BLASTING IN A LAYERED ROCK

by

Andres Centonzio Segovia
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Signed: [Signature]
Andres Centonzi Segovia

Golden, Colorado
Date: May 4, 1971

Approved: [Signature]
W.A. Hustrulid
Thesis Advisor

Golden, Colorado
Date: May 4, 1971

Head of Department
A.M. Keenan
Para Bélgica mi esposa
quien soportó estos duros
años y me ayudó en este
regreso forzado a una
etapa que requería
una mente más ágil y
un corazón menos viejo...
To the Copiapó School of Mines

who gave me the magnificent opportunity to learn and to teach and it was the first step to realize this 20 years old dream to be a Colorado Miner.
ABSTRACT

This thesis presents the results of an investigation performed at a limestone quarry with the following purposes:

a) to improve the technique used to blast layered limestone

b) to develop a fragmentation index considering blasting as a part of a total system (drilling, blasting, loading, hauling, and crushing)

c) to determine the ground motion constants in order to predict vibration levels from blasting

d) to predict air blast from blasting and to compare noise levels from explosive charges and common noise sources

Different drilling patterns, delay times and shooting directions were tried. The best result was with the following:

1. Hole spacing equal to twice the burden
2. Stemming height equal to one half the bench height
3. Delay time equal to 9 milliseconds
4. Arrangement of the rows parallel to the structural planes of weakness and perpendicular to the dip of the formation
5. Direction of shooting perpendicular to the dip of formation
6. A staggered drilling pattern of 8 ft. by 16 ft. This pattern resulted in a Powder factor equal to 6 to 8 ton./lb

7. The fragmentation index was based upon the time required to load the broken rock and also in the power required by primary crushing. The variations observed in loading times were found to be small. The differences in energy consumed for the primary crushing were very small.

8. Ground motion constants were determined with the use of seismological records. The constant of particle velocity from the ground motion and the respective equation were used to construct tables for the prediction of vibration from blasting based on the weight of explosive per delay and the distance from the shot.

9. Air blast pressures were determined with the aid of published nomograms. Equations were derived from the nomograms and tables were calculated to forecast the air-blast pressure. The best relationship between amount of explosive and distances in order to produce the least annoyance, if any, to the neighborhood was determined.
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1. INTRODUCTION

In mining and rock mechanics meetings in which papers on rock blasting are included and discussed one or more of the following statements is often heard:

1.- "More research is needed in the laboratory, field, and mine if the potential for increased efficiency in the use of explosive energy is to be realized."

2.- "A study of the explosive fragmentation processes should include consideration of the related problems of drilling and handling the broken material after it is blasted."

3.- "If explosive fragmentation techniques are going to help solve problems such as efficiently mining large deposits of low grade materials or dramatically increasing rates of advance in tunneling, they must be considered as part of a total system."

4.- "Future research should be conducted with realization that new demands are being made on fragmentation methods."

5.- "An unbiased criteria to define good fragmentation is necessary in order to properly evaluate and improve drilling and blasting techniques"

The above statements point out that although considerable efforts have been expended on understanding each part of the total process of extracting and processing to a useable form rock materials from the earth's surface, (i.e. drilling, blasting transporting and crushing) rather little is presently known
concerning optimization of the system as a whole. One reason for this is undoubtedly the high cost involved in performing experiments of a scale large enough so that the results could be considered meaningful.

One such experiment however was performed in 1966, at Quebec(1) in order to evaluate the effect of expenditures for explosives on the total cost.

The ore was a coarse grained quartz specular hematite having average grade of approximately 30 per cent Fe₂O₃.

Production requirements at that time were in excess of 8 million tpy of iron concentrates grading 66 percent Fe.

Stripping volumes were 8.5 million yd of waste rock including some overburden stripping, making a total mine production of ore and waste in excess of 35 million tpy or over 100,000 tpd.

The experiment included the mining of approximately 41 millions of tons of ore and waste over a period of 14 months.

This was accomplished using:

Drilling.- Electric rotary machines were used to drill 9 7/8-in. and 12 1/4-in. diameter holes.

Blasting.- All of the various types of explosives presently available on the market were tested.

Loading.- Eight-yd electric shovels were used exclusively.

Hauling.- Haulage was done using a mixed fleet of diesel-powered 40-45, and 64-ton trucks.

Crushing.- Primary crushing was done using two 66 in.
by 84 in. jaw crushers and secondary crushing by two 30 in.
by 70 in. gyratory crushers.

Bench.- The bench height was a standard 40 ft.

From their tests they found that a square pattern
(29x29 ft.) with a hole diameter of 12 1/2 in. charged with
a 50/50 TNT slurry-ANFO combination resulted in the minimum
total cost.

The individual dependence of the costs of drilling,
blasting, loading, hauling and crushing as a function of the
degree of fragmentation (as described using shovel loading
rates) are shown in figure 1. The total cost as a function
of fragmentation is shown in figure 2. The results show
that the overall minimum cost does not coincide with the
minimum explosive cost but rather gives proof to the popular
saying among mine management that the place for primary
crushing is in the mine, not in the crushing plant. This is
said because smaller fragments, sized regularly are indeed,
able to decrease the cost of loading (maintenance, repairs,
waiting time for secondary breakage) the cost of hauling (less
waiting time) and the cost of the crushing plant (bridging
delays, more material passes through as undersize).

This thesis present the results of a similar although
smaller scale experiment performed at the Dewey Rocky
Mountain Cement company's limestone quarry at Lyons, Colorado.

The objectives of the study were:
1. To study the presently used methods of drilling, blasting, hauling and crushing and to recommend means of reducing their total overall cost.

2. To study and compare methods of describing rock fragmentation.

3. To determine the ground motion constant for the quarry so that predictions of vibration levels from any blast could be made.

4. To predict the air blast pressure from blasting.

Although only one combination of explosives was used, different drilling patterns, loading, sequence delay times, and shooting directions were tried.

Some idea of the degree of fragmentation may be obtained by using any of the following indicators:

1. Shovel loading speed
2. Quantity of secondary breakage required
3. Bridging delays at the crusher
4. Total cost to obtain a regular predetermined size.
5. Sampling, or
6. Photoplanimetric method (2)

In practice, the most effective of these indicators is probably the shovel loading rate, but on an experimental scale the total cost of drilling, blasting, and crushing may be a better indicator. Each of these indicators will be discussed in later sections.
In this thesis, "optimum blasting" will be defined as the blasting practice which gives the degree of fragmentation necessary to obtain the lowest unit cost of combined operation of drilling, blasting, loading, hauling and crushing.

In actual practice, however the particular combination giving the least cost must sometimes be modified to one resulting in a lesser disturbance to the surrounding environment (noise, vibration, fumes, dust, etc.). This situation will be discussed in a later section.

The fragmentation index most often quoted in the literature is in terms of the volume or weight of rock broken per pound of explosive used (cu. yds/lb., ton/lb.). This however is concerned with only one part of the total process, and a more complete and accurate index can be obtained by considering drilling, blasting and crushing operations as a unit. The index could for example be expressed in cents/ton crushed to ½ in.

This index is unbiased, explicit and accurate by itself and can be compared with others. Therefore, the broken rock in every experiment should be weighed, screened and crushed to a final predetermined size and the total cost to obtain this could be considered a measure of the performance of the explosive. How this performance is affected by the explosive distribution, firing sequence, and firing direction can be determined by comparison.
FIGURE 1  DRILLING, BLASTING, LOADING, HAULING AND CRUSHING VS. SHOVEL LOADING RATE (1)

c/ cu yd.

DRILLING

BLASTING

LOADING

HAULING

CRUSHING

Shovel loading Rate(x1000 cu yd./hr)
FIGURE 2 TOTAL COST vs. SHOVEL LOADING RATE (Fragmentation)

$\$/cu yd.$

SHOVEL LOADING RATE:

$(x1000 \text{ cu yd./hr})$
2. GENERAL DESCRIPTION (3)

Located about 45 miles northwest of Denver, a major market for construction material, is the tenth cement manufacturing facility of Martin-Marietta. The Dewey Rocky Mountain Cement Plant occupies a 1400-acre site and represents a capital investment of about $20 millions. More than 42,000 cu yd of concrete were poured when the Plant was built. Its design is to withstand 125 mph winds and to resist a zone three earthquake.

An outstanding building is the clinker storage, claimed to be one of the largest A-frame structure in existence (Apex 80 ft., 200 ft., wide, 288 ft.long) and able to store 300,000 bbl of clinker.

2.1 Quarrying

Four major raw mix components are recovered from the quarry: low calcium, high calcium, high calcium limestone and weathered cement rock. Silica sand is obtained from a local pit and gypsum from a quarry 25 miles away.

A Niobrara formation of Cretaceous age is the source of four cement rocks used in the Plant's process. Quarrying is planned according to the chemical content of the different materials. Four benches (8- to 35-ft high) are the result of planning.

Physical properties of the rocks determine when they are going to be ripped (single-blade-tooth ripper mounted
on a bulldozer) or blasted (ANFO and/or dynamite in 3-in. diameter holes).

The test were made in the high calcium limestone quarry or pit "B".

In this area of 200x600 ft., the height of the bench varies from 8 to 14 ft. The number of layers forming the bench varies from 15 to 21 with the thickness of the layers (rangeing from 3 to 24 in.) generally increasing with the depth.

Wet clay normally found between layers varies in thickness from 1/2 to 2 1/2 in. The quarried layer dips from 10 to 15 degrees from the horizontal plane, with 10 degrees being predominant. Three principal structural weakness planes were found; the most important having a direction of N 38° W.

A good natural separation plane exists between the limestone and the sandstone stratum immediately below.

All these features are advantageous from the blasting viewpoint making possible shots with few problems other than occasional toes and small but consistent backbreak.

At the beginning of testing, a square drilling pattern of 11x11 ft., using 3-in. diameter holes, resulted in a power factor varying from 2 to 4 ton./pound (the lower figure used for drop cut).

Blasting was carried out in an elongated arrangement starting the shot at the protruding corner of the three-
free face bench as shown in figure (3). A 9 or 17-ms delay was used between any two successive rows. M-second connectors were used as timers for the E-cord trunkline.

Good fragmentation was obtained, and loading, hauling and primary crushing of the limestone was done without any special difficulty.

2.1.1 Drilling

Drilling is carried out using a Gardner Denver model RDC-11 Rotary blast-hole Drill rig. This crawler mounted, self-propelled drilling machine is powered by a GMC 3-53 diesel engine rated at 78 HP, (2200 RPM). Fuel consumption is 3-4 gallon per hour. The rig is designed to drill a 40-ft. hole in a single pass. It has a pull down weight of 1200 pounds, and will drill holes from 2\(\frac{1}{2}\) to 4\(\frac{1}{2}\) inches. It is equipped with a bit grinder and an all weather cab.

The original drill was able to drill angled holes. However when this rig was modified to drill 40 ft. instead of 30 ft. in a single pass, an iron counterweight was added and located in the base of the mast in such a way that the drilling of vertical holes only is now possible. A heavier metal counterweight (made for example out of lead) might be used to avoid this restriction. This should be considered, as there are indications the angled holes can lead to improved results.

The cost of the RDC 11 is approximately $40,000.
FIGURE 3 BLASTING PATTERN

Burden = 11 ft.
Spacing = 11 ft.

Direction of shooting
Direction of the dip

Free face

Initio

Bench walls (free face)
2.2 Milling

After two stages of crushing the rock is roasted (to eliminate an organic hydrocarbon-kerogen- and sulphur) and then cooled.

Stone, silica and iron go to the raw mill. Mill discharge is air-separated; coarse material is recycled and the fine is mixed. The mixed material goes to the rotary kiln. The clinker produced is discharged to coolers. Clinker, gypsum, and limestone are transported to the finish mill and after that air-separated. Fine material from the separators is cooled before being pumped pneumatically to storage silos.

The capacity of the storage silos is more than 200,000 bbls of various types of cement plus masonry cement.

The shipping areas are equipped with truck and track scales for loading of either trucks or rail cars from each area.

Raw mix blending, burning, clinker, cooling, and milling operations are controlled by a direct digital control (DDC) computer system.
3. Design of the blasting round

3.1 Introduction

In the design of a blasting round there are many variables which must be included such as type of explosive, burden, spacing, etc. The concept of blasting as part of a total system make the problem much more complicated. Some analytical approaches developed to help solve this problem will be considered in the following pages. Their results form a good basis upon which to perform the required experimental tests.

European approaches have been included in this study as well as those of Americans. A Russian method is also cited because it begins with the fragmentation analysis and prediction of sizes and after that the calculations of drilling and blasting parameters.

The approach of Gerbella (4) was selected for calculating burden, spacing and stemming because, in spite of being quite old (1950), takes into account the rock characteristics, the structural features of the body to be blasted, and the explosive characteristics.

The direction of shooting was selected according to Atchison (5) and the delay time according to Langefors (6).

At the limestone quarry the following conditions were considered as fixed: 1) hole diameter, 2) bench height and 3) explosive type.
3.2 Physical Properties of the Rock

Limestone (tensile strength between 350 and 1380 psi.) was considered to be a good rock to experiment with. Its strength might be considered representative of a wide range of rocks and minerals. From the standpoint of energy spent in crushing, limestone (work index = 12.74 kwh. per ton crushed to 67% passing 200 mesh) is also representative of copper ore (12.73 kwh.) and hematite (12.84 kwh.).

Compressive, tensile, and shear strength properties are sometimes used to classify rock with regard to ease of breaking with explosives or to calculate explosive charges to break them.

As most rocks are very weak in tension, this strength measures to an extent the susceptibility of the rock to tensile failure by stress pulse reflection. The ratio of compressive to tensile strength which normally varies between 10 to 100, has been called Blastability coefficient (7).

These values as well as the rock density were either determined or calculated.

The compressive strength was obtained by averaging the results of 5 samples (Cores 2\(\frac{1}{16}\) in. diameter, 6 in. length) air dried for a week. Under a loading rate of 100 psi per second the rupture load was between 34,700 and 38,500 lbs.

Average compressive strength thus determined is 11,200 psi.
The tensile strength was determined indirectly with the Brazilian test. Eight cores air dried for a week were loaded along the longitudinal axis directly between the plates of a testing machine. Three of the cores collapsed before starting the test. Five cores failed under an average load of 2600 lbs. Therefore the indirect Tensile strength ($T_o$) is:

$$T_o = \frac{2F}{\pi DL} = \frac{2 \times 2600}{3.14 \times 2 \times 6} = 138 \text{ psi.}$$

Where:

- $F$ = applied force (lb.)
- $D$ = diameter of the core (in.)
- $L$ = length of the core (in.)

The value of $T_o$ seems low as compared with other usual limestone values. Some fossil inclusions (kerogen) could be an explanation.

The shear strength was calculated from compressive and tensile values using the Wuerker (8) approach (intersection of common tangent with the shear axis from the plot of $C_o$ and $T_o$ on Mohr's Circle). A value of 500 psi was selected.

The density was determined to be 2.6 gr/cc.

