 - 18 \]

\[ = 42 \text{ ft.} \]

\[ \therefore \text{ Total Charge} = 10 \times 42 \]

\[ = 420 \text{ lb.} \]
The volume of rock from such a hole would be:

\[ V = \frac{18 \times 18 \times 60}{27} \text{ cu.yd.} \]

\[ = \quad 720 \text{ cu.yd.} \]

At 2 tons/cu.yd. = 1440 tons

At a loading ratio of 5 tons/lb. rock

Total Charge = \( \frac{1440}{5} \) lb.

\[ = \quad 285 \text{ lb.} \]

This gives a large difference in charges between the two methods of calculation but the difference is due to the length of hole filled with explosive. The M.E. formula gives a charge of 10 lb./ft. run which from Table 2 it will be seen that this would require a \( \frac{1}{2} \)" diameter explosive cartridge. In practice a 5" diameter explosive would be used in a 6" borehole giving 12.5 lb./ft. run. For a 285 lb. charge this would give an explosive column of only 22.4 ft. but experience in well hole blasting has shown that it is not necessary to have a continuous column of explosive in the hole to secure satisfactory blasting and it is usual to place about \( \frac{2}{3} \) of the charge in the bottom of the hole and space the remainder as one or two deck charges further up the hole. On this basis, therefore, the M.E. formula for charges per foot run of charge is very close to practice. From the quarry operator's point of view, however, the total charge per hole is required and the most satisfactory method of obtaining this is by calculating the volume of rock to be blasted, converting this to tonnage and then dividing by the blasting ratio, which in normal conditions would be 5 tons/lb. of rock. For hard rock or where it is desired to get better fragmentation with the primary blast, ratio of 4 tons/lb. can be/
be used.

The effectiveness of this method of charging in well holes confirms the theory that much more energy is required to shear the rock off at floor level than is needed to shear along the face from hole to hole.

**Application of Formula to Practice**

The foregoing formulae depend for their use on a variable factor $K$ in addition to other factors for the rock and explosive. The calculation of charges is therefore made fairly complicated and for that reason the charges determined are liable to wide variation. In practice some simple straightforward method is required and the method of calculating the tonnage of rock to be blasted and dividing by 4 or 5 according to the strength of the rock meets these conditions. It has been shown (page 5) that the charge is proportional to the square of the burden multiplied by the length of borehole.

$$C \propto B^2D$$

In effect this formula is being used when the volume calculation is made.

Taking the charge for one of a line of holes, the volume on the hole is approximately a rectangle with the burden $B$ forming one side, the spacing $S$ (which is approximately equal to the burden) the second side, and the depth of hole $D$, the third:

$$V = B \times S \times D$$

if $S = B$ then $V = B^2D$

If $B$ and $D$ are in feet and $V$ is converted to weight (at 2 tons of rock per cu.yd.) then:

$$W = B^2D \times \frac{2}{27} \text{ tons}$$
At a charging ratio of 5 tons rock per pound explosive
the charge is:

\[ C = B^2 \times D \times \frac{2}{27} \times \frac{1}{5} \]

\[ = \frac{2}{135} B^2 \times D = \frac{1}{67} B^2 D \text{ (approx.)} \]

at 4 tons of rock per lb.

\[ C = B^2 \times D \times \frac{2}{27} \times \frac{1}{4} \]

\[ = \frac{1}{54} B^2 D \]

If a formula is required, therefore, the volume method can
be stated thus:

\[ C = k B \times S \times D \quad \text{(14)} \]

where \( C \) = Charge in lb.

\( k \) = A constant varying from \( \frac{1}{54} \) to \( \frac{1}{67} \) or in
round figures \( \frac{1}{50} \) - \( \frac{1}{70} \) depending upon the
hardness of the rock.

\( B \) = Burden at bottom of shothole in feet

\( S \) = Spacing of shothole in feet

\( D \) = Depth of shothole in feet

The best charging ratio is a matter of opinion but generally
in Great Britain it is about 5 tons/lb. explosive, whereas in the
United States it is nearer 4. For most conditions the writer
believes that a figure of about 4.5 tons/lb. gives best results
and this would give a factor \( k \) in formula 14 of approximately \( \frac{1}{50} \).
It is, therefore, recommended that for jackhammer, wagon drill
and well drill holes, the charge should be calculated from

\[ C/ \]
C = \frac{B \times S \times D}{60} \tag{15}

where C = \text{total charge in lb.} \\
B = \text{burden in feet} \\
S = \text{spacing in feet} \\
D = \text{depth in feet}

RECOMMENDED CHARGE WEIGHTS, BURDENS AND SPACINGS

Jackhammer or Wagon Drill Holes

These types of machines are normally used to drill vertical holes up to 20 feet in depth with diameters varying from 1" to 3". Table 2 gives the recommended burdens and spacings for various depths of hole and the total charge using a gelatinous explosive is shown as calculated from:

C = \frac{B \times S \times D}{60} \tag{15}

This gives a ratio of 4.45 tons of rock per pound of explosive used which may be higher than is common in some quarries but which the writer believes should be used to get the best fragmentation. With these drills, with proper spacing and charging, primary fragmentation should be such that practically no secondary blasting is necessary. Too large burdens and spacing will result in large pieces of stone being blown down and it is false economy to try to cut down on the number of holes drilled.

Table 2/
<table>
<thead>
<tr>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
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<th></th>
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<th></th>
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<tr>
<td>6</td>
<td>3</td>
<td>3</td>
<td>2</td>
<td>4</td>
<td>0.9</td>
<td>3</td>
<td>1</td>
<td>7/8</td>
<td>1</td>
<td>4.45</td>
</tr>
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<td>8</td>
<td>4</td>
<td>4</td>
<td>4.7</td>
<td>9.4</td>
<td>2.1</td>
<td>4</td>
<td>1/2</td>
<td>1 1/2</td>
<td>1 1/2</td>
<td>4.45</td>
</tr>
<tr>
<td>12</td>
<td>6</td>
<td>6</td>
<td>16</td>
<td>32</td>
<td>7.2</td>
<td>6</td>
<td>1.2</td>
<td>1 1/2</td>
<td>1 1/2</td>
<td>4.45</td>
</tr>
<tr>
<td>16</td>
<td>8</td>
<td>8</td>
<td>38</td>
<td>76</td>
<td>17</td>
<td>8</td>
<td>2.1</td>
<td>2</td>
<td>2 1/8</td>
<td>4.45</td>
</tr>
<tr>
<td>20</td>
<td>10</td>
<td>10</td>
<td>74</td>
<td>148</td>
<td>33 1/2</td>
<td>10</td>
<td>3.3</td>
<td>2 1/2</td>
<td>2 1/2</td>
<td>4.45</td>
</tr>
</tbody>
</table>
Well Hole Drills

There are two common sizes of well drills in operation in Britain drilling 6⅞" and 9" diameter holes. With these machines the burden on the shothole is usually much less than half the depth and it is not necessary to fill the hole with explosive in order to get satisfactory blasting. In normal practice the charges are "deck" loaded approximately two thirds of the charge being placed at the bottom of the hole and the remaining third spaced out according to hard bands in the rock. The charge required per hole can be calculated from:

\[ C = \frac{B \times S \times D}{60} \]  

(15)

and distribution of the charge may be varied to suit local conditions. Table 3 gives examples for different types of blasts.

When a large number of well holes is detonated simultaneously there will be a considerable ground vibration set up and this may cause disturbance to private property if close to the site of the blast. Reference is made to this question of vibration in Part II and permissible charges for different distances given in Appendix I.

Table 3
<table>
<thead>
<tr>
<th>Rock</th>
<th>No. of holes</th>
<th>Dia. of holes</th>
<th>Hole Placement</th>
<th>Base Charge</th>
<th>Deck Charge</th>
<th>Top Stemming</th>
<th>Total Charge</th>
<th>Total Charge</th>
<th>Total Charge</th>
<th>Total Tonnage</th>
<th>Ratio</th>
<th>Remarks</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>lbs</td>
<td>lbs</td>
<td>lbs</td>
<td>lbs</td>
<td>lbs</td>
<td>lbs</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Limestone</td>
<td>6</td>
<td>6.5</td>
<td>ft.</td>
<td>20</td>
<td>20.0 lbs</td>
<td>10.0 lbs</td>
<td>30.0 lbs</td>
<td>5000.0 lbs</td>
<td>5000.0 lbs</td>
<td>3000.0 lbs</td>
<td>15,500.0 lbs</td>
<td>4.50</td>
</tr>
<tr>
<td>Limestone</td>
<td>12</td>
<td>9</td>
<td>ft.</td>
<td>112</td>
<td>22.0 lbs</td>
<td>25.0 lbs</td>
<td>15.0 lbs</td>
<td>1360.0 lbs</td>
<td>1360.0 lbs</td>
<td>16800.0 lbs</td>
<td>72,000.0 lbs</td>
<td>4.29</td>
</tr>
<tr>
<td>Limestone</td>
<td>8</td>
<td>6.5</td>
<td>ft.</td>
<td>75</td>
<td>18.0 lbs</td>
<td>25.0 lbs</td>
<td>30.0 lbs</td>
<td>495.0 lbs</td>
<td>495.0 lbs</td>
<td>4000.0 lbs</td>
<td>17,700.0 lbs</td>
<td>4.41</td>
</tr>
<tr>
<td>Limestone</td>
<td>4</td>
<td>6.5</td>
<td>ft.</td>
<td>101</td>
<td>20.0 lbs</td>
<td>25.0 lbs</td>
<td>25.0 lbs</td>
<td>708.0 lbs</td>
<td>708.0 lbs</td>
<td>2800.0 lbs</td>
<td>12,700.0 lbs</td>
<td>4.54</td>
</tr>
<tr>
<td>Granite</td>
<td>2</td>
<td>6.5</td>
<td>ft.</td>
<td>94.0</td>
<td>15.0 lbs</td>
<td>25.0 lbs</td>
<td>20.0 lbs</td>
<td>1033.3 lbs</td>
<td>1033.3 lbs</td>
<td>3,800.0 lbs</td>
<td>3.70</td>
<td>Deck loaded</td>
</tr>
<tr>
<td>Basalt</td>
<td>5</td>
<td>9</td>
<td>ft.</td>
<td>90.0</td>
<td>21.0 lbs</td>
<td>25.0 lbs</td>
<td>15.0 lbs</td>
<td>850.0 lbs</td>
<td>850.0 lbs</td>
<td>1625.0 lbs</td>
<td>19,000.0 lbs</td>
<td>4.1</td>
</tr>
</tbody>
</table>

N.B.  FAGD = Polar Ammon Gelatine Dynamite  
O.G. = Opencast Gelignite
Small Diameter Deep Holes

When using such holes the necessity for horizontal holes in conjunction with the vertical holes has been mentioned (page 12). The charge required can again be calculated from the formula:

\[ C = \frac{B \times S \times D}{60} \]  

but in this case part of the charge will be placed in the horizontal holes, e.g. for a face 60 feet high with vertical holes \( \frac{3}{8} \)" diameter, 3'6" apart and with a burden of 10 feet,

\[ C = \frac{B \times S \times D}{60} \]
\[ = \frac{10 \times \frac{3}{8} \times 60}{60} \]
\[ = 35 \text{ lb.} \]

In practice with a machine drilling deep holes \( \frac{1}{8} \)" diameter it is generally necessary to use \( \frac{1}{8} \)" dia. cartridges in order to have sufficient clearance between the hole and cartridge to allow it to drop freely to the bottom of the hole. From Appendix 1, \( \frac{1}{8} \)" dia. explosive gives .6 lb. per foot run and the charge in the vertical hole would be:

\[ C = D - B \times 0.6 \]
\[ = 50 \times 0.6 \]
\[ = 30 \text{ lb.} \]

This leaves 5 lb. to be placed in the horizontal hole giving a total charge of 35 lb. per pair of holes.

