SHOCK COUPLING, LOADING DENSITY AND THE EFFICIENCY OF EXPLOSIVES IN COMMERCIAL BLASTING

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Factors considered most important in the blasting of rock are:

1. The maximum available energy A - determined by the heat of explosion Q and the mechanical efficiency, a factor intimately associated with the mode of application, ($A \sim Q$ at highest gas concentrations).

2. The "borehole pressure" p_b which is the maximum pressure attained in the borehole after passage of the detonation wave and before the burden has had time to move or become compressed appreciably. (Owing to the short duration of the detonation wave at any particular point in the borehole, the fact that the explosive may not always fill the borehole completely and the further fact that the burden may not actually "see" the detonation pressure, the borehole pressure is considered more significant than the detonation pressure p_2 in borehole blasting) The borehole pressure is determined by the explosion or adiabatic pressure p_3 and the loading density Δ, or the fraction of the borehole filled by explosive.

3. The physical conditions important in the application of the explosive:

   a. The "powder factor" (W_e/W_r), or the ratio of the weight of the explosive to that of the rock being blasted expressed as pounds per ton (more generally in pounds/cubic yard).

   b. The relative impedance R, or the ratio of the (effective) impedance of the explosive $(\rho V)_e$ to that of the rock $(\rho V)_r$.

   c. The "burden" or "line of least resistance", the spacing between boreholes and the geometry of the borehole pattern.

   d. The physical and chemical properties of the rock, most significant of which are possible heterogeneities, such as faulting, pre-fracture, and greater than micro-scale chemical heterogeneities.

All of these factors need to be carefully considered in the most economical engineering of a blast. It is possible today to give a relatively satisfactory scientific outline of the mechanism of blasting even though much remains to be learned regarding the detailed mechanism. Here is considered firstly an outline of the present status of rock mechanics as it pertains to blasting. The factors pertaining to the most efficient application of blasting agents are then considered followed by a discussion of methods of application to achieve optimum explosives performance.

Rock Mechanics

Rock mechanics is currently a rapidly developing science contributing greatly to a better understanding and consequently the more effective application of explosives in blasting of rock[1-14]. Basic to the development of the science of rock mechanics were the advances of Goransen[15] and the Los Alamos and NOL groups[16-20] concerning shock wave phenomena and the transmission and reflection characteristics of shocks at interfaces between different media. Basing considerations on this new knowledge as well as new experimental methods of study (ultra-high speed streak and framing cameras and electronic timers) the theory of fracture and failure of solids under impulsive loading by shock waves developed rapidly[6,8,10,19-21]. Also the recent development of accurate methods for measuring the pressures in high intensity shock and detonation waves has made possible more quantitative work in this field[22,23].

Fragmentation of hard rock by explosives occurs predominantly in stress relief and in tension waves created in yet incompletely defined ways by a blast. Tensile fragmentation may really be the only means of breaking the hardest rocks although fragmentation directly by the compression wave may be important in the softer and lower density rocks. Tension waves develop prominently by reflection of compression waves at free surfaces. The <u>shock wave</u> theory of blasting[1-6] therefore has emphasized, perhaps too strongly, the phenomenon of successive "scabbing" by shock wave reflections at free surfaces as described by Rinehart and Pearson[14,19]. Some investigators considered the scabbing process to be practically the only means by which hard rocks are fragmented in blasting, although others[7,8,11-13] described other mechanisms of fragmentation by tensile forces some of which may prove to be of much greater importance than fragmentation by means of release waves reflected from free surfaces. Shock waves from a high explosive separate into the P-waves (longitudinal) and the lower velocity S-waves (shear). The latter is considered to cause appreciable radial fracture[7a,11,12] and possibly close-in shattering of the rock. The predicted "shatter zone" near the borehole does not actually occur in the hardest rocks as seen by the presence of semi-boreholes on a new free face in certain good blasts, namely those that produce good fragmentation and no "back break", i.e., no gross fracture of the rock on the inward side of the borehole.

Kovazhenkov[7b] described an <u>energy theory</u> of rock fragmentation similar to the model wherein rock breakage is a "release of load" or stress relief effect following the temporary transfer of the energy of the blast from the explosive gases into potential energy by powerful compression under the sustained pressure of the products of detonation and the great inertia of the burden. This is the concept described as the "rock bursting" mechanism of fragmentation[13]. The initial shock wave from a detonation carries into the rock generally less than 0.1 (sometimes less than 0.05) of the blast energy, whereas the total energy transferred to the burden by the initial compression of the rock may be a much larger fraction of the available energy of the explosive at some critical early stage of the blast, e.g., the instant the initial shock wave reaches the free surface. Kovazhenkov postulated that there will be numerous means of creating the necessary tensile forces for fragmentation once the rock has first been excessively compressed. Even a relatively long duration stress relief may be the source of most of the fragmentation. The period of the main over-all relief actually involved in a blast is between 5 and 10 times the period between detonation and the emergence of the shock wave at the free surface. The rise time of the stress wave turns out to be appreciably greater in general than the time required for detonation of the charge, and the fall of the stress wave is even longer. The total time the main rock mass is under compression is several times greater than the stress wave rise time. Moreover, a long-time stress relief fracturing of rock seems to have been amply verified by recent studies by Obert[1a] who found that stressed rock fractures in proportion to the magnitude of the stress by simply cutting it away from the source of stress. One may demonstrate this effect by pressing fine powder at very high pressures; upon stress relief the specimen often fractures into layers perpendicular to the axis of the die. The number of such fractures is proportional to the magnitude of the initial stress. Additional evidence for long time stress relief fragmentation is the "step" (and perhaps sometimes continuous) increase in the velocity of fragments ejected from the free surface of the blast in release wave fragmentation discussed below.

In the shock wave theory of fragmentation three zones of a blast are described: 1) the <u>fragmentation zone</u> beginning at the free surface and extending inward to, 2) an <u>unfragmented zone</u> which in turn is sandwiched between the fragmentation zone and, 3) a <u>shatter zone</u> adjacent to the borehole. The unfragmented zone is, of course, absent when the burden/charge ratio is sufficiently small. (It could not be tolerated in commercial blasting.) The energy theory of fragmentation does not deny the release wave fragmentation zone and the shock wave shatter zone (occurring in porous and soft rocks) but replaces the unfragmented zone by a release of load type fragmentation zone. An unfragmented zone will, of course, occur in blasts with excessive burden/charge ratios, but, when it does, release wave fracturing is usually also absent. Kovazhenkov added (to the predicted conical-shaped crater of a spherical charge, or wedge-shaped crater of a cylindrical charge running parallel to a free face, which he called the <u>ejection zone</u>) a <u>fracture zone</u> associated exclusively with shear type release-of-load or stress-relief fragmentation. Whereas the shock wave theory predicts the conical or wedge type craters, ellyptical craters actually occur due, according to Kovazhenkov, to the fracture zone inside the ejection zone.