### 3.3. Determination of the Burden

#### 3.3.1. Langefor's approach

According to Langefor's (6) the burden for bench blasting can be determined using

$$V = \frac{dp}{33} \left( \frac{p}{c_f E/V} \right)^{\frac{1}{2}} \tag{1}$$
Where:

\[ V = \text{Burden (meters)} \]
\[ d_b = \text{diameter at bottom of the drill hole (mm)} \]
\[ P = \text{degree of packing of the explosive (kg/}\text{decimeter}^3) \]
\[ s = \text{strength of the explosive (as compared with dynamite where, } s = 1) \]
\[ c = \text{rock factor (amount of explosive required to break a cubic meter of rock)} \]
\[ \bar{c} = \text{same as } c \text{ plus a technical margin (} \bar{c} = c + 0.05) \]
\[ f = \text{fixation factor (1 for vertical holes; 0.90-0.80 for slopes 3:1 - 2:1)} \]
\[ E = \text{hole spacing (meters)} \]

For this quarry these parameters take the following values:

\[ P = 0.82 \text{ kg/cubic decimeter (ANFO prills)} \]
\[ s = 0.86 \text{ (ANFO prills)} \]
\[ \bar{c} = 0.28 \text{ (lowest limit from Langefors figures)} \]
\[ E = 2 V \text{ (assuming a staggered pattern is used)} \]
\[ d_b = 75 \text{ mm} \]
\[ f = 1 \text{ (not exactly true because the drill holes have some inclination)} \]

Therefore equation (1) becomes:

\[ V = \frac{75}{33} \left[ \frac{0.82 \times 0.90}{0.23 \times 1 \times 2} \right]^{1/2} \]
and:

\[ V = 2.63 \text{ m.} = 8.6 \text{ ft.} \]

The practical burden (\( V_1 \)) is less than \( V \) because hole placement is not exact and hole direction can be in error. Using the equation of Langefors one find that:

\[ V_1 (\text{Practical burden}) = V - 0.05 K \]

where:

\[ K = \text{the height of the bench} \]

thus:

\[ V_1 = 8.6 - 0.05 \times 10 = 8.1 \text{ ft.} \]

3.3.2 Approach of the burden according to Pearse, Allsman and Speath (9)

According to these authors, the following formula can be used to calculate the burden for bench blasting.

\[ B = \left( \frac{K D_e}{12} \right) \left( \frac{P_e}{S_t} \right)^{\frac{1}{2}} \]

(2)

where:

\[ B = \text{burden (ft.²)} \]

\[ K = \text{constant estimated to be about 0.8 for most rocks.} \]

\[ D_e = \text{charge diameter (in.)} \]

\[ P_e = \text{peak explosion pressure of the explosive (psi.)} \]
$S_t =$ ultimate tensile strength of the rock (psi).

For the limestone it is assumed that:

\[ K = 0.8 \]

\[ D_e = 3 \text{ in.} \]

\[ P_e = 13.5 \text{ kilobars}, \ P_{e1} = 40 \text{ kbars (12)} \]

\[ S_t = 138 \text{ psi.} \]

(average from 5 samples. Brazilian test)

*\( P_e \) may represent the poorest confinement

*\( P_{e1} \) is a hydrodynamic value

Substituting in equation (2) one finds that:

\[
B = \left( \frac{0.8}{12} \right)^{1/2} \left( 3 \right) \left( 13.5 \text{kbar.} \times 14.2 \times 10^3 \text{ psi./kbar.} \right)^{1/2}
\]

\[
B = 7.5 \text{ ft. and } B_1 = 12.5 \text{ ft. if } P_e = 40 \text{ kbars.}
\]

By taking into account some of the favorable features of the limestone formation in this quarry, such as angled holes, well defined planes of separation between layers and between the limestone body and the underlying sandstone it appears that the \( K \) could be increased to 1.0.

The burden in this case would be:
\[ B = \left( \frac{1 \times 3}{12} \right) \left( \frac{13.5 \times 14.2 \times 10^3}{138} \right)^{\frac{1}{2}} \]

\[ B = 9.2 \text{ ft. and } B_1 = 15.3 \text{ ft} \]

3.3.3 Approach of Ash (9)

Due to difficulties in assigning values to \( P_e \) and \( S_t \) (peak explosive pressure and ultimate tensile strength), Ash (9) simplified equation (2) by replacing \( (P_e/S_t)^{1/2} \) and \( K \) by a burden ratio \( K_b \). Thus:

\[ B = \frac{K_b \times D_e}{12} \quad (3) \]

where:

\[ B = \text{burden (ft.)} \]

\[ K_b = \text{burden ratio} \]

The burden ratio \( K_b \) varies from 14 to 49, with 30 being a reasonable estimate for blasting material weighing 160 pcf. using an explosive having properties similar to 60% ammonia dynamite (S.G. = 1.3, \( V = 12000 \text{ ft./sec.} \)). No consideration is given to the competency of the rock strength, or structural features of the material. The blasting effectiveness of an explosive is considered only as a function of its kinetic energy release and the weight of material to be fragmented and moved.

To take into account these two variables, adjustment with multiplying factors is suggested for \( k_b = 30 \) (average).
The rock adjustment factor is given by

\[ \text{Raf} = \frac{\text{Dsr}^{1/3}}{\text{Dsm}} \]

(4)

Where:

- \( \text{Raf} \) = rock adjustment factor
- \( \text{Dsr} \) = density of standard rock (160 pcf.)
- \( \text{Dsr} \) = density of material being blasted (pcf)

Thus:

\[ \text{Raf} = \frac{5.43}{(\text{Dsr})^{1/3}} \]

Since at the quarry \( \text{Dsr} = 156 \text{ pcf} \)

Then:

\[ \text{Raf} = \frac{5.43}{5.38} \]

\[ \text{Raf} = 1.008 \]

If:

- \( \text{Eaf} \) = explosive adjustment factor
- \( \text{Ds} \) = density of standard explosive
- \( \text{Vs} \) = detonation velocity of standard explosive
D\textsubscript{c} = density of the explosive to be used

V\textsubscript{c} = detonation velocity of explosive to be used

Then the explosive energy potential of the standard explosive (EPS) is

\[ \text{EPS} = D\textsubscript{s} \times V\textsubscript{s}^2 \]  

(5)

and the energy potential of the explosive to be used, is

\[ \text{EPu} = D\textsubscript{e} \times V\textsubscript{e}^2 \]  

(6)

The explosive adjustment factor becomes:

\[ E\textsubscript{af} = \frac{\text{EPS}}{\text{EPu}}^{1/3} \]  

(7)

Explosive energy potential is considered to be equal to

\[ 45 \times 10^6 \left( \frac{\text{gm}}{\text{cm}^3} \times \text{sq ft.} \right) \text{ weakest explosive} \]

\[ 167 \times 10^6 \]  

" standard "

\[ 706 \times 10^6 \]  

" strongest "

Assuming for ANFO a detonation velocity of 12000 ft/sec, the energy potential is: (from 5)

\[ \text{Energy Pot} = 0.83 \times 12000^{1/2} \times 118 \times 10^6 \left( \text{gm/cc x sq ft} \right) \]  

(13)

The burden ratio \( k\textsubscript{b} \) is defined as the product of average \( K\textsubscript{b} \) and \( E\textsubscript{af} \). Then:

\[ k\textsubscript{b} = 30 \left( \frac{118}{187} \right)^{1/3} \]

or:

\[ K\textsubscript{b} = 25.8 \]
By making the necessary adjustment for rock density, the final burden ratio is obtained:

$$K_b = 25.0 \times 1.003 = 26$$

and the burden is:

$$b = \frac{k_b \times D_0}{12} = \frac{26 \times 3}{12} = 6.5 \text{ ft.}$$

Should the ANFO have a detonation velocity of 13,500 ft./sec. The burden for this case will be:

$$b = 7.10 \text{ ft.}$$

3.3.2 Serbellon's Approach (4)

As a first approximation, a blast hole can be thought of as a cylindrical container with a gas under pressure. For the collapse of the container and surroundings it is necessary to know at least approximately the tensile and shear strength of the rock.

By assuming that the explosive loaded in a vertical hole, located parallel to the pit wall, has to be able to develop a larger force than the tensile and shear loads at failure of planes 1 and 2 in fig. (4) and if:

$$h = \text{height of the bench}$$

$$s = \text{spacing between holes}$$

$$t = \text{burden}$$
d = blasthole diameter  
1 = length of the explosive load  
$F_t =$ Tensile load at failure  
$F_c =$ shear load at failure  
$T_o =$ ultimate tensile strength  
$S_o =$ ultimate shear strength  

Then:

$$F_t = T_o \, h \, s$$  \hspace{1cm} (8)  
$$F_c = S_o \, b \, s$$  \hspace{1cm} (9)  

And

$$F_t + F_c = T_o \, h \, s + S_o \, b \, s \text{ (Tensile and shear load at failure)}$$  \hspace{1cm} (10)  

The force developed by the explosive ($F_E$) is:

$$F_E = P_E \times \text{Area}$$  \hspace{1cm} (11)  

where:

$$\text{Area} = d \times l \text{ (Longitudinal section of loaded part of the blasthole)}$$  

To produce failure the force of explosive must be greater than the sum of tensile and shear loads.

$$F_E > T_o \, h \, s + S_o \, b \, s$$  \hspace{1cm} (12)  

Equation (12) can be written as an equality with the introduction of reduction factor

$$F_E \, n = T_o \, h \, s + S_o \, b \, s$$  \hspace{1cm} (13)  

where:

$$n = 0.15 \text{ to } 0.35 \text{ (According decreasing difficulty to break the rock)}$$
It will be assumed that:

\[ s = 2b \]
\[ H = 3.05 \text{ m (10 ft)} \]
\[ h = 1.52 \text{ m (5 ft) (height of explosive load)} \]
\[ d = 7.62 \text{ cm (3 in.)} \]
\[ S_0 = 138 \text{ psi. (90.5 kbar.)} \]
\[ T_0 = 138 \text{ psi. (90.5 kbar.)} \]

From Table (2) dynamite of density .82 grm/cm\(^3\) develops a pressure of 11340 kg/cm\(^2\) (11.34 kbar). As the strength of ANFO is equivalent to 0.86 as compared with dynamite (6), its pressure \(P_{\text{ANFO}}\) is:

\[ P_{\text{ANFO}} = 11.34 \times 0.86 = 9.2 \text{ kbar.} \]

The area of the hole \(A_H\) over which the gas pressure is assumed to act is given by:

\[ A_H = l \cdot d \]

Where:

- \(l\) = length of the explosive charge
- \(d\) = hole diameter

Thus:

\[ A_H = 152 \text{ cm} \times 6.72 \text{ cm} = 1155 \text{ cm}^2 \]

The explosive force is given as:

\[ E_F = \frac{P_{\text{ANFO}} \cdot A_H}{h} \]
\[ E_F = 9.2 \text{ kbar} \times 1155 \text{ cm}^2 = 11403 \text{ ton} \]

The effective force is obtained by multiplying \(E_F\) by a reduction factor depending upon the structural features of the rock. It varies from 0.15 to 0.35 with the later value
applying for optimum conditions.

A reduction factor of 0.35 has been assumed according to the particular situation at the limestone quarry.

Therefore:

\[ F_E (0.35) = 11.403 (0.35) = 3991 \text{ ton.} \]

Substituting this into equation (13) one finds that:

\[ 3991 = 90.5 (3.05) (2b) + (90.5) 2b^2 \]

Solving for \( b \) one finds

- Burden, \( b = 2.56 \text{ m. (8.24 ft)} \)
- Spacing, \( s = 5.12 \text{ m. (16.48 ft)} \)

Garbella used tables from A. Grenon for approximate values of tensile and shear strength of the rocks and the tables of Tafanel and Dautrice to estimate the explosive pressure developed by the charge according to its density. These are given as Tables 1, 2.
### TABLE 1  ULTIMATE TENSILE AND SHEAR STRENGTH

*(After A. Grenon)*

<table>
<thead>
<tr>
<th>ROCK</th>
<th>S.G.</th>
<th>ULTIMATE TENSILE STRENGTH</th>
<th>ULTIMATE SHEAR STRENGTH</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>gm/cm³</td>
<td>ton/m² (psi)</td>
<td>ton/m² (psi)</td>
</tr>
<tr>
<td>DIABASE</td>
<td>3.2</td>
<td>1800 (2554)</td>
<td>3000 (4256)</td>
</tr>
<tr>
<td>BASALT</td>
<td>3.0</td>
<td>500 (709)</td>
<td>2600 (3838)</td>
</tr>
<tr>
<td>GRANITE</td>
<td>2.8</td>
<td>800 (1135)</td>
<td>1000 (1417)</td>
</tr>
<tr>
<td>SANSTONE</td>
<td>2.6</td>
<td>300 (425)-600 (850)</td>
<td>1400 (1986)</td>
</tr>
<tr>
<td>QUARTZ</td>
<td>2.65</td>
<td>300 (425)-500 (709)</td>
<td>1000 (1417)</td>
</tr>
<tr>
<td>LIMESTONE (compact)</td>
<td>2.65</td>
<td>500 (709)</td>
<td>1100 (1560)</td>
</tr>
<tr>
<td>MARBLE (Carrara)</td>
<td>2.70</td>
<td>450 (638)</td>
<td>450 (638)</td>
</tr>
<tr>
<td>LIMESTONE (ordinary)</td>
<td>2.50</td>
<td>300 (425)</td>
<td>300 (425)</td>
</tr>
</tbody>
</table>

### TABLE 2  DYNAMITE PRESSURE vs. LOADING DENSITY

*(After Taffanel and Dautrice)*

<table>
<thead>
<tr>
<th>LOADING DENSITY</th>
<th>PRESSURE</th>
</tr>
</thead>
<tbody>
<tr>
<td>gm/cm³</td>
<td>kbar</td>
</tr>
<tr>
<td>0.7</td>
<td>9</td>
</tr>
<tr>
<td>0.8</td>
<td>11.2</td>
</tr>
<tr>
<td>0.9</td>
<td>12.6</td>
</tr>
<tr>
<td>1.0</td>
<td>14.0</td>
</tr>
<tr>
<td>1.1</td>
<td>15.4</td>
</tr>
<tr>
<td>1.2</td>
<td>16.8</td>
</tr>
<tr>
<td>1.3</td>
<td>18.2</td>
</tr>
</tbody>
</table>
PLANE 1  A-D-I-E (TENSILE PLANE) = Hs

PLANE 2  A-B-C-D (SHEAR PLANE) = bs

H = height of the bench
b = burden
s = spacing

FIGURE 4  DIAGRAMATIC REPRESENTATION OF TENSILE AND SHEAR
PLANE S IN GERBELLA'S APPROACH
Calculations of burden and spacing for four cases using the method of Gerbella\(^{(4)}\) are given below.

