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Hard Rock Blasting Developments and Possibilities*)

Introduction

It is now generally agreed that rock fragmentation by industrial explosives results from the combined effects of the strain wave and the high pressure gases generated by the explosion. Although the strain wave represents a small fraction (3%~8%) of the total explosive energy it plays an important role in "pre-conditioning" the rock for subsequent fragmentation effects of the high pressure gas.

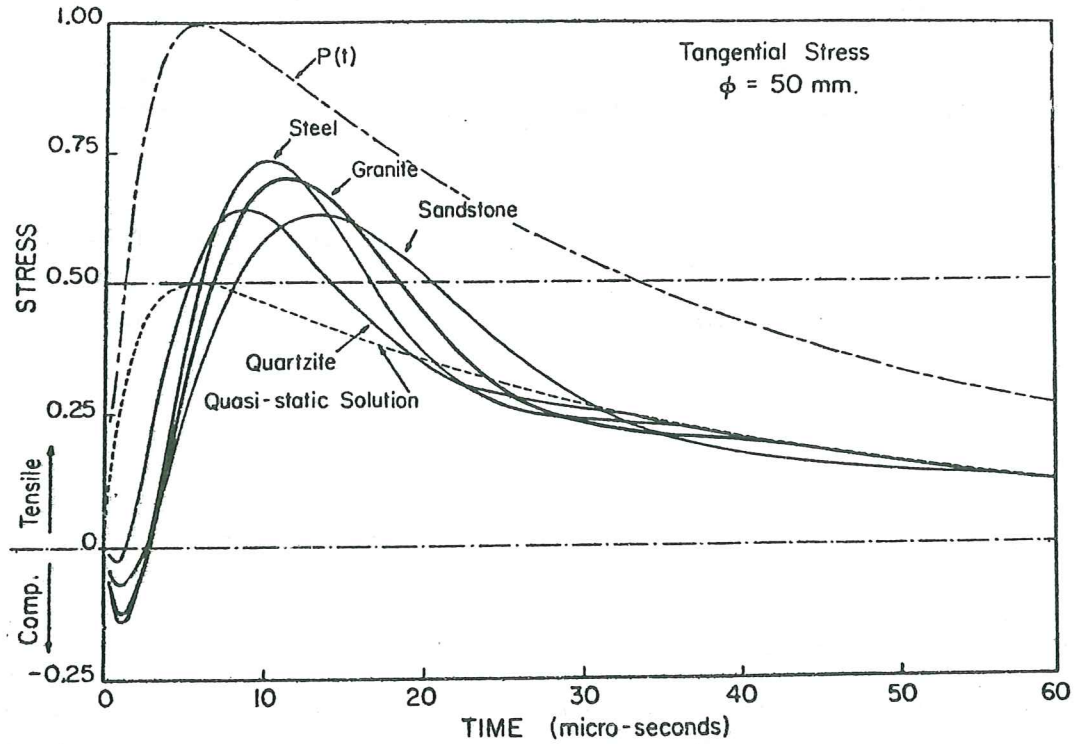
This paper reviews current thinking of the author and his colleagues at the University of Minnesota with regard to the mechanics of blasting. Some of the ideas are conjectural and have not been tried in practice; they are presented to stimulate discussion and, perhaps, to learn of any practical evidence of their validity. The paper concludes with a brief discussion of current bench blasting practice in the taconite mining operations of Northern Minnesota, U.S.A.

Strain Pulse Effects in Blasting

a) Effects at the Borehole Wall

Detonation of an explosive in a borehole produces a very rapid rise in pressure (and temperature) on the walls of the borehole. Pressures of the order of 10^5bars (10^6 lb. per square inch) are generated in times of the order of 10^{-5}seconds. This results in a dynamic impact on the borehole wall. The rate of pressure rise is so great that it is 'felt' to full intensity on the borehole before the disturbance has been transmitted beyond a few cm from the borehole.

BLIGH (1) has analyzed the problem of dynamic loading of a borehole and makes the very significant observation that, under dynamic pressurization, the applied loads are fundamentally different from those generated by static or quasi-static (i.e. slow by comparison with the speed of sound in the rock) pressures. As is seen from Fig. 1, the dynamic tangential stress is initially COMPRESSIVE before changing to tension. In the case of static or quasi-static pressurization, the tangential stress is always tensile.



— Dynamic tangential stress produced by pressure P (t) applied to 50 mm diameter cavities in different materials.

Fig. 1. Tangential (or "hoop") stresses generated around a borehole due to very rapid, dynamic pressurization. (Note that the stresses are initially compressive.)

*) To be inserted under tab 4

Dynamically, the rock is subjected to very high triaxial confinement, which results in compressive pulverization of the rock rock, all around the hole. The temperatures are also very high at the immediate borehole wall and tend to reduce the rock strength. By contrast, under static loading a single crack is initiated under tangential tension at much lower pressures. Figures 2 and 3 indicate the difference between the two cases.