If \( \frac{1}{4} \)" dia. cartridges of explosive could be used Appendix 1 shows .8 lb. explosive per foot run of charge and the charge in the vertical hole would be:

\[ C = 50 \times 0.8 \]
\[ = 40 \text{ lb.} \]

Under such conditions the spacing of the holes could be increased to 4'6" ft. as with 5 lb. in the horizontal hole the total charge would be/
be 45 lb.

It has been found that where these machines can drill successfully, higher blasting ratios can be obtained than with other types of drilling. In some cases the spacing of the shotholes has been increased to 5 ft. with good results. In the foregoing example a 5 ft. spacing would increase the rock blasted per pair of shotholes from 155 tons to about 250 tons and as the same charge would be used in both cases the blasting ratio would be increased from 4.5 to 7.1 tons/lb. explosive. This higher blasting efficiency is most likely due to the close drilling of the holes, bringing the explosive cartridges in close proximity to the bulk of the rock to be blasted. To ensure success with this method of blasting it is essential to drill sufficient breast holes to break out the bottom rock.

In another quarry 40 to 45 ft. deep holes are being drilled with a 2-3/8" diameter bit, the burden being 10 ft. and spacing 8'6". Breast holes are also drilled 11 ft. deep at 8'6" spacing. Calculating the charge required from:

\[
C = \frac{B \times S \times D}{60}
\]

\[
= \frac{10 \times 8\frac{1}{2} \times 45}{60}
\]

\[
= 63\frac{3}{4} \text{ lb.}
\]

This charge is for a vertical and horizontal hole and would be split up into 58 lb. for the vertical and 6 lb. for the horizontal. In practice, 62 to 68 lb. is used in the vertical holes and 6 lb. in the horizontal. This is a slightly higher loading figure than is really necessary but fragmentation of the stone is excellent and the extra charge is considered justified by the quarry operators.

The rock in this quarry lends itself to chambering of the shotholes,
shotholes, and a trial was made with this system. A 2 lb. charge of high explosive was lowered and fired in the bottom of five holes, and then a second 2 lb. charge lowered and fired in each hole. Grain explosive was poured down the shotholes into the cavities thus formed and two holes took 20 lb., two 25 lb. and one 30 lb. before the cavities were full. Explosive cartridges were then loaded into the holes in the normal manner.

Table 4

<table>
<thead>
<tr>
<th>Hole No.</th>
<th>B.S.D. ft.</th>
<th>Theoretical Charge BSD 60 lb.</th>
<th>Actual Charge lb.</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td></td>
<td>Cavity Hole Total</td>
</tr>
<tr>
<td>1</td>
<td>12 9 53</td>
<td>95.4</td>
<td>25 84 109</td>
</tr>
<tr>
<td>2</td>
<td>12 9 53</td>
<td>95.4</td>
<td>25 78 103</td>
</tr>
<tr>
<td>3</td>
<td>12 9 53</td>
<td>95.4</td>
<td>20 88 108</td>
</tr>
<tr>
<td>4</td>
<td>12 9 55</td>
<td>99</td>
<td>30 86 116</td>
</tr>
<tr>
<td>5</td>
<td>12 9 57</td>
<td>102.6</td>
<td>20 94 114</td>
</tr>
</tbody>
</table>

Table 4 gives details of the holes and charges used and also the theoretical charges as calculated from BSD 60, which are in reasonable agreement with the actual charges. The five holes were fired and the results compared favourably with those obtained using vertical and horizontal holes.

This method of blasting has a limited application as only hard rock can be chambered successfully but where it can be employed, the elimination of the horizontal holes reduces the cost of drilling and the cleaning down of the face.

In a limestone quarry where a 20 ft. bench was being blasted a trial blast was arranged on the formula given in this paper. Nine holes were drilled for 2" diameter explosive, 20 ft. deep with/
with 9 ft. burdens and 10 ft. spacing. From formula BSD the charges required were 30 lb. in each hole and this amount was used in the seven centre holes with 36 lb. in the end holes. The nine holes were fired simultaneously with Cordtex and after firing there was a clean space of 3 ft. between the face and the rock which was blasted right down to the bottom of the face. Fragmentation was excellent and the rock pile was from 20 ft. to floor level over 30 yards. There was no flying material and the quarry manager stated that "it was the most perfect blast he had ever seen".
The heading or coyotte system of blasting stone has been carried out for many years and consists of large charges of explosives placed in the rock at predetermined intervals according to the conditions. The chambers for the charges are obtained by the driving of small tunnels into the rock face and these are filled up again with broken material when the explosive is in position. Generally, such blasts have been employed to break up large masses of rock in the one operation but over the last five years in Great Britain the writer has been associated with the development of the system as a standard production method of stone getting and where the rock conditions are suitable excellent results have been obtained. Heading Blasting is not suited for every quarry but experience has shown that it has a much wider application than was generally thought a decade ago.

The object of this paper is to examine the work which has been done on Heading Blasting and to record the experience gained.

GENERAL DESCRIPTION AND APPLICATION OF HEADING BLAST METHOD

The Heading Blast Method is carried out by the driving of small, 4\(\frac{1}{2}\) ft. high by 2\(\frac{1}{2}\) ft. wide, tunnels driven into the face of the rock for a distance depending upon the height of the face. Side tunnels are then driven parallel to the outside face of the rock and chambers are made at intervals on these side or back drives to hold the explosive charges. The charges are coupled up for instantaneous firing and the headings back filled with stone which acts as stemming.

The method can be applied to any type of rock but the limiting factor/
factor is the bedding or joint planes. Where these are vertical or nearly so, the conditions are the most suited to this system and the best blasts have been obtained with "on face" blasts in such conditions. The worst circumstances are where the bedding planes are almost horizontal and the Heading Blast method is not recommended unless the face is low and the horizontal beds are combined with almost vertical secondary joints. It is difficult to generalize on the limiting angle of bedding planes but where these are well defined the Heading Blast system can be considered if the bedding planes are at a greater angle than $45^\circ$ to the horizontal.

Fragmentation of the rock with Heading Blasts depends entirely on the jointing and on the nature of the rock. The use of bulk charges means that only a small percentage of the rock is close enough to the charge to be affected by the explosive and the general effect is to deposit the mass of rock on the quarry floor, the rock being broken up by the fall. This has proved to give satisfactory fragmentation in many igneous rock and limestone quarries but in other cases the cost of secondary blasting has been too great and other methods of blasting have had to be adopted.

Heading Blasts have been fired in Great Britain since the end of the 19th Century when they were used with Blackpowder in Granite quarries. As most of the rock was required for "dressed" stone, the larger the pieces the better and while the demand for this type of stone has fallen, several granite quarries continue to use the system. For road or crushed stone quarries a high explosive of the Ammonal type is used. Heading Blasts with this explosive have been used since the 1914-18 war and in fact it was an adoption of the Heading Blast system which was used by the tunnelling companies of/
of the Royal Engineers for the blowing up of Hill 60 and the Vimy Ridge. In the writer's experience a few Heading Blasts were fired in quarries each year up to the finish of the 1939-45 war and in 1945 five such blasts known to the writer took place. Since then the development of the Heading Blast system has been rapid and in the last two years the writer has been associated with 33 and 38 Heading Blasts respectively. These blasts have been for the routine production of stone and several quarries throughout Great Britain have adopted the method as standard.

There are various reasons for the changeover from conventional methods to Heading Blasting but the main ones are as follows:–

1) Present day requirement for large tonnages per blast to suit mechanical loading.

2) Difficulty in getting sufficient skilled drillers and face strippers.

3) Necessity for cheapest possible blasting cost.

4) Greater knowledge of the Heading Blast Method.

CONDITIONS NECESSARY FOR HEADING BLASTING

There are many factors to be considered before deciding that the Heading Blast Method is suitable for any particular quarry but the following are the most important:–

1) Suitable rock face as regards (a) height, (b) length, (c) shape.

2) Sufficient landing space for stone.

3) Sufficient distance from private property to avoid trouble from ground vibration.

1) Suitable Rock Face

(a) Height of face.

As a general rule the writer considers that the minimum height of face for a heading blast should be 50 ft. as below this height
the amount of rock produced is not usually sufficient to justify the cost of making the tunnels.

It is difficult to give a definite maximum height as faces over 200 ft. high have been blasted with charges at ground level by the heading method. Except in special circumstances such blasts are not recommended as there is the danger of the rock being projected from the bottom and even if this does not happen the resulting rock pile will be too high for safe and economical working. The main factor in determining the maximum height for a single row of chamber is the jointing of the rock. If the joints are numerous, well defined and almost vertical then heights of up to 140 ft. can be taken in one lift. Where the jointing is not so favourable it is preferable to limit the height to about 100 ft. and in flat bedded strata, unless this is associated with some vertical jointing, to about 70 ft. Where the face heights are greater than these given, it is necessary to consider using the two tier principle as described later.

The above figures relate to the actual blasting but quarry operators must also consider the loading out of the stone. With a properly designed heading blast the height of the rock pile will be from a half to two thirds of the original face height. Where the stone is clean it will slide down as loading proceeds but if it is dirty or mixed with overburden, "hang ups" can occur which are difficult and dangerous to work. High faces give high rock piles which can mean expensive loading and the ideal height for heading blast faces lies in the range 80 to 100 feet.

(b) **Length of Face**

Heading blasts can be used for any length of face from about 60 ft. upwards but where the method is to be applied for routine production/
production, sufficient face length should be available for at least two blasts and preferably three as once a blast has been fired most of the stone has to be cleared before tunnelling for the blast behind it can begin. The size of each blast can be varied as required but the usual blast consists of four chambers requiring a back drive of 80 to 90 feet. Allowing for the angle of break out of the rock, a face length of 130 to 150 feet is necessary for such a blast and to allow of room for two blasts the quarry face should be at least 260 to 300 feet long.

(c) Shape of Rock Face

For best results the rock face should be straight and nearly vertical, with the joints "on face". A straight face is advisable to enable the back drive of the tunnel to be driven straight and parallel to the outside face and thus ensure equal burdens on each explosive charge.

With heading blasts the charges of explosive are placed near quarry floor level and their size depends on the burden on them. The straighter the face, therefore, the greater amount of rock which will be produced by the blast.

Heading blasts can be fired in both "on end" and "on face" faces but the break out angle is much greater in the latter case resulting in a higher output for the same quantity of explosive.

2) Sufficient Landing Space for Stone

Heading blasts are designed to produce large quantities of stone and it is essential that there is sufficient landing space on the quarry floor for the material broken out by the blast. Where the jointing is vertical or nearly so, the bulk of the rock pile will lie within one and a half times the height of the face but to give a safety margin it is advisable to allow at least twice the face height. With horizontal bedding, particularly if the joints are
close together, there is the possibility that the bottom beds may be forcibly shot out and at least 150 yards of clear floor space should be allowed.

3) **Distance from Private Property**

The large charges used in heading blasting do not cause a loud explosion and there is practically no air blast but they do set up ground vibrations which might give rise to complaints from owners of private property. The ground vibration from blasting charges has been extensively studied over the past years and sufficient knowledge is now available to lay down blasting limits for property at various distances from the blast point. The writer has worked on the basis of a formula derived by one of his associates (Ref. 1 and 2) and has found that while it was originally built up from theory, it has proved most satisfactory in practice. Many blasts have been fired and vibration measurements taken to build up a safe working rule and it has been possible on one occasion to get permission to use charges heavy and close enough to a building to damage it (Ref. 3). The damaging vibration in this case was found to approximate very closely to the theoretical damaging vibration and has shown the factors of safety previously adopted to avoid damage to be sound.

The formula used is

\[ A = \frac{K \sqrt{E}}{D} \]

where

- \( A \) = amplitude of ground movement in thousandths of an inch
- \( K \) = site factor (varies from 50-100)
- \( E \) = explosive charge in pounds
- \( D \) = distance of property in feet from shot point.

The amplitude about which damage could occur is 0.040 inches but it is recommended that the following amplitudes should not be exceeded:
Amplitudes of .004 inches can just be felt by the average person and where it is desired to fire blasts so that the ground movement is insufficient to be felt, then the amplitude would require to be less than .003 inches.