**Case 1.** Assume:

\[
T_o = 90.5 \text{ ton/m}^2 \quad \text{(determined value)}
\]

\[
S_o = 352 \quad \text{"} \quad \text{(value calculated from uniaxial compressive strength 11200 psi and tensile strength 138 psi, according Wuerker \((8)\))}
\]

\[
s = b
\]

\[
H = 3.05 \text{ m} = 10 \text{ ft.}
\]

\[
F_e(0.35) = T_o \cdot h \cdot b + S_o \cdot b \cdot s
\]

\[
3991 = 352 \cdot b^2 + 296.5 \cdot b
\]

\[
b = 2.56 \text{ m} = 9.56 \text{ ft.}
\]

\[
s = 2.56 \text{ m} = 9.56 \text{ ft.}
\]

\[
b \cdot s = 91.40 \text{ sq ft.}
\]

**Case 2.** Assume:

\[
T_o = S_o = 90.5 \text{ ton/m}^2 \quad \text{and} \quad 2s = b
\]

\[
3991 = 790 \cdot b^2 + 583 \cdot b
\]

\[
b = 1.87 \text{ m} = 6.15 \text{ ft.}
\]

\[
s = 3.74 \text{ m} = 12.30 \text{ ft.}
\]

\[
b \cdot s = 75.64 \text{ sq ft.}
\]

**Case 3.** Assume \(T_o = S_o = 300 \text{ ton/m}^2\) (Table values) and \(s = b\)

\[
3991 = 300 \cdot b^2 + 915 \cdot b
\]

\[
b = 2.39 \text{ m} = 7.82 \text{ ft.}
\]

\[
s = 2.39 \text{ m} = 7.82 \text{ ft.}
\]

\[
b \cdot s = 61.15 \text{ sq ft.}
\]

**Case 4.** Assume same as above and \(s = b\)

\[
3991 = 600 \cdot b^2 + 1830 \cdot b
\]

\[
b = 1.47 \text{ m} = 4.73 \text{ ft.}
\]

\[
s = 2.93 \text{ m} = 9.46 \text{ ft.}; \quad b \cdot s = 44.75 \text{ sq ft.}
Calculation of the powder factor for a bench height of 10 ft. (3.05 m.) and an explosive load height of 5 ft (1.52 m.) in a 3-in. diameter hole for the previous four cases follow.

Volume of explosive = \( \pi/4 \times d^2 \times h = 6932 \text{ cm}^3 \)

1. Weight of explosive = Vol x S.G. = 6932 x 0.82 = 5684 g = 13.1 lb.

Tonnage yield per hole = \( b \times s \times H \times S_g \text{ rock} \)

\[ = 2.56 \times 2.56 \times 3.05 \times 2.6 = 52 \text{ ton.} \]

Powder Factor

\[ = \frac{52}{13} = 4 \text{ ton./lb.} \]

2. Yield per hole = 55.5 ton.
   Powder factor = 4.27 ton./lb.

3. Yield per hole = 45.3 ton.
   Powder factor = 3.48 ton./lb.

4. Yield per hole = 34.27 ton.
   Powder factor = 2.67 tons./lb.

For the proposed pattern (8 x 16 ft) and the same conditions.

Yield per hole = 94.23 ton.
Powder factor = 7.24 ton/lb.

3.3.5 Summary

These four approaches, all with rather different assumptions, suggest burdens varying from 5.4 to 10.5 ft.

A final burden of 8 ft. was chosen since this value is equal to the lowest height of the variable bench, and because spacing became 16 ft. for the earlier assumption made that \( s = 2b \). This gives an area of 128 sq. ft. which is similar to the pattern used in normal blasting (11x11) having an area of 121 sq. ft. This similarity in areas is
desirable because it was intended to see how the change in direction of shooting following helpful structural planes should affect the final fragmentation.

This selection leads to the following situation:

<table>
<thead>
<tr>
<th>Present Pattern</th>
<th>Proposed Pattern</th>
</tr>
</thead>
<tbody>
<tr>
<td>Diameter of the hole</td>
<td>3 in.</td>
</tr>
<tr>
<td>Bench's height</td>
<td>same</td>
</tr>
<tr>
<td>Burden</td>
<td>11 ft.</td>
</tr>
<tr>
<td>Spacing</td>
<td>11 ft.</td>
</tr>
<tr>
<td>B x S</td>
<td>121 sq ft.</td>
</tr>
<tr>
<td>Rock volume</td>
<td>V</td>
</tr>
</tbody>
</table>

Practically only the weight of explosive, its distribution and direction of shooting are going to be different.
3.4 Determination of Hole Spacing

After burden, the spacing is the next most important parameter in blasting design. It is normally considered to be the distance between charges aligned in rows parallel with the pit wall.

The interaction between charges in adjacent holes depends on the spacing, the burden, height of bench, timing between adjacent charges and direction of the structural planes of the rock.

Usually the value of the spacing/burden ratio varies from 1 to 2.

Because of the presence of partings at floor level and favorable features of the limestone formation, the creation of large stresses should be the most effective way to insure maximum fragmentation.

According to Ash (9) these differential shearing stresses are maximized when blastholes which are aligned with the principal rock structural weakness planes are fired simultaneously.

This differential shearing is obtained when the time interval for initiating adjacent charges is small (as simultaneous firing with E-cord). Maximum stress wave interaction occurs in the zone between blast holes.

Hino (7) determined that a full composite crater is obtained with simultaneous firing of two adjacent charges located in such a way that $S = 1.4B$. Ash (9) found that for...
desert alluvium $S = 1.0$ to $1.25$, but in blasting experiences with rocklike material it was found that $S = 1.5B$.

Laboratory and field experience (9) from full scale industrial blasts confirm the validity of this relationship particularly when charge alignment closely follows the structural planes of a material. The optimum spacing would be twice the burden dimension when apex angles are 90 degrees $S = 2B$.

If failure does not follow structural planes then Ash (9) recommends that $S = 1.8B$ for proper shearing occurs between blastholes.

For sequence delays in the same row:

$$S = B$$

and for simultaneous timing in the same row:

$$S = 2B$$

In this latter situation, when all holes in a single row are fired simultaneously, but timing between rows is delayed, the rectangular pattern is possible but the staggered pattern is preferred. This last configuration not only provides the best balance, but insures there will be charges located directly in back of the zones between blastholes of the previously initiated row.
3.5 Explosives

Low density and low cost Hercomix (Hercules' ANFO version) is usually used as a principal 1-deck charge for blasting the limestone rock. This is primed with a gelatine type (Hercules Gelamite) dynamite initiated by E-cord (25 grains low charge primacord type).

The explosives have the following characteristics and prices (1).

<table>
<thead>
<tr>
<th>Explosive</th>
<th>Density</th>
<th>Velocity</th>
<th>Energy</th>
<th>Price</th>
</tr>
</thead>
<tbody>
<tr>
<td>Hercomix</td>
<td>0.82 g/cm³</td>
<td>11 - 13 ft/s (x 1000)</td>
<td>900 Kcal/Kg</td>
<td>5.0 $/lbs.</td>
</tr>
<tr>
<td>Gelamite</td>
<td>1.48 g/cm³</td>
<td>15.500 ft/s</td>
<td>1160 Kcal/Kg</td>
<td>22.5 $/lbs.</td>
</tr>
</tbody>
</table>

Detonating Device

(E-Cord) 25 grain/ft; 21,000 ft/sec and 2.65 $/ft.

Timing Device

Millisecond delay connectors type 9 and 17 millisecond at 6.35 $/piece.

The E-cord is detonated by a pair of number 6 electric blasting caps costing 26 $ each.
3.6 Stemming

At the beginning of the testing program at the limestone quarry two or three ft. of loose stemming was used in each of the 8 to 14-ft. depth holes without regard for the burden.

It is known that energy travels faster as density of the material through which it is travelling increases. The difference in density of the rock and stemming is responsible for confinement since a delayed action occurs as energy travels across the stemming compared with the velocity of travel in solid rock. The longer the stemming, the greater the difference in travel time and the better chance for the gases to perform fracturing of rock before movement or stemming ejection.

Stemming has been a controversial matter, but laboratory and field experience has generally shown that when done with care it can greatly improve explosive performance (13).

The importance of maintaining pressure and controlling the gas release increases as the weakness of fracturing of the rock increases.

Several criteria exist regarding the amount of stemming necessary. A majority of authors (6) recommend that the stemming should be equal in length to the burden. Some (4, 9) say that it should vary from 3/4 to 4/5 of the burden. Others (9, 13) say that it should be at least 2/3 in normal conditions, but it can be as low as a half of the burden when discontinuities are pronounced (9).
Stemming has an influence on the breakage, air blast, flyrock, and overbreak in the collar zone of the blastholes.

From the above it follows that better stemming can improve fragmentation and control of air-blast, flyrock and overbreak.

A stemming of at least a half of the burden was chosen for the tests:

3.7 Direction of shooting

According to Atchison (5), in many blasting situations the structural pattern of the rock exerts a major control on the resulting fragmentation. He states:

"As a general principle, more effective breakage is accomplished by placing explosive charges within the solid blocks bounded by such discontinuities rather than attempting to transfer explosive energy across them. Blasting patterns can be designed to take advantage of rock structure, for example, by planning a free face parallel rather than perpendicular to marked vertical joint planes or in rock with well developed bedding or schistosity planes, by keeping the free face perpendicular rather than parallel to the direction of the dip".

At the limestone quarry the principal vertical joints and dip direction are roughly coincident, and therefore the parallel alignment of the rows with these planes gives a free face perpendicular to the dip. This arrangement satisfied both situations described above. The alignment is optimum regarding not only the fragmentation but the loading as well.
since advantage can be taken of the floor grade. This allows easier digging for the front end payloader and aids in fully loading buckets. Visibility and maneuverability are also better.

3.8 Use of delays

Langefors (6) has found that for burdens between 2 and 25 ft. a linear relationship exists between the burden and the delay time required to obtain the best breakage.

If $\tau$ is the delay time in milliseconds, $B$ the burden in ft. and $k$ a factor varying between 0.9 and 1.5 milliseconds per ft. the equation can be written as:

$$\tau = k B$$

For a burden of 8 ft. and assuming a $k$ value of 1.2 ms./ft., the delay time become 9 ms.

This delay time was chosen and used in almost all the tests with good results.
4. EXPERIMENTAL RESULTS

4.1 Introduction

The drilling, blasting, loading, hauling, and crushing operations at the Lyons quarry were observed over a period of several months. From these observations it was possible to suggest improvements in the operation. These are discussed in detail in the following sections.

From timing the drilling, loading, and hauling operations it was clear that each of the many men in the different tasks tried to do his best under observation. Moreover, the challenge of working under any such measurer as a simple stop watch was led to the recording of many minimum times. A man produces more in amount and/or quality if he has determined goals or standards to meet.

Bonuses and/or more than one-man supervision could improve productivity by a factor more than enough to pay for the additional labor costs. Requirements for full capacity production may demand one or both alternatives. Some experience of the author in similar mining operations supports this opinion.

Changes in the round design suggested by a combination of published theory and practice for other rock types resulted in substantial improvement in blasting practice at the Lyons quarry. Using such available information can save much time in the development of a best round.
Although Langefors, Ash, Fearse, and Gerbella have all calculated the burden in somewhat different ways, the final values were all found to be similar. An average value used in the round design gave excellent results.

4.2 Drilling

The drilling under three operators was observed. The highly skilled operator was able to get twice as much footage as the medium-skilled one, and four times more footage than the least skilled (beginner). A smoother operation, less wear on the equipment and better footage with a better quality hole (right location and depth) obtained by a good operator lead one to conclude that a bad operator is a luxury.

Two and three-winged tungsten carbide bits had been used for 3-in. diameter holes. In spite of its better performance and cheaper cost the use of the two-winged bit was discontinued because of the special skill required of the driller when collaring the holes.
This type of bit was able to drill 160 ft. per hour and support 5-6 regrinds giving a useful life between 500-600 ft. The down force used was between 600 and 700 lbs.

Three winged KAY brand, medium type, 3-in. diam. bits are currently in use. The performance depends largely on the operator. Better footage in limestone is obtained with hold down forces ranging from 800-1000 lbs. though forces as high as 1200 lbs. are commonly used.

For the 3-in. diameter holes the drilling rates vary from 3 to 8 fpm. for new bits and 1.5 to 6.8 for reground ones.

Ideal performance is 1000 ft. per a 7½-hr. working shift. Average performance is 50% to 60% of this.

4.3 Drilling Patterns

The 11 ft. x 11 ft. square pattern in use at the beginning of the study was eventually changed to a staggered 8 ft. x 16 ft. pattern.

Intermediate trials were made starting with 7.5 x 15 and finishing with a 9 ft. x 18 ft. pattern. Fragmentation at the beginning was evaluated using the loading time of the
broken rock. No differences were observed, however. The final pattern was chosen based upon visual observation of the broken pieces.

No difference in cost of drilling per hole for the various patterns was observed.

The depth of the holes was set equal to the bench height. In practice this was measured indirectly by inspection of the color of the cuttings which change from white to light brown when the sandstone base is reached.

4.4 Drilling Cost

Drilling cost can be estimated using the formulas developed by Williamson (14) given below.

\[ R_h = 1.5 M \times 10^{-4} + 1.25 E \]  
(14)

\[ R_f = \frac{R_h}{0.7 Z} = \frac{1.5 M \times 10^{-4} + 1.25 E}{0.7 Z} \]  
(15)

\[ DC = R_f + \frac{B}{L} = \frac{1.5 M \times 10^{-4} + 1.25 E + B/L}{0.7 Z} \]  
(16)

where:

- \( R_h \): rig hourly operating cost ($/hr.)
- \( M \): machine or rig delivered cost ($)
- \( E \): sum of direct wages of operator and helper ($/hr.)
- \( R_f \): rig operating cost per ft. of hole ($/ft.)
- \( Z \): Drilling rate (ft/hr) (0.7 \( Z \) is effective drilling rate, allowing for 30% down time)
Assuming that

\[ M = \$ 40,000. \]
\[ E = \$ 4.00 \]
\[ Z = 130 \text{ ft/hr} \]
\[ L = 600 \text{ ft.} \]
\[ B = \$ 25.00 \]

One finds that:

\[ R_h = \$ 11.00 \text{ per hour} \]
\[ R_f = \$ 0.121 \text{ per ft}^2 \]
\[ DC = \$ 0.163 \text{ per ft}^2 \]

This drilling cost of 16.3 $/ft. results in a cost of 1.88 $/ton for an 11 x 11 ft. pattern and 1.72 $/ton for an 8 x 16 ft. pattern.

4. **Comparative blasting cost**

<table>
<thead>
<tr>
<th></th>
<th>Final Pattern</th>
<th>Initial Pattern</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>8 x 16 ft.</td>
<td>11 x 11 ft.</td>
</tr>
<tr>
<td>Blast-hole diameter (in.)</td>
<td>3</td>
<td>3</td>
</tr>
<tr>
<td>Depth (ft.)</td>
<td>10</td>
<td>10</td>
</tr>
<tr>
<td>Area (sq. ft.)</td>
<td>128</td>
<td>121</td>
</tr>
<tr>
<td>Volume (cu yd)</td>
<td>47.3</td>
<td>44.8</td>
</tr>
<tr>
<td>Tonnage</td>
<td>94.6</td>
<td>89.6</td>
</tr>
<tr>
<td>Powder Factor (Ton./lbs.)</td>
<td>7</td>
<td>4</td>
</tr>
</tbody>
</table>
Explosive Charge (lbs.)