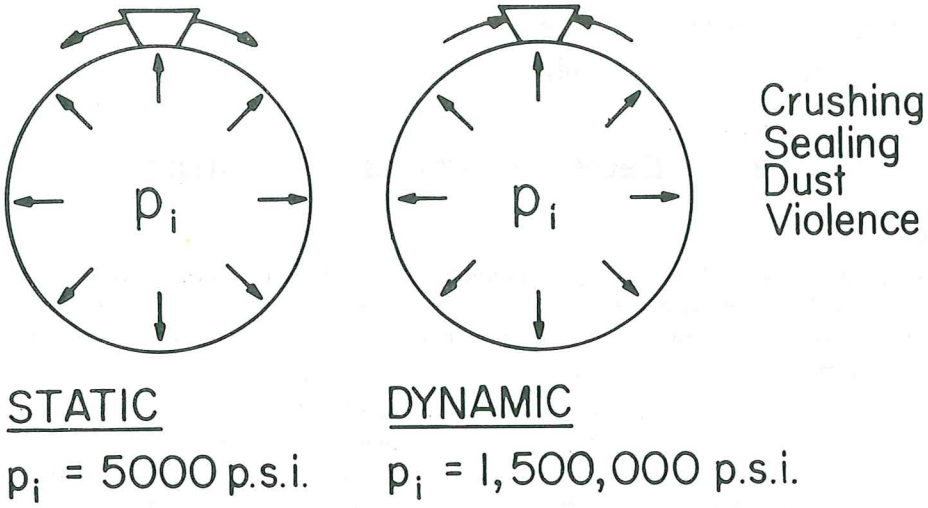


Fig. 2. Comparison of dynamic and static pressurization conditions in a borehole

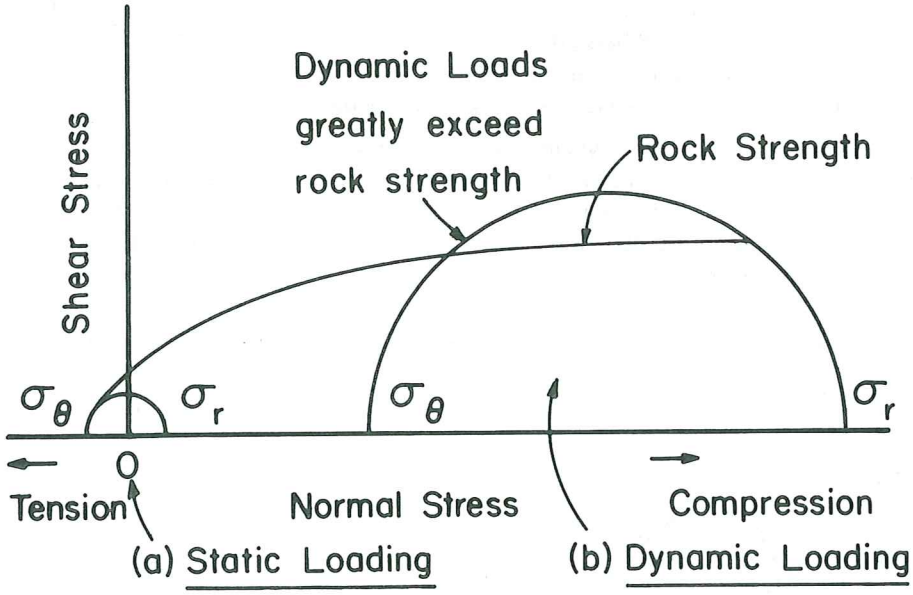


Fig. 3. Rock fracture conditions under (a) Static and (b) Dynamic loading conditions

Dynamic stresses greatly exceed confined rock strength in compression and cause intense pulverization around the wall of the borehole.

The specific energy absorbed in compressive pulverization is relatively very high so that the strain wave becomes rapidly attenuated and the rise time increases (i.e. high frequency components of the wave are most rapidly attenuated) and the pressure rise, although still rapid, becomes "less dynamic" in nature. Thus, as the wave proceeds from the hole the intensity of crushing is reduced to be replaced eventually by quasi-static **multiple** radial cracking, at some distance from the hole. The density of radial cracking decreases progressively, the eventual extent in any direction becoming governed by rock strength anisotropy and the intensity of the 'in-situ' ground stresses acting on the rock. In isotropic rock the pattern of crack extension is roughly elliptical in section, as shown in Fig. 4.