A serious difficulty in the shock wave theory is seen in the frontal fragment velocity measurements of Petkof, et. al.[1e] taken from the free surfaces of quarry blasts. By focusing a high speed camera on a given spot on the free surface, they were able to follow the distance-time behavior of particular fragments during the blast. They observed "step" velocity curves for these fragments in which the initial velocity was sometimes only a fraction of the ultimate velocity, velocity apparently increasing discontinuously or in steps due to collision from behind by faster moving fragments. The significance of stepwise (sometimes fairly continuous) acceleration of frontal rock fragments in a blast may be better appreciated by a brief consideration of the simplified multiple scabbing model of fragmentation by release waves at free surfaces.

A shock wave generally is considered to have a pressure-distance characteristic in which the pressure falls exponentially with distance behind the shock front following an equation of the form

$$p = p_m e^{-t/\tau} \tag{1}$$

where p_m is the pressure at the shock front, τ is the relaxation time and t the time for a given characteristic in the wave to pass a fixed point. (The stress waves observed in blasting are not actually of this type: they exhibit a relatively long rise time of the order of 0.1 to 0.2 m sec or more.) If desired, t/τ may be replaced by αx where α is also a relaxation constant and x the distance behind the wave front at a given instant. The velocity of a fragment ejected from the free surface by reflection of a shock was as a release wave of intensity greater than the tensile strength S_t of the rock is given by the equation

$$V_{ti} = 2p_i/(\rho V)_i \tag{2}$$

where V_{ti} is the free surface velocity of the i^{th} fragment and $(\rho V)_i$ is the impedance of the rock (ρ = density; V = velocity). If a shock wave enters a free face from within the condensed medium with a peak pressure p_m, it has the potential of generating N fracture planes by successive tensile scabbing as the release wave moves back into the solid, N being given simply by

$$N \leq p_m/S_t \tag{3}$$

The upper limit condition corresponds to no losses in the wave due to friction, viscosity and heating of the solid during scabbing. Owing to the exponential decay form of the incident shock wave, which is simply mirrored into the tension region during reflection, a tensile failure should occur in the solid for each increment of decay in the net pressure in the amount $\Delta p = S_t$. The velocity of the first fragment must be the highest because p_i is greatest for this fragment. On the basis of these considerations the only apparent way to account for the step (or continuous) acceleration of a rock fragment scabbed off the free surface in the shock wave theory of fragmentation is for the pressure to rise discontinuously following a decay by at least the amount $\Delta p = S_t$. This would require, in effect, multiple shock waves of progressively increasing intensity, a condition which seems unlikely, and in fact, is not apparent in the strain wave measurements described by Bureau of Mines investigators. As a matter of fact, the strain waves observed by Atchison, Duvall, et. al.[1] showed relatively long rise time and even longer, more gradual decay. Therefore, the scabs should be much thicker than the observed fragments unless there were to exist much higher frequency wave components with pressure fluctuating in magnitude by at least S_t.

From the magnitude of the pressure at the free face, the observed initial velocity V_t of the fragments at the free surface and the known tensile strength of rock also one may show the relative unimportance of free surface fracturing. The observed initial fragment velocities are in the range 3 to 7 m/sec for the shots studied by Petkof, et. al. Equation (2) thus gives about 0.2 to 0.4 kb for the peak pressure in the shock wave upon striking the free surface. From equation (6) and reasonable values of S_t ($\geq$0.05 kb) one can account for only 4 to 8 successive scabs which is much too small a value to account for the fragmentation observed in normal blasts.

Massive acceleration of the burden provides an explanation for the acceleration of fragments at the free surface following their ejection by release wave scabbing. In the relatively much slower stress relief following a relatively long duration of sustained pressure in the borehole, the whole burden accelerates to reach an ultimate velocity around 30 m/sec, appreciably greater than the free surface velocity of fragments, ejected from the free surface. Calculations were made applying Newton's equation

$$Md^2r/dt^2 = \text{force} \qquad (4)$$

using a method of stepwise integration[20]. Upper limits were computed by assuming idealized perfect confinements, incompressible and suitably prefractured rock to permit uniform acceleration under a hypothetical hemi-cylindrical expansion of the products of detonation perfectly confined within the rock mass. Velocity-time V(t) curves for the burden were plotted along with pressure-time p(t) curves for the products of detonation and the maximum available energy-time A(t) curves for the energy transferred from the hot gases to the burden. Comparison of these velocity-time curves with those observed by Petkof et. al. shows an interesting correlation for times following detonation of the order of 20 to 50 m sec and greater. To account for this seemingly fortuitous order-of-magnitude agreement, let us consider more carefully the actual conditions that may exist in the rock and borehole at various stages of the blast beginning with the instant t_o the initial shock wave reaches the free surface. The $p_m(r)$ relationships that should exist at this instant and at various times thereafter are illustrated in Figure 1, based on an application to the case of hemicylindrical symmetry and a negligible detonation time along the depth interval under consideration. Even after taking into account the compressibility of the burden the pressure in the expanding borehole should still be an appreciable fraction (evidently about a quarter) of the borehole pressure at the time t_o. One may therefore expect a pressure of about 25 kb (using the highest pressure explosives) for the gases in the borehole at this stage based on an upper limit borehole pressure p_b of about 100 kb. The assumption of an exponential decay in pressure with distance permits one to draw a straight line between the two points p_m^o and $p_m(r_o)$ in log p vs r plots.

Following the initial emergence of the shock wave the release wave moves back into the rock mass at a velocity comparable to that in the initial shock wave. Shocks and release waves thus have time to rebound enough times in say 20 m sec effectively to smooth out pressure and velocity gradients in the rock. Therefore, assuming that the energy associated with fragmentation is, for example, half of the total blast energy, at about 30 m sec after detonation the velocity of the whole burden would be about 0.7 of that computed by equation (4) for the idealized conditions there mentioned. Since half of the blast energy is probably a fair representation to that going ultimately into fragmentation and surface tension in the rock, one thus accounts approximately for the observed acceleration.

Explosives Performance

a. Shock Coupling

According to the theory of "impedance mismatch", the initial pressure p_m^o in the rock next to the borehole is related to the detonation pressure p_2 for a loading density Δ of unity by the relationship

$$p_m^o/p_2 = 2/(1+R) \qquad (5)$$

where

$$R = (\rho V)_e/(\rho V)_r \qquad (6)$$

Measurements of the strain energy-distance relationships in rock were shown(1b-d) to follow the relationship

$$\epsilon = (W^{1/3}K/r)\exp(-\alpha r/W^{1/3}) \qquad (7)$$

where K is a constant considered to be approximately directly proportional to p_m^o and α is a constant independent of the explosive but dependent on the nature of the rock being blasted. Thus K should be a function of the detonation pressure p_2 and the impedance ratio R. (It is also a function of the heat of explosion since the weight factor W alone does not account completely for the extensive property of the explosive. One must expect also that α is a function of the available energy A or heat of explosion Q because explosives differ sometimes appreciably in available energy. This difficulty may be avoided if W is taken to be the TNT-equivalent weight rather than the actual weight.) Theoretically, one may express these parameters by the functions