<table>
<thead>
<tr>
<th>Material</th>
<th>HER</th>
<th>JLR</th>
</tr>
</thead>
<tbody>
<tr>
<td>Hercomix (ANFO)</td>
<td>13.54</td>
<td>22.4</td>
</tr>
<tr>
<td>Gelamite (primer)</td>
<td>1.83</td>
<td>1.83</td>
</tr>
<tr>
<td>E-Cord (ft./hole)</td>
<td>26</td>
<td>30</td>
</tr>
<tr>
<td>MS-connectors (units/100holes)</td>
<td>18</td>
<td>36</td>
</tr>
</tbody>
</table>

Cost of Hercomix (¢/hole) | 67.7 | 112.0 |
Cost of Primer (¢/hole)   | 41.17| 41.17 |
Cost of Connectors (¢/hole)* | 11.4  | 21.6  |
Cost of E-cord (¢/hole)*  | 68.90| 79.50 |
EXPLOSIVE COST (¢/hole)   | 189.17 | 254.27 |
LOADING EXP. COST (¢/hole)** | 4.0  | 4.0  |
DRILLING COST (¢/hole)   | 163.0 | 163.0 |
DRILLING AND BLASTING (¢/hole) | 356.17 | 421.27 |
(¢/ton)                  | 3.77  | 4.70  |

The pattern 8 x 16 is approximately 20% cheaper than the 11 x 11 pattern.

*It has been calculated from a 100-hole round.

**It has been calculated using a round of 100 holes loaded by 1 man during a working shift.

4.6 Blasting

The burden and spacing were selected from several approaches described earlier by taking into account the following ideas:

a) The burden has to be equal or less than the lowest bench height.
b) The area of the burden times the spacing has to be approximately equal to the area of the pattern used earlier in order for comparisons to be made.

c) Alignment of the rows has to be parallel to a principal plane of weakness.

d) Drilling costs should be not greater than the costs already obtained.

It was found that when the direction of shooting was perpendicular to the dip and 9 ms delays were used between successive rows which were shot simultaneously, the staggered 8ft x 16ft was the best. (Fig. 5)

For the staggered 11ft x 11ft and equilateral staggered 9 ft x 11 ft patterns shot in the direction of the dip, the powder factors were the same (4 ton/lb.) as for the blasting currently practiced. From an inspection by the author and the loader operator it was concluded that these trials were inferior to the straight 11 x 11 pattern. Back break and some big boulders were present. Differences in loading time were not observed, but the loading operation was more difficult. Crushing energies were not measured in these early tests.

Blasting costs were lower for the staggered 8ft x 16ft pattern as compared with the straight 11ft x 11ft.
FIGURE 5  BLASTING PATTERN (PROPOSED)

Burden = 8 ft.
Spacing = 16 ft.

Approximate direction of weakness planes
Direction of the dip

E-Cord
ms-delays

INITIO

DIRECTION OF THE DIP
SHOOTING
4.7 Loading

The loading of the broken limestone is carried out with an International Harvester and Hough model 400 Articulated Tractor Shovel. It is rubber-tired with a toothed 10-cu yd bucket. Timing on different parts of the loading cycle, including loading and waiting times, are given below:

Loading cycle

Digging .................. 14 to 18 sec
Transport ............... 15 to 18 sec
Dump time ............... 5 to 7 sec
Return time ............. 11 to 12 sec
Spotting the truck ...... 3 to 6 sec

<table>
<thead>
<tr>
<th></th>
<th>Minimum</th>
<th>Maximum</th>
<th>Average</th>
</tr>
</thead>
<tbody>
<tr>
<td>Loading</td>
<td>(502 readings)</td>
<td>90 sec</td>
<td>255 sec</td>
</tr>
<tr>
<td>Waiting</td>
<td>(210 readings)</td>
<td>30 sec</td>
<td>480 sec</td>
</tr>
</tbody>
</table>

As hourly operating costs are usually higher for loaders than for trucks, the average waiting time suggest the idea of another truck being added to the two already in existence. Also it is not unusual at present for one of the two trucks be down for repairs, at any one time. A third truck might also be justified for this reason alone.
An estimation of the loading cost following Killebrew (15) criteria and data was done. The results are given below.

**Hourly ownership cost:**

- Delivered price: $104,585
- Tires (four): 10,244
- Bare machine cost: 94,341

Assume:

- Depreciation period 5 years or 10000 hr.
- Hourly cost of Interest, Insurance, Taxes is 1 % each.

**Hourly ownership costs** ($94,341/1000) = 9.43

**Interest, Insurance, Taxes** ($104,585 x 0.03 / 10000) = 3.12

**TOTAL HOURLY OWNERSHIP COST** = $12.55

**Operating costs:**

- Fuels and lubricants: 1.705
- Repairs parts and labor ($4.72/10000) = 0.0472

**Tire cost** (Purchase cost/3000 hr.) = 3.415

**Total estimated HOURLY OPERATING COST** = $14.20

**TOTAL ESTIMATED HOURLY OWNING AND OPERATING COST** = $26.75

- Cost per ton @ 800 ton/hr. = 3.34 $
- Cost of waiting time per minute = 44.6 $

4.8 Hauling

The hauling of the broken rock is done using two International-Harvester and Hough 50-ton trucks.

The round-trip of one mile was timed $174$ times.

Results follow:

<table>
<thead>
<tr>
<th>Minimum</th>
<th>Maximum</th>
<th>Average</th>
</tr>
</thead>
<tbody>
<tr>
<td>189</td>
<td>360</td>
<td>225.7</td>
</tr>
</tbody>
</table>

Waiting time for trucks was rarely observed; loading capacity is higher than the haulage capacity.

A rough estimate of cost for hauling was $20$ per hour. This costs includes ownership cost, operating cost, Interest, Taxes, Insurance and Storage cost, Tire replacement and Repairs.

Tire replacement and repairs is an important item. The calculation of the total cost of haulage and the determination of tire life values are extremely difficult. The values of tire life which were obtained range from $2300$ to $3000$ miles.

The factors in Haulage unit tire life are:

- Maintenance
- Maximum speed
- Curves
- Surface
- Loads
- Wheel position

Average
30 m/hr
Moderate
Hard packed earth
20 % underload
Rear dump
Bishop's (16) criteria and data were followed to get an estimation for the cost of hauling:

Hourly ownership cost

Machine cost. ....................................... $ 92150
8 tires ................................................. $ 7384
Bare machine cost .................................... $ 84766

Assume:

Depreciation (5 years @ 3000 hr./yr)

$84,766/15000 ......................................... $ 6.651

Interest, Taxes Insurance and Storage

\[
= \frac{13\% \times \text{average yearly investment}^*}{\text{operating hours per year}}
\]

* Yearly investment is assumed 60% of the bare machine cost

\[
= \frac{0.13 \times 0.60 \times $92150 - $7384}{3000}
\]

$2.204

Operating cost

Hourly tire = \[
\frac{\text{Tire replacement cost}}{\text{Estimated tire life}^*}
\] = $7384 = 1.758

4200

Tire repair cost = 15% hourly tire cost

0.264

Repairs (Parts and labor in average job conditions) = \[
\frac{\text{Purchase price} \times 0.60}{15,000}
\] = $3.690

Engine fuel consumption (10 gal/hr)

@ 0.14 a gal = 10 \times 0.14

1.40

Lubricants (oil, grease) + labor

(thumb rule) = 1/3 cost diesel fuel

0.47

Operator's wages and fringes benefits

5.00

* Calculated according to haulage factors
TOTAL HOURLY OPERATING COST

12.583

OWNERSHIP COST PER HOUR

7.855

TOTAL ESTIMATE HOURLY OWNERSHIP AND

OPERATING COSTS

$ 20.44

COST PER TON ≥ 400 TON/HR.

5.44 c

4.9 Crushing

The primary crushing of limestone and the other rocks necessary for making cement is done using a Single Roll Primary Crusher (Pennsylvania Crusher Corporation, 30" x 50" rated at 150 tph.) The feed has a maximum size of 24 in. and the product is a nominal minus 3 in.

A power-meter was installed to measure the energy spent in crushing, for it was hoped that these data would identify the pattern producing the best fragmented material.

No significant difference in energy consumption was found by comparing various rocks types.

Over 2100 power meter readings were taken over a period of three months. These observations were made at different times and shifts and for several loader operators to assure a sort of random sample.

One observation means the time required for the disk on the meter to make one complete revolution. According to the meter features and the potential and current transformers used, 10,000 revolutions are equivalent to 960kwh. Therefore one revolution is equal to 0.096 kwh.
Comparative energy consumption in Primary Crushing:

<table>
<thead>
<tr>
<th>Crushing</th>
<th>Observations</th>
<th>Drilling Pattern</th>
<th>Blasting Powder Factor (Ton./lb.)</th>
<th>Power Meter Disk (Sec./rev.)</th>
<th>Total Power (KW)</th>
<th>Power Consumption (Kwh/ton.)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Limestone</td>
<td>600</td>
<td>11x11 ft.</td>
<td>4.3 to 4.5</td>
<td>13.41</td>
<td>25.82</td>
<td>0.218</td>
</tr>
<tr>
<td>Limestone</td>
<td>600</td>
<td>8x16 ft.</td>
<td>5 to 7.5</td>
<td>14.15</td>
<td>24.00</td>
<td>0.204</td>
</tr>
<tr>
<td>Weathered Limestone</td>
<td>600</td>
<td>---------------</td>
<td>Material Ripped</td>
<td>14.40</td>
<td>23.90</td>
<td>0.200</td>
</tr>
<tr>
<td>Crusher Idling</td>
<td>---</td>
<td>---------------</td>
<td>---</td>
<td>---</td>
<td>---</td>
<td>---</td>
</tr>
</tbody>
</table>

\[
(a) \ KW = \frac{\text{Kwh x rev.}}{\text{rev hour}} \\
(b) \ Kwh/\text{ton.} = \frac{\text{Kwh x hour}}{\text{Ton. hour}}
\]

* If the cost of one Kwh is assumed to be 1 cent (a reasonable figure), this column represents the cost of the crushing energy.
This method of timing was necessary because the primary crusher was used with five different rock types which were scheduled according to plant necessities.

Long run measurements were also made and the results roughly match those for short runs.

Measurements revealed that on the average 120 tons of crushed limestone were produced per hour. This was used to calculate the Kwh required per ton of material.

These data differ somewhat from the plant data of 101.6 ton/hr. which were obtained by dividing the total production of limestone (140,089 tons) by the number of hours of operation (1466.1). This latter figure however includes the idle time of the crusher.

For all the different rock types, the total production of 400,000 tons required approximately 4000 operation hours. This gives a throughput figure of 100 ton/hr. through the primary crusher.

Measurements were made for about 100,000 tons. Energy spent on the total material (including four different kinds of rocks) was found to be 0.165 Kwh/ton. Additional data are in table 3.

A check was made by means of Bond's equation (17), with data from a hand sample. Acceptable agreement was found.
Bond's theory (17) asserts that when crushing feed particles of a given size to a smaller average size, an energy inversely proportional to the square root of the diameter of the product particles has to be used.

Bond's equation is usually written as:

\[
KWH = \frac{10 W_i}{\sqrt{P}} - \frac{10 W_i}{\sqrt{P}}
\]  

(17)

where:

- \( KWH \) = Gross power input in Kwh/short ton
- \( W_i \) = work index
- \( P \) = square-mesh aperture through which 80% of the feed will pass (microns)
- \( P \) = square-mesh aperture through which 80% of the product will pass

The value of \( P \) is calculated because its measurement is dependent upon several very variable factors.

Bond (18) gives an experimentally obtained equation to calculate \( P \).

\[
P \text{ (in)} = \text{Oss} \times (0.04 W_i + 0.40)
\]  

(18)

where:

- \( W_i \) = work index factor
- \( \text{Oss} \) = open side setting of the crusher at the bottom of the crushing chamber in inches.
At the limestone quarry

\[ W_i = 12.74 \text{ kwh/ton (Limestone)} \]

\[ O_{ss} = 3 \text{ in.} \]

Then, from equation (18)

\[ P = 3(0.04 \times 12.74 + 0.46) \]
\[ P = 2.7 \text{ in.} = 68580 \text{ microns} \]

From the hand sampling

\[ F = 2.90 \text{ in.} = 72500 \text{ microns} \]

Using equation (17) the power expended per ton of feed becomes

\[ \text{Kwh/ton} = \frac{10 \times 12.74 - 10 \times 12.74}{1/2 - 1/2} \]
\[ 68580 - 72500 \]

\[ \text{Kwh/ton} = 0.1335 \]

By considering an idling power of 37%, the total calculated power becomes:

\[ \text{Kwh/ton} = 0.1335 \times 1.37 = 0.183 \text{ Kwh/short ton} \]
\[ = 0.187 \text{ Kwh/ton} \]

The measured power was (table 3) \[ = 0.204 \text{ Kwh/ton} \]

*Percentage of idling power was calculated by dividing the measured idling power at the Roll Crusher by the measured power for crushing the limestone (Table 3).*
5. FRAGMENTATION COMPARISON

5.1 Fragmentation as Determined through Hand Sampling

With the purpose to evaluate the blasting efficiency a "random" sample of the broken material at the pit was taken. Such a sample was a sixth part of a fully loaded bucket spread over the floor. The sample has the following features:

Number of pieces = 2010
Weight = 6105 lbs (2.775 tons)
Volume = 1.454 cu yd (1.11 cu m.)

The smallest and largest samples according to length were:

<table>
<thead>
<tr>
<th>Weight</th>
<th>length</th>
<th>width</th>
<th>thickness</th>
<th>area</th>
</tr>
</thead>
<tbody>
<tr>
<td>lbs.</td>
<td>in.</td>
<td>in.</td>
<td>in.</td>
<td>sq. in.</td>
</tr>
<tr>
<td>0.006</td>
<td>0.5</td>
<td>0.487</td>
<td>0.25</td>
<td>0.98</td>
</tr>
<tr>
<td>90.00</td>
<td>35.0</td>
<td>24.0</td>
<td>1.20</td>
<td>1820.0</td>
</tr>
</tbody>
</table>

Whereas the smallest and largest samples according to weight were:

<table>
<thead>
<tr>
<th>Weight</th>
<th>length</th>
<th>width</th>
<th>thickness</th>
<th>area</th>
</tr>
</thead>
<tbody>
<tr>
<td>lbs.</td>
<td>in.</td>
<td>in.</td>
<td>in.</td>
<td>sq. in.</td>
</tr>
<tr>
<td>0.006</td>
<td>0.5</td>
<td>0.487</td>
<td>0.25</td>
<td>0.98</td>
</tr>
<tr>
<td>255.00</td>
<td>35.0</td>
<td>20.0</td>
<td>4.71</td>
<td>1671</td>
</tr>
</tbody>
</table>

Specific Area (SPA) and Reciprocal Mean diameter (DAV) were calculated to be

\[
SPA = \frac{TS}{TV} = 80.64 \text{ sq yd} \quad (67.42 \text{ m}^2 \text{ cu yd})
\]

\[
DAV = \frac{6}{SPA} = 0.87 \text{ m} \quad (2.21 \text{ in})
\]

where:

TS = overall surface of fragments
TV = total volume of fragments
To evaluate the size distribution, the three criteria given below were selected from among the several sometimes used.