Appendix 1 shows the maximum charges which can be used at different distances from property. It should be noted that short delay firing gives no vibration advantage with heading blast charges.

DESIGN OF HEADING BLASTS

Once it has been decided that conditions are suitable for the heading blast system and the rock face has been straightened up, the appropriate design of blast can be laid out. There are several types to suit the various face conditions and each type will be described separately.

1) Standard Heading Blast Design

The standard heading blast consists of drivages driven in the form of a T, the main drive being driven at right angles to the face line and the back drive parallel to it. Experience has shown that the main drive should be driven to a depth of 0.6 of the height of the face over the chambers with a maximum depth of 50 feet, Fig.1. In special circumstances the main drive has been driven up to 80 ft. deep but for routine working the writer is not in favour of this as the large charges required may be difficult to control and the resulting rock pile will be high and may cause trouble in loading out.

There is no definite restriction on the length of the back drives but the cost of driving increases with the length. On the other hand, the/
the longer the back drives the greater the tonnage of rock produced.

As stated earlier, it is preferable to have sufficient face room for three blasts and, using a four chamber design, this would require 400 to 450 feet of face. Figure 2 shows a sequence of blasts to give regular production under such conditions. Blast No.1 has been fired and is being loaded out. No.2 has been driven and is ready for firing, No.3 is being driven. No.2 blast would be fired when No.1 was almost finished so that output could be maintained from a large rock pile, while the remains of blast No.1 were being cleared away and the face made safe for the tunnel crew to begin driving No.4 when they completed No.3.

An alternative is to fire the first blast in the centre and follow with one on either side. Whichever method is adopted, the first blast will have two tight ends and the second and third will have one free end.

Where the quarry face is not of sufficient length for three blasts in a row, the system can be worked with two which would reduce the length of face necessary to 250 to 300 feet. Blasts would then alternate right and left but good organisation would be necessary to keep up the full quarry output when one blast is almost finished and the other has to be fired to keep up the supply of stone.

Figure 3 shows how heading blasts can be used to extend the face of a narrow quarry, the blasts being fired in the order shown. No.1 blast need not be as large as the others if it is not desired to open up to the side, but a small two chambered blast could be fired to give more face room in preparation for blast No.3.

Examples of standard type heading blasts are given in Table 1.

Table 1/
LOADING OUT

READY FOR FIRING

DRIVING

FIG. 2

SCALE: 1" = 50'

400' - 450'

4. 5. 6.

1. 2. 3.
### Table 1: Standard Heading Blasts

<table>
<thead>
<tr>
<th>Height</th>
<th>Jointing</th>
<th>Tonnage</th>
<th>Estimated Charge</th>
<th>Tons/lb.</th>
<th>Remarks</th>
</tr>
</thead>
<tbody>
<tr>
<td>54'</td>
<td>Good</td>
<td>1. 5,100</td>
<td>1,050</td>
<td>4.75</td>
<td>Good blast, excellent fragmentation.</td>
</tr>
<tr>
<td></td>
<td></td>
<td>2. 3,775</td>
<td>750</td>
<td>5.18</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>3. 4,400</td>
<td>900</td>
<td>4.9</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>4. 5,300</td>
<td>1,200</td>
<td>4.44</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>18,575</td>
<td>3,700</td>
<td>4.62</td>
<td></td>
</tr>
<tr>
<td>70'</td>
<td>Rectangular</td>
<td>1. 4,900</td>
<td>1,100</td>
<td>4.46</td>
<td>Good displacement of stone. Fair fragmentation.</td>
</tr>
<tr>
<td></td>
<td></td>
<td>2. 6,700</td>
<td>1,350</td>
<td>4.97</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>3. 6,400</td>
<td>1,350</td>
<td>4.74</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>4. 7,200</td>
<td>1,400</td>
<td>5.14</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>25,200</td>
<td>5,200</td>
<td>4.80</td>
<td></td>
</tr>
<tr>
<td>80'</td>
<td>Parallel</td>
<td>1. 8,000</td>
<td>1,700</td>
<td>4.7</td>
<td>Good blast. Fragmentation excellent.</td>
</tr>
<tr>
<td></td>
<td></td>
<td>2. 8,750</td>
<td>1,850</td>
<td>4.7</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>3. 7,950</td>
<td>1,700</td>
<td>4.7</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>4. 6,380</td>
<td>1,350</td>
<td>4.7</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>Almost vertical</td>
<td>31,800</td>
<td>6,600</td>
<td>4.7</td>
</tr>
<tr>
<td>90'</td>
<td>Good</td>
<td>1. 11,000</td>
<td>2,800</td>
<td>4.0</td>
<td>Good in every way</td>
</tr>
<tr>
<td></td>
<td>vertical jointing</td>
<td>2. 11,000</td>
<td>2,000</td>
<td>5.5</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>3. 8,000</td>
<td>2,000</td>
<td>4.0</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>30,000</td>
<td>6,800</td>
<td>4.4</td>
<td></td>
</tr>
</tbody>
</table>
### Table 1 (Contd.)

<table>
<thead>
<tr>
<th>Height</th>
<th>Jointing</th>
<th>Estimated Tonnage</th>
<th>Explosive Charge lb.</th>
<th>Ratio tons/lb.</th>
<th>Remarks</th>
</tr>
</thead>
<tbody>
<tr>
<td>110'</td>
<td>Irregular</td>
<td>1. 12,000</td>
<td>2,500</td>
<td>4.8</td>
<td>Chamber 3 slight undercharged. Rock pile out to 160'</td>
</tr>
<tr>
<td></td>
<td>vertical</td>
<td>2. 10,000</td>
<td>2,000</td>
<td>5.0</td>
<td></td>
</tr>
<tr>
<td></td>
<td>jointing</td>
<td>3. 9,000</td>
<td>1,800</td>
<td>5.0</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>31,000</td>
<td>6,300</td>
<td>4.9</td>
<td></td>
</tr>
<tr>
<td>140'</td>
<td>Good</td>
<td>1. 18,500</td>
<td>4,100</td>
<td>4.5</td>
<td>Excellent blast. Rock pile out to 220 ft.</td>
</tr>
<tr>
<td></td>
<td>vertical</td>
<td>2. 19,000</td>
<td>4,000</td>
<td>4.75</td>
<td></td>
</tr>
<tr>
<td></td>
<td>jointing</td>
<td>3. 14,000</td>
<td>2,900</td>
<td>4.82</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>51,500</td>
<td>11,000</td>
<td>4.7</td>
<td></td>
</tr>
<tr>
<td>80'</td>
<td>Good, almost</td>
<td>1. 8,000</td>
<td>1,800</td>
<td>4.44</td>
<td>Fragmentation good. Rock pile out to 80 ft.</td>
</tr>
<tr>
<td></td>
<td>vertical</td>
<td>2. 8,400</td>
<td>1,700</td>
<td>4.38</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>3. 7,400</td>
<td>1,700</td>
<td>4.45</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>4. 8,000</td>
<td>1,800</td>
<td>4.50</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>31,800</td>
<td>7,000</td>
<td>4.50</td>
<td></td>
</tr>
<tr>
<td>85'</td>
<td>Irregular</td>
<td>1. 10,300</td>
<td>2,400</td>
<td>4.3</td>
<td>Excellent blast.</td>
</tr>
<tr>
<td></td>
<td></td>
<td>2. 9,700</td>
<td>2,000</td>
<td>4.85</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>3. 9,500</td>
<td>2,100</td>
<td>4.5</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>4. 11,500</td>
<td>2,500</td>
<td>4.4</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>40,800</td>
<td>9,000</td>
<td>4.5</td>
<td></td>
</tr>
<tr>
<td>98'</td>
<td>Irregular</td>
<td>1. 13,000</td>
<td>2,800</td>
<td>4.71</td>
<td>Excellent blast.</td>
</tr>
<tr>
<td>115'</td>
<td></td>
<td>2. 11,500</td>
<td>2,400</td>
<td>4.79</td>
<td></td>
</tr>
<tr>
<td>100'</td>
<td></td>
<td>3. 10,500</td>
<td>2,500</td>
<td>4.2</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>35,000</td>
<td>7,700</td>
<td>4.75</td>
<td></td>
</tr>
</tbody>
</table>
2) "Half" Face or Two Tier Heading Blasts

Where the height and condition of the quarry face is such that it is inadvisable to attempt a heading blast with all the chambers at floor level, the tunnels can be driven and the charges placed up the face. In many British quarries continuous working over the years has resulted in high faces which are difficult to work and can only be kept safe by regular dressing. Skilled labour for this class of work is scarce and expensive and a few years ago consideration was given to the possibility of using heading blasts to blow down the top rock from such faces. Information on such blasts was rare but it was decided to experiment and several blasts have now been successfully fired in various types of rock. Details are given in Table 2.

The "half" face or two tier heading blast is designed in the same way as the standard blast but the tunnels are driven into the quarry face at a point above floor level. It is usually possible to make access to the tunnelling point from the top of a previous rock pile or to make a path up the face, though in one instance (example (4) Table 2) in an Aberdeenshire granite quarry, where the face was over 500 ft. high, holes were drilled into the face and a stairway fixed from the top of the quarry down to the tunnel. The materials were lowered by means of a small crane from the top of the quarry and this was also used to lower the explosive for charging and rock for stemming.

The actual costs of driving such blasts are not appreciably greater than those at floor level but loading the explosive and the stemming costs more.

In Figure 4 is shown the layout for a "half" face blast. It will be seen that the tunnel is driven 20 ft. above the half way line, and this is to allow for the breaking out angle. The joint planes
QUARRY TOP

SECTION

ELEVATION.

QUARRY FLOOR LEVEL.

PLAN

FIG. 4

SCALE: 1" = 50'
in most of these blasts have been almost vertical and experience has shown that the rock breaks out downwards at an angle of approximately $45^\circ$. This must be taken into account in placing the top blast as, if it is too far down the face, the down break might be such as to spoil the face for the next blast below.

When a top blast is fired as shown, the bulk of the stone is blown down to quarry floor level and only a small portion remains on the sloping face. It is necessary to make this safe before workmen or machines approach the bottom of the face, but so far it has been found quite easy to do this. Once the stone has been cleared and the face made safe a standard heading blast can be driven and fired at floor level. If desired two top blasts, one behind the other, can be fired before the bottom one but the amount of stone blown down to quarry floor level with the second top blast will be less than with the first and usually it is better to alternate top and bottom blasts.