$$K = f(p_2,\Delta,R) \tag{8}$$

$$\alpha = g(W_e/W_r,A,R) \tag{9}$$

K and α are, of course, different for each type of rock. If on the other hand, one uses the concept of borehole pressure p_b instead of detonation pressure p_2 for cases in which the explosive does not completely fill the borehole, i.e., $\Delta < 1.0$, equation (8) may be written in the form

$$K = f(p_b,R) \tag{8a}$$

Atchison and Duvall[3] attempted to modify equation (5) based on results with four explosives using measured detonation velocities to compute R and p_2. They suggested the following modified impedance mismatch equation

$$p_m^o/p_2 = (1+N)/(1+NR) \tag{10}$$

They proposed the value N = 5 based on results with these four different explosives. Since two of these explosives were non-ideal their detonation velocities in the borehole should not be the same as the measure (unconfined) velocities. Therefore the basis for equation (10) seems questionable in addition to uncertainties in the meaning of $R(p_2)$. One is, in general, concerned with the application of cylindrical charges. In case the explosive does not completely fill the borehole, there is a serious ambiguity in the use of the detonation pressure p_2 as being truly representative of the pressure applied to the rock and conditions contributing to the impedance ratio R. Even when the explosive fills the borehole completely, there is no assurance that the detonation wave will extend all the way to the rock-explosive interface owing to an observed "edge effect" which does not always disappear even under strong confinement, especially in the most non-ideal explosives. Moreover, the detonation pressure p_2 is very short in duration or transient; the borehole pressure p_b exists unattenuated for a much longer time.

(This is based on an assumed negligible compressibility of the rock; when compressibility is taken into account, the borehole pressure is found also to be relatively transient in favor of a still lower, more sustained pressure.) The (ideal) borehole pressure is identically the adiabatic or explosion pressure p_3 at $\Delta = 1.0$, but at lower loading densities it is related to p_3 by a relation of the form

$$p_b = p_3 \Delta^n \qquad (11)$$

where n is a constant between 2.0 and 3.0[20]. Detonation pressure is given (neglecting ambient pressure) by the relation

$$p_2 = \rho_1 DW \qquad (12)$$

where D is the detonation velocity, W is the particle velocity and ρ_1 is the initial density of the explosive. Using the (good) approximations $p_2 \doteq 2p_3$ and $W/D \doteq 0.25$, the impedance of the explosive in terms of the explosion pressure for impulse transfer through the end of a detonating charge becomes approximately

$$(\rho_1 D) = (8\rho_1 p_3)^{1/2} \qquad (13a)$$

The effective borehole impedance should more properly be related, not to p_2 or p_3, but to the actual pressure which the borehole experiences, namely p_b. Therefore, in the general case borehole impedance should be given by the relationship

$$(\rho V)_e = (4\Delta \cdot \rho_1 p_b)^{1/2} \qquad (13)$$

The relative borehole impedance R_b should, therefore, be given approximately by the relation

$$R_b \doteq (0.4\Delta \cdot \rho_1 p_b)^{1/2} / (\rho V)_r \qquad (14)$$

for $(\rho V)_r$ in g/cc · km/sec, p_b in kb and ρ_1 in g/cc. The initial peak pressure p_m^o in the rock is then given by

$$p_m^o = 2p_b/(1+R) \qquad (15)$$

The (shock) coupling should therefore be related to Δ, or the "decoupling" factor $\Delta^{1/2}$ defined by Atchison[5], by the relationship

$$p_m^o(\Delta) = p_m^o(1)\cdot\Delta^n\left[1+R(1)\right]/\left[1+R(1)\cdot\Delta^{(n+1)/2}\right] \qquad (16)$$

where R(1) and p_m^o are values of the relative impedance and initial peak pressure in the rock at $\Delta = 1.0$. Taking $n = 2.5$ one finds that for $R(1) \ll 1.0$ "decoupling" should vary as $\Delta^{2.5}$; for $R = 1.0$ it should vary as $2\Delta^{2.5}/(1+\Delta^{1.75})$ and for $R \gg 1.0$ it should be given approximately by $\Delta^{2.5}\ R(1)/\ 1+R(1)\Delta^{1.75} \approx \Delta^{0.75}$. Based on upper and lower limits of R, one thus expects p_m^o to vary within the limits $\Delta^{0.75}$ and $\Delta^{2.5}$ for values of Δ not too far below unity.

Studies of decoupling in limestone by Atchison[5] showed it to vary about as $\Delta^{0.75}$, and comparable results were obtained in granite by Atchison and Duvall[3]. On the other hand conditions employed in these investigations were such that results should have varied as $\Delta^{2.0+}$ because $R_b < 0.5$ in all cases considered by them.

b. Energy Coupling

In the theory of energy coupling the impedance mismatch equation is not applicable; if one neglects compressibility, the borehole pressure p_b will be the actual pressure applied on the inner (borehole) boundary at all stages prior to emergence of the shock wave into the free surface of a properly loaded blast. This will then be the initial pressure p_m^o in the rock. In the idealized case, therefore, the energy theory predicts that the (effective) decoupling should vary also approximately as $\Delta^{2.5}$. The observed decoupling factor $\Delta^{0.75}$ may, however, be accounted for in the energy theory by taking into account rock compressibility.

The strain energy density ϵ in the compression of rock is approximately $\overline{\beta}p^2/2$, where $\overline{\beta}$ is the average compressibility at pressure p. Let us assume a pressure distribution function for rock compression in cylindrical expansion of the compression wave in a long borehole to be given by the equation

$$p_m = p_m^o e^{-a(t)x} \qquad (17)$$

where $x = r/r_o$ and $r_o = V_r t_o$. The constant a is given by the equation

$$a = \ln p_m^o/p(r_o) \qquad (18)$$

The total energy of compression is then obtained by integrating over the whole volume of the rock under compression giving

$$E_T = p_m^{o2}\overline{\beta}\pi r_o^2 \cdot L/8a^2 \tag{19}$$

The maximum available energy density of an explosive at its maximum density is approximately p_3[24]. Therefore, for a charge of $\triangle = 1.0$ and $\rho_1 = 1.5$ g/cc one obtains, using equation (18) and this approximation for A, the result

$$r_o/r_b = 4.6(E_t p_b/\rho_1 W_e A\beta p_m^{o2})^{1/2} \log p_m^o/p_m(r_o) \tag{20}$$

for the ratio of the burden to the borehole radius.