1) Size = Width

2) Size = $\sqrt{\text{Length} \times \text{width}}$

3) Size = $\frac{\text{Length} + \text{width} + \text{thickness}}{3}$

A criterion $\text{Size} = \text{length}$ was also used to determine the number of pieces longer than the 24 in. length considered the maximum for adequate feeding of the primary crusher.

A size distribution percentage, a weight distribution per size, and a percentage of weight distribution per size were calculated. The results are given in Tables (8) through (16).

The range for the above calculations was for 1 to 24-in. sizes.

Passing dimension correspond to 36.8 per cent and 80 per cent were selected from the 2010 fragments of the hand sample. This selection was made based on the assumption that all samples are different in size in spite of the fact that for calculation the samples weighing less than 0.006 lbs. were considered to have the same dimensional parameters.

These oversize dimensions were determined for further use of Bond's equation (to estimate power requirements for commercial crushing and grinding installations) and the Rozin-Rammler equation to evaluate size distribution after blasting.
5.2 **Photoplanimetric Method**

The relationship between the fragment-size composition of blasted rock and a given oversize fragment dimension was found, and a method to calculate it was developed by Sirotyuk (2). According to this author the method is based upon energy consumed in crushing. The most suitable representative fragment size index being in this case reciprocal mean fragment diameter.

This reciprocal mean fragment diameter is equal to

\[ d_{AV} = \frac{\text{S}}{\text{NA}} \]  \hspace{1cm} (23)

where \( \text{NA} \) is the specific newly-formed surface area in \( m^2/m^3 \).

To determine the newly-formed surface and the reciprocal mean diameter, direct measurements after a blast have to be made.

A method to avoid this direct measurement and calculate them instead is based on the use of Rozin-Rammler equation in the form:

\[ R = 100 e^{-\left(\frac{x}{x_e}\right)^n} \]  \hspace{1cm} (24)

where:
- \( R \) = total yield of fractions of size \( x \), (percentage)
- \( n \) = index of distribution of the crushed material
- \( x_e \) = fragment dimension for which 36.8% of the crushed mass is larger (millimeter),
- \( x \) = is an arbitrarily set fragment dimension in the range between \( x_{\text{min}} \) and \( x_{\text{max}} \) (millimeter).
Because this reference was originally published in Russian, it is felt that some description of the method should be given.

Experimental tests were performed which showed that the Rozin-Rammler equation and the characteristic parameters $x_e$ and $n$ could be used to analyze the broken rock size either in mining or in the construction of underground structures.

Machines for loading and hauling the broken rock determine an $x_{\text{max}}$ value which is used as the initial quantity for the calculations.

Thirteen different rock types with a wide range of properties, variable explosive consumptions between 3.12 and 7.58 kg/m³ (Ammonite N 6) and different drilling patterns were used to determine the relationship between the reciprocal mean diameter, the oversize dimension ($x_{\text{max}}$) and the parameters $x_e$ and $n$.

The resistance of the rock to be broken was estimated according to a "crushability index" $v_{\text{max}}$ which was found to vary from 1.47 to 17.15 cm³, corresponding respectively to the classes: "most difficult to break" and "easy to break".

The fragment size composition of blasted rock was determined by the "point version of the photoplanimetric method" through pictures of surface of the rock heap in the mine cars or, alternatively, the surface of each new exposure of rubble directly at the face.

From the analysis of photoplanograms, it was found that the type of curve of the fragment size distribution (determined by blasting in mining conditions) was the same in all cases.
regardless of the wide range of variables involved in their production. It was inferred that for given blasting conditions, rock properties and explosive consumption have the same type of influence on the fragmentation.

After plotting and least squares fitting of experimental data the following empirical formulas were developed by Sirotyuk

\[ d_{av} = 0.26 x_e + 0.021 \]  
\[ d_{av} = 0.02 n + 0.03 \] (26)

after using Rozin-Rammler equation, it was found that:

\[ d_{av} = 0.003 + 0.094 x_{max} \] (27)

or

\[ d_{av} = 0.1 x_{max} \]

An important conclusion of Sirotyuk was that, given the dimension of oversized fragments when their content is 1%, it becomes possible to determine the reciprocal mean diameter from equation (27) and from equations (25) and (26) the correspondent \( x_e \) and \( n \) values.

The method has been shown to apply to test result for several rocks, ranging from gabbrodiabase (resistant to blasting) to high grade apatite (easily blasted).

Another Sirotyuk method (19) was used to calculate the drilling and blasting parameters and the necessary amount of explosives to get a given degree of crushing.
Excellent agreement between the given and measured crushing quality indices ($d_{av}$ and the fragment size composition of the blasted rock) was obtained when the consumed amount of explosive was close to that calculated.

Feasibility of applying the method to blasting with deep boreholes in open cut and underground ore mining was also shown.

Taking the log of both sides of equation (24) one finds that:

$$ x = x_{e} \frac{n \log \frac{100}{R}}{\log e} $$

(28)

When R is a constant, then $x$ is directly proportional to $x_{e}$.

The same is true for the relationship between $d_{av}$, $x_{max}$ and $x_{e}$ because these quantities are also linear dimensions of the fragment size.

Trials of 40 variants (for ranges of $n$ from 0.8 to 1.2 and $x_{max}$ from 0.2 to 1.6 m) show that the direct proportionality between the parameters is also conserved over a wider range of fragment sizes than for the fragment sizes formed in blasting of borehole charges.

One can thus write:

$$ d_{av} = k \cdot x_{e} \frac{n \log \frac{100}{R}}{\log e} $$

(29)

where:

$$ k = \text{Proportionality coefficient.} $$
For deep boreholes, two Rozin-Rammler type equations must be solved:

\[
\log \left[ \log \frac{100}{R} \right] - \log \left[ \log \frac{100}{R_{\text{max}}} \right] = n \left( \log \frac{x}{x_e} - \log \frac{x_{\text{max}}}{x_e} \right)
\]  

(30)

Where:

- \( x \) = any given fragment dimension (in the range between \( x_{\text{max}} \) and \( x_{\text{min}} \))
- \( R \) = overall yield of \( x \) expressed as percentage.

If:

- \( x = x_e \) then \( R = 36.8 \% \)

Then from equation (24)

\[
-n = -0.362 - \log \left( \log \frac{100}{R_{\text{max}}} \right) - \log \frac{x_{\text{max}}}{x_e}
\]

(31)

Since \( x_e \) is found from a given oversize fragment dimension, and \( x_{\text{max}} \) is that giving a yield \( R = 1\% \), finally

\[
-n = -0.663 - \log \frac{x_{\text{max}}}{x_e}
\]

(32)
References about the relationship between the fragment-size distribution and the explosive powder factor were not available. However, with the fragmentation parameters from the hand sampling data, the equations of Sirotuyuk and the Rozin-Rammler equation could be applied. The calculated fragment-size distribution compositions were compared with the fragment size distribution obtained by hand sampling.

The reciprocal mean fragment diameter $D_{av}$ calculated using equation (23) for the blasting at the limestone quarry is:

$$D_{av} = \frac{6}{S_{NA}} = 0.089 \text{ m}$$

Where

$$S_{NA} = 67.42 \text{ m}^2$$

From equation (25)

$$X_e = \frac{0.089 - 0.021}{0.26} = 2.62 \text{ cm}.$$  

The index of fragmentation $n$ was obtained from equation (32)

$$n = -\frac{0.663}{-\log \frac{0.25}{0.0262}} = 0.677$$

Equation 28 (Rozin-Rammler) was used to get the fragment-size distribution in the form

$$R_i = 100 e^{-\left(\frac{X_i}{X_e}\right)^n}$$

$i = 1, 2, 3, \ldots, 24 \text{ in.}$
The followings results were obtained:

<table>
<thead>
<tr>
<th>Sizes range (inches)</th>
<th>0 - 2</th>
<th>4 - 6</th>
<th>4 - 8</th>
<th>8 - 12</th>
<th>12 - 16</th>
<th>16 - 24</th>
</tr>
</thead>
<tbody>
<tr>
<td>From Sirotuyk and Rozin-Rammler equations</td>
<td>58.46</td>
<td>20.92</td>
<td>13.4</td>
<td>3.48</td>
<td>1.059</td>
<td>0.409</td>
</tr>
<tr>
<td>From hand Sampling</td>
<td>58.25</td>
<td>29.50</td>
<td>9.55</td>
<td>2.50</td>
<td>0.90</td>
<td>0.25</td>
</tr>
</tbody>
</table>

It is felt that a better hand sampling could give a closer result. Only one sample was taken whereas the Sirotuyk experiments included over one hundred.
6. VIBRATION FROM BLASTING

6.1 Introduction

Vibration is a by-product of blasting which affects the neighborhood; it may damage buildings or other structures and presents an annoyance to people. Therefore it is important to be able to minimize it.

There have been many investigations as to the type and magnitude of motion due to blast-generated waves necessary to cause damage to buildings. Safe limits which have been defined in terms of acceleration, energy, or a combination of amplitude and frequency of vibratory motion have been widely tested in practice.

Vibration due to normal use in different types of buildings have been analyzed and compared with vibration in the same buildings from blast-generated waves.

One of the most recently published studies on this topic is a paper written by members of the United States Bureau of Mines. This paper presents the results of an investigation of the problem of vibration from quarry blasting and the effects that these vibrations have on residential structures. It is an empirical approach, done by collecting and analyzing data from blasting records, such as distances,
explosive charges and peak particle velocities.

The principal purpose of the investigation was to obtain a method for making reasonable estimates of the vibration levels produced by quarry blasting of different sizes at various distances in diverse rock and soil conditions.

The final criteria are summarized in the following quotation:

If one or more of the three mutually perpendicular components (radial, vertical, and transverse) of the vibration in the ground near a structure have peak velocity less than 2 in./sec there is a low probability that damage to the structure will occur. The probability of damage increased as the vibration level increases above 2 in./sec, and the probability of damage decreases as the vibration level falls below 2 in./sec.

Also, the presumptiveness or likelihood involved in the approach is pointed out by:

Damage will not occur necessarily because a vibration level of 2 in./sec is exceeded. Figure 1 summarizes all of the published data where the vibration level was about 2 in./sec and no damage to the structure was detected. Also just because the vibration level is below 2 in./sec does not mean that damage will occur in some structures. Poorly constructed structures or structures previously stressed from settlement or unstable soil condition can be damaged by low vibrations that normally occur in a structure.
FIGURE 6 RELATIONSHIP BETWEEN THE DISPLACEMENT AND FREQUENCY WITH THE DAMAGE CRITERION
SUPERIMPOSED (After Duvall and Devine)

@ LANGEFORS DATA
* EDWARDS AND NORTHWOOD DATA
+ U.S.B.M DATA
Through a statistical analysis of the data, it was found that

1) Damage correlates best with particle velocity of the ground

2) A relationship exists between some particular values of the particle velocity and either major or minor damage. On the average major damage (defined as serious cracking and fall of plaster) occurs at a peak particle velocity of 7.6 in/sec and minor damage (defined as opening of old cracks and creation of new fine cracks) occurs at 5.4 in./sec.

3) A reasonable separation line between a relatively safe zone and a probable damage zone, corresponds to a particle velocity of 2 in./sec.

4) The vibration level for any given quarry blast can be predicted using amplitude and frequency data from past shots.

5) A general propagation equation relating the explosive used and the distance to any point where the vibration level is desired can be developed.

6) A graphical method of determining the safe blasting procedures from a vibration standpoint was given.

All of these conclusions have been used in the evaluation of the blasting operations at the limestone quarry, and to hopefully resolve the problems, if any, caused from ground motion at Dewey Rocky Mountain Cement Plant, Lyons, Colorado.
6.2 **Particle velocity, scaled-distance equation**

The U.S.B.M. (20) has determined that a general propagation equation exists, relating peak particle velocity of the ground, weight of explosive per delay, and distance from the blast to the point where the vibration determination is needed.

The general form of the equation is:

\[
V = H \left( \frac{D}{W^{0.5}} \right)^\phi
\]  

(33)

where:

\[
\begin{align*}
V & = \text{peak particle velocity of the ground vibration (in./sec)} \\
D & = \text{distance from shot (ft.)} \\
W & = \text{weight of explosive used per delay (lbs.)} \\
H, \phi & = \text{constants which are different for each place.}
\end{align*}
\]

The U.S.B.M. also indicates a procedure to derive such an equation using the method of least squares. The constants \( H \) and \( \phi \) are obtained from the straight fit of a log-log plot of the experimental points of the particle velocity produced by every shot in in./sec and the scaled-distance parameter, obtained by dividing the distance \( D \) in ft. by the square root of the explosive weight per delay used.

In the particular case of the limestone quarry, data from 21 shots, including the weight of explosive per delay, the respective figures for ground displacement and frequencies, were used to get the particle velocity and scaled distance.
These parameters, for every shot, were plotted on log-log paper. A least square fit was done and the slope of the straight line was obtained together with \( H \), the intersection of the line with the y-axis (velocity-axis).

The resulting equation is:

\[
V = 634 \left( \frac{D}{W} \right)^{-1.90}
\]

(34)

where:

- \( V \) = peak particle velocity, (in./sec).
- \( D \) = distance from shot to the recorder, (ft.).
- \( W \) = weight of explosive per delay, (lbs).
FIGURE 7 LIMESTONE QUARRY PARTICLE VELOCITY CURVE

Limestone Quarry Curve

EQUATION

\[ v = 634 \left( \frac{D}{W} \right)^{1.9} \]

+ = S. Dist., Part. Vel.)
This particular equation used to calculate peak particle velocity of the ground motion gives results accurate enough to be useful for the practical work. It may be considered a preliminary equation since it can be improved as more data becomes available.

The seismological records used to obtain the equation were those registering the vertical component of the ground vibration. Langefors (6) states:

"In simultaneous measurement the three components of the ground vibration it has been found that these are generally of the same order of magnitude. In many cases it is quite sufficient to register only the vertical or the longitudinal component, but now and then a check should be made of both of them. According to Northwood the transversal component adds little to the information."

Only the largest values among different velocities for the same scaled-distance were chosen so that a very conservative equation would result. This means that the equation will give higher velocity values for most of the calculations, compared with velocities obtained from seismological records.