\textbf{TABLE 2/}
<table>
<thead>
<tr>
<th>Height</th>
<th>Stone</th>
<th>Jointing</th>
<th>Estimated Tonnage</th>
<th>Explosive Charge lb.</th>
<th>Ratio tons/lb.</th>
<th>REMARKS</th>
</tr>
</thead>
<tbody>
<tr>
<td>65'</td>
<td>Blue Whin</td>
<td>contorted bedding</td>
<td>1. 6,000</td>
<td>1,200</td>
<td>5</td>
<td>Fragmentation very good</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>2. 6,500</td>
<td>1,300</td>
<td>5</td>
<td>very good</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>3. 6,500</td>
<td>1,500</td>
<td>5</td>
<td>indeed, debris 125' from quarry toe.</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>80'</td>
<td>Limestone</td>
<td>Rectangular</td>
<td>1. 8,650</td>
<td>1,850</td>
<td>4.65</td>
<td>Very successful 45' from crest to crest</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>2. 6,100</td>
<td>1,500</td>
<td>4.05</td>
<td>125' from quarry toe.</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>3. 6,300</td>
<td>1,400</td>
<td>4.5</td>
<td>Top of rock pile lay with 150' of face.</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>4. 6,300</td>
<td>1,400</td>
<td>4.6</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>5. 7,350</td>
<td>1,600</td>
<td>4.6</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>6. 7,600</td>
<td>1,600</td>
<td>4.75</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>7. 6,900</td>
<td>1,650</td>
<td>4.2</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>4.45</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>49,200</td>
<td>11,000</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>117'</td>
<td>Granite</td>
<td>Massive</td>
<td>1. 11,000</td>
<td>2,100</td>
<td>4.25</td>
<td>Fragmentation excellent.</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>2. 9,000</td>
<td>1,800</td>
<td>5.00</td>
<td>Debris thrown 150'. Yield approx. 100,000</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>3. 10,000</td>
<td>3,100</td>
<td>3.20</td>
<td>tons. Whole toe affected by the blast.</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>4.3</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>30,000</td>
<td>7,000</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>Granite</td>
<td>Vertical</td>
<td>1. 5,180</td>
<td>1,250</td>
<td>4.15</td>
<td>Excellent blast.</td>
</tr>
<tr>
<td></td>
<td>Chambers</td>
<td>jointing</td>
<td>2. 5,130</td>
<td>850</td>
<td>6.04</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>3. 5,360</td>
<td>900</td>
<td>5.95</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>4. 4,100</td>
<td>1,000</td>
<td>4.1</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>4.95</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>19,770</td>
<td>4,000</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>
3) **Double Heading Blasts, Vertical**

Where the size of explosive charge is not restricted by a vibration problem it is possible to fire a top and bottom heading blast simultaneously. Several such blasts have been fired and Figure 5 gives details of one in the writer's experience. This blast was fired with Blackpowder to give monumental stone in a granite quarry and the result was excellent. For special cases such blasts are satisfactory but they are not recommended for routine production work. The main disadvantages are that a considerable time is necessary to drive the tunnels with the consequent "freezing" of that part of the quarry and the resulting rock pile is high and can give trouble in loading out. Generally it is better with such high faces to fire the top blast by itself.

4) **Double Heading Blasts, Horizontal**

It has already been suggested that the maximum depth of the main drive should be limited to 50 feet but occasions arise when it is desired to move a greater burden than this at the one operation and in such circumstances a double line of back drives can be made. This has only been done a few times in Great Britain but several such blasts have been reported in American journals. For normal quarry work this system has little application and it has the disadvantages of taking a long time to prepare and giving a high rock pile. On the site of a dam, however, excavations for the abutments had to be made in a hard almost vertically jointed sandstone and the double heading blast system proved satisfactory in blowing down the rock to the breaking line at the one operation. Circumstances were such that it was necessary to have the burden on the chambers greater than .6 the height and while the charges used gave good breakage of the rock, the
the rock pile was higher than in a normal blast. An example of one of these blasts is shown in Figure 6.

5) **Well Holes and Heading Blasts**

In very high faces where the rock is suitable for well drilling, a combination of well holes and heading blasts can be adopted. The top part of the face can be blown down with well holes before the heading blast or the two can be fired together. In designing such a blast the well holes would be drilled to within about 90 feet of the heading blast chambers and vertically above the line of the back drive as shown in Figure 7. The spacing of the well holes will depend upon their diameter, the burden at the bottom of the holes and the cleavage planes.

With well holes taking 6" diameter cartridges the usual spacing is 16 to 18 ft. and the burden varies from 20 to 30 ft. With holes taking 8" diameter cartridges the figures would be increased to 23 to 25 ft. and 25 to 40 ft. respectively (Ref.4). Where well holes are used in conjunction with heading blasts, however, the burden at the bottom of the holes is likely to exceed these figures and it may be necessary to decrease the spacing to get the required breakage. If the cleavage planes are such that the blast is "on face" then wider spacing is possible than when the blast is "on end". The actual spacing is determined from the charge of explosive required and this is calculated in the normal way for well holes (Ref.4).

6) **Special Heading Blasts**

From the experience gained with standard heading blasts it has been possible to apply the heading system to break down unusual masses of rock in various quarries. Details of a few of those blasts/
FIG. 6
blasts are as follows:

1) **Limestone Quarry**

The rock in this quarry showed a marked unconformity with well defined horizontal beds from the quarry floor level to a height of 16 feet and above this the beds were "on end" and almost vertical. Some years ago a heading blast had been tried with the tunnel at floor level in the horizontal beds with very poor results. The bottom beds were shot out and the floor of the quarry heaved up some yards in front of the face. Some of the top rock was brought down but a bad face was left and the total tonnage brought down was less than estimated. Working of the face was proving difficult and it was decided to try a second heading blast. In this case the tunnel was driven in on top of the horizontal beds about 16 feet above floor level with excellent results. In this case some voids were encountered when driving the tunnels but it was possible to get the explosive chambers in solid rock and the voids were filled up with stemming. The tunnels were driven and the charges calculated in the standard manner with a ratio of 4.5 tons/lb. Following this successful blast a second blast to break into the space left by the first was designed and the plan of this blast is shown in Figure 8. The main drive was again started above the horizontal beds and ran into badly broken rock and a cavity. This was filled up as well as possible and the side drives set off as shown on the plan, the left-hand side followed the bad rock but it was possible to keep the tunnel in the solid and drive forward for the chambers and get these in the solid rock.

On/
FIG. 8
SCALE: 50' = 1"
On the right-hand side a further large cavity was encountered below the level of the tunnel but the tunnel was continued and the rock from the driving tipped into the cavity. Chambers were put in as shown and were in solid rock but to prove they had sufficient good rock all round, 5 feet deep drill holes were put in and proved satisfactory. The charges were calculated at a ratio of 4.55 tons/lb. and an excellent blast resulted. The rock pile was within 200 feet of the original face and the blast broke through cleanly to the free end.

2) **Granite Quarry**

The heading blast system had been in regular use at this quarry using blackpowder to give monumental stone and it was decided to use this system to clear a high face of poor quality rock. The face was from 190 to 220 ft. high and the rock formed a projection as shown in Figure 9. On the left-hand side of this projection there was a bench 90 feet above quarry floor level with a pile of broken rock lying against the face. It was impossible to say if this rock was broken right into the face and it was thought that there would be solid rock underneath.

The face was examined thoroughly and a survey made of the top and of the bottom. It was decided that it was out of the question to tackle the full height of the face by driving in a tunnel at floor level, and a suitable bench 90 feet above floor level was selected as the starting point for the tunnel and an access path made to this bench. It was difficult to get an accurate survey of the mass of rock at this level but the outline was interpolated from the top and bottom surveys and the blast designed as shown in Figure 9. Three chambers were used and these were placed so that they would be approximately in the centre of the/
BENCH
80 FT. HIGH.

LOOSE
AND
SOLID ROCK

QUARRY FLOOR

3 3,100 lb.
(+130')

2 1,800 lb.
(+120')

1 2,100 lb.
(+100')

TUNNEL MOUTH
90 FT. UP FROM
FLOOR.

FIG. 9

SCALE: 1" = 20'

7,000 LBS AMMONAL No. 3
the mass of rock to be broken down.

The charges were calculated on the basis that the rock would be broken out forward in the direction of the tunnel and a total of 7,000 lb. of Ammonal No.3 was used. This was based on an estimated rock production of 30,000 tons but it was hoped that a much greater amount would be produced by the side and back break of the charges. The blast was loaded and fired and the rock broke out normally in the direction of the tunnel but it also loosened the rock all round the blast to a considerable extent. An examination at the top of the face showed a wide crack as indicated by the dotted line on Figure 9, and the mass of rock could be heard "working". It was obvious that this rock was liable to go down and everyone was withdrawn from the vicinity. The rock went on breaking for two or three hours when it eventually dropped leaving a clean break along the dotted line, Figure 9. The total amount of rock eventually brought down was just over 100,000 tons.

3) Granite Quarry

Continued working over many years had resulted in a very high face at this quarry and difficulty was experienced in keeping the face safe for men working on the quarry floor. It was decided that the heading blast system should be used to shoot the top part of the face but the difficulty was to get a point of access for the main tunnel. The rock was well jointed and the total face height was over 200 ft. The mass of rock involved is shown on Figure 10, and it was finally decided that a tunnel should be started from beyond the face where a suitable bench provided an access point. Driving from this bench would give a height above the chambers of 80 ft. plus 10 ft. of overburden. In order to get chambers near the front of the face to be blasted, the/
the main tunnel had to be driven 118 ft. long and a side tunnel 165 ft. long as shown on the plan. This amount of drivage was considered justified as it was hoped that the tunnels could be used for a succession of blasts and that the total amount of rock produced by using these tunnels would justify the cost.

The tunnels were driven in the normal manner and while it was expected that some form of artificial ventilation would be necessary, it was found possible to finish the main drive and about half the side drive without any difficulty. At this stage when ventilation at the face was becoming poor the tunnel encountered a joint plane down which air was passing. This joint plane was exposed to the left-hand side of the tunnel and resulted in a considerable flow of air which gave good ventilation and thus proved satisfactory for the rest of the drivage. A careful survey had been made of the mass of rock and this was checked when the tunnel had been driven its calculated distance by drilling a borehole through to the front face as shown on the plan. The length of this borehole was within a foot of the calculated distance and the side drives were then started. It was expected that a chamber for explosives would be required on each side of the main tunnel but the rock on the left-hand side was found to be badly broken and it was therefore decided to put two chambers on the right-hand side as shown. The charges for these chambers were calculated in the normal manner and 4,200 lb. of Ammonal No.3 were used. The blast was then stemmed with loose rock and this stemming was brought back along the main tunnel for a distance of 50 ft. and the blast fired electrically/
electrically. The result was most satisfactory, the blast breaking out cleanly in the tight corner and breaking backwards for a considerable distance on the free end side. The line of the face after the blast is shown by a dotted line on Figure 10. An examination of the tunnel afterwards showed that it had not suffered by the blast and the stemming was hardly moved, so that there will be no difficulty in using the tunnel for a further blast.

4) Slate Quarry

In this quarry wire saws are used to undercut the slate. These involve the driving of tunnels and the question was raised as to the possibility of using these tunnels for the placing of charges as in heading blasts. It was decided that this idea had possibilities and a small experimental blast was fired from some tunnels previously driven. These were not in an ideal position but the results obtained were sufficiently encouraging to justify a larger blast and a design was prepared as shown in Figure 11.

The main tunnel was driven in 45 ft. and branch tunnels 36 ft. to the left and 18 ft. to the right. These tunnels were used for the saw equipment and the limit of the saw undercut is shown. The last blast to complete the drivage on the right hand side broke through into a cavity and when this was examined it was found to be fairly extensive. Further examination revealed that a considerable number of old workings existed below the bench at which the tunnel had/
FIG. 10.

SCALE: 30' = 1"

LINE OF FACE AFTER BLAST.

POSSIBLE POSITION OF NEXT BLAST.

HEIGHT ABOVE CHAMBERS 90'

DIRECTION OF BOREHOLE

BRANCH TUNNEL 165'

LINE OF CREST

LINE OF FACE AT DRIVE ENTRANCE

FLAT LEDGE.
had been driven and these are shown in Figure 11. It was felt that it would be a pity to stop the blast at this stage and that while the results might not be up to expectations, it was worth while risking charging the chambers. The cavities were too large to fill up with material but the right hand chamber was brought back 5 ft. from the cavity and this 5 ft. packed as solid as possible. It was felt that as the slate had been undercut a chamber in this position would have a reasonable chance of breaking forward and while some energy would most certainly be lost in the cavity, sufficient work would be done on the slate to move it forward. Chambers Nos.1 and 2 were 14 ft. above another cavity but again it was felt that with the assistance of the saw cut the rock should be broken out reasonably well. The charges were calculated and 4,800 lb. of Blackpowder were used. Care was taken in the stemming to make this as tight as possible and the blast fired. The results exceeded expectations and the slate was broken outwards to form an excellent pile for loading. Some rock broke through into the cavities but the loss of energy in this direction had obviously not interfered with the working of the blast.