At r_o it has been observed that $p_b/p_m(r_o) \sim 500$ for blasts employing an explosive of density around $\rho_1 = 1.5$ g/cc, and borehole pressure $p_b \doteq 100$ kb ($\Delta = 1.0$) corresponding to the best modern blasting agents. Since E_T may be known as a function of p_m^o from the theory of the maximum available work function A (cf. Figure 11.6, ref. 20) one may therefore use it to obtain, via equation (20), the ratio p_m^o/p_b. For $E_T/AW_e \sim 0.5$, for example, $p_m^o \sim 0.25\ p_b$ as seen in the above reference. Then equation (20) becomes roughly $r_o/r_b = 25 \log p_b/4p(r_o)$. Taking $p_b/4p_m(r_o) = 100$ one then finds $r_o/r_b = 50$. This agrees essentially with the observed distance to the free surface of a properly loaded blast. It thus justifies qualitatively the p(r) relations depicted in Figure 1 and shows that a large portion of the energy of the blast is, indeed, stored temporarily as compression energy in the rock at the instant that fragmentation begins at the free surface. Of course, half of this energy then is in the region back of the borehole, but in the subsequent stress relief this energy is partially, at least, transferred to the other half of the blast to assist in further stress relief fragmentation.

The above considerations apply in the case of an ideally loaded blast. Let us now consider conditions existing (a) in an underloaded blast (excessively low W_e/W_r) and (b) in an overloaded blast (excessively high W_e/W_r). In considering poorly loaded blasts, note that $a(t_o)$ will not be changed if all conditions are the same except that the burden has been increased (a) or decreased (b). However, p_m at the free surface will be different in each case.

Let us consider the case in which all conditions of a blast are the same as those depicted in Figure 1 except that the burden is 20 percent greater ($r_o' = 1.2\ r_o$). At the time t_o after the blast, therefore, the rock in the region between the borehole and the distance r_o will be under identically the same pressure as in the case in which the free surface is encountered at r_o (dotted line, Figure 2). Since no free surface is encountered at this stage there is no fragmentation except in the borehole fracture zone. Instead, the compression wave must continue on the additional time $t_o/5$ before fragmentation can commence. When it reaches the free surface there has been considerable attenuation of compression throughout the burden. Moreover, the amplitude of the wave when it reaches r_o' may have dropped below that required to produce release wave fragmentation. Stress relief then occurs much more gradually so that fragmentation in the stress-relief zone becomes much less extensive. This type of shot produces excessive back break owing to stress relief then tending to become symmetrical around the borehole, and fragmentation is poor.

Conditions will be opposite in case (b). Figure 3 depicts conditions for the case in which again all conditions are identical with those in Figure 1 except that the burden is now 20 percent too small. Free surface fragmentation then begins at $t_o'' = 0.8\ t_o$ at which point the pressure $p_m(r_o'')$ is considerably higher. This results in greater release wave fracturing and in ejection of rock at much higher velocity; the burden experiences excessive "throw". The stress-relief zone fragmentation becomes excessive at $r_o'' = 0.8\ r_o$ owing to more sudden release from higher pressures.

Consider now the case of a blast in which all conditions are the same except that $\Delta = 0.75$ and, therefore, $p_b \sim 0.4\ p_3$. That is, the "powder factor" and the "burden" are the same as in Figure 1, but the borehole is a third greater in volume than in the blast depicted in Figure 1. The result of the blast is almost the same as that depicted in Figure 2. That is, decoupling to the extent $\Delta = 0.75$ has nearly the same effect on fragmentation as increasing the burden about 20 percent. This situation is depicted in Figure 4.

The above illustrates the theory qualitatively; in a quantitative application there are three unknowns in equation (20), namely $E_T(t)$, $p_m^o(t_o)$ and $p_m(r_o)$ which must be defined. They are interrelated through the theory of the maximum available energy A because in reversible expansion, A is a single-valued function of the specific volume v of the gaseous products of detonation. Any loss of energy from the gases to the burden is associated with an increase in the specific volume $\Delta v(t)$ of the gases in the borehole. By knowing $\Delta v(t)$ one may then compute $A(t)$ by the method given in ref. 20, p. 267. (The ratio E_T/W_eA in equation (20) is identically $A(t)/A$, where $A(t)$ is the work done on the burden per unit weight of explosive during the time t and A is the ultimate work per unit weight of explosive over the whole period that the gases are able to do work on the burden.) One is thus primarily interested in $\Delta v_b(t_o)$, the increase in specific volume of the borehole at t_o. This may be obtained from the equation

$$W_e \Delta v_b(t_o) = -W_r \bar{\Delta} v_r = \int_o^{r_b} \int_o^{p_m} 2\pi r \beta\, dp\, dr$$

giving

$$\Delta v(t_o) = 2\pi r_o^2\ L\bar{\beta} p_m^o / W_e a^2 \qquad (21)$$

where minus the total increase in volume of the borehole at time t_o has been equated to the total decrease in volume of the rock due to compression at this particular stage of the blast.

Since p_m^o may be derived from p_b and $v_b(t)$ once $A(t)$ is known the only remaining unknown in equation (20) is $p_m(r_o)$. For a properly loaded blast $p_m(r_o)$ must be large enough to permit fragmentation to begin at the free surface, i.e., it must be several times greater than S_t. From the Bureau of Mines studies $p_m(r_o)$ for a good blast is $N_i S_t$ where $N_i \sim 4$. For example, for granite $p_m(r_o) \sim 0.25$ kb and $S_t = 0.075$ kb. If a borehole in granite is loaded with an explosive of $p_b = 100$ kb one then finds, by an iterative simultaneous solution of equations (20) and (21), that $p_m^o \sim 25$ kb and $p_m^o/p(r_o) = e^a \sim 100$, giving $a \sim 4.6$. Therefore $\Delta v_b(t_o)/v_b \sim 3$; the borehole apparently has increased in diameter by approximately a factor of 2.0 (Figure 5) at the instant (t_o) that the shock wave reaches the free surface in a properly loaded shot in granite using an explosive with $p_b = 100$ kb.

In applying the stress-relief theory here outlined to various types of rock to be blasted with different explosives one should first determine more accurately the ratio $p_m(r_o)/S_t$ for best results. Then p_m^o, the ratios $p_m^o/p(r_o) = e^a$ and $A(t_o)/A$, and $\Delta v_b(t_o)$ may be obtained with good accuracy by an iterative procedure. This should permit one to predict optimum loading ratios W_e/W_r for the various combinations of rock and explosive. It will thus be necessary in the further development of the theory to provide more reliable data on $p(r_o)/S_t$ and $\overline{\beta}$ for various rocks and on p_b, A and p_m^o for various explosives.

c. Explosives

The parameters of equation (17) have a complex dependency; they depend ultimately not only on the burden, spacing, borehole diameter and depth and the properties of the rock, but also on loading density Δ and the explosive density ρ_1 (or pressure) and A. To emphasize the part played by the explosive let us express $p_m^o(t)$ and $a(t)$ in the form

$$p_m^o(t) = (\Delta \cdot \rho_1)^n \; f(\emptyset_r, X) \tag{23}$$

$$a(t) = g(\emptyset_r, X)/A^m \tag{24}$$

where $n \sim 2.5$, $m \sim 1/3$, f and g are functions of the physical and chemical properties of the rock $\emptyset_r$, and on the geometrical factors X (burden, spacing and borehole diameter and depth). Thus, maintaining f and g constant p_m^o is found to vary essentially as $(\rho_1 \cdot \Delta)^n$ and $\underline{a}$ as $A^{-1/3}$, which are the important factors pertaining to the explosive irrespective of whether one accepts the shock wave theory, the energy theory or both.