Inspection of the equation in its graphical form, fig. (7) reveals that 4500 pounds of explosive per delay is the maximum amount to be used if a peak particle velocity less than 1 in./sec is desired at a distance of 2000 ft.

Table 4 shows particle velocities of ground vibration calculated with amplitudes and frequencies from seismological records, and those calculated with the derived equation and respective differences are shown in Table 5.
Table 4

PARTICLE VELOCITIES FROM SEISMOLOGICAL RECORDS AND RESPECTIVE VELOCITIES CALCULATED WITH DERIVED EQUATION

<table>
<thead>
<tr>
<th>FREQUENCY, CYCLES/SECOND</th>
<th>DISPLACEMENT, INCHES</th>
<th>PARTICLE VELOCITY, IN./SEC (1)</th>
<th>DISTANCE FROM SHOT, FT.</th>
<th>WEIGHT EXP. PER DELAY, LBS</th>
<th>PARTICLE VELOCITY IN./SEC (2)</th>
</tr>
</thead>
<tbody>
<tr>
<td>9.3</td>
<td>0.0038</td>
<td>0.0368</td>
<td>1500</td>
<td>125</td>
<td>0.0572</td>
</tr>
<tr>
<td>11.9</td>
<td>0.0012</td>
<td>0.0897</td>
<td>1200</td>
<td>125</td>
<td>0.0874</td>
</tr>
<tr>
<td>8.6</td>
<td>0.0019</td>
<td>0.1027</td>
<td>1500</td>
<td>280</td>
<td>0.1230</td>
</tr>
<tr>
<td>10.0</td>
<td>0.0015</td>
<td>0.0942</td>
<td>2560</td>
<td>650</td>
<td>0.0992</td>
</tr>
<tr>
<td>11.6</td>
<td>0.0013</td>
<td>0.0948</td>
<td>1310</td>
<td>228</td>
<td>0.1309</td>
</tr>
<tr>
<td>10.2</td>
<td>0.0037</td>
<td>0.2371</td>
<td>1265</td>
<td>400</td>
<td>0.2387</td>
</tr>
<tr>
<td>11.4</td>
<td>0.0025</td>
<td>0.1791</td>
<td>1250</td>
<td>300</td>
<td>0.1858</td>
</tr>
<tr>
<td>11.1</td>
<td>0.0012</td>
<td>0.0837</td>
<td>1610</td>
<td>275</td>
<td>0.1057</td>
</tr>
<tr>
<td>31.3</td>
<td>0.0007</td>
<td>0.1377</td>
<td>1205</td>
<td>204</td>
<td>0.1381</td>
</tr>
<tr>
<td>9.7</td>
<td>0.0025</td>
<td>0.1524</td>
<td>1260</td>
<td>315</td>
<td>0.1917</td>
</tr>
<tr>
<td>10.0</td>
<td>0.0044</td>
<td>0.2765</td>
<td>1160</td>
<td>286</td>
<td>0.2046</td>
</tr>
<tr>
<td>11.6</td>
<td>0.0039</td>
<td>0.2843</td>
<td>1080</td>
<td>300</td>
<td>0.2453</td>
</tr>
<tr>
<td>10.0</td>
<td>0.0043</td>
<td>0.2702</td>
<td>1100</td>
<td>225</td>
<td>0.1802</td>
</tr>
<tr>
<td>11.9</td>
<td>0.0023</td>
<td>0.1720</td>
<td>1090</td>
<td>212</td>
<td>0.1733</td>
</tr>
<tr>
<td>4.6</td>
<td>0.0007</td>
<td>0.0202</td>
<td>6300</td>
<td>650</td>
<td>0.0180</td>
</tr>
</tbody>
</table>

Particle Velocity (1) = \(2\pi \times \text{Amplitude} \times \text{Frequency}\)

\[
\text{Particle Velocity (2)} = 633.4 \times \frac{\text{Exp. Weight}}{\text{Distance}} \times 0.5 \times 1.90
\]
### Table 5

**Comparison Between Particle Velocities**

<table>
<thead>
<tr>
<th>FROM SEISMOLOGICAL DATA, in./sec</th>
<th>FROM DERIVED EQUATION, in./sec</th>
<th>DIFFERENCE, in./sec x 10</th>
</tr>
</thead>
<tbody>
<tr>
<td>0.0468</td>
<td>0.0572</td>
<td>10</td>
</tr>
<tr>
<td>0.0897</td>
<td>0.0874</td>
<td>2</td>
</tr>
<tr>
<td>0.1027</td>
<td>0.1230</td>
<td>20</td>
</tr>
<tr>
<td>0.0942</td>
<td>0.0992</td>
<td>6</td>
</tr>
<tr>
<td>0.0948</td>
<td>0.1309</td>
<td>36</td>
</tr>
<tr>
<td>0.2371</td>
<td>0.2387</td>
<td>16</td>
</tr>
<tr>
<td>0.1791</td>
<td>0.1858</td>
<td>7</td>
</tr>
<tr>
<td>0.0837</td>
<td>0.1057</td>
<td>24</td>
</tr>
<tr>
<td>0.1377</td>
<td>0.1381</td>
<td>1</td>
</tr>
<tr>
<td>0.1524</td>
<td>0.1917</td>
<td>39</td>
</tr>
<tr>
<td>0.2765</td>
<td>0.2046</td>
<td>72</td>
</tr>
<tr>
<td>0.2843</td>
<td>0.2453</td>
<td>39</td>
</tr>
<tr>
<td>0.2702</td>
<td>0.1802</td>
<td>90</td>
</tr>
<tr>
<td>0.1720</td>
<td>0.1733</td>
<td>1</td>
</tr>
<tr>
<td>0.0202</td>
<td>0.0180</td>
<td>2</td>
</tr>
</tbody>
</table>
Table 6

EXPECTED VIBRATION

Distance from shot = 1500 ft

<table>
<thead>
<tr>
<th>Weight of explosive per delay, lbs</th>
<th>Scaled-Distance ft/lbs 0.5</th>
<th>Particle velocity in./sec</th>
</tr>
</thead>
<tbody>
<tr>
<td>50</td>
<td>212.1</td>
<td>0.024</td>
</tr>
<tr>
<td>300</td>
<td>86.6</td>
<td>0.131</td>
</tr>
<tr>
<td>500</td>
<td>67.1</td>
<td>0.213</td>
</tr>
<tr>
<td>950</td>
<td>48.7</td>
<td>0.393</td>
</tr>
<tr>
<td>1000</td>
<td>47.4</td>
<td>0.413</td>
</tr>
<tr>
<td>2000</td>
<td>33.5</td>
<td>0.797</td>
</tr>
<tr>
<td>2550</td>
<td>29.7</td>
<td>1.004</td>
</tr>
<tr>
<td>4000</td>
<td>23.7</td>
<td>1.541</td>
</tr>
<tr>
<td>5600</td>
<td>20.0</td>
<td>2.121</td>
</tr>
</tbody>
</table>

Distance from shot = 2000 ft

<table>
<thead>
<tr>
<th>Weight of explosive per delay, lbs</th>
<th>Scaled-Distance ft/lbs 0.5</th>
<th>Particle velocity in./sec</th>
</tr>
</thead>
<tbody>
<tr>
<td>50</td>
<td>353.6</td>
<td>0.009</td>
</tr>
<tr>
<td>300</td>
<td>144.3</td>
<td>0.050</td>
</tr>
<tr>
<td>500</td>
<td>111.8</td>
<td>0.081</td>
</tr>
<tr>
<td>1000</td>
<td>79.1</td>
<td>0.156</td>
</tr>
<tr>
<td>2650</td>
<td>48.6</td>
<td>0.394</td>
</tr>
<tr>
<td>6000</td>
<td>32.3</td>
<td>0.858</td>
</tr>
</tbody>
</table>

Tables of expected vibration were calculated for a range of distances from 100 to 6000 ft. and of 50 to 6000 lbs. of explosive charge per delay. The distance is in increments of 100 ft. and the charge per delay in 50 lbs. increment.

Table 6 is a sample of some of those values.
6.3 Graphical Method

This method to determine a safe scaled-distance for blasting from the U.S.B.M. standpoint, requires two steps:

a) To determine the velocity, scaled-distance curve for a particular site

b) To compare this curve with the general plot of curves determined by the U.S.B.M.

If the curve for that particular site falls on the left side of the general plot, a scaled-distance of $20 \text{ ft/}lb^{0.5}$ can be used. If the same curve falls on the right side of the general plot a scaled-distance greater than $20 \text{ ft/}lb^{0.5}$ should be used.

Sites with high vibration capabilities (their velocity curves are going to fall on the right side of the general plot) should use 40, 50, or more $\text{ft/}lb^{0.5}$. Using these values the probability is small of finding a site that produces a vibration level that exceeds 2 in./sec of particle velocity (a level considered safe by the Bureau).

At the Dewey Rocky Mountain Cement plant limestone quarry the scaled-distance should be more than $20 \text{ ft/}lb^{0.5}$ because its curves do not, as seen in fig. 8, fall on the left side of the general plot. As a matter of fact, the limestone quarry velocity crosses the U.S.B.M. plot, the upper part falling on the right side and the lower part on the left.
FIGURE 8 PARTICLE VELOCITY, SCALED-DISTANCE CURVE
AS COMPARED WITH THE U.S.B.M. GENERAL PLOT

Particle velocity in/sec

Scaled-Distance (ft./lb.)
According to this, the scaled-distance has to be over 20 ft./lb. If the maximum peak particle velocity is desired to be less than 1 in/sec. at 2000 ft., a safe scaled-distance should be 30 ft./lb. 0.5.

With this scaled-distance the probability of exceeding a particle velocity of 1 in/sec. is very small, so the recommended figure has already a safety factor compared with the safe level of 2 in./sec. given by the U.S.B.M.

The recommended scaled-distance means that no charges greater than 4500 pounds per delay should be used at any place closer than 2000 ft. to any building.

The safety factor mentioned appears necessary because a house more than 100 years old is in the neighborhood within a radius of 2000 to 5000 ft. of the working places.

In short, to set the limit of vibration at less than 1 in/sec. particle velocity is to account with a very safe vibration limit since the proved damage occurs at 5.76 in./sec. (fig.9-10) and the safe limit from the U.S.B.M. is 2 in./sec. Incidentally, the highest particle velocity ever reached by any of the limestone quarry blasting is 0.28 in./sec, at 2000 ft., far below the safe level recommended by specialists.

However, there is another situation to be considered. That is the subjective sensation of the ground motion. (see fig. 11).

From Langefors (6) one finds the following
"A great deal of irritation which arises between the blaster and his environment is due to the fact that a layman is very apt to acquire a wrong conception of the risk of damage. Another aspect is that the blasting is regarded in itself as something disagreeable.

Reiher and Meister (6) have studied human reaction to vibration. Their results are reproduced in fig. (11). The lines indicated when the effect has been considered to be scarcely noticeable (a), clearly noticeable (b), irritating (c) and disagreeable (d).

This information is of great value in dealing with complaints that the ground vibration are injurious. The diagram shows that a man normally reacts strongly long before there is a reason to fear damage to his house. In fact a very large share of common complaints have proved to be unwarranted. If people are told that suspicion is aroused long before any damage can possibly be expected. This should elucidate matters and reduced the subjective reaction. But even so, there may still be discomfort, which would be avoided as much as possible. The number of blast can often be reduced, due warning of them given, and above all, blasting at night should be avoided.

In comparison with other causes of discomfort in a modern city, we may well ask whether a blast carried out in a reasonable manner at a reasonable time may not be regarded by a normal human being as a healthy and distinct tone in a blurred cacophony."

According to the Reiher and Meister criterion, (6) 50% of the blasting done at the limestone quarry falls between the ranges "irritating" and "disagreeable" if they are felt from 2000 ft. This explains the complaints from the neighborhood but it does not mean that any danger is involved. The irritating reaction is reached with particle velocities from 0.80 to 0.63 in./sec. velocities 3 to 20 times less than the safe level of 2 in./sec. from the U.S.B.M.
Velocities that should give no reaction have occurred with the rest of the shots, but even these have been considered as disagreeable or irritating by some neighbors. Professor Duvall (21) advises a particle velocity less than 0.4 in./sec. should not be considered as cause for complaint.

Therefore, this velocity value is going to be recommended to hopefully avoid complaints. Such a velocity requires that no more than 2400 pounds of explosive per delay should be used at 2000 ft. from the surrounding dwellings.

No vibration and human reaction to blasting would require zero pounds of explosive, meaning no blasting at all, and therefore no mining. There is presently no other economic method to break rocks.
Particle velocity in./sec.

7.6 Velocity able to produce major damage

5.4 Velocity able to produce minor damage

3.0 Proved velocity without damage

2.0 Velocity recommended by U.S.B.M

1.0 Velocity recommended for Limestone Quarry

0.4 Velocity recommended to avoid complaints

0

FIGURE 9 COMPARATIVE VELOCITY OF GROUND MOTION AND ITS CLASSIFICATION IN ORDER OF DANGER.
Amplitude (in.)

V = 7.6 in./sec Average velocity causing major damage

V = 5.0 in./sec Average velocity causing minor damage

V = 1 in./sec Recommended according to ground motion constant

V = 0.4 in./sec Recommended to avoid complaints

SAFE VIBRATION LEVEL (U.S.P.M.) V = 2 in./sec

FIGURE 10 COMPARATIVE AMPLITUDE FREQUENCY CURVES CRITERIA AND RECOMMENDED VALUES FOR THE QUARRY.
Subjective estimate of ground vibration
(after Reihor and Meister)

a=slightly noticeable  b=clearly noticeable  c=irritating
d=disagreeable  f=recommended level to decrease the
subjective reaction to blasting  r=level recommended
according vibration criteria  h=level recommended by
U.S. Bureau regarding vibration
6.4 Future critical spots

Some buildings are located as close as 550 ft. from future shots. These places determine a sort of critical spot and are: the primary crusher located at 550 ft. from the closest shot at Pit C; dwellings located at 650, 950, and 2100 ft. from future shots at Pit B.

For these future shots a careful blasting pattern should be designed because the possibility exists that a shot might alter the level of the roll crusher axis, and the dwellings are permanent sources of complaints, even to the lowest vibration levels that the normal shots produce.

Langefors (6) states that a relationship exists between time delay, number of holes, frequency, and amplitude. The relationship is made clear in the following quotation:

"The ground vibrations which are caused by a single charge acquire a maximum amplitude $A$ only after one or several previous minor deflections. The vibration is ordinarily of relatively short duration, and in most cases only three full vibrations can be expected to have an amplitude greater than $A/2$; all the others can more or less be ignored. This means that at intervals greater than $3T$ ($T$ is time for a full period =1/frequency) it can be reckoned that there is no collaboration between two different shots."

The average frequency obtained from shots at the limestone quarry is 11 cps., therefore the time for a full period is 90 msec. The non-collaborating situation could be reached with delays of 270-300 msec., which is far greater than the 9 msec. delay that is used to get the best fragmentation with the
burden established for the blasting of the benches.
Fragmentation is therefore to be lower than the normal one.