5) Gravel Quarry

Under certain conditions gravel deposits can be loosened for loading out by means of heading blasts. Usually such deposits have no obvious stratification and only require shaking up to permit of easy loading. Where the height of face is over 30 feet the heading blast system can be used, provided the gravel is strong enough to permit of a tunnel being/
SECTION ALONG C - D

SCALE: 20' = 1

FIG II
being driven into it. Some gravels are too soft and would require tunnels to be timbered to keep them open but in other cases the tunnel will stand unsupported. It is recommended that if tunnelling is considered, a trial tunnel should be put in say 15-20 ft. deep and left for 5-6 weeks to see if it will remain open. If it does so then the remainder of the tunnel can be completed.

The design and layout of heading blasts in gravel are exactly the same as for hard rock but in this case a much lighter charging ratio can be used. The average gravel weighs about 30 hundredweights per cubic yard and an overall charging ratio of 6 tons per pound will give good results.

7) Positioning of Heading Blast Charges

Next to the amount of explosive used the actual positioning of the chamber for the charge is the most important factor in the success of any heading blast. Before any tunnel driving is begun the site of the blast should be accurately surveyed at floor level and on top and sections should be made of the face. Particular attention should be paid to any irregularity on the face such as benches at different levels, as these may indicate lines of weakness which could cause the blast to "blow out".

When the survey has been completed and drawn out, the design of the blast can be determined and pegs, at least three, should be placed in the quarry floor to give the line of the main drive. When the main drive has been driven the required distance, the branches should be driven off on either side approximately at right angles for a distance of 6 to 8 ft. and then a further survey made and the exact line of the back drives set off. Pegs should be/
be put into the roof of the tunnel on the correct line to be used by the tunnel drivers to keep the back drives straight. Once the tunnelling has been completed a check survey should be made to ensure that the drivages are correct and if so the position and size of the chambers for the explosive can be marked out.

The following are the main factors to be considered in the positioning of the chambers.

a) Spacing

b) Breaking out angle — straight face
c) " " " — fast end
d) Blasting to a free end.

a) Spacing of Chambers

As in all simultaneous blasting proper spacing of charges is essential for good results, with shot holes the spacing approximates to the burden but in heading blasting the spacing is generally appreciably less than the burden. The main reason for this is that as the burden on the charge increases, the break out angle decreases. Figure 12(a) shows that with a breaking out angle of 45°, which is normal for drill holes when spacing equals burden, the break out angles of any two holes intersect at a distance of $\frac{B}{2}$ from the free face, and this gives good breakage between holes (Ref. 4). Experience with heading blast charges shows that the breaking out angle is about 35° and to maintain the intersection point at $\frac{H}{2}$ from the free face, Figure 12(b), it is necessary to reduce the spacing. In this case

$$S = \tan 35^\circ = .7002$$

$$\therefore S = .7B$$

The burden on a heading blast charge should be taken as the distance from the front of the chamber to the free face/
(a) \( S = B \)

(b) \( S \)

(C) CHAMBER \( 2\frac{1}{2} \) \( 3\frac{1}{2} \) BACK DRIVE FREE FACE

FIG. 12
face, Figure 12(c). When the main drive is 50 feet deep, then with a 2\(\frac{1}{2}\) feet wide back drive and a 3\(\frac{1}{2}\) feet deep chamber, the burden on the charge would be:

\[ S = (50 - (2\frac{1}{2} + 3\frac{1}{2})\text{ft}) = 44\text{ ft.} \]

\[ S = 30.8\text{ ft.} \]

In practice where the blast is "on face" the chambers are spaced at 30 feet centres and when "on end" at 25 feet centres, such spacings have given satisfactory results and broken out the rock cleanly between chambers. It is not recommended that these distances be exceeded as wide spacing results in bad breakage between chambers with the rock being fractured but not displaced. On several occasions the writer has seen the centre charges spaced at 40 to 50 feet apart and while the blasts looked most successful on firing, heavy costs were incurred to straighten and clean up the face at the end of the blasts, due to standing pillars of broken rock.

Another bad feature of wide spacing is that the explosive charges are greater due to the amount of rock on each chamber and this makes the liability of a "blow out" greater. Better control of the blast is obtained with several small charges closely spaced than with a few large charges widely spaced.

b) Breaking Out Angle - Straight Face

As already stated the usual breaking out angle of a heading blast chamber is about 35°. This angle varies slightly according to the nature of the jointing of the rock but it is obtained where the blast is "on face" and almost so with "on end" blasts.
blasts unless the joints are very well defined. In this latter case one blast was seen which broke out directly in line with the end of the back drives with no side break, and while the bottom stone was blown out normally the top part of the face locked itself on falling and made quarrying out very difficult. This blast was fired in a very tight corner of the quarry and the possibility of a blast locking itself on a straight face is very rare.

c) **Breaking Out Angle - Fast End**

Where heading blasting is employed for routine production in quarries most blasts will have one fast end, for example with the layout shown in Figure 2 two blasts out of three will have a fast end. In order to maintain the length of quarry face it is essential that the corner be blasted out each time and from much trial and error experience the conclusion has been reached that the end chamber should not be greater than 10 feet from the extension of the side line of the quarry, Figure 13.

This distance will give satisfactory breaking out irrespective of the line of the joint planes and will ensure maintenance of the face length.

d) **Blasting to a Free End**

Where the end chamber of a blast is next to a free end the chamber should be so placed that the rock is completely broken out and a straight line of face left. This is not always easy to achieve because the break out line from the previous blast is at an angle of about 35° to the line of face, Figure 14, and if the chamber at X is positioned so that \( XY = B \), then \( B_1 \) will be the line of least resistance and the blast may break out on this line. Various distances for \( XY \) in relation to the burden/
burden $B$ have been tried from $XY = \frac{3}{2}B$ to $XY = B$ but it has been found that for almost all conditions the most satisfactory breakage is obtained when the chamber is placed so that $B = B_1$. Where the angle $XYZ$ is $90^\circ$ the same rule applies unless the joint planes along the line $XY$ are very well defined, in which case the distance $XY$ could be increased up to $1.25B$.

Broken rock from the previous blast should always be cleared away from the free corner to permit of an accurate survey which is the only way to ensure correct positioning of the end chamber.

**CALCULATION OF HEADING BLAST CHARGES**

This is the most important part of successful heading blasting and two methods are used in Great Britain.

a) **Total Tonnage Method.**

b) **Bursting Tonnage Method.**

a) **Total Tonnage Method**

This is the method most commonly used and is the one preferred by the writer. It consists of calculating the total volume of rock on each chamber and multiplying this by the weight of the rock per cu. yd. to give the theoretical tonnage. This is then divided by the loading factor to obtain the charge. For example Figure 1 shows a four chambered heading blast in a face 85 ft. high with a main drive of 50 ft. and back drive of 90 ft.

To calculate the volume on each chamber lines $BC$, $EF$ and $GH$ are drawn half way between each chamber and the bursting angle for the end chambers is taken as $35^\circ$. The areas so set out are then multiplied by the height to give the volume as follows:

<table>
<thead>
<tr>
<th>Chamber</th>
<th>Calculation</th>
</tr>
</thead>
</table>

...
Chamber 1  \[ \text{Vol.} = \frac{AB + DG}{2} 	imes BC \times \text{height} \]
\[ = \frac{15 + 50}{2} \times 50 \times 85 \text{ cu.ft.} \]
\[ = 138,125 \text{ cu.ft.} = \frac{138,125}{27} \text{ cu.yd.} \]
\[ = 5,120 \text{ cu.yd. approx.} \]

@ 2 tons per cu.yd.
Tonnage = 10,240 tons.

Chamber 2  \[ \text{Vol.} = BE \times EF \times \text{height} \]
\[ = 30 \times 50 \times 85 = 127,500 \text{ cu.ft.} = \frac{127,500}{27} \text{ cu.yd.} \]
\[ = 4720 \text{ cu.yd.} \]

@ 2 tons per cu.yd.
Tonnage = 9,440 tons.

As the blast is symmetrical, chamber 3 will have the same tonnage as No.2 and chamber 4 as No.1.

The total tonnage for this blast is therefore
\[ 2 \times 10,240 + 2 \times 9,440 \text{ tons} = 39,360 \text{ tons.} \]

**Charging Ratio**

Having obtained the tonnage per chamber this is divided by the charging ratio to obtain the amount of explosive required. The correct charging ratio to use is always a difficult question, as successful blasts have been fired with ratios varying with high explosive from 3 tons/lb. to 6 tons/lb. These ratios were based on the theoretical tonnage and higher ratios have been obtained on actual tonnage due to overbreak. From 3 to 6 is a great variation and considering that the cost of the explosive constitutes the major part of the cost of a blast the correct assessment of the charging ratio may make the difference between an economic blast and an uneconomic one. With a view to obtaining some definite answer/
answer to this question the writer has tried over the last few years to standardise the blasts as much as possible. Over one hundred blasts have been fired in this time and the tonnage per chamber has been calculated as shown. The ratio of depth to height \((D = 0.6H)\) has been maintained up to faces 85' high and the spacing between chambers kept as near as circumstances would permit in the range 25 to 30 ft. Charging ratios from 4 to 5 tons per lb. high explosive have been used and good results have been obtained in every case. It has been difficult to distinguish any effect on the fragmentation of the stone in this charging range, but a definite difference has been noticed in the extent of the rock pile, this being greatest with the heaviest charges. e.g. At an igneous rock quarry where several blasts have been fired Table 3 shows the difference in the throw of the rock pile with heavy and light charging ratios.

From a study of the results of many blasts it is recommended that for high faces 80 ft. -140 ft., the total charging ratio should be between 4.5 and 4.75 tons per lb. to obtain a flat rock pile and for faces below 80 ft. between 4.75 and 5.0 tons per lb. Faces above 140 ft. high with single tier firing require special consideration.

Referring to the example given in Figure 1, as the face is over 80 ft. high the total blasting ratio would be between 4.5 and 4.75 tons per lb. and the estimated tonnage has been calculated to be 40,000 tons approximately, so the total charge will lie between 8,400 and 8,900 lb. This charge has to be distributed between the four chambers and the end chambers will require a greater charge than the centre ones as the former have a greater rock/
<table>
<thead>
<tr>
<th>Height</th>
<th>Jointing</th>
<th>Estimated Tonnage</th>
<th>Explosive Charge (lb.)</th>
<th>Charging Ratio (tons/lb.)</th>
<th>REMARKS</th>
</tr>
</thead>
<tbody>
<tr>
<td>95'</td>
<td>Irregular</td>
<td>1,10,900</td>
<td>2,400</td>
<td>4.5</td>
<td>Fragmentation good. Extent of rock pile 160 ft. High rock pile. Blast too lightly charged.</td>
</tr>
<tr>
<td></td>
<td></td>
<td>2. 9,900</td>
<td>1,900</td>
<td>5.2</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>3. 8,500</td>
<td>1,900</td>
<td>4.5</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>4. 10,700</td>
<td>1,800</td>
<td>5.9</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>40,000</td>
<td>8,000</td>
<td>5.0</td>
<td></td>
</tr>
</tbody>
</table>

| 1.     | 90       | 1. 10,300         | 2,800                  | 3.7                      | Fragmentation good. Extent of rock pile 230 ft. Flat rock pile. |
|        |          | 2. 10,000         | 2,200                  | 4.55                     |         |
|        |          | 3. 14,300         | 3,000                  | 4.77                     |         |
|        |          | 34,600            | 8,000                  | 4.35                     |         |
rock surface to shear. Referring to Figure 1, chambers 1 and 4 have to shear the rock to the side, at the back drive, along the floor and share the shearing areas between the chambers with Nos. 2 and 3. The centre chambers Nos. 2 and 3 have to shear the rock at the back drive, along the floor and share the shearing between the chambers. The area to be sheared by each of No. 1 and No. 4 chambers is therefore:

\[
AD \times \text{height} + AB \times \text{height} + \frac{1}{2}BC \times \text{height} + \frac{AB + DC}{2} \times 50
\]

\[= (61 \times 15 + \frac{1}{2} \times 50)85 + \frac{15 + 50}{2} \times 50 \]

\[= 10,210 \text{ sq.ft.} \]

Area to be sheared by No. 2 and No. 3 chambers

\[= \frac{1}{2}BC \times \text{height} + BE \times \text{height} + \frac{1}{2}EF \times \text{height} + BE \times EF \]

\[= \left(\frac{1}{2} \times 50 + 30 + \frac{1}{2} \times 50\right)85 + 30 \times 50 \]

\[= 8,300 \text{ sq.ft.} \]

The end chambers have therefore each to shear 1,910 sq.ft. more than the centre chambers and this equals 23%. The end charges should therefore be 23% heavier than the centre charges.