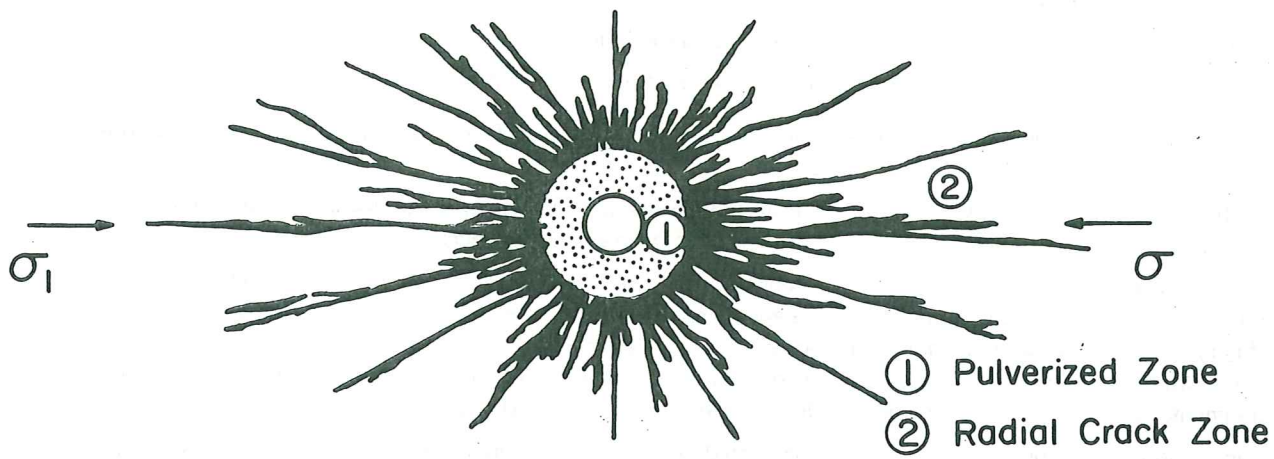


Fig. 4. Extension of radial fractures due to borehole blasting in stressed rock

BLIGH (1) contends that the pulverized rock formed in dynamic loading acts as a very effective barrier against penetration of the borehole gases into the radial cracks. Further extension of the radial cracks towards a free surface must then proceed under the forces at the borehole (i.e. gas pressure x borehole diameter, per unit length of charge) whereas in the static, or quasi-static, case the radial cracks begin from the hole and are accessible to the high pressure gases. The force exerted by the gas on the rock increases rapidly as the pressurized gas extends into the (**low volume**) cracks [i.e. the gas pressure should not drop greatly, even for a sizeable degree of gas extension into cracks.] Fig. 5. (after OUCHTERLONY and HARDY) illustrates the force required for radial crack extension in each case [i.e. (a) no gas in cracks; (b) gas pressure fully active in cracks).

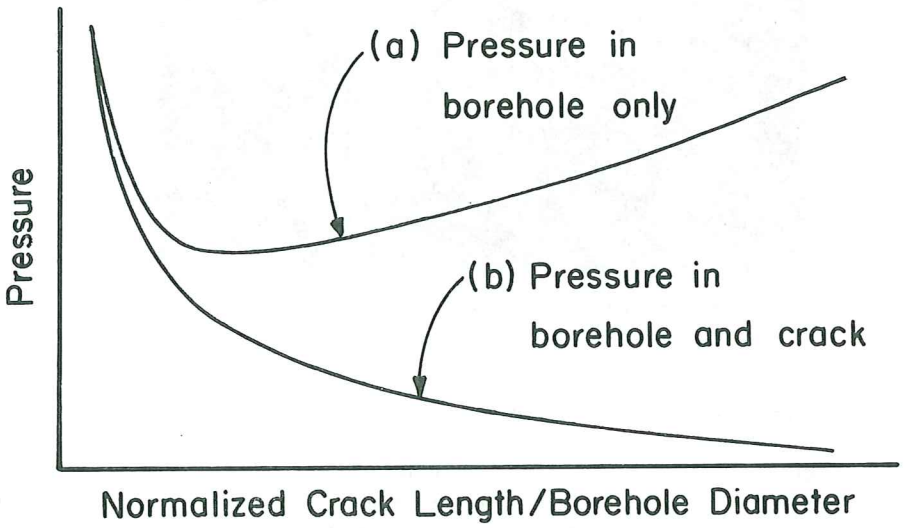


Fig. 5. Comparison of Pressure Necessary to extend cracks from an internally pressurized hole for the two cases:
(a) where pressure exists in the borehole only
(b) where pressure exists in the borehole and in the crack

It would appear that effective fragmentation could be achieved at significantly lower gas pressures if the intense pulverization process is avoided. BLIGH's (1) experiments with controlled pressure explosions (acetylene air mixtures) tend to confirm this conclusion. De-doupling of explosives – another method of reducing the peak pressure and increasing the rise-time of an explosive – also appears to reduce or eliminate pulverization and generate effective radial cracking from the borehole (e.g. as in pre-splitting). It should be noted, however, that de-coupling by use of charges of smaller diameter than the borehole has at least two possible drawbacks:

a) The total amount of explosive gas energy in the hole is reduced.

b) Some explosives may not detonate as efficiently if not fully confined.

[see ATCHISON and DUVAL (4)]

Thus, any experiment to examine the effect of eliminating or reducing the pulverization zone on fragmentation should:

i) maintain the total energy of the explosive by increasing the hole size,

ii) use an explosive or a confining arrangement that ensures effective detonation.

BLIGH suggests several possible improvements in blasting that could result from more careful control of peak pressure rise-time in a borehole explosion. These include:

1. High and rapidly rising explosion pressures result in intense and damaging or disturbing ground vibrations. It may be possible to reduce these vibrations without adverse rock fragmentation effects.
2. The sealing effect of the pulverized region tends to prevent release of the high pressure gases into the rock fractures until fragmentation is well advanced, and the energy required to complete rock breakage is relatively small. The (high-pressure gas) energy available is consequently excessive and results in violent ejection of the broken rock. [In multiple row blasting, of course, some of this excessive kinetic energy may be recovered through additional fragmentation when moving fragments collide with rock previously blasted.]
3. Elimination of a major source of explosion generated dust (i.e. the pulverized zone). This is most important in tunnel blasting where it can be a serious health hazard as well as a cause of delay.

Fig. 6, taken from the paper by PERSSON, LUNDBORG, and JOHANSSON (5), graphically illustrates the intense 'crushing' zone immediately adjacent to the borehole, and the sealing of the hole from the radial cracks.