Langefors(6) also points out that the conditions became more complicated if the interval is shorter than 3T as will be the cooperation between the different waves system.

He also states:
An interference effect is obtained which implies that when the delay time \( \tau \) is as great as the vibration time T, or an integral multiple thereof, cooperation is obtained between the different delays so that the vibration effects add-up. This applies when \( H \) is an integer in the ratio

\[ \tau = H \frac{T}{n} \]

On the other hand, when \( H \) is an odd number of semivalues the different wave system extinguish or weaken one another. This takes place almost entirely when \( H = \frac{1}{2} \).

Using this and by knowing the average frequency at the quarry, a delay time of 45 msec can be recommended for those critical spots that were mentioned earlier, since

\[ \tau = H \frac{T}{n} \]
\[ \tau = 90 / 2 \]
\[ \tau = 45 \text{ msec.} \]

With this time per delay, vibration is going to be less and values of particle velocities will be lower than those given earlier in the tables which apply to blasting with 9 msec. delays.
According to Langefors, (6) in order to decrease vibration levels an arranged interference can be obtained if the spread of the round in time is distributed over one or a number of full periods of the vibration. This means that:

\[ \tau n = k T \]  

where:

- \( n \) = number of intervals
- \( \tau \) = delay time
- \( T \) = time for a full period
- \( k \) = integer factor

The condition that must be fulfilled is that the ratio between \( k \) and \( n \) is not an integer.

The time for a full period at the limestone quarry is 90 msec, and the delay time that best matches the blasting conditions is 9 msec. Therefore the equation for this case becomes:

\[ \tau n = k T \]

\[ 9 n = 90 k \]

\[ n = 10 k \]

Hence the number of intervals has to be any multiple of 10 to get the lowest vibration level by weakening the amplitudes by interference.
For values of \( n = 10, 20, 30, 40, \ldots \) the respective values of \( k \) are 1, 2, 3, 4, \ldots. The condition that \( k \) has to be an integer, but not the ratio \( k/n \), is so fulfilled.

Increasing the size of the shot by increasing the number of delays will decrease the vibration if the recommended values are used.

A decrease of the amplitude of the ground motion by doing so was shown in several experimental tests. The delay time used was 9 msec, and the delay numbers were 3, 5 and 7.

With these \( n \) values, the resulting \( k \) was:

\[
T' = \frac{T}{n} \Rightarrow k = T' = \frac{9 \times 3}{90} = 0.3 \text{ or } 0.5 \text{ or } 0.7
\]

For the \( k \) values 0.3 to 0.5, higher values of amplitudes were obtained. This is explained by the fact that \( k \) is not an integer.

For the \( k \) value of 0.7 a lower value of vibration was obtained. In this case \( k \) approached the value of an integer. All the shots in which the \( k \) values were integers or approached integers gave lower amplitude values than those with non-integer values. The vibration values were measured practically at the same distance for all shots and the following results were obtained.
**TABLE RELATIONSHIP BETWEEN AMPLITUDE AND K VALUE**

<table>
<thead>
<tr>
<th>WEIGHT OF EXPLOSIVE LBS.</th>
<th>EXPLOSIVE PER DELAY LBS.</th>
<th>NUMBER OF DELAYS</th>
<th>K VALUE</th>
<th>AMPLITUDE IN.</th>
<th>FREQUENCY CPS</th>
<th>PARTICLE VELOCITY IN./SEC</th>
</tr>
</thead>
<tbody>
<tr>
<td>675</td>
<td>125</td>
<td>7</td>
<td>0.7</td>
<td>0.0012</td>
<td>11.9</td>
<td>0.090</td>
</tr>
<tr>
<td>1900</td>
<td>156</td>
<td>9</td>
<td>0.9</td>
<td>0.0014</td>
<td>8.5</td>
<td>0.075</td>
</tr>
<tr>
<td>2100</td>
<td>160</td>
<td>24</td>
<td>2.4</td>
<td>0.0039</td>
<td>11.6</td>
<td>0.284</td>
</tr>
<tr>
<td>1200</td>
<td>212</td>
<td>11</td>
<td>1.1</td>
<td>0.0023</td>
<td>11.9</td>
<td>0.172</td>
</tr>
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<td>675</td>
<td>225</td>
<td>5</td>
<td>0.5</td>
<td>0.0014</td>
<td>10.0</td>
<td>0.276</td>
</tr>
<tr>
<td>1700</td>
<td>228</td>
<td>13</td>
<td>1.3</td>
<td>0.0013</td>
<td>11.6</td>
<td>0.095</td>
</tr>
<tr>
<td>5150</td>
<td>250</td>
<td>50</td>
<td>5</td>
<td>0.0037</td>
<td>10.2</td>
<td>0.237</td>
</tr>
<tr>
<td>*5750</td>
<td>280</td>
<td>20</td>
<td>2.0</td>
<td>0.0019</td>
<td>8.6</td>
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</tr>
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<td>825</td>
<td>297</td>
<td>3</td>
<td>0.3</td>
<td>0.0044</td>
<td>10</td>
<td>0.276</td>
</tr>
<tr>
<td>4050</td>
<td>300</td>
<td>17</td>
<td>1.7</td>
<td>0.0016</td>
<td>5.3</td>
<td>0.053</td>
</tr>
</tbody>
</table>

* This shot had several failures.
6.5 Vibration Dip Characteristic

Jenkins (22) has suggested that the displacement produced by blasting increases in proportion to the explosive blasted until a certain point at which it begins to decrease to a value approximately 30% lower than the peak at which time it begins to increase again.

This decrease in amplitude has been called the "dip vibration characteristic." Jenkins suggests that a knowledge of this characteristic is important in blasting practice in order to use the right amount of explosive to get better fragmentation and explosive performance while decreasing vibration amplitude. An explanation for the occurrence of waste of energy is given in the following terms:

"Blast generated earthborne vibration transmitting beyond the shot point area classifies as residual energy. This residue may be considered as being composed of two parts: the irreducible minimum and waste. Since for every action there must be an equal and opposite reaction, the explosive forces acting to dislodge the earth materials from the blasting face also exert a force against the earth mass behind the shot point. This force compresses the earth mass giving rise to the elastic waves, the motion of which creates the phenomenon known as vibration. When the amplitude of these elastic waves is excessive, it may be presumed that such excess is the result of a part of the source energy having been wasted."

When the amplitude is or is not excessive can be determined if enough seismic data are available to plot a curve through points of minimum amplitudes. Such a curve is approaching the irreducible minimum. Amplitudes over these minimum may be presumed to represent waste."
From a large volume of information, data earlier considered as inconsistent (observed in the records of a few random and widely scattered seismological records) became basic knowledge to determine the existence of dip vibration. It has been proved to be measurable and useful for obtaining better explosive performances and lesser by-products (vibration, noise).

The results of cases histories follow:

<table>
<thead>
<tr>
<th>Operation</th>
<th>Explosive load (lbs)</th>
<th>Vibration dip lies between (in.)</th>
<th>Drop percentage</th>
</tr>
</thead>
<tbody>
<tr>
<td>Small</td>
<td>up to 200</td>
<td>0.0029 and 0.0021</td>
<td>26</td>
</tr>
<tr>
<td>Medium</td>
<td>1000 to 4000</td>
<td>0.013 and 0.005</td>
<td>38</td>
</tr>
<tr>
<td>Large</td>
<td>over 10000</td>
<td>0.006 and 0.004</td>
<td>33</td>
</tr>
</tbody>
</table>

According to Jenkins (22), burden and spacing as well as the optimum of holes and charge per hole can be determined by seismographic analysis. Blast efficiency it is claimed may be improved by 50% if the analysis is performed. The analysis should be strictly controlled in its field application. Burden and spacing should be measured with tape, not pacing distances, powder ratio for each hole should by computed in advance, not following the blast, and rules of the thumb method should be discarded. Supervision should avoid the words "sinks back into routine procedures, old habits again predominate and details are forgotten."
FIGURE 12 Illustration of the "vibration dip"

Dip representing increased efficiency attainable through seismic control methods.
FIGURE 13 Maximum and minimum amplitudes for explosive loadings of different ranges.

Small operation

Medium operation

Large operation

Displacement, in.

Explosive weight, Lb.

Weight of explosive, In.
7. DAMAGE FROM AIR-BLAST

7.1 Introduction

Recent research (23) has shown that window panes fail before any structural damage, even though the damage most frequently mentioned in air-blast complaints is cracked plaster. In fact, a poorly mounted window pane can fail when subjected to peak overpressure as low as 0.1 psi. Damage to properly mounted window panes can be expected when the air-blast pressures approach 1 psi.

Also it is pointed out that an air-blast pressure of only 0.03 psi can cause a loose window sash to rattle against the window frame; no actual damage occurs in spite of the loud noise produced. The loud noise is a source of annoyance and hence complaints.

If the air-blast pressures exceed a level of 1 psi, cracking of plaster will develop.

Environment, state of repair, and construction methods are some of the factors which can contribute to plaster damage and its evaluation is very difficult.

7.2 Prediction of Air-blast Pressure

Perkins and Jackson (24) developed a monogram for predicting air-blast pressure, as a function of distance for a surface blast. It is, however, pointed out here that a large reduction in the level of the air-blast would be expected when the explosive is placed in drillholes and confined by stemming. The monogram in fig. (14) shows the distance-pressure relation from explosives blasted on surface.
Weight of explosive blasted on surface

a = 10,000 pounds
b = 4,000 "
c = 2,000 "
d = 1,000 "
e = 400 "
f = 200 "
g = 100 "
h = 40 pounds
i = 20 "
j = 10 "
k = 4 "
l = 2 "
m = 1 "

FIGURE Summary of air-blast Pressure vs. Distance
(after Perkins and Jackson)
The U.S.B.M. has developed a nomogram to estimate the reduction of air-blast pressure when the explosive is buried. The nomogram shows the relationship of the pressure to distance and to depth of burial for spherical charges. Recent data (23) from multiple hole quarry blast suggests that the depth of burial for spherical charges can be replaced by the burden when cylindrical charges are used (this is the case in quarry blasting). Therefore the nomogram can be used to estimated an average air-blast pressure for quarry shots when normal conditions are found.

This nomogram (as given in fig.15) has been used to obtain tables for predicting air-blast pressures from shots at the limestone quarry.

From the nomogram (figure 15) the equations of the pressure scaled-distance curves for differen scaled-depth can be determined.

In general form they may be expressed as:

\[ P = F \left( \frac{D}{W} \right)^{-b} \]  \hspace{1cm} (37)

where:

\( P \) = air-blast pressure (psi)
\( F \) = intersection of the curve with the pressure axis (Y-axis)
\( D \) = distance from the shot (ft)
\( W \) = explosive weight per delay (lbs)
Figure 15: Summary of Air Blast Data from Quarry Blasts (U.S.B.)

(after Duvall and Devine)
The scaled-distance is calculated by dividing the distance by the cube root of the explosive weight. The scaled-depth is obtained by dividing the burden by the cube root of the explosive weight.

If the explosive charge per delay varies from 100 to 5000 pounds and the burden is 8 ft., the range of the scaled-depth is 0.467 to 1.730. For a range of distance from 100 to 7500 ft. and the already mentioned explosive range, the scaled-distance varies from 6 to 1616 (ft./lbs.1/3).

The equations derived for 5 values of scaled-depth are:

1. For 0.50 scaled-depth:

\[ P = 18.6 \left( \frac{D}{W^{1/3}} \right) - 1.26275 \]  \hspace{1cm} (38)

2. For 0.75 scaled-depth:

\[ P = 6.4 \left( \frac{D}{W^{1/3}} \right) - 1.1918 \]  \hspace{1cm} (39)

3. For 1.00 scaled-depth:

\[ P = 2.6 \left( \frac{D}{W^{1/3}} \right) - 1.2644 \]  \hspace{1cm} (40)
4. For 1.25 scaled-depth:

\[ P = 0.83 \left( \frac{D}{W^{1/3}} \right) - 1.063 \quad (41) \]

5. For 1.50 scaled-depth:

\[ P = 0.28 \left( \frac{D}{W^{1/3}} \right) - 1.02041 \quad (42) \]
For scaled-depth values different from those used above the next lowest scaled-depth will be chosen. For instance, if the scaled-depth value was 0.66, the equation used to calculate the air-blast pressure for this value would be that derived for an scaled-depth of 0.50. An exception to this criteria was that for values lower than 0.50. The equation derived for a scaled depth of 0.50 was used.

According this criteria, in general, table values represent an upper limit of air-blast pressure expected for every shot.

7.3 Damage Criterion

The damage criterion should be set at 1030 psi. for all shots. From the table this means that 1200 lbs. of explosive per delay should be the maximum amount shot if a dwelling is located at 1000 ft. For this case an air-blast pressure of .028 psi. will be expected.

Should the shot be located 500 ft. from the nearest dwelling; the maximum amount of explosive is 500 lbs. The expected air-blast pressure will be .024. For distances greater than 3000 ft. the amount of explosive per delay can be 5000 lbs. Expected air-blast pressure are always going to be less than .026 psi.

All those air-blast pressures are less than those required to poorly mounted window pane fail and less than that needed to make a loose window sash rattle against the window frame.
Since the cracking of plaster occurs at pressures above 1 psi, the damage criterion chosen is lower than this by a factor of 33. Therefore, even under the most unfavorable conditions that kind of damage has a very small chance of occurring. Moreover, a good stemming and covered millisecond connectors and detonating fuse are able to decrease the airblast pressure by a factor of 10 or more.

Although the damage criterion chosen is very conservative, this is necessary because other parameters such as atmospheric conditions play some role regarding the airblast pressure. This parameter has not been taken into account; therefore, a safety factor is the only resource of the blasting technique to approach the problem.

Blasting with poor atmospheric conditions should be avoided. The Dupont's blaster handbook defines poor atmospheric conditions as "those days when the air is relatively still...usually foggy, hazy, or smoky."

The same handbook points out:

An indication of unfavorable conditions may be drawn from the behavior of smoke from a nearby smoke-stack. If the smoke fans out horizontally after the initial rise with little looping or vertical motion evident as the smoke moves away from the stack a poor condition exists and very likely, a temperature inversion. Also, clear, somewhat hazy days with fairly constant temperatures and possible very light winds.
Winds, if they are not directed toward population centers, are an indication that blasting can be carried out since they tend to prevent the formation of inversions. This inversion (air becomes warmer as the altitude increases), increases the sound velocity and reduces the chance of transformation of the sound in heat so it is back to earth, producing louder noises in certain areas.

Conversely, at normal conditions (air temperatures decrease as the altitude from the earth's surface increases) a decrease in sound velocity results. Bending of sound rays is upward and the noise is directed away from the earth's surface.

Favorable or normal conditions for blasting are clear to partly cloudy skies with fleecy clouds and relatively warm daytime temperatures or, cloudy days with rapidly changing winds, perhaps accompanied by brief showers.