The calculation of the charge for each chamber can now be made as follows:

The tonnages on each chamber have already been calculated, page 61, and the charges are obtained by dividing first the tonnages for the centre chambers by the blasting ratio 4.80. The charges for the end chambers are then determined by adding 23% to those of the centre chambers. These are the theoretical charges and the actual charges are taken to the nearest 50 lb.

The results are then set out:
<table>
<thead>
<tr>
<th>Chamber No.</th>
<th>Estimated Tonnage</th>
<th>Theoretical Charge [lb.]</th>
<th>Blasting Ratio [tons/lb.]</th>
<th>Actual Charge [lb.]</th>
<th>Blasting Ratio [tons/lb.]</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>10,240</td>
<td>2,419</td>
<td>4.25</td>
<td>2,400</td>
<td>4.27</td>
</tr>
<tr>
<td>2</td>
<td>9,440</td>
<td>1,967</td>
<td>4.8</td>
<td>1,950</td>
<td>4.83</td>
</tr>
<tr>
<td>3</td>
<td>9,440</td>
<td>1,967</td>
<td>4.8</td>
<td>1,950</td>
<td>4.83</td>
</tr>
<tr>
<td>4</td>
<td>10,240</td>
<td>2,419</td>
<td>4.25</td>
<td>2,400</td>
<td>4.27</td>
</tr>
<tr>
<td></td>
<td>39,360</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

N.B. The percentage increase for the end over the centre charges is 23% in the example taken but this percentage will vary with the burden and spacing, it must therefore be calculated for each blast.

Where gunpowder is used for heading blasts it is necessary to have heavier charges as compared to High Explosive as the gunpowder contains less explosive energy. The same system of calculating the tonnage per chamber is used and while experience with gunpowder has been more limited, total charging ratios of 3-3.5 tons per lb. of powder have given good results.

It should be noted that undercharging of a heading blast is liable to give projection of the rock pile and may in fact be more dangerous than overcharging. A heading blast charge breaks out the rock by shearing it off at floor, back, and sides and displacing the body of rock. In Figure 15(a) a charge at A would shear the rock along the lines AB and AC and for a successful blast it would require to have sufficient energy to lift the mass of rock ABDEC and displace it. The resultant of the forces along lines AB and AC will be approximately along AD and Figure 15(b) shows the position the rock will have taken up in the first stages of breaking out. The face at D will have bulged outwards and the mass of rock will have lifted, the rock at D continues to move outwards, Figure 15(c), and determines largely the extent of the rock pile. The top rock falls down into the space left by the ejection of the rock from D and the rock pile will be spread out to about \( \frac{1}{3} \) times the height of the face and have a height of about half that of the original face.
If the blast is overcharged or is given the effect of an overcharge by the nature of the jointing the effect is to increase the bursting angle BAD, Figure 15(a), and displace more stone. This has the effect of spreading the rock pile farther than normal and of moving the bulk of the stone away from the face. The final rock pile from such blasts takes the form shown in Figure 16.

Where the blast is undercharged the usual result is that there is not sufficient energy available to lift the mass of rock above the chamber and the energy is concentrated in ejecting the wedge of rock in front of the charge. In such cases the angle BAD, Figure 15(a), is decreased and the rock may travel outwards several times the height of the face. A portion of the top rock will fall due to the effects of gravity but only a fraction of the theoretical tonnage will be obtained and a very bad overhang will remain.

b) Bursting Tonnage Method

This method of calculation is based on the wedge of rock in front of the explosive charges. Figure 17 shows in perspective the wedge ABCDEF which is calculated to determine the bursting tonnage. Various angles might be taken for the horizontal and vertical breaking lines but usually the horizontal angle at both ends of the blast is taken at 35° and the vertical angle at 45°. In Figure 17 angle KEJ = 45° and angle KED = 35°, if both angles were the same the volume enclosed in EDAJK would be one quarter of a pyramid but in making this calculation the volume taken is the area of AJKD multiplied by 1/3 EK. The bursting volume therefore is the volume of the prism EKJ, PHG plus the two volumes EDAJK and FHCBG.
FIG. 16

MUCK PILE

2 - 2 1/2 H
If as before the main drive is 50 feet deep and the back drive 90 feet long then the bursting volume would be calculated as follows:

\[
\text{Vol. of prism } EKJ,FHG = \text{Area } EKJ \times KH
\]

\[
= \frac{1}{3} \times 50 \times 50 \times 90
\]

\[
= 112,500 \text{ cu.ft.}
\]

\[
\text{Vol. of } EDAJK = \text{Area ADKJ} \times \frac{1}{3} \times 50
\]

\[
= 50 \times 50 \tan 35^\circ \times \frac{1}{3} \times 50
\]

\[
= 29,166 \text{ cu.ft.}
\]

\[
\text{Bursting Volume. } = 112,500 \times 2(29,166) \text{ cu.ft.}
\]

\[
= \frac{170,832}{27} = 6,300 \text{ cu.yd.}
\]

@ 2 tons per cu.yd.

\[
\text{Bursting Tonnage } = 12,600 \text{ tons}
\]

The total tonnage for the one blast has been found to be 39,360 tons so the ratio of the 2 tonnages is:

\[
\frac{T_T}{B_T} = \frac{39,360}{12,600} = 3.12
\]

The charge is obtained by dividing the tonnages by the blasting ratios

\[
\text{Charge } C \text{ lb. } = \frac{T_T}{T_R} = \frac{B_T}{B_R}
\]

or \[
\frac{T_T}{B_T} = \frac{T_R}{B_R} = 3.12
\]

Where total ratio \( T_R = 5 \text{ tons/lb} \). Bursting ratio \( B_R = 1.60 \text{ tons/lb} \) and \( T_R = 4 \text{ tons/lb} \) and \( B_R = 1.28 \text{ tons/lb} \).

The bursting ratio therefore lies between 1.28 and 1.60 tons/lb. when the total ratio varies from 4 to 5 tons Elb. These figures only apply when the depth of the main drive is 0.6 the height of the face.
To get the charges for each chamber of a blast the bursting tonnage should be calculated as follows:

**Figure 17 End Chambers**

Bursting Vol. = Vol. of EDAJK + Vol. of triangular prism EKJ,LMN

\[ = AD \times DK \times \frac{1}{3}EK + \frac{1}{2}EK \times KJ \times KM \]

\[ = 35 \times 50 \times \frac{50}{3} + \frac{50}{2} \times 50 \times 15 \]

\[ = 29166 + 18750 \]

\[ = 47916 \text{ cu.ft.} \]

Bursting tonnage (at 2 tons/cu.yd.) = 3550 tons

**Middle Chambers**

The bursting volume in the middle chambers is volume of prism LNMQ.

\[ \text{Vol.} = \text{Area} \times \text{MP} \]

\[ = \frac{1}{2} \times 50 \times 50 \times 30 \]

\[ = 37,500 \text{ cu.ft.} \]

@ 2 tons/cu.yd. = 2,780 tons

When the calculation for all chambers for a blast have been made they should be set out as in Table 4.

<table>
<thead>
<tr>
<th>Chamber No.</th>
<th>Total Tonnage</th>
<th>Bursting Tonnage</th>
<th>Charge Tonnage</th>
<th>Total Ratio tons/lb.</th>
<th>Bursting Ratio tons/lb.</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>10,240</td>
<td>3,550</td>
<td>2,400</td>
<td>4.27</td>
<td>1.48</td>
</tr>
<tr>
<td>2</td>
<td>9,440</td>
<td>2,780</td>
<td>1,950</td>
<td>4.83</td>
<td>1.42</td>
</tr>
<tr>
<td>3</td>
<td>9,440</td>
<td>2,780</td>
<td>1,950</td>
<td>4.83</td>
<td>1.42</td>
</tr>
<tr>
<td>4</td>
<td>10,240</td>
<td>3,550</td>
<td>2,400</td>
<td>4.27</td>
<td>1.48</td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td><strong>39,360</strong></td>
<td><strong>12,660</strong></td>
<td><strong>8,700</strong></td>
<td><strong>4.52</strong></td>
<td><strong>1.45</strong></td>
</tr>
</tbody>
</table>

From Table 4 it will be seen that for the example chosen when the calculations are made as indicated, the results are the same for both the total tonnage and bursting tonnage method. This will apply for all blasts where the depth of the main drive is made .6 of the height of the face over the chambers. In
faces over 85 feet high unless conditions are favourable, the main drive should not exceed 50 ft. so that the bursting tonnage will remain constant at the figure calculated for 50 feet. The total tonnage, however, will increase with the height of face and experience has shown that the total ratio of 4 to 5 tons/lb. remains constant even for higher faces. In such cases therefore the bursting ratio will decrease because the bursting tonnage remains constant and the charges increase. Table 5 shows details of blasts which were very similar in plan but the heights were different and consequently different charges were used. It will be seen how the total ratio is approximately the same in each case while the bursting ratios are quite different. Because of this and the fact that the bursting volume calculation is more complicated, the writer prefers to work always with the total tonnage method.

DRIVING THE TUNNELS

To drive the tunnels for heading blasts a team of two men is sufficient. They should be provided with the following equipment:

a) Light-weight jackhammer or airleg combination.
b) 5/8" steel rods, 18" and 30" lengths.
c) Gelignite 13/4", diameter cartridges.
d) Capped Fuses 36" long.
e) Small wheelbarrow with pneumatic tyre or small trolley on rails.

Two suggested types of hole placement for a 2'6" x 4'6" tunnel are shown in Figure 18. These are designed to give a pull per round of 2 feet as experience has shown this to be the most economical for these small tunnels though longer rounds are sometimes obtained when jointing and rock is favourable. The holes/
CHARGES,
Polar Ammon. Gel. 7" Dia.
Cut shots 8 oz.
Others 6 oz.

Top shots 6 oz.
Top cut shots 8 oz.
Bottom cut shots 6 oz.