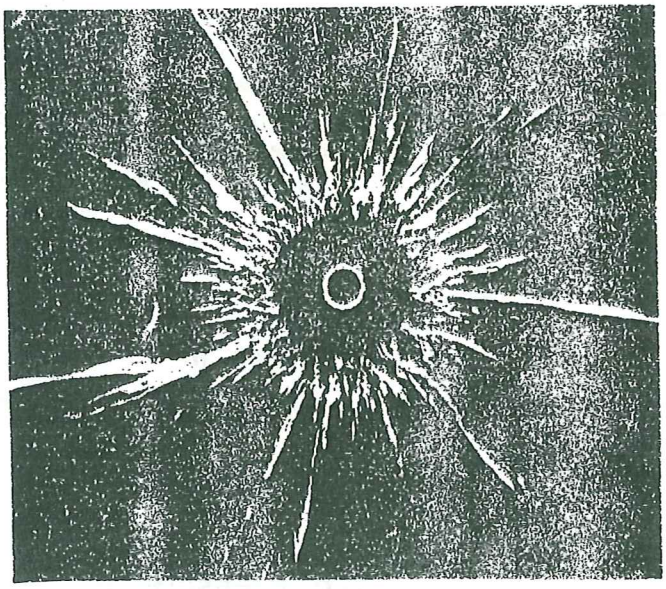


Fig. 6. Appearance of the region around a hole in Plexiglass after detonation of an explosive in the hole

After: Persson, P.A., et. al.
[Explanation on following page.]

The "unbroken" region adjacent to the hole has been intensely deformed and 'rehealed' under high pressure and temperature. Radial cracking begins only after the strain wave energy density (i.e. stress amplitude) has fallen substantially due to heavy energy dissipation and high frequency attenuation (i.e. increasing the rise time) in the pulverization zone.

b) **Strain Pulse Radiation from Columnar Explosive Charges**

The above discussion leads to the further interesting conclusion that increased explosion pressure and faster rise times will result principally in a more extensive zone of pulverization, this zone extending until the increased strain wave amplitude has been dissipated and the wave is of the form (pressure and rise time) appropriate to develop radial cracking. The pressure at which this stage ensues will depend on the particular rock being blasted. This implies that the strain waveform emerging from the pulverized region is determined principally by the rock type, and not by the explosive.

Supporting evidence for this suggestion is provided in the analysis by STARFIELD and PUGLIESE (6) of tests conducted by the U.S. Bureau of Mines to compare the strain wave forms measured by strain gauges at a distance 'h' (Fig. 7) from the center of a columnar explosive charge.

It is seen (Fig. 7) that the High Pressure Gelatin explosion produced higher strain amplitudes at the gauge than did the Prills and Fuel Oil explosion. This is well known and usually assumed to be the result of the higher explosion pressure (in the borehole) of the HPG. STARFIELD and PUGLIESE propose the following alternative explanation:
(continued on P 5)

