DIVISION DE EDUCACION CONTINUA

CURSOS ABIERTOS

VI CURSO INTERNACIONAL DE INGENIERIA GEOLOGICA APLICADA A OBRAS SUPERFICIALES Y SUBTERRANEAS

MODULO IV: TECNOLOGIA SOBRE EL USO DE EXPLOSIVOS

Del 20 al 24 de junio de 1994

DIRECTORIO DE PROFESORES

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MODULO IV: TECNOLOGIA SOBRE EL USO DE EXPLOSIVOS

Del 20 al 24 de junio de 1994

FECHA	HORARIO	TEMA	PROFESOR
Lunes 20	9;00 a 19;00 hrs.	Propiedades físicas y químicas de los explosivos	Ing. Raúl Cuéllar borja
Martes 21	9;00 a 19;00 hrs.	Propiedades geométricas y mecánicas de las rocas	Ing. Raúl Cuéllar Borja
Miércoles 22	9;00 a 19;00 hrs.	Mecanismos de fragmentación	Ing. Raúl Cuéllar Borja
Jueves 23	9;00 a 19;00 hrs.	Fundamentos sobre el diseño de voladuras.	Ing. Raúl Cuéllar Borja
Viernes	9;00 a 15;00 hrs.	Fundamentos sobre el diseño de voladura.	Ing. Raúl Cuéllar Borja
	15;00 a 19;00 hrs.	Efectos adversos de las voladuras	Ing. Jaime Ruíz Reyes

EVALUACION DEL PERSONAL DOCENTE

CURSO: Módulo IV: Tecnología sobre el uso de explosivos.
FECHA: Del 20 al 24 de junio de 1994.

CONFERENCISTA	DOMINIO DEL TEMA	USO DE AYUDAS AUDIOVISUALES	COMUNICACION CON EL ASISTENTE	PUNTUALIDAD
Ing. Raúl Cuéllar Borja				
Ing. Jaime Ruiz Reyes				

EVALUACION DE LA ENSEÑANZA

ORGANIZACION Y DESARROLO DEL CURSO	
GRADO DE PROFUNDIDAD LOGRADO EN EL CURSO	
ACTUALIZACION DEL CURSO	
APLICACION PRACTICA DEL CURSO	

EVALUACION DEL CURSO

CONCEPTO	CALIF.
CUMPLIMIENTO DE LOS OBJETIVOS DEL CURSO	
CONTINUIDAD EN LOS TEMAS	
CALIDAD DEL MATERIAL DIDACTICO UTILIZADO	

ESCALA DE EVALUACION: 1 A 10

1.- ¿LE AGRADO SU ESTANCIA EN LA DIVISION DE EDUCACION CONTINUA?

SI	NO

SI INDICA QUE "NO" DIGA PORQUE.

2.- MEDIO A TRAVES DEL CUAL SE ENTERO DEL CURSO:

PERIODICO EXCELSIOR		FOLLETO ANUAL		GACETA UNAM		OTRO MEDIO	
PERIODICO EL UNIVERSAL		FOLLETO DEL CURSO		REVISTAS TECNICAS			

3.- ¿QUE CAMBIOS SUGERIRIA AL CURSO PARA MEJORARLO?

4.- ¿RECOMENDARIA EL CURSO A OTRA(S) PERSONA(S)?

SI		NO	

5.- ¿QUE CURSOS LE SERVIRIA QUE PROGRAMARA LA DIVISION DE EDUCACION CONTINUA.

6.- OTRAS SUGERENCIAS:

DIVISION DE EDUCACION CONTINUA
CURSOS ABIERTOS

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MODULO IV: TECNOLOGIA SOBRE EL USO DE EXPLOSIVOS

Del 20 al 24 de junio de 1994.

DIRECTORIO DE ASISTENTES

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FACULTAD DE INGENIERIA U.N.A.M.
DIVISION DE EDUCACION CONTINUA

CURSOS ABIERTOS

VI CURSO INTERNACIONAL DE INGENIERIA GEOLOGICA APLICADA A OBRAS SUPERFICIALES Y SUBTERRANEAS

MODULO IV: TECNOLOGIA SOBRE EL USO DE EXPLOSIVOS

USO DE EXPLOSIVOS EN ROCA

ING. RAUL CUELLAR BORJA

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USO DE EXPLOSIVOS EN ROCA

ANTECEDENTES

El uso de los explosivos es más una técnica que un arte. Hasta ahora el método más económico para fragmentar la roca es mediante el uso de explosivos.