FIG. 18.
### Table 5

<table>
<thead>
<tr>
<th>Type of Rock</th>
<th>Height of Face at Chamber</th>
<th>Burden to Back of Drive</th>
<th>Estimated Tonnage</th>
<th>Charge</th>
<th>Charging Ratio</th>
<th>Bursting Tonnage</th>
<th>Bursting Ratio</th>
</tr>
</thead>
<tbody>
<tr>
<td>Limestone</td>
<td>1. 80</td>
<td>50</td>
<td>8,600</td>
<td>1,800</td>
<td>4.77</td>
<td>3,320</td>
<td>1.84</td>
</tr>
<tr>
<td></td>
<td>2. 80</td>
<td>50</td>
<td>7,500</td>
<td>1,700</td>
<td>4.40</td>
<td>2,320</td>
<td>1.35</td>
</tr>
<tr>
<td></td>
<td>3. 80</td>
<td>50</td>
<td>6,750</td>
<td>1,700</td>
<td>4.0</td>
<td>2,080</td>
<td>1.22</td>
</tr>
<tr>
<td></td>
<td>4. 80</td>
<td>50</td>
<td>8,250</td>
<td>1,800</td>
<td>4.58</td>
<td>3,080</td>
<td>1.71</td>
</tr>
<tr>
<td>Limestone</td>
<td>1. 92</td>
<td>50</td>
<td>11,100</td>
<td>2,600</td>
<td>4.27</td>
<td>3,540</td>
<td>1.31</td>
</tr>
<tr>
<td></td>
<td>2. 96</td>
<td>49</td>
<td>10,500</td>
<td>2,000</td>
<td>5.25</td>
<td>2,660</td>
<td>1.33</td>
</tr>
<tr>
<td></td>
<td>3. 90</td>
<td>49</td>
<td>10,500</td>
<td>2,200</td>
<td>4.76</td>
<td>3,370</td>
<td>1.53</td>
</tr>
<tr>
<td>Limestone</td>
<td>1. 113</td>
<td>51</td>
<td>13,900</td>
<td>3,000</td>
<td>4.63</td>
<td>3,750</td>
<td>1.25</td>
</tr>
<tr>
<td>(Granite)</td>
<td>2. 113</td>
<td>52</td>
<td>12,950</td>
<td>2,200</td>
<td>5.88</td>
<td>3,000</td>
<td>1.36</td>
</tr>
<tr>
<td></td>
<td>3. 113</td>
<td>51</td>
<td>11,650</td>
<td>2,300</td>
<td>5.05</td>
<td>2,650</td>
<td>1.16</td>
</tr>
<tr>
<td></td>
<td>4. 113</td>
<td>50</td>
<td>12,450</td>
<td>3,000</td>
<td>4.15</td>
<td>3,300</td>
<td>1.10</td>
</tr>
<tr>
<td>Limestone</td>
<td>1. 115</td>
<td>51</td>
<td>16,000</td>
<td>3,200</td>
<td>5.0</td>
<td>4,120</td>
<td>1.29</td>
</tr>
<tr>
<td></td>
<td>2. 115</td>
<td>48</td>
<td>12,000</td>
<td>2,300</td>
<td>5.22</td>
<td>2,520</td>
<td>1.10</td>
</tr>
<tr>
<td></td>
<td>3. 117</td>
<td>52</td>
<td>13,000</td>
<td>2,800</td>
<td>4.65</td>
<td>3,460</td>
<td>1.24</td>
</tr>
</tbody>
</table>

```
### TABLE 5 (Contd)

<table>
<thead>
<tr>
<th>Type of Rock</th>
<th>Height of face to back of drive</th>
<th>Burden at chamber</th>
<th>Estimated Tonnage</th>
<th>Charge</th>
<th>Charging Ratio</th>
<th>Bursting Tonnage</th>
<th>Bursting Ratio</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Igneous 140'</td>
<td>1. 140</td>
<td>50</td>
<td>19,800</td>
<td>4,100</td>
<td>4.83</td>
<td>4,100</td>
<td>1.00</td>
</tr>
<tr>
<td></td>
<td>2. 140</td>
<td>52</td>
<td>17,500</td>
<td>4,000</td>
<td>4.38</td>
<td>3,250</td>
<td>0.81</td>
</tr>
<tr>
<td></td>
<td>3. 140</td>
<td>48</td>
<td>14,200</td>
<td>2,900</td>
<td>4.90</td>
<td>2,925</td>
<td>1.01</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>51,500</td>
<td>11,000</td>
<td>4.68</td>
<td>10,275</td>
<td>0.93</td>
</tr>
<tr>
<td>Igneous 146'</td>
<td>1. 146</td>
<td>55</td>
<td>20,400</td>
<td>3,500</td>
<td>5.85</td>
<td>4,580</td>
<td>1.3</td>
</tr>
<tr>
<td></td>
<td>2. 146</td>
<td>57</td>
<td>16,000</td>
<td>3,000</td>
<td>5.3</td>
<td>3,300</td>
<td>1.1</td>
</tr>
<tr>
<td></td>
<td>3. 146</td>
<td>52</td>
<td>14,600</td>
<td>3,700</td>
<td>3.95</td>
<td>2,600</td>
<td>0.76</td>
</tr>
<tr>
<td></td>
<td>4. 146</td>
<td>49</td>
<td>17,600</td>
<td>3,800</td>
<td>4.65</td>
<td>3,370</td>
<td>0.89</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>68,600</td>
<td>14,000</td>
<td>4.90</td>
<td>14,050</td>
<td>1.0</td>
</tr>
</tbody>
</table>
holes should all be drilled and charged for firing at one operation. The required delay for the explosive charges can be got by safety fuse and plain detonators, the fuses being shortened by about 3" between each set of shots, e.g. No. 0 2'6", No. 1 2'9", No. 2 3'0", No. 3 3'3", No. 4 3'6" fuse. By lighting up in the order the shots will fire, a 3" difference in length will prevent any overlap of the shots. If desired delay electric detonators can be used but these are a bit more expensive. The tunnel should be made to rise slightly to drain off any water.

For lighting in the tunnel miners carbide lamps, electric hand lamps or power lighting can be used. The latter two are generally preferred as they are also suitable for use when loading the explosive charges. Where an electric cable is used shotholes should be drilled near the roof of the tunnel at 6 to 8 feet intervals and wood plugs inserted, to which the cable can be tied. The same plugs will be suitable for tying up the detonating fuse.

The speed of driving depends on the nature of the rock and on the system of payment of the men. If they are on piece work the drivage in normal rock will be between 15 and 20 feet, per week on single shift, explosive will be about 2-2½ lb. per foot run and overall the cost of driving will be from 25/- to 30/- per foot.

It is not usual for the tunnels to require any support or any special means of ventilation. The use of the air hose after blasting is sufficient to clear out the fumes and keep the air in the tunnel fresh.

**LOADING HEADING BLASTS**

When a heading blast is ready for loading the explosive should/
should be ordered so that it can be delivered for transfer direct from the railway or motor van into the quarry. The explosive cases should be counted and stacked on the quarry floor in the numbers required for each chamber, to ensure that the correct weight of charge is used. The explosive should be in 25 lb. tins for High Explosive and 25 lb. bags for Blackpowder. Where the latter is used the chambers should have some pieces of wood on the bottom and the whole chamber should be lined with waterproof felting unless the chamber is absolutely dry. With High Explosives no waterproofing precautions are necessary.

To put the explosive into the chambers as many men as are available should be collected and the 25 lb. cases or bags passed from hand to hand along the tunnels and into the chambers. A barrow or truck can be used to carry cases from the stack to the tunnel mouth and perhaps into the junction of the main and side drives but the human chain is most efficient.

When the explosive is all in the chambers the charges are ready for priming. The charges can be fired either by electric detonators for High Explosives or electric powder fuses for Blackpowder or by detonating fuse. The latter is generally preferred for its simplicity, strength and because it is free from the static electricity or lightning hazard of electric detonators.

For heading blasts double taped detonating fuse is now available and is recommended though single fuse has been successfully used for many blasts. The detonating fuse should be run in from the mouth of the tunnel along the main drive then along the back drives and out again to the mouth of the tunnel as shown in Figure 19 slack fuse being left at each chamber. A priming charge of at least/
least 2 lb. of Gelignite should be tied to the fuse at each chamber
and this charge with the fuse should then be placed between two
tins or bags of the main charge. As an extra precaution an
additional piece of detonating fuse with a 2 lb. primer at either
end can be run between the two chambers on each side of the main
drive.

When the primers have been placed the detonating fuse should
be fastened to the side of the back drives and to each side of
the main drive. This can be done either by tying to the pegs
previously put in the drilled plug holes or by using building lathes
which can be sprung across the roof of the tunnel. The tunnel
is now ready for back filling or stemming.

With Blackpowder it is advisable to stem the full length
of the back and side drives but with High Explosive recent
experience indicates that this is not necessary. The practice
now adopted by the writer is to stem opposite the charges and
have continuous stemming only from the charges nearest the main
drive and in the main drive itself, Figure 19. The stemming
can be strengthened at the junctions by placing old railway
sleepers across or some quick setting cement can be poured in
at this point. Steel rails are not recommended as these might
cause damage to the crusher.

The stemming material should consist of pieces of stone
which can be easily passed in by hand. The quickest method
is to build a dry stone wall about half the height of the tunnel
and fill up behind, the best distance between the walls being
2 to 3 feet. The wall should then be raised again and filled
up and so on. When the stemming approaches the detonating fuse
this/
this must be protected to prevent breakage and the best means of doing this is to surround the fuse with turves. The two fuses should be kept separate by at least 12" and the set up is shown in Figure 20. When the fuses have been protected the stemming should be built up to the roof.

With the detonating fuse properly placed and protected and stemming carried out carefully there should be no risk of any chamber misfiring but a further safeguard can be obtained by crossing over extra fuse leads at the junction as shown in Figure 20. This protects against a break in either of the leads in the main tunnel.

To prepare the blast for firing the Cordtex leads at the mouth of the tunnel should be joined together and two detonators securely attached by insulating tape or wire. The detonators can be initiated by safety fuse or electrically. In the latter case the electric detonators should be coupled in parallel. The firing circuit should be carefully prepared and checked before the detonators are attached to the Cordtex and the source of electric power must be sufficient to pass the required current through to the detonators.

Large blasts usually draw spectators and every precaution must be taken to see that they are in a safe position. They must always be kept at the side and behind the line of the face to be blasted and on no account be allowed to remain in front of the blast even if some hundreds of yards distant. Where roads pass in front of a blast all traffic must be stopped. As stated earlier, heading blasts land the stone in from one and a half to twice the height of the face but on the odd occasion stones may fly and precautions must therefore always be taken.

COSTS/
FIG. 20.
COSTS OF HEADING BLASTING

The major item in the cost of producing rock by the heading blast method is the cost of explosives. For calculation purposes charging ratios of from 4.5 to 5 tons of rock per pound of explosives have been recommended though in most blasts the actual tonnage will exceed the estimated tonnage.

With explosive costing 146/6 per 100 lb. the cost of the main charge of explosive would therefore vary from 3.52 to 3.91 pence per ton.

The cost of driving the tunnels is usually 25/- per foot for the main drive and 30/- per foot for the back drives and chambers. Referring to the blast described in Figure 1 the estimated tonnage is 40,000 tons approximately and the costs of the drives would be:

- Main drive 50 ft. at 25/- per ft. = £62.10/-
- Side drives 90 ft. at 30/- = £135
- Chambers, 4 at 30/- per ft. = £18

Total = £215.10/-

= 1.29d./ton.

In charging and stemming the tunnels the time taken will depend upon the length of the drivages and the number of men employed. Usually about 6 feet of stemming per hour can be placed and with say 10 men 100 ft. of tunnel could be done in about 17 hours, so the cost would be approximately £25 which is equal to .15d./ton.

For initiation 300 ft. of double taped Cordtex at 36/- per 100 ft. would cost £5.8/- or .03d./ton.

Taking the average cost for explosive, the cost of the primary blasting would therefore be 5.18d./ton.

Detailed costs for a blast in a limestone quarry are given in Appendix 2.

It should be noted that a four chamber blast is slightly cheaper.
cheaper than a two chamber as the cost of the main drive is common. In the example referred to a two chamber blast would produce approximately half the quantity of the four chamber and the cost of £62.10/- for the main drive would therefore be .75d./ton as compared with .375d./ton for the four chamber blast.
CONCLUSIONS

Over the last few years observations have been made and experiments carried out with a view to determining the best methods of placing and assessing the explosive charges required to break down rock.

There are two methods of placing explosive charges:
1. By means of shotholes drilled into the rock.
2. By means of small tunnels driven into the rock.

1. By means of Shotholes drilled into the Rock