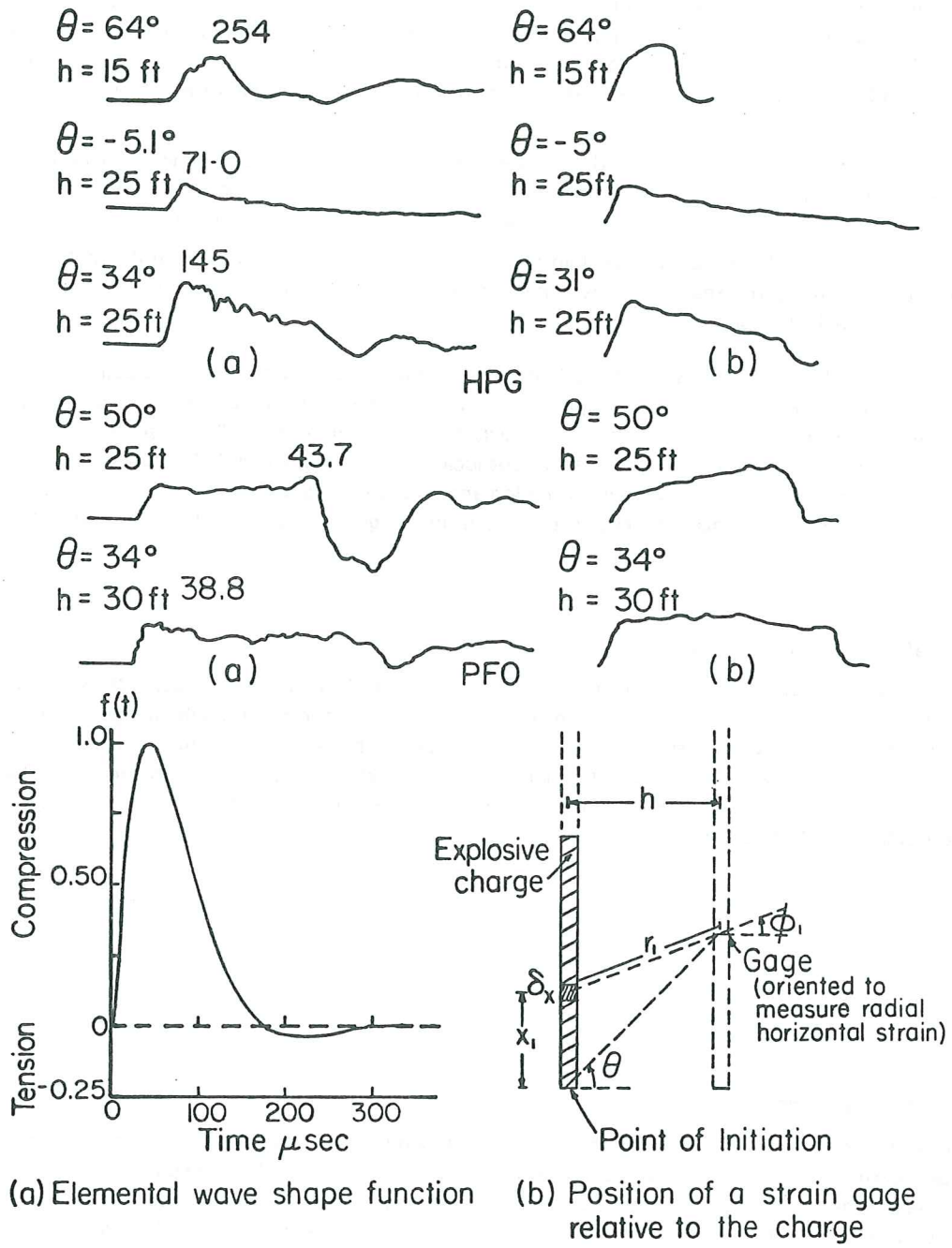


Fig. 7. Strain waves generated by a 27 ft. explosive charge of high pressure gelatin (HPG), prills and fuel oil (PFO).

The waves in column (a) are those actually observed. The waves in column (b) are those calculated on the assumption that the elemental wave form emanating from the immediate rock wall zone was the same (as shown) for each explosive.

Consider the explosive column to be made up of a series of small (elemental) explosive charges (each δX in length) stacked on top of each other to form the column. Each element acts as a point charge in the hole and, when detonated, radiates a strain wave into the rock, this wave diverging spherically from the point, traveling at the velocity of sound for the rock. Successive elemental wave-forms are generated at time intervals determined by the Velocity of Detonation for the explosive. Thus, for the cases shown in Fig. 7, the explosive column is detonated at the bottom, initiating an elemental wave that is transmitted into the rock mass, reaching the gauge location after having traveled a radial distance $h/\cos\theta$ (see diagram). At a small time interval ($\delta X \cdot$ Velocity of Detonation) later, the second element of charge is detonated, generating a wave into the rock. This will travel a somewhat shorter distance to reach the gauge location. The process is repeated until the entire explosive column has been detonated. The nett strain wave observed at the gauge location (not that the gauge was oriented to detect only the components of compressional strain normal to the axis of the explosive column) will consist of the sum of the components (normal to the column axis) of all of the elemental wave-forms passing through the gauge point, each having traveled its own path at the same velocity and each having been initiated in a sequence determined by the Velocity of Detonation.

If we accept this model of wave propagation from the explosive column it is seen that the higher peak strain observed at the gauge for the HPG is due to the higher Velocity of Detonation – the fact that, being initiated in more rapid succession, the elemental wave-forms arrive at the gauge over a shorter time span and hence reinforce each other more giving a nett total strain that has a higher peak and is less extended in time than the nett wave form resulting from the PFO detonation.

It is seen that the agreement between the predicted and observed wave forms using the assumed elemental wave form shown in Fig. 7 (the waveform was found as a 'best fit' to the observed data) is very good, and is convincing evidence of the validity of the rock type control on the wave form radiated from the explosive column.

STARFIELD and PUGLIESE did not suggest a mechanism whereby the elemental wave-form in the rock would be the same for different explosives, but the earlier discussion in this paper suggests that attenuation in the pulverization region is probably the principal cause.

STARFIELD and PUGLIESE's hypothesis has interesting practical implications. If it is accepted that radial cracking by strain waves is an important mechanism of rock fragmentation (at least as a "pre-conditioning" of the rock to allow further fragmentation by subsequent action of the explosive gas), then it should be possible to selectively intensify strain wave amplitudes for increased fragmentation at various locations in the rock (e.g. in hard layers or at the toe of the burden) by variation of the point of explosive detonation and by multiple detonation points. The possibility of varying the point of detonation in adjacent holes in order to improve fragmentation may also be considered.

c) Strain Pulse Reflection from a Free Surface

The phenomenon of tensile reflection slabbing of rock at a free face [DUVALL and ATCHISON (7)] is well known, and has been considered to play an important part in effective rock fragmentation. Although now generally considered to be less significant than gas pressure, it is perhaps worth noting that a small amount of tensile slabbing could be critical in reducing the effective burden to a value that can be effectively removed by the gas energy [e.g. removal of 10% of the burden by tensile slabbing reduces the volume of rock (in a crater) to be removed by the energy of the high pressure gases by 27%. $(1-0.1)^3 = 0.73$]