DuPont's Blaster's Handbook (19) covers these situations and the "mixed" ones based on experiments made in the company research facilities and tests carried out in actual quarries or open-pits.
Noise from blasting at limestone quarry

In spite of the low vibration level maintained in all the past shots, complaints have arisen from all of them. As the general case, it is the associations of noise with vibration that is the culprit of the situation.

Noise levels ranging from 85 to 114 decibels have been measured from some of the shots at those places from which complaints have been forthcoming.

An evaluation of these figures can be made by comparing these noises with those from other sources familiar to people living in this country and which they consider as a little share of annoyance to pay for progress.

a) 1 decibel is the lowest sound to be heard in a silent place

b) 55-60 decibels is the range able to awake someone

c) 75 " is the noise level from traffic on a relatively quiet street

d) 75-90 " is the level sought as a limit if industrial noise

e) 96 " noise from a household blender

f) 114 " noise from some lawn mowers

g) 114 " noise from electric guitars playing rock music
h) 115 decibels noise from sport-car or motocycle
i) 120 " noise able to cause pain
j) 150 " noise from a Jet airplane

The shots at the limestone quarry are of the same intensity as items d, e, f, g. Noise from some of the shots were actually below that of a blender.

The shot noises are of short duration (of the order of one or two seconds) and shorter by a factor of 10 to 1000 as compared to the blender or a lawn mower.

In order to improve the Public Relations situation the following suggestion can be made:

1. If shots have made at distances less than 1000 ft. from the nearest dwelling, Low Energy Detonating Cord (LEDC from Dupont) should be used. This detonating device has 96% less charge explosive than the standard promacord so 150 ft. of LEDC makes no more noise than one electric blasting cap or two inches of primacord. Surface noise can be almost completely eliminated. For distances from 1000 to 2000 ft. blasting can be carried out with covered millisecond connectors and covered detonating fuse. For larger distances no major changes are suggested but only covering the millisecond connectors.

2. Measurements should be taken and evaluated for several shots before a permanent policy about noise is developed.

Several tests has already been done at limestone quarry (September 29 and October 20, 1970). The conclusions were
that by covering the detonating fuse to a depth of 18 in. with dirt one could decrease the sound intensity by a factor of 1000 (decibel levels decreasing from 106 to 70). However this is a time-consuming and expensive job. Further investigation is necessary to determine the effect on the covering depth and the practical depth on the sound level.

Electric blasting caps might be used to decrease the noise levels from the surface blast of the detonating fuse but varying weather conditions around the year at the quarry do not lend themselves for such a change of initiating devices.

The risk involved with the use of explosive is high compared with another kind of job. From the safety point of view, risks of premature explosion increases when a high level of static electricity, snow storms or lightning are present if any kind of electrical cap is being used. Therefore at the quarry the safest method at present for initiating the explosive charges is already in use.

From the economic point of view casualties are quite expensive and the public image of the company can suffer a deteriorating impact.

No warranted reason can be seen to justify taking chances.

The right of a dwelling owner to protect his privacy or enjoy his residence is as respectable as the right of the men working in the explosive crew to have the safest method of working.
CONCLUSIONS

By comparing both the presently used and the proposed blasting system, the following statements can be made:
1) The drilling patterns used with all patterns tried were alike regarding drilling rates and costs.
2) Using the proposed blasting system (8 x 16) the fragmentation was improved, the broken rock is easier to load and better performance was obtained from the explosives charges. Explosive consumption was reduced by 50 per cent.

Regarding fragmentation and according table (8) the following results can be given:

<table>
<thead>
<tr>
<th>Size</th>
<th>Percentage of fragments</th>
<th>Percentage of Total Ton.</th>
</tr>
</thead>
<tbody>
<tr>
<td>minus 4 in.</td>
<td>87.75</td>
<td>19.74</td>
</tr>
<tr>
<td>minus 12 in.</td>
<td>98.85</td>
<td>71.00</td>
</tr>
<tr>
<td>minus 24 in.</td>
<td>100</td>
<td>100</td>
</tr>
</tbody>
</table>

Fewer boulder were left for secondary blasting with only approximately 0.5% of the total tonnage rejected for loading (500 ton. out of 100,000).

Nine milliseconds delays were shown better than the previously used 17 milliseconds regarding fragmentation and backbreak.

The change made in the direction of shooting (to N 38° W) was thought to be the predominant factor in achieving the better results.
3) No significant difference in loading and hauling time were found for the various patterns tried.

4) In spite of the fact that a smaller amount of explosive was used for the final blasting pattern than for the original, no difference in energy consumption in primary crushing was observed. A statistical treatment of the crushing data shows a better distribution of the fragments around a central size value with the proposed blasting system if energy of crushing is considered to be proportional to feeding size (fig. 20).

Scattering of sizes of fragmented rock feeding the primary crusher was higher for the original pattern (fig. 21).

5) Fragmentation analysis can be made by hand sampling (to get the new formed surface after blasting) and further use of this figure with Sirotuyk's and Rozin-Rammler equations. This suggested method does not need the use of "photoplanograms" (not used in this country yet) and it can be worked out at any place without any special equipment.

6) Ground vibration and air blast pressure are two by-products of blasting which are directly proportional to the weight of explosive per delay. With the 50% reduction in the explosive used per ton in the new pattern, the magnitudes of both undesired by-products are reduced assuming the same number of holes per delay is used. For the same magnitudes of these quantities a larger number of holes per delay can be blasted.

7) No conclusions regarding the fragmentation index can be made because the loading time does not vary according size of the broken rock when the ratio bucket size / fragment size is too large.
9. **RECOMMENDATIONS FOR FUTURE STUDIES**

The following recommendations for future studies can be made on the basis of the work performed in this thesis:

1. A analysis of fragmentation of the rock broken by blasting using hand sampling and a statistical treatment of the data of common minerals (copper, iron) with related relationship to explosives consumption should be done to enable forecasting of the size distribution.

2. A complete study of Sirotyuk's photoplanimetric method to evaluate fragmentation, to determine the average diameter of broken rock, and to further design blast according to maximum sizes of loaders and transportation devices should be performed.

3. A study of the economic relationship between cost for breaking rock by means of explosives and mechanical primary crushing to establish the cheapest way, specially for copper and iron ores should be made.

4. An examination of "dip" of vibration curves obtained by seismological recording of blasting should be done to obtain optimum utilization of energy released from explosives. This waste of energy is transformed into undesired by-products such as ground vibration, air-blast pressures, and noises.
5. A determination of the relationship between bucket-size and fragment-size to obtain the best loading time and the lowest cost of crushing should be performed.

6. Additional studies are needed for the prediction of noises and air-blast pressures for predetermined weather conditions and blasting parameters.

7. A testing program on the use of water stemming (water in plastic bags) may lead to better confinement and reduction of dust, noise and vibration from blasting.
LITERATURE CITED


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10. Grenon A., Perforation mecanique et abatage des roches (Drilling and blasting of rocks), Eyrolles, France, (1933)


18. ___________, Crushing and grinding calculations, Allis-Chalmers.

19. Sirotyuk, N.G., Investigation of the gradient of the specific explosives consumption as the principal criterion for breaking down by blasting (in Russian), Abstract of dissertation Apatity (1968). (It was not available)


11. APPENDIX I

DATA FROM TIME STUDIES

(From a set of observations were selected 100 random data)
FIGURE 16

A. LOADING TIME

<table>
<thead>
<tr>
<th>Number of data</th>
<th>100</th>
</tr>
</thead>
<tbody>
<tr>
<td>Number of classes</td>
<td>15</td>
</tr>
<tr>
<td>Class size</td>
<td>7.33</td>
</tr>
<tr>
<td>Mean value</td>
<td>131.83</td>
</tr>
<tr>
<td>Standard deviation</td>
<td>21.63</td>
</tr>
<tr>
<td>Standard error of the mean</td>
<td>2.16</td>
</tr>
</tbody>
</table>

BAR DIAGRAM OF CLASSIFIED SAMPLE DATA

ON LOADING TIME

(sec. for the payloader load a 50-ton truck)
FIGURE 17

B. WAITING TIME (PAYLOADER)

Number of data: 100
Number of classes: 15
Class size: 18.57
Mean value: 144.03
Standard deviation: 42.47
Standard error of the mean: 4.25

BAR DIAGRAM OF CLASSIFIED SAMPLE DATA ON WAITING TIME

(only waits longer than 1 minute were considered waiting time)
C. HAULAGE TIME

Number of data .................................. 100
Number of classes ................................ 15
Class size ........................................ 18.667
Mean Value ....................................... 236.320
Standard deviation .............................. 46.43
Standard error of the mean ................. 4.64

BAR DIAGRAM OF CLASSIFIED SAMPLE DATA
ON TRUCK'S ROUND TRIP (1 mile)
FIGURE 19

D. PRIMARY CRUSHER IDLING TIME

Number of data: 100
Number of classes: 15
Class size: 0.380
Mean value: 36.716
Standard error of the mean: 2.273
Standard error of the mean: 0.227

BAR DIAGRAM OF CLASSIFIED SAMPLE DATA
ON PRIMARY CRUSHER IDLING TIME
(secs. for the disk meter make a revolution)
FIGURE 20

E. PRIMARY CRUSHER CRUSHING TIME

Number of data.................100
Number of classes...............15
Class size.......................1.20
Mean value....................14.692
Standard deviation.............2.033
Standard error of the mean.....0.203

BAR DIAGRAM OF CLASSIFIED SAMPLE DATA

ON PRIMARY CRUSHER CRUSHING TIME

(limestone blasted using
8x16 ft. pattern)

sec./rev.

<table>
<thead>
<tr>
<th>Lower Limit</th>
<th>Mid-Point</th>
<th>Upper Limit</th>
</tr>
</thead>
<tbody>
<tr>
<td>sec.</td>
<td>rev.</td>
<td>sec.</td>
</tr>
<tr>
<td>7.00</td>
<td>8.20</td>
<td>9.40</td>
</tr>
<tr>
<td>10.40</td>
<td>10.80</td>
<td>12.00</td>
</tr>
<tr>
<td>15.60</td>
<td>16.00</td>
<td>17.80</td>
</tr>
<tr>
<td>19.00</td>
<td>20.20</td>
<td>21.40</td>
</tr>
<tr>
<td>22.60</td>
<td>23.80</td>
<td>25.00</td>
</tr>
</tbody>
</table>
F. PRIMARY CRUSHER CRUSHING TIME

Number of data: 100
Number of classes: 15
Class size: 0.493
Mean value: 14.28
Standard deviation: 2.091
Standard error of the mean: 0.209

BAR DIAGRAM OF CLASSIFIED SAMPLE DATA

ON PRIMARY CRUSHER CRUSHING TIME

(limestone blasted using 11 ft x 11 ft. pattern)
SIZE S

(fragments after blasting)

NUMBER OF CLASSES ..................= 15
CLASS SIZE ..........................= 1.780
NUMBER OF DATA ......................= 100
MEAN VALUE ..........................= 4.465
STANDARD DEVIATION .................= 5.127
STANDARD ERROR OF THE MEAN .......= 0.513
Bar diagram of classified Sample Data on sizes after Blasting (Limestone)

FIGURE 22
WEIGHT OF THE FRAGMENTS
AFTER BLASTING

NUMBER OF CLASSES = 15
CLASS SIZE = 16.947
NUMBER OF DATA = 100
MEAN VALUE = 9.950
STANDARD DEVIATION = 28.465
STANDARD ERROR OF THE MEAN = 2.846
BAR DIAGRAM OF CLASSIFIED SAMPLE DATA ON WEIGHT OF FRAGMENTS AFTER BLASTING

FIGURE 23
12. APPENDIX II

DATA FROM HAND SAMPLING OF BROKEN ROCK
<table>
<thead>
<tr>
<th>SIZE (in.)</th>
<th>PERCENTAGE</th>
<th>CUMULATIVE PERCENTAGE</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>26.75</td>
<td>26.75</td>
</tr>
<tr>
<td>2</td>
<td>31.50</td>
<td>58.25</td>
</tr>
<tr>
<td>3</td>
<td>23.10</td>
<td>81.35</td>
</tr>
<tr>
<td>4</td>
<td>6.40</td>
<td>87.75</td>
</tr>
<tr>
<td>5</td>
<td>4.60</td>
<td>92.20</td>
</tr>
<tr>
<td>6</td>
<td>2.85</td>
<td>95.20</td>
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<tr>
<td>7</td>
<td>1.15</td>
<td>96.35</td>
</tr>
<tr>
<td>8</td>
<td>0.95</td>
<td>97.30</td>
</tr>
<tr>
<td>9</td>
<td>0.40</td>
<td>97.70</td>
</tr>
<tr>
<td>10</td>
<td>0.45</td>
<td>98.15</td>
</tr>
<tr>
<td>11</td>
<td>0.15</td>
<td>98.30</td>
</tr>
<tr>
<td>12</td>
<td>0.55</td>
<td>98.85</td>
</tr>
<tr>
<td>13</td>
<td>0.05</td>
<td>98.90</td>
</tr>
<tr>
<td>14</td>
<td>0.40</td>
<td>99.30</td>
</tr>
<tr>
<td>15</td>
<td>0.25</td>
<td>99.55</td>
</tr>
<tr>
<td>16</td>
<td>0.20</td>
<td>99.75</td>
</tr>
<tr>
<td>17</td>
<td>0.05</td>
<td>99.80</td>
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<tr>
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<td>0.00</td>
<td>99.80</td>
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<td>99.80</td>
</tr>
<tr>
<td>20</td>
<td>0.10</td>
<td>99.90</td>
</tr>
<tr>
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<td>0.10</td>
<td>100.00</td>
</tr>
<tr>
<td>24</td>
<td>0.00</td>
<td>100.00</td>
</tr>
<tr>
<td>SIZE</td>
<td>WEIGHT LBS.</td>
<td>CUMULATIVE WEIGHT LBS.</td>
</tr>
<tr>
<td>------</td>
<td>-------------</td>
<td>------------------------</td>
</tr>
<tr>
<td>1</td>
<td>17.40</td>
<td>17.44</td>
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<tr>
<td>2</td>
<td>292.92</td>
<td>310.36</td>
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<td>543.59</td>
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<td>350.94</td>
<td>1204.89</td>
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<td>449.24</td>
<td>1664.13</td>
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<td>446.00</td>
<td>2100.13</td>
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<td>7</td>
<td>418.50</td>
<td>2518.63</td>
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<tr>
<td>8</td>
<td>488.80</td>
<td>3007.43</td>
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<td>9</td>
<td>264.00</td>
<td>3271.43</td>
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<td>10</td>
<td>262.75</td>
<td>3534.18</td>
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### APPENDIX III

WORK INDEXES FOR DRY CRUSHING

(From Allis-Chalmers, Milwaukee, Wisconsin)

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$W_i$ is a work index or comminution parameter expressed as the gross power in Kwh required to reduce one ton of material from a theoretical infinite size to 80% passing a 100 micron (About 67% passing 200 mesh) square mesh aperture.