La teoría está soportada por la práctica, de tal manera que el diseño de voladuras se realiza más por la relación entre parámetros que mediante fórmulas teóricas, por ejemplo: la relación entre el diámetro y el bordo. Es necesario comprender cómo trabaja el explosivo en la roca, para lo cual se requiere del conocimiento de las propiedades de los dos elementos, *la roca* y *los explosivos*.

EN RELACION A *LA ROCA* SE PUEDE DECIR LO SIGUIENTE:

Calidad

Tenemos una gran variedad en la calidad de los macizos rocosos en función de su estructura y resistencia (*caracterización del macizo rocoso*). Este término de calidad involucra muchas propiedades del macizo rocozo, por ejemplo: velocidad de transmisión de ondas de compresión P, resistencia en compresión simple, densidad, dureza, anisotropía, homogeneidad, flujo de agua, temperatura y estado de esfuerzos interno, son algunas de las propiedades más importantes de las rocas para su utilización en el diseño de voladuras.

Mecanismo de fragmentación

En todos los tipos de roca tenemos que la resistencia en compresión simple es mucho mayor que la resistencia en tensión, cortante o flexión (del orden de 10 veces para tensión y cortante y 4.5 veces para flexión).

De acuerdo con lo anterior, los mecanismos de fragmentación están diseñados para *romper la roca por tensión, corte y flexión* más que por compresión.

Cuando existe una cara libre se produce el fenómeno de reflexión y refracción de las ondas de choque de compresión o primarias P, creándose vibraciones de alta frecuencia (150 a 200 c.p.s.) que dán lugar a impactos de tensión intermitentes por razón de la fuerza centrífuga hasta que estas fuerzas de inercia vencen la resistencia a la tensión de la roca y entonces se produce el desprendimiento de fragmentos de roca a partir de la periferie hacia el centro.

Por otro lado, las fracturas de tensión en el cilindro de pared gruesa avanzan y los gases penetran en ellas produciendo el desplazamiento de los fragmentos de roca. También se produce un efecto combinado, semejante a una viga con un apoyo empotrado y otro libre bajo la carga de presión producida por el explosivo.

EN RELACION AL EXPLOSIVO SE TIENE LO SIGUIENTE:

Que la generación de la explosión o voladura ocurre por oxidación o reducción de combustible a alta presión. Durante esta reacción se producen temperaturas de 5000°C y gases a presiones muy altas que varían entre 15 000 y 150 000 kg/cm^2.

Esta presión se produce súbitamente en forma de impacto, propagándose las ondas de choque a velocidades entre 2000 y 7000 m/seg.

El trabajo realizado por 1 kg de TOVEX es de 580 ton-m/seg, o sea, que puede levantar 1 ton a una altura de 580 m en un segundo, equivalente a 5800 KW y 100 kg a 580 000 KW.

INGREDIENTES Y COMPOSICION DE LOS EXPLOSIVOS

La mayor parte de los explosivos comerciales son mezclas de compuestos que contienen 4 elementos básicos: carbón, hidrógeno, nitrógeno y oxígeno.

Otros compuestos con elementos tales como sodio, aluminio y calcio, se incluyen para producir ciertos efectos deseados.

Como regla general estos componentes deben dar un balance de oxígeno correcto.

Esto significa que durante la reacción todo el oxígeno disponible en la mezcla reaccione solamente para formar vapor de agua (H_2O) y que con el carbón reaccione para formar únicamente bióxido de carbono (CO_2) en forma de gas y el nitrógeno quede libre formando sólo gas nitrógeno (N).

Cuando hay otros elementos además de los cuatro básicos, por ejemplo sodio, deberá incluirse suficiente oxígeno adicional para lograr una combinación balanceada.

Cuando hay exceso de oxígeno disponible, se producen gases altamente venenosos, como los gases nitrosos NO o NO_2 (óxidos de nitrógeno). Estos gases son fácilmente detectables por su olor y color café-rojizo.

Por otro lado, si estamos en defecto de oxígeno, se forma el mortal gas monóxido de carbono (CO), el cual desafortunadamente no es detectado por olor ni color.

Además de la formación de gases venenosos por exceso o deficiencia de oxígeno, se produce una disminución de temperatura con una consecuente reducción en la presión de los gases producidos.