Prior to the advent in 1955 of "prills and oil" (the 94/6 prilled ammonium nitrate/fuel oil[13]) commercial explosives comprised principally dynamites (based on nitroglycerin) and the NCN ("nitro-carbo-nitrate") explosives comprising essentially fuel-sensitized ammonium nitrate. The average density in these products ranged from 0.6 to 1.4 g/cc, but owing to their application in cartridge form Δ was generally appreciably below 1.0. Thus seldom was their $\Delta \cdot \rho_1$ product much above that for prills and oil in which ρ_1 is only 0.85 g/cc, but Δ is generally 1.0; it is generally used in bulk form such that it always fills the borehole completely. A/A_0 for prills and oil is close to unity and therefore also about the same as for an average commercial explosive, A_0 being the maximum available energy of TNT or the "TNT equivalent". In retrospect it is thus understandable that prills and oil, generally previously regarded by explosives technologists as an inferior blasting agent, has actually performed far above expectations. Owing to its low cost, roughly only a quarter that of an equal (weight) strength dynamite, and the great importance of Δ on performance, prills and oil has replaced well over half of the dynamites and NCN explosives previously comprising the commercial market in America.

Shortly after the discovery of prills and oil the "slurry" explosives were discovered by Cook and Farnam[13,25,26]. Slurries are based on thickened or gelatinized aqueous ammonium nitrate solutions; they differ in type depending on the sensitizer employed in them, types being:

a) Coarse TNT and TNT-aluminum slurries

b) Smokeless powder slurries

c) The NCN slurries in which the sensitizer is a non-explosive "fuel", e.g., aluminum, emulsified fuel oil, etc.

Slurries are characterized by their high density and fluidity which makes it relatively easy for them to attain $\Delta = 1.0$ in the borehole. The $\Delta \cdot \rho_1$ product is thus generally 1.5 to 2.5 times greater for slurry than for prills and oil. The A/A_0 ratios of slurries range from 0.85 to 1.5, those with the highest aluminum contents developing borehole pressures about five times greater and A/A_0 values up to 1.5 times greater than for prills and oil.

On an equal strength basis the cost of slurries averages about half to two-thirds that of the older commercial explosives and about twice to three times that of prills and oil.

By virtue of the much higher $\rho_1 \cdot \Delta$ of slurry compared with prills and oil and the older commercial explosives, and excellent new methods for economic, rapid and safe loading of them at $\Delta = 1.0$ they represent a major advance in the commercial explosives field and should not only replace dynamites and other explosives in wet hole and underwater blasting where prills and oil is inapplicable, but may even replace much of the prills and oil in dry hole blasting. Already slurries are being produced in quantities exceeding 10^8 pounds per year in the U.S.A., Canada and foreign countries.

Novel Loading Method

Until recently the importance of $\Delta \cdot \rho_1$ was not fully appreciated, operators frequently using larger and larger boreholes to obtain their necessary high powder factors rather than taking full advantage of best methods for maximizing $\Delta \cdot \rho_1$. With the current much better appreciation of coupling and high borehole pressures many are turning to the slurry explosives to achieve high powder factors in blasts of high burdens without having to increase borehole diameters. In fact, some are even now contemplating reduction in borehole diameters. The savings resulting from this more scientific application of explosives are not only reduced explosives and drilling costs, but also reduced shovel, crushing and grinding costs, and sometimes also large increases in the rates of production.

Successful bulk loading units and field mixing methods for prills and oil developed rapidly after its introduction(13,27). On the other hand, considerable research was required for the development of bulk handling methods for slurries; field mixing and loading proved much more difficult with slurries because 1) slurries with suitable properties (water resistance, plasticity and high density) require much more accurate control in their formulation, and 2) they are usually applied under more difficult conditions, e.g., in water-filled holes. After several years of research an excellent new principle of handling slurries was recently introduced by IRECO having great potential for rapid development and extensive application especially in the large operations. This is a unique on-site mixing and loading method called the "pump truck" method. Pump truck units were introduced early in 1963 simultaneously at the Kaiser Steel Corporation's Eagle Mountain Mine in California, at the Iron Ore Company of Canada's Nob Lake and Carrol Lake Mines in Northern Quebec and on the Northern Minnesota and upper Michigan iron ranges.

A photograph of a unit now in routine operation is shown in Figure 6. The pump truck method utilizes a hot, preconditioned, aqueous solution of ammonium and sodium nitrate with other additives, e.g., one for prevention of the aluminum-water reaction, and another for promoting rapid slurry gelatinization. Solid ingredients are fed by vibrators from one, two or three storage bins depending on the particular slurry desired. The hot solution is fed into the slurry stream by an especially designed pump. All ingredients come together in a second specially designed pump which quickly mixes them and forces them rapidly through a long nose into the borehole. The rate of mixing and loading averages more than 400 lbs/min. The truck may be loaded at a nearby storage facility, e.g., with about 25,000 lbs of ingredients, in about 15 minutes, or it may be loaded continuously during loading of boreholes depending on the method desired. It is thus possible to mix and load with a single such unit up to about 50 tons of slurry per eight-hour shift.

The products produced by the pump truck method are generally superior to corresponding plant-manufactured products because they are generally higher in density, they are more water resistant and require less of the explosively ineffective ingredients required for products that need to be stored, transported and handled. A valuable feature of the pump truck method is that it can load more than one type of slurry into the same borehole by merely switching vibrators and adjusting their speeds. For example, the most powerful, high density aluminized slurry may be loaded at the bottom or "toe" of the hole where it is most needed, and the much less costly, less powerful NCN type slurry may be used in the upper part of the borehole where less power is required. In this combination both slurries are of the NCN type, i.e., they are properly called slurry blasting agents[13] for which no storage or transportation of actual explosive is required. Thus safety is maximized; when only NCN type slurries are used, the blasting agent made by the pump truck method may be formulated to become a detonatable explosive only when it is actually in place in the borehole. Such NCN-slurries, moreover, may be formulated for use in water-filled boreholes by the same mixing procedures used in dry boreholes, except that the product is then extruded from the loading hose into the bottom instead of the top of the borehole. This procedure does not reduce the loading rate appreciably, but prevents the finished slurry from mixing with water as it would do if it were required to fall through water. One may provide additional water resistance for the slurry by extruding it into a large diameter, polyethylene tube that may be quickly and easily raveled over the end of the hose, pushed to the bottom of the borehole, then pushed off the end of the loading hose by the extruded slurry to line the borehole as the slurry fills it.

Possibly the future of NCN-slurry blasting agents may be judged by the fact that one such type (designated DBA-KS) made available for use only by the ideal conditions made available only by the use of pump truck mixing and loading, actually costs less than prills and oil when both explosive and loading costs are considered. Despite its low cost this slurry is considerably better than prills and oil, e.g., its properties are $\rho_1 \cdot \Delta \sim 1.4$ g/cc, $p_b \sim 50$ kb and A/A_0 about 0.9. Reduction in the costs of explosive, drilling, shovel, crushing, and grinding are all phenomenal with this product especially when used in conjunction with the powerful aluminized slurry (DBA-10) in bottom loads.