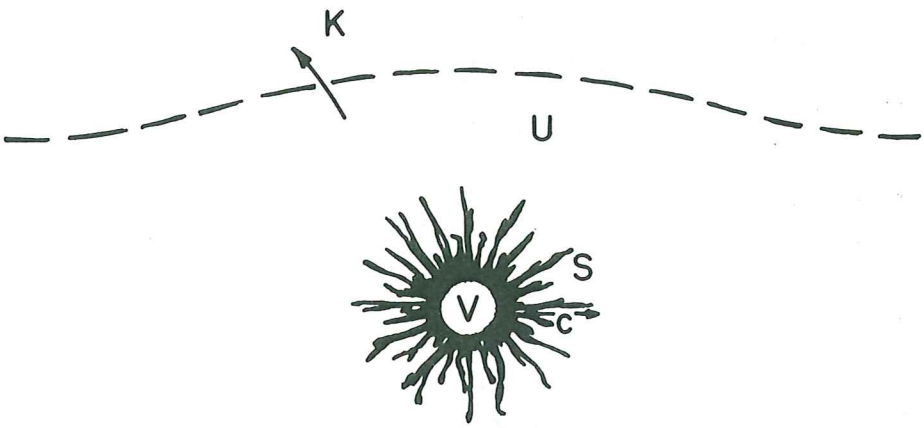
Gas Pressure Effects in Blasting

In discussing the effects of strain pulse propagation in rock it was seen that the rate of pressure rise was such that the initial loading produced effects that were signigicantly different than similar statically applied loads. Following the initial pressure rise the gas reaction proceeds and exerts a sustained pressure on the borehole wall. Although the time during which the gas pressure acts is still of the order of thousandths of a second it is still much longer than the time required for the pressure effects to be distributed throughout the burden, and it is hence not unreasonable to analyze the effect of the gas pressure in terms of statics, i.e. as a "quasi-static" system.

The action of the gas pressure is simply to generate tangential tension stresses around the wall of the borehole, which tend to extend the radial cracks towards the free surface. As already mentioned, the gas may or may not enter the radial cracks, depending on conditions at the borehole wall. This, however, does not change the basic tendency to extend the cracks but rather makes this extension less or more difficult to accomplish.

Stability Analysis of Radial Crack Growth due to Borehole Gas Pressure

The problem of crack growth under the influence of borehole explosion pressure may be examined in terms of energy changes that occur as the cracks are extended. This approach, essentially an application of the GRIFFITH (8) criterion for rupture of brittle solids, has been suggested and used by several authors, including OUCHTERLONY (2), HARDY (3), and PORTER and FAIRHURST (9). As shown in Fig. 8, the total potential energy (P) of the rock strained by the gas pressure involves energy in several different forms.



P = Total Energy of System

V = Gas Pressure Energy

U = Strain Energy of Rock

S = Surface Energy (Work of Fracture) of Cracks

K = Kinetic Energy

c = Length of Extending Crack

Fig. 8. Energy Changes during Crack Extension by Gas Pressure

$P = V + U + S + K$

Thus, we have [neglecting energy forms (e.g. potential energy of position) that do not change with fragmentation]:

V – the energy stored in the high temperature, high pressure, gas in the borehole

U – the strain energy stored in the rock mass in 'containing' the gas pressure

S – the energy used in developing the surfaces in the rock, through which fracture extension and fragmentation develop

K – the kinetic energy of the rock accelerated by the gas pressure

Denoting the arbitrary crack length as 'c', we assert, following GRIFFITH that, according to the Theorem of Minimum Potential Energy, cracks will extend when the Potential Energy (P) of the system under load **decreases** with crack extension, i.e. when

$$\frac{\partial (P)}{\partial c} \leq 0$$

Detailed analysis of the energy changes, considering the simultaneous extension of several cracks in various directions is complicated. It seems probable that the crack or cracks which produces the most rapid drop in potential energy with crack extension will propagate sooner than others, but that these may stabilize (i.e. $\frac{\partial P}{\partial c} > 0$) after extending some distance allowing cracks in other orientations to propagate further, as shown in Fig. 9.

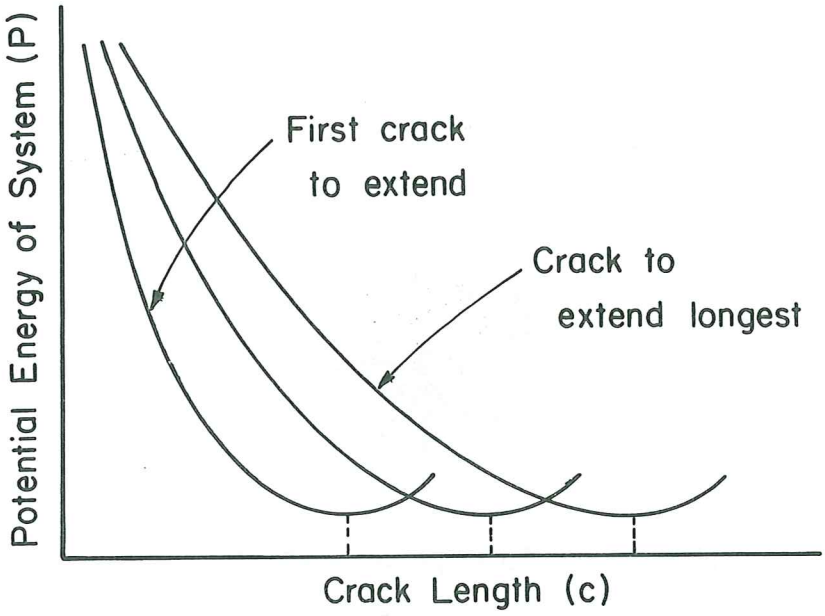


Fig. 9. Unstable Propagation of Cracks of Various Orientations due to Borehole Gas Pressure

Theoretical studies are underway to examine the stability of crack propagation from a borehole following this method. At the present stage the analysis involves several simplifications. Thus,

i) Kinetic effects are considered negligible. (This seems reasonable during fracture extension, i.e. before rapid ejection of rock begins.)

ii) Symmetrical pairs of cracks are examined separately, i.e. each crack pair of various orientations is extended alone, on the supposition that no other cracks exist around the borehole. The crack pair which remains "most unstable" to the free face is considered to define the boundary of the fragmented (ejected) zone produced by the explosion.

Fig. 10 illustrates the application of the method to the determination of optimum "spacing to burden" ratios in multiple hole (simultaneous) blasting.