Para ilustrar los efectos del balance de oxígeno en el AN-FO (nitrato de amonio-aceite combustible) como agente explosivo, tenemos:

1. Oxígeno balanceado

$$3NH_4NO_3 + CH_2 \cdot\cdot\ 7H_2) + CO_2 + 3N_2$$

$$3(80) \quad + 14$$

$$\frac{240}{254} = 94.5\% \; ; \; \frac{14}{254} = 5.5\% \Rightarrow 0.94 \text{ K cal/gr}$$

nitrato de amonio + aceite combustible (diesel)

2. Oxígeno en exceso (positivo)

$$5NH_4NO_3 + CH_2 \quad 11H_2O + CO_2 + 4N_2 + 2NO$$

$$96.6\% \quad = 3.4\% \rightarrow 0.61 \text{ K cal/gr}$$

Además de que se produce menos temperatura y presión se produce gas nitroso (NO), que es un gas venenoso.

3. Oxígeno deficiente (negativo)

$$2NH_4NO_3 + CH_2 \rightarrow 5H_2O + 2N_2 + CO$$

$$92\% \qquad 8\% \rightarrow 0.82 \text{ K cal/gr}$$

Se tiene menor temperatura y presión y se produce monóxido de carbono (CO) que es mortal.

La reacción química más eficiente para el ANFO es 94% de nitrato de amonio y 6% de aceite combustible diesel.

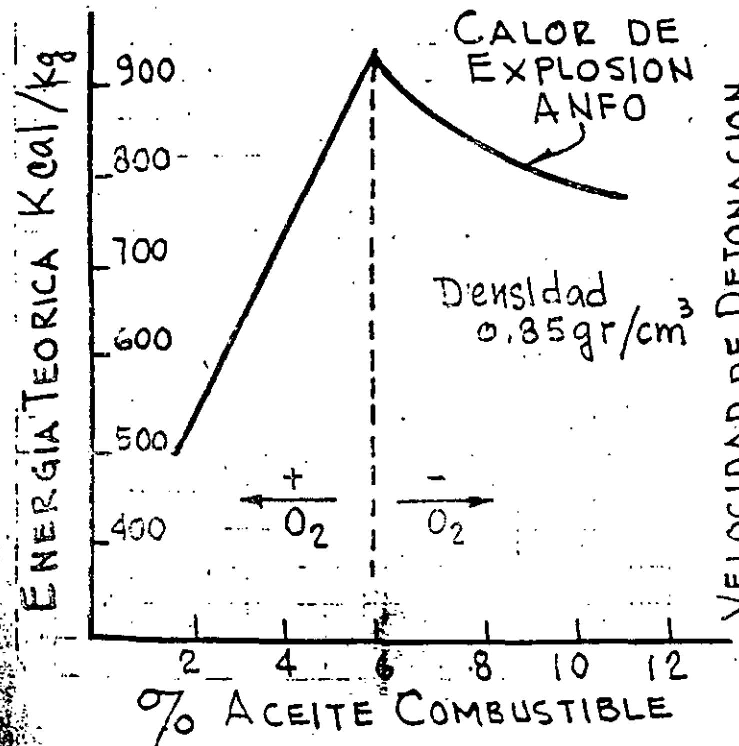
Se pueden producir otros agentes explosivos más potentes, por ejemplo utilizando aluminio:

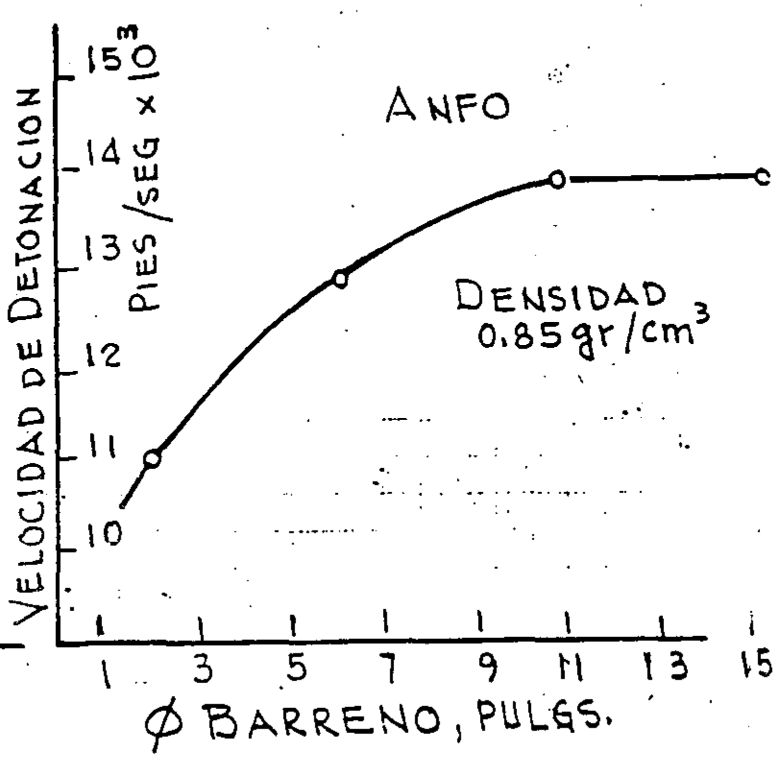
$$3NH_4NO_3 + 2A\ell \rightarrow 6H_2O + A\ell_2O_3 + 3N_2$$

240   2(27)

81.5%   18.5% → 1.5 K cal/gr

La desventaja de este compuesto para uso comercial es su alto costo. Se usa sólo para explosivos militares.





CLASIFICACION DE LOS EXPLOSIVOS

Los ingredientes usados en la fabricación de explosivos se definen como: explosivos bases, oxidantes, antiácidos y absorbentes.

Un explosivo base es un sólido o líquido que bajo la acción de suficiente calor o impacto se transforma en un producto gaseoso con acompañamiento de energía calorífica. Los combustibles y oxidantes se agregan para lograr el balance del oxígeno.

Un antiácido se agrega para incrementar la estabilidad en almacenaje y un absorbente se agrega para absorber o proteger los explisvos bases.

Un agente explosivo es cualquier material o mezcla compuesto por un combustible y un oxidante, de tal modo que ninguno de sus ingredientes sea explosivo base.

En este caso la mezcla ANFO no puede ser detonada por un estopín No. 8, que contiene 2 gr. de una mezcla de 80% de fulminato de mercurio y 20% de clorato de potasio.

El ANFO tiene baja resistencia al agua y deflagrante.

La adición de un ingrediente explosivo como el TNT, cambia la clasificación de la mezcla de agente explosivo a explosivo.

Los agentes explosivos pueden ser clasificados como -agentes explosivos secos- o -agentes explosivos "slurry"-. El ANFO (agente explosivo seco) se inició en 1950.

Hidrogeles

Los hidrogeles son los explosivos más recientemente desarrollados y actualmente son los más utilizados. Se fabrican en formulaciones tanto de agentes explosivos como de explosivos.

Contienen alta proporción de nitrato de amonio, parte del cual está en solución acuosa y dependiendo del resto de los ingredientes, puede ser clasificado como agente explosivo o explosivo.

Los agentes explosivos contienen ingredientes no sensibilizadores, como aceite combustible, carbón, azufre o aluminio, y no constituyen cápsulas-sensitivas, mientras que los explosivos hidrogeles sí contienen ingredientes como TNT que los transforma en cápsulas-sensitivas, el TNT sólo es una cápsula-sensitiva. Las mezclas del nitrato de amonio y los aceites o los sensibilizadores se espesan o gelatifican con gomas para proporcionar resistencia al agua.

Los hidrogeles son más seguros y no detonan aún barrenando sobre ellos, lo cual no sucede con las gelatinas.

Dinamita pura

La dinamita pura está compuesta por: nitroglicerina (NG) y sílice (SiO_2) en proporción 50% (NG) y 50% (SiO_2) hasta 25% (NG) y 75% (SiO_2) (Kieselgur o tierra de diatomeas o infusorios). Normalmente se fabrica en 20 a 60% (NG) y 40 a 80% (NS + C, donde NS = Nitrostarch.