TNT slurries are sometimes preferred to the NCN type because they are more reliable under difficult conditions, e.g., running water and in very cold boreholes. The TNT most suitable for pump truck handling is the coarse, largely +10 mesh "pelletol" TNT manufactured by the duPont Company or the

"Nitropel" TNT made by CIL in Canada. This coarse TNT product is not cap sensitive and has a critical diameter of 2". Since it is therefore no more sensitive than some of the field-mixed and bulk loaded prills and oil products, particularly those used in small diameter, underground blasting, and since the handling of this product comprises simply the periodic reloading of the bulk TNT bin on the pump truck from which it is fed into the slurry stream via a safe vibrator feed, the TNT slurry made and loaded by the pump truck presents no hazard greater than the use of corresponding plant-manufactured products. The pump truck method of mixing and loading slurry explosives and blasting agents should thus prove to be a major advance in commercial blasting that will result in increased safety, improved performance, reduced costs and greater rates of production.

REFERENCES

1. Bureau of Mines Reports of Investigations:
   a. "Effects of Stress Relief...on...Rock", L. Obert, 6053 (1962)
   b. "Comparative Studies of Explosives in Salt", H. R. Nichols and V. F. Hooker, 6041 (1962).
   c. "Comparative Studies of Explosives in Marble", T. C. Atchison and J. Roth, 5797 (1961).
   d. "Comparative Studies of Explosives in Granite", T. C. Atchison and W. G. Tourney, 5509 (1959).
   e. "Photographic Observations of Quarry Blasts", B. Petkof, T. C. Atchison and W. I. Duvall, 5849, (1961).
   f. "Strain Energy in Explosion-Generated Strain Pulses", D. E. Fogelson, W. I. Duvall and T. C. Atchison, 5514 (1959).
   g. "Spherical Propagation of Explosion-Generated Strain Pulses in Rock", W. I. Duvall and B. Petkof.
   h. "Rock Breakage by Explosives", W. I. Duvall and T. C. Atchison, 5356, (1957).
   i. "...Strain Waves in Rock", L. Obert and W. I. Duvall, 4683, (1950).

2. Duvall, W. I., Geophysics 18, 310 (1953).

3. Atchison, T. C. and Duvall, W. I., "Effect of Characteristic Impedance on....Strain Pulses in Rock", Rock Mechanics, Pergamon Press, p 331, 1963.

4. Nichols, H. R., and Duvall, W. I., ibid, p331, 1963.

5. Atchison, T. C., "Effects of Coupling on Explosive Performance", Drilling and Blasting Symposium, Colorado School of Mines, Quarterly, Jan. 1961.

6. Hino, K., "Theory and Practice of Blasting", Nippon, Kayaku Co., Ltd., 1959.

7. Twentieth Congress, Acad. Sci. USSR Min. Inst., "Problems of the Theory of Destruction of Rocks by Explosives", Publishing House of the Academy of Sci., USSR, Moscow, 1958
   a. A. N. Khanukayev, "Physical Nature of Rock Breakage", pp 6-56.
   b. A. V. Kovazhenkov, "Investigation of Rock Breakage", pp 99-130.

8. Poncelet, E. F., "Theoretical Aspects of Rock Behavior Under Stresses", Fourth Symp. on Rock Mechanics, Science Press, Inc., Ephrata, Pa, 1961; Inst. Min. Met. Engrs. Techn. Publ. No. 1684 (1944).

9. Bauer, A., "Blasting Characteristics of Frozen Ore and Overburden", Canadian Industries Ltd, Montreal, 1961

10. Livingston, C. W., "Theory of Fragmentation in Blasting", Ann. Drilling and Blasting Symp., University of Minn., Oct., 1956.

11. Johansson, C. H., and Langefors, U., Mine and Quarry Engg 17, 287 (1951).

12. Selberg, H., "Transient Compression Waves from Spherical and Cylindrical Cavities", Kangl. Svenska Vetenskopsakal. Arkiv for fysik (Stockholm) 4, (1951); 5(1952).

13. Cook, M. A., Science 132, 1105 (1960).

14. Rinehart, J. S., Quarterly of the Colorado School of Mines, Vol. 55, No. 4, October 1960.

15. Duff, R. E., and Houston, E. E., "Measurement of the C-J Pressure", Second ONR Symp. on Detonation, Washington D.C., Feb. 1955, p. 225.

16. Mallory, H. D., and Jacobs, S. J., "The Detonation Reaction Zone in Condensed Explosives", ibid., p. 240.

17. Walsh, J. M., and Christian, R. H., Phys. Rev. 97, 1544 (1955).

18. Rice, M. H., and Walsh, J. M., J. Chem. Phys. 26, 824 (1957).

19. Rinehart, J. S., and Pearson, J.,"Behavior of Metals under Impulsive Loading", Am. Soc. Met., Cleveland, 1954.

20. Cook, M. A., "The Science of High Explosives", Reinhold Publishing Corp., New York, 1958.

21. Kolsky, H., Stress Waves in Solids, Claredon Press, Oxford (1953).

22. Cook, M. A., Keyes, R. T., and Ursenbach, W. O., J. Appl. Phys. 33, 3413 (1962).

23. Bauer, A., and Cook, M. A., Trans. Cana. Inst. Min. and Met. 61, 62(1961).

24. Vance, R. W., Cryogenic Technology, Chapter 13, (M. A. Cook), John Wiley and Sons, Inc., New York, 1963.

25. Cook, M. A., Farnam, H. E., USP 2,930,685, Mar. 29, 1960; Canadian Patents 619,653, May 9, 1961, 605,314, Sept. 20, 1960, 597,177, May 3, 1960, (Other countries).

26. Cook, M. A., "Water Compatible Ammonium Nitrate Explosives for Commercial Blasting", Univ. of Missouri School of Mines and Metallurgy Bulletin, Technical Series No. 97, 1958.