In the case of shotholes the depth is independent of the explosive and depends on the type of drilling equipment available and on the height of the rock face to be blasted. Shotholes vary in diameter from 1" to 9" and the diameter of the explosive charge will be from $\frac{1}{2}$" to 1" less than the diameter of the shothole. Both theory and practice show that the burden of rock which can be blasted is directly proportional to the diameter of the explosive cartridge used, Formula 5, p. 7, and if tests are made with explosive of any diameter in a particular rock the burden for any other diameter can be calculated from $\frac{d}{d_1} = \frac{B}{B_1}$.

Once the depth and burden have been fixed the spacing between shotholes can be determined. This has been found to vary somewhat according to whether the rock to be broken was "on face" or "on end" but on the average, spacing has been found to be equal to the burden on the shotholes:

$$S = B$$

One exception to this rule is the case of small diameter deep holes where the burdens are high in relation to the diameter of explosive used. To give satisfactory breakage these vertical/
vertical holes are supplemented by horizontal holes and the spacing between the vertical holes is about half the burden. The spacing is actually approximately equal to the burden which would be blasted if the vertical holes were unassisted by horizontal holes.

To assess the charge of explosive for shotholes three formulae have been examined and compared with practice. All the formulae include a series of constants which vary according to the conditions of the rock to be blasted, and the varying values given for the constants give a wide variation in charge. Two of the formulae examined do not give charges which are related to practice but the third formula, that used in Military Engineering, approaches practice for holes up to 20 feet deep. It is unsuited for deeper holes, particularly those of 6" and 9" diameter but such holes are not used in normal military work so the formula can be said to suit the purpose for which it was designed.

In the wider field of rock breaking the writer has found that the best method of assessing the charge required in a shothole is to calculate the volume of rock which is to be broken by that shothole. The shotholes must, of course, be of the correct diameter and properly placed in relation to the work that has to be done. The volume of rock is then multiplied by the weight of the rock per cubic yard to get the tonnage and this figure is divided by the loading ratio to give the charge required.

From the results of many blasts the best charging ratio for most rocks appears to lie between 4 and 5 tons of rock per pound of explosive. The writer recommends a figure near/
near the average of these two, viz. 4.45 tons/lb. and to simplify calculation the charge can be determined from the simple formula:

\[ C = \frac{B \times S \times D}{60} \]

where  
- \( C \) = total charge in pounds  
- \( B \) = burden in feet  
- \( S \) = spacing in feet  
- \( D \) = depth in feet.

In the blasts examined the explosives used have been mainly of the gelatinous type with a density of 1.4 to 1.5 grms/cc, a weight strength of 80% to 90% taking Blasting Gelatine as 100% and a velocity of detonation of 2500 metres per second. These are the explosives normally used in Great Britain in quarries and construction work. Weaker explosives are not recommended for hard rock breaking but if softer rocks have to be blasted, weaker explosives could be used on the basis set out in this paper.

2. By means of Small Tunnels driven into the Rock

Where the rock is to be broken by charges placed in chambers from tunnels driven into the rock the positioning and amount of the charges has been found to follow a similar pattern to charges placed in shotholes. Observations have been made of about one hundred Heading Blasts, as such chamber blasts are called, and while this method of blasting is not suitable for every quarry it has a very wide application.

For the best results from heading blasting the rock face should be:

a) over 50 ft. high  
b) over 150 ft. long  
c) joints nearly vertical.

There must be sufficient landing space for the rock blasted down and the minimum space is twice the height of the face.
81.

Where large charges are being used either in shothole or heading blasting the ground vibrations set up by the explosion of the charges will be noticeable in property close to the site of the blast. To avoid trouble with the property owners it is essential that the size of the blasts should be kept within safe limits and a table of maximum charges for different distances is given in Appendix 2.

The standard type of Heading Blast consists of tunnels 2½ ft. wide by 4½ ft. high driven into the rock face at quarry floor level.

Such blasts are satisfactory where the rock face is from 50 ft. to 100 ft. high or if the joints are well defined and almost vertical 140 ft. high. The depth of the main drive should be .6 x the height of the face with a maximum depth of 50 ft.

The back drive or tunnel should be parallel to the rock face and of sufficient length to permit of at least two chambers, though four chambers are more economical.

Various spacings for chambers have been tried generally in the range 25 to 35 feet centres. Blasts have been fired with chambers up to 50 feet centres but such wide spacing is not recommended. Theoretically the spacing between the chambers is .7 x the burden on the chamber, S = .7B.

\[ S = \text{distance between centre line of chambers} \]
\[ B = \text{distance from front of chamber to nearest free face.} \]

In practice this gives a spacing of 25 to 30 feet and good results have been obtained with chambers spaced in this range.

Details are given of various adaptations of the standard heading/
heading blast which have been tried out with successful results and the positioning of the charges is similar to that of the standard type blasts.

With a properly designed blast the calculation of the explosive charge per chamber is the most important factor for a good result. Two methods of calculating the charges have been used in Great Britain:

a) The total tonnage method.
b) The bursting tonnage method.

a) The total tonnage method is the one most commonly used and is recommended by the writer. It consists, as in shothole blasting, of calculating the total volume of rock to be blasted by each charge and multiplying this volume by the weight of the rock per cu. yd. to give the theoretical tonnage. Wide variations are possible in making this calculation as rock faces are rarely vertical and usually irregular in outline. The amount of rock in the side break is also considerable and the theoretical tonnage will therefore depend upon the values taken for these varying distances. In order to obtain definite information and comparative results the writer has adopted the following standards for assessing the theoretical volume:

\[
\begin{align*}
\text{Height} &= \text{Height of face at chamber} \\
\text{Depth} &= \text{Distance from free face to back of drive at chamber.} \\
\text{Sidebreak} &= 35^\circ \text{ each side.}
\end{align*}
\]

When the tonnage on each chamber has been determined the charge is obtained by dividing this by the charging ratio which is the weight of rock per pound of explosive. With the foregoing method of calculation for the tonnage it has been found that for high faces 80-140 ft. the total charging ratio using/
using the high explosive Ammonal should be 4.5 to 4.75 tons/lb. and for faces 50-80 ft., 4.75 to 5 tons/lb. Where Blackpowder is used the ratio should be 3-3.5 tons/lb.

The end chambers have more work to do than the centre ones and it is recommended that the shearing area for each chamber be calculated.

In practice it has been found that the end chambers require from 15% to 25% more charge than the centre chambers so that the charging ratio is lower for the end chambers.

b) The bursting tonnage method is usually taken as a check on the results obtained with the total tonnage calculation and as far as can be found out, it is not used by itself to determine charges. An explanation of the method and the results from some blasts are given but because of the variations and the complicated calculation the writer does not feel that the bursting tonnage method is worth using.

The methods of driving the tunnels, charging, stemming and firing them have been examined and recommended actions made as to the best methods to be employed.

A representative cost has been given in detail and this is about average for a large number of blasts. The costs obtained with these blasts have been satisfactory to the owners of the quarries so much so that about twenty quarry companies are now employing the system under the supervision of the writer as a routine method of getting stone.

In addition the heading blast system is in use for the excavation of abutments for a large dam in India.
In this paper an attempt has been made to bring theory and practice together in order to determine the most efficient methods of placing and calculating explosive charges for breaking down rock and the writer is confident as a result of many blasts that the rules laid down in this paper for both shotholes and tunnels will give satisfactory results.

PHOTOGRAPHS

Photographs of representative heading blasts are given in Appendix 4.
APPENDIX 1

Gelatinous Explosives - Density 1.5 grams/cc.

<table>
<thead>
<tr>
<th>Cartridge Diameter in.</th>
<th>lb. explosive/ft. run</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>.5</td>
</tr>
<tr>
<td>1 1/8</td>
<td>.6</td>
</tr>
<tr>
<td>1 1/4</td>
<td>.8</td>
</tr>
<tr>
<td>1 1/2</td>
<td>1.2</td>
</tr>
<tr>
<td>1 3/4</td>
<td>1.5</td>
</tr>
<tr>
<td>2</td>
<td>2.1</td>
</tr>
<tr>
<td>2 1/4</td>
<td>2.6</td>
</tr>
<tr>
<td>2 1/2</td>
<td>3.3</td>
</tr>
<tr>
<td>2 3/4</td>
<td>4.0</td>
</tr>
<tr>
<td>3</td>
<td>4.6</td>
</tr>
<tr>
<td>3 1/2</td>
<td>6.2</td>
</tr>
<tr>
<td>4</td>
<td>8.0</td>
</tr>
<tr>
<td>4 1/2</td>
<td>10.1</td>
</tr>
<tr>
<td>5</td>
<td>12.5</td>
</tr>
<tr>
<td>6</td>
<td>15.1</td>
</tr>
<tr>
<td>7</td>
<td>18.0</td>
</tr>
<tr>
<td>8</td>
<td>24.5</td>
</tr>
</tbody>
</table>
## APPENDIX 2

### Maximum Explosive Charge with Normal Conditions K=100 for a Ground Movement of $A = 3 \times 10^{-3}$ in. and $A = 8 \times 10^{-3}$ in.

<table>
<thead>
<tr>
<th>Distance Ft.</th>
<th>$A = 3 \times 10^{-3}$ in.</th>
<th>$A = 8 \times 10^{-3}$ in.</th>
</tr>
</thead>
<tbody>
<tr>
<td>600</td>
<td>320</td>
<td>2300</td>
</tr>
<tr>
<td>700</td>
<td>440</td>
<td>3140</td>
</tr>
<tr>
<td>800</td>
<td>580</td>
<td>4100</td>
</tr>
<tr>
<td>900</td>
<td>730</td>
<td>5200</td>
</tr>
<tr>
<td>1000</td>
<td>900</td>
<td>6400</td>
</tr>
<tr>
<td>1500</td>
<td>2000</td>
<td>14400</td>
</tr>
<tr>
<td>2000</td>
<td>3600</td>
<td>25600</td>
</tr>
<tr>
<td>2500</td>
<td>5600</td>
<td>40000</td>
</tr>
<tr>
<td>3000</td>
<td>8100</td>
<td>57600</td>
</tr>
<tr>
<td>4000</td>
<td>14400</td>
<td>102400</td>
</tr>
</tbody>
</table>

**N.B.** In the case of well hole blasts where the holes are fired singly with a short delay interval between each two thirds of the above charges can be fired on each delay period.
Date of Blast: 20th July, 1951

1) COST OF DRIVING TUNNEL

<table>
<thead>
<tr>
<th>Description</th>
<th>£</th>
<th>s.</th>
<th>d.</th>
</tr>
</thead>
<tbody>
<tr>
<td>Wages, cost of living</td>
<td>116</td>
<td>5</td>
<td>-</td>
</tr>
<tr>
<td>Explosives, fuse and detonators</td>
<td>28</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Drills, Electricity, lamps, etc.</td>
<td>10</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

Total footage: 115 ft.

Cost of tunnel per foot: £1.6.10.

2) CHARGING COSTS

<table>
<thead>
<tr>
<th>Description</th>
<th>£</th>
<th>s.</th>
<th>d.</th>
</tr>
</thead>
<tbody>
<tr>
<td>Wages</td>
<td>40</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Cement, sleepers, etc.</td>
<td>40</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

3) FINAL EXPLOSIVES COSTS

<table>
<thead>
<tr>
<th>Description</th>
<th>Quantity</th>
<th>£</th>
<th>s.</th>
<th>d.</th>
</tr>
</thead>
<tbody>
<tr>
<td>Ammonal</td>
<td>4,900 lb.</td>
<td>301</td>
<td>4</td>
<td>-</td>
</tr>
<tr>
<td>Gelignite</td>
<td>25 lb.</td>
<td>1</td>
<td>9</td>
<td>-</td>
</tr>
<tr>
<td>Cordtex</td>
<td>273 ft.</td>
<td>4</td>
<td>18</td>
<td>3</td>
</tr>
<tr>
<td>Connecting wire</td>
<td>200 yd.</td>
<td>-</td>
<td>13</td>
<td>-</td>
</tr>
</tbody>
</table>

TOTAL COMBINED COST OF TUNNEL £502 9 3

Estimated Yield 21,000 tons

Estimated Cost per ton 5.74 pence
1. Heading Blast in Process – Reference p.65, Fig.15(c)

2. Photographs of Heading Blast Sequences.
   (a) Before
(b) First stage showing initial breaking.
(c) Second stage showing rock on the move.
(d) Rock pile after blasting.
3. Photographs of Heading Blast Sequences.

(a) Before
(b) First stage showing initial breaking.
(c) Second stage showing rock on the move.
(d) Rock pile after blasting.
96.

REFERENCES