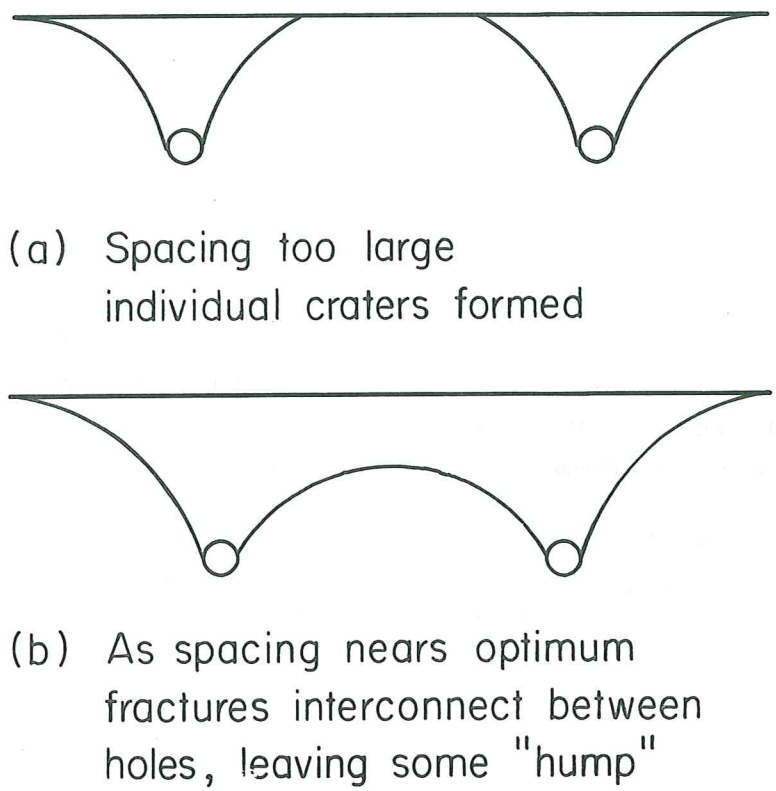


Fig. 10. Crater Formation with Variable Burden-Spacing Ratio

The energy changes associated with crack propagation are examined using the "displacement discontinuity" method [see SALAMON (10)] and the computer program for the method developed by CROUCH (11). A crack is first assumed to start at some point on the borehole and then the direction of extension that maximizes the rate of decrease of potential energy of the system is determined for successive incremental extensions of the crack.

From preliminary results obtained to date it appears that the method of analysis used by PORTER and FAIRHURST (9) is a reasonably good approximation in most cases. In this method the path taken by the fractures is assumed to follow **approximately** the direction of maximum principal stress from the point of initiation (maximum tangential tension) on the borehole.

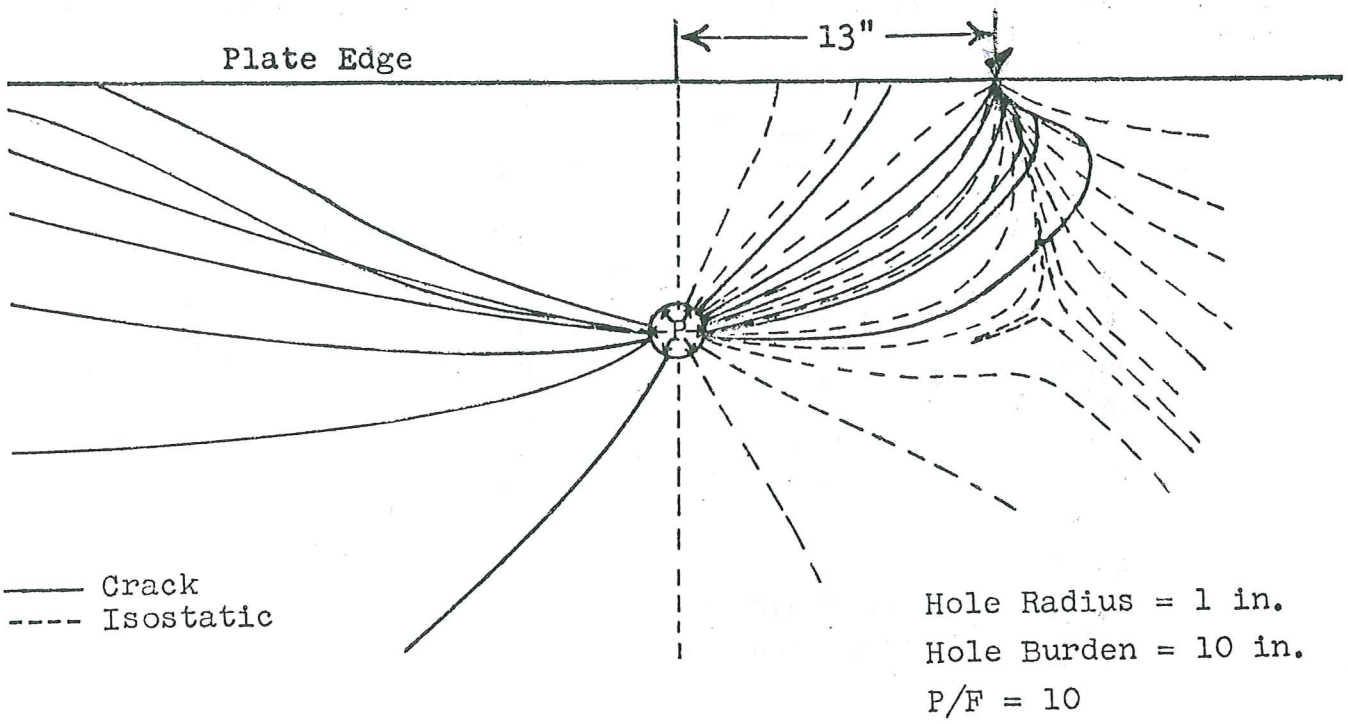


Fig. 11. Composite of 7 Pairs of Cracks Formed with Surface Line-Load Spacing of 13 Inches