27. Farnam, H. E., Jr., J. Am. Min. Congr., Mar. 1958; Ann. Drilling and Blasting Symp., Univ. of Minn., 1958.

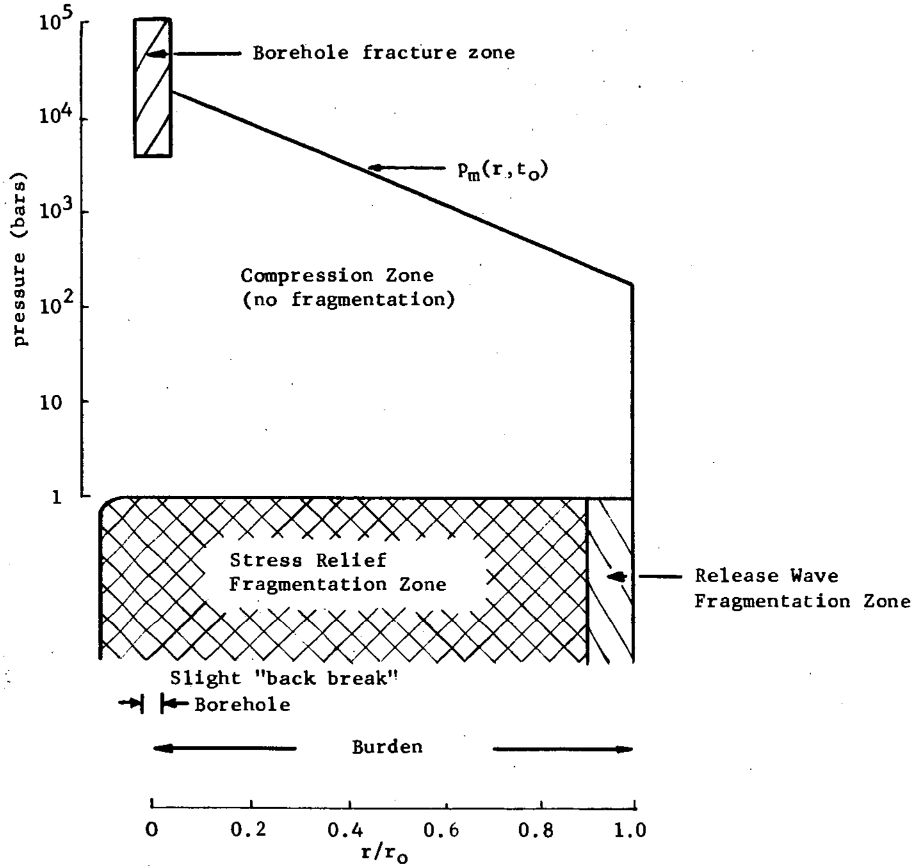

Figure 1. The $p_m(t_o)$ curve and fragmentation zones.
(p_b = 100 kb, Δ = 1.0, ρ_1 = 1.5 g/cc, W_e/W_r - ideal)

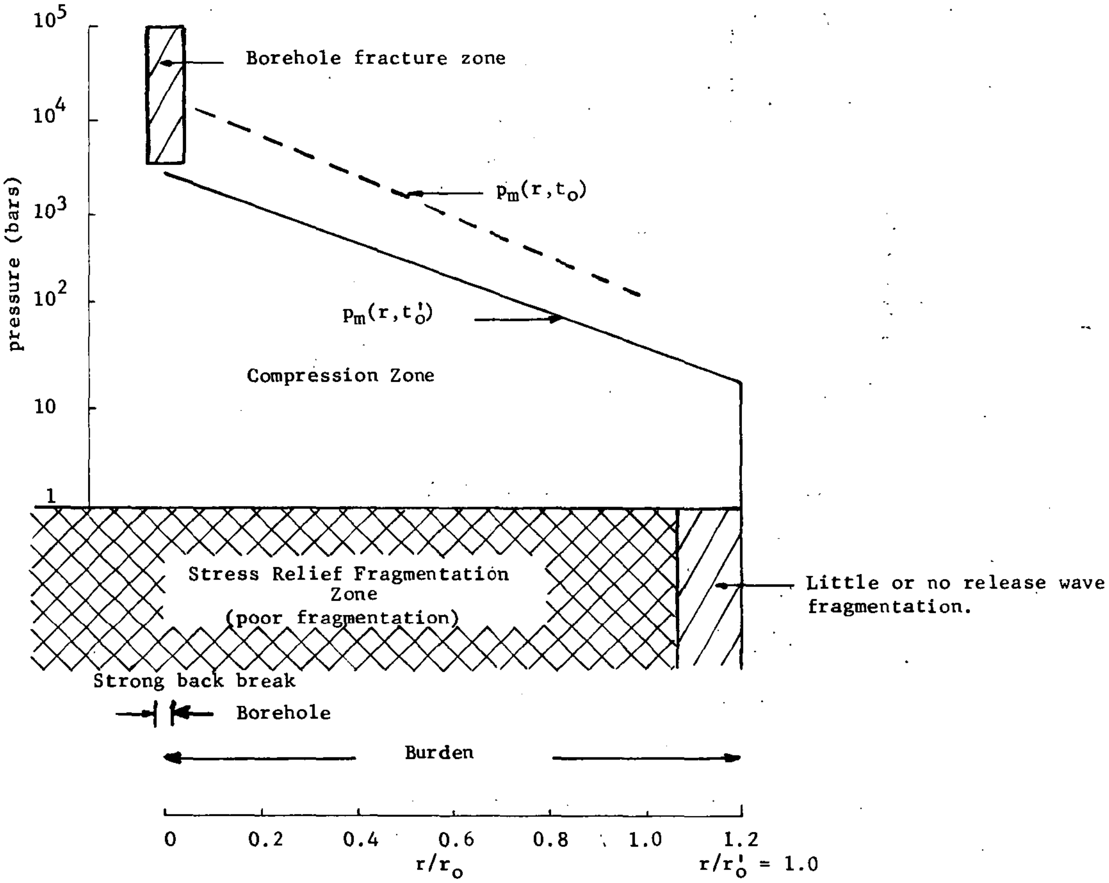

Figure 2. The $p_m(t_o')$ and $p_m(t_o)$ curves and fragmentation zones. (p_b = 100 kb, ρ_1 = 1.5 g/cc, Δ = 1.0, W_e/W_r too low)

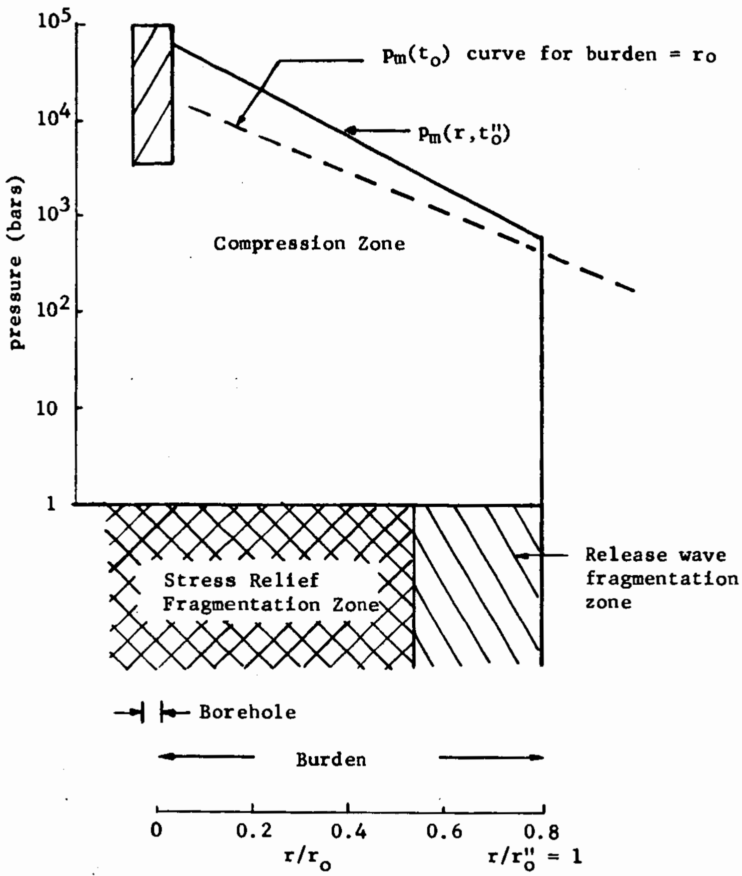

Figure 3. The $p_m(t_o'')$ curve and fragmentation zones for $t_o'' = 0.8\ t_o$ (p_b = 100 kb, ρ_1 = 1.5 g/cc, Δ = 1.0, W_e/W_r too large)

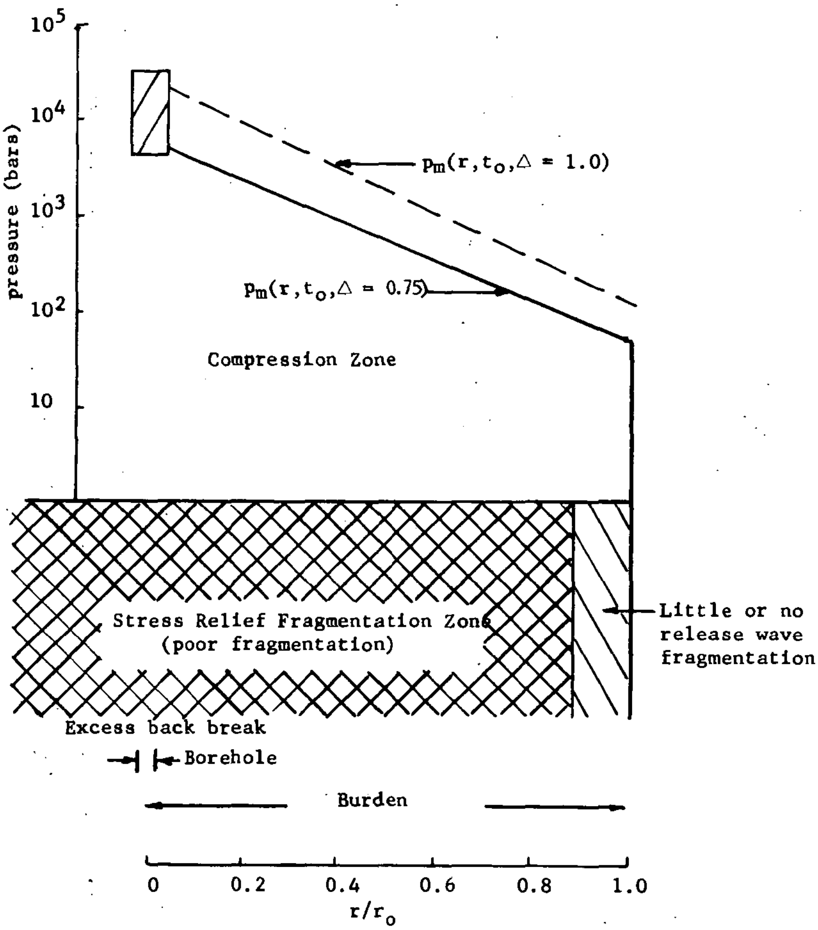

Figure 4. The $p(t_o)$ curve and fragmentation zone. (p_b = 40 kb, ρ_1 = 1.5 g/cc, Δ = 0.75, W_e/W_r - same as in Figure 1)

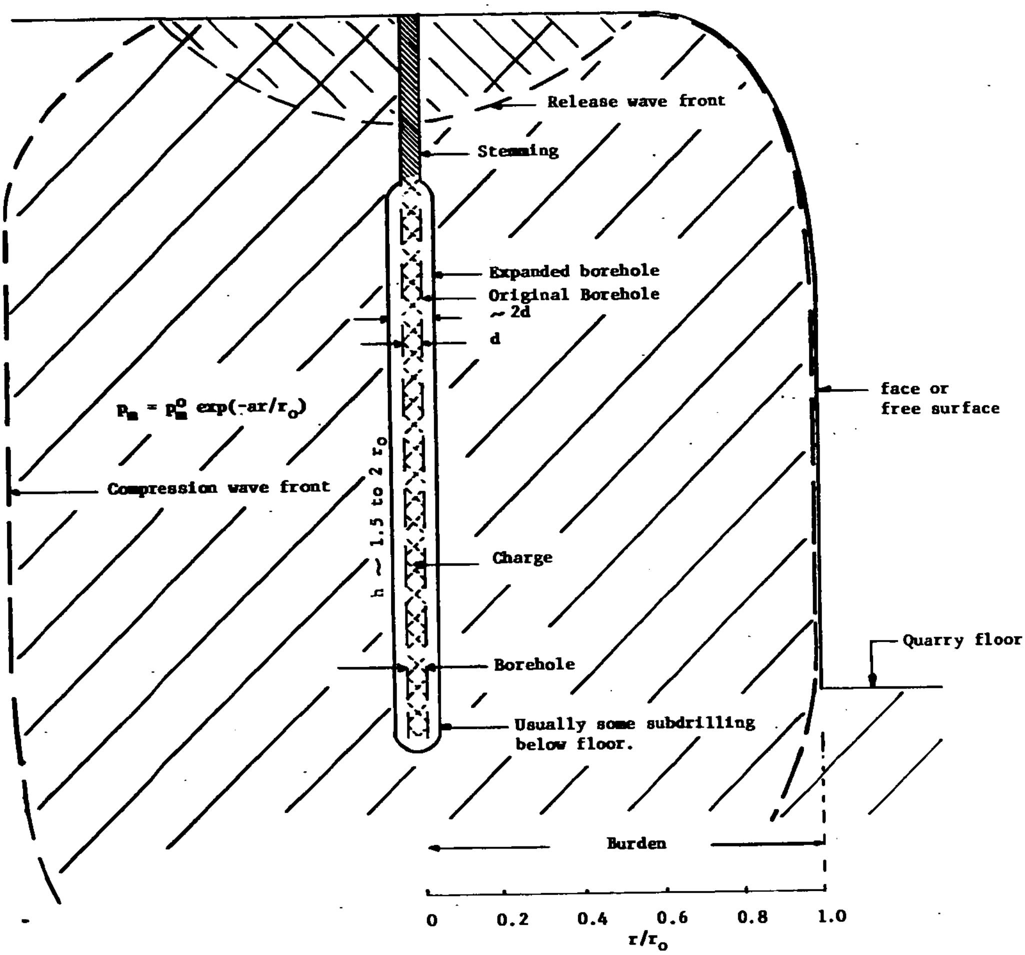

Figure 5. Diagram of borehole and compressed burden at t_o for 9" borehole filled with "slurry".