Once the method has been demonstrated for homogeneous isotropic materials it is **in principle** possible to modify the computer program to allow consideration of anisotropic (e.g. jointed or bedded) materials. While it is quite likely that the computer results will confirm the rules derived empirically by ASH and shown below in Table 1, demonstration that crack propagation in blasting can be examined in terms of unstable energy changes should represent significant advance, and would facilitate studies of optimization of blasting patterns [such as the 4:1 and 6:1 spacing-burden ratios found to be effective in Sweden, PERSSON and JOHANSSON (12)], methods for controlling crack directions, the influence of gas penetration on crack extension, etc.

Table 1
Blast Design Ratios

	Average	Minimum	Maximum
Burden (B/D)	25–30	20	40
Subdrill (J/B)	0.2–0.3	0	0.5
Collar (T/B)		0.7	1.0
Spacing (S/B)	1.5	1.0	2.0+
Hole depth (H/B)		1.5	4–5

B = Burden
D = Hole Diameter
J = Depth of Sub-Grade Drilling
T = Collar (unloaded section of hole)
S = Hole Spacing
H = Hole Depth

(after ASH)

Current Blasting Practice in Open-Pit Taconite (Iron-ore) Mining in Northern Minnesota, U.S.A.

A principal feature of the taconite mining operations in Minnesota is the use of jet-piercing drills. These allow holes to be chambered (i.e. enlarged with respect to the initial diameter) at the bottom, facilitating concentration of explosive.

Although most major operations use jet-piercer drills 12 ~15 inch (Bucycrus Erie, Gardner-Denver) rotary drills are being increasingly used, particularly on the middle and west end of the Iron Range, where the taconites are somewhat less hard and drilling rates of 40 ~45 ft/hour may be achieved. High strength aluminized slurry explosives are used in the bottom of the holes, especially where wet conditions are encountered. The drilling pattern tends to be a square 25 x 35 ft with 12 inch rotary drills or 30 x 30 ft with 15 inch drills. Sometimes "en echelon" blasting is used, giving an effectively larger spacing and reduced burden. Bottom hole initiation is perhaps the most popular practice, although top-hole and multiple initiation blasting is sometimes used. Bench heights are generalled 35 ~40 ft.

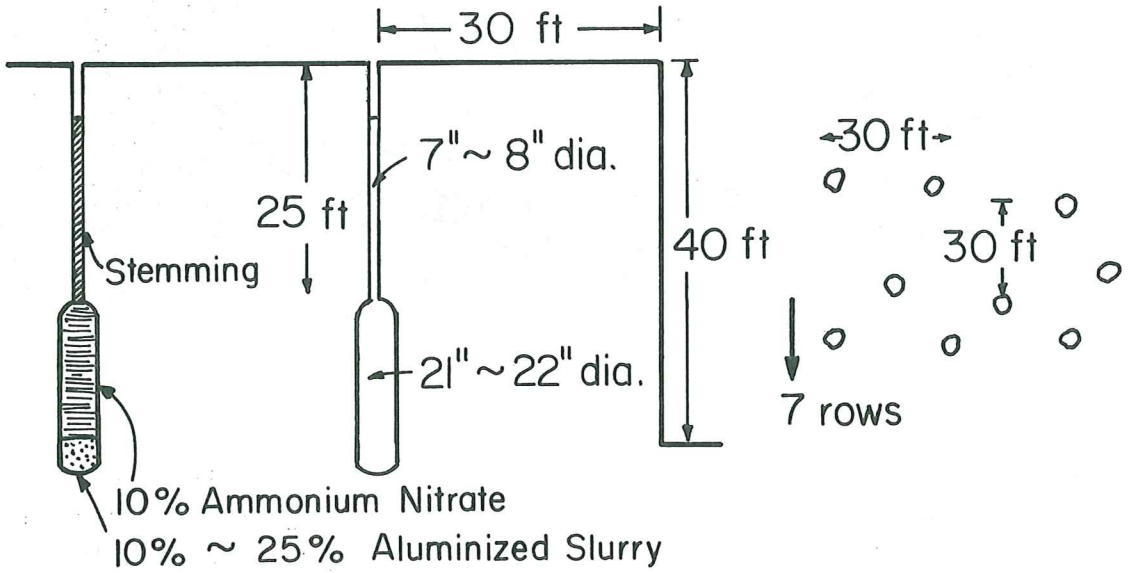


Fig. 12. Layout of Chambered Jet Piercer Drill Holes for Taconite Blasting in Minnesota

Conclusion

Large scale experimental research on blasting is very costly and sometimes difficult to evaluate, so that much of the development in blasting practice tends to occur in a relatively slow evolutionary manner. Conjectural theoretical notions such as expressed in this paper are no substitute for large scale tests – but perhaps they may help give some guidance to the continuing evolutionary improvements.

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