TABLA 1 INGREDIENTES USADOS EN LOS EXPLOSIVOS

INGREDIENTE	FORMULA	FUNCION
Nitroglicerina (NG)	$C_3H_3(NO_3)_5$	Explosivo base
Trinitrotolueno (TNT)	$C_6H_2CH_3(NO_2)_3$	Idem
Dinitrotuleono (DNT)	$C_7N_2O_4H_6$	Idem
Glicol de etileno dinitrato (EGDN)	$C_2H_4(NO_3)_2$	Idem, anticongelante
Nitrocelulosa	$C_6H_7(NO_3)_3O_2$	Idem, gelatilizante
Nitrato de amonio (NA)	NH_4NO_3	Idem + oxidante
Clorato de potasio	$KCIO_3$	Idem + oxidante
Perclorato de potasio	$KC\ell O_4$	Idem + oxidante
Nitrato de sodio (SN)	$NaNO_3$	Oxidante, reduce congelación
Nitrato de potasio	KNO_3	Oxidante
Pulpa de madera	$C_6H_{10}O_5$	Absorbente, combustible
Aceite combustible	CH_2	Combustible
Parafina	CH_2	Idem
Aceite para lámpara	C	Idem
Gis	$CaCO_3$	Antiácido-estabilizador
Oxido de zinc	ZnO	Idem
Aluminio (metal)	$A\ell$	Catalizador
Magnesio (metal)	Mg	Catalizador
Kieselgur	SiO_2	Absorbente anti-cake diatomeas o infusorios
Oxígeno líquido	O_2	Oxidante
Azufre	S	Combustible
Sal	$NaC\ell$	Anti-inflamante
Compuestos orgánicos nitrosos		Explosivo base, sensibilizadores, anticake

TABLA 2 ENERGIA CALORIFICA (Q) PARA ALGUNOS EXPLOSIVOS

EXPLOSIVO	DENSIDAD	Q (cal/gr)
Nitroglicerina (NG)	1.6	1420
PETN Pentaeritritetetranitrato	1.6	1400
RDX	1.6	1320
Compuesto B	1.6	1140
Tetril	1.6	1010
NG, Gelatina 40%	1.5	820
Slurry (TNT-AN-H_2O) 20 - 65 - 15	1.5	770
NG, Gelatina 100%	1.4	1400
NG, Gelatina 75%	1.4	1150
AN, Gelatina 75%	1.4	990
NG, Dinamita 40%	1.4	930
AN, Gelatina 40%	1.4	800
NG, Dinamita 60%	1.3	990
PETN	1.2	1200
Semigelatina	1.2	940
Dinamita extra 60%	1.2	880
Amatol, 50/50	1.1	890
RDX	1.0	1280
DNT	1.0	960
TNT-AN (50-50)	1.0	900
TNT	1.0	870
ANFO (94-6)	0.9	890
AN	0.8	350

Pólvora negra

Es el explosivo comercial más antiguo. Originalmente era una mezcla de nitrato de potasio, carbón vegetal y azufre; ahora se usa nitrato de sodio en lugar del nitrato de potasio.

Composisión:	Nitrato de potasio	75%
	Carbón vegetal	15%
	Azufre	10%

Cuando se usa nitrato de sodio se disminuye un poco su porcentaje aumentando el carbón y el azufre.

Tiene propiedades indeseables para su uso, razón por la que ha sido sustituida.

Es extremadamente sensible al deflagarse o quemarse explotando a baja velocidad (1300 pies/seg). Se usa en forma limitada en rocas blandas en canteras.

VELOCIDAD DE DETONACION

La propiedad sola más importante a considerar al evaluar la potencia de un explosivo es su velocidad sónica y puede ser confinada o no confinada.

La velocidad de detonación confinada es una medida de la velocidad con que viaja las ondas de compresión a través de una columna de explosivo dentro de un barreno u otro espacio confinado, mientras que la velocidad no confinada se obtiene cuando se detona el explisvo a cielo abierto.

Como los explosivos se usan con cierto grado de confinamiento, es más significativa la velocidad confinada.

La velocidad de detonación confinada en los explosivos comerciales varía entre 5000 y 25 000 pies/seg.

Las velocidades no confinadas son del orden de 70 a 80% de la velocidad confinada.

PRESION DE DETONACION

La presión de detonación es una función de la velocidad de detonación y de la densidad del explosivo. Usualmente no se menciona como una propiedad, pero es muy importante en la selección del explosivo. Cuando se tiene una cara libre se producen esfuerzos por impulso que son reflejados en la roca y son parte importante del mecanismo de rotura o de fragmentación.

La relación entre la velocidad de detonación, la densidad y la presión de detonación es compleja. La siguiente expresión es una de las aproximaciones obtenidas:

$$P = \frac{4.18 \times 10^{-7} DC^2}{1 + 0.8D}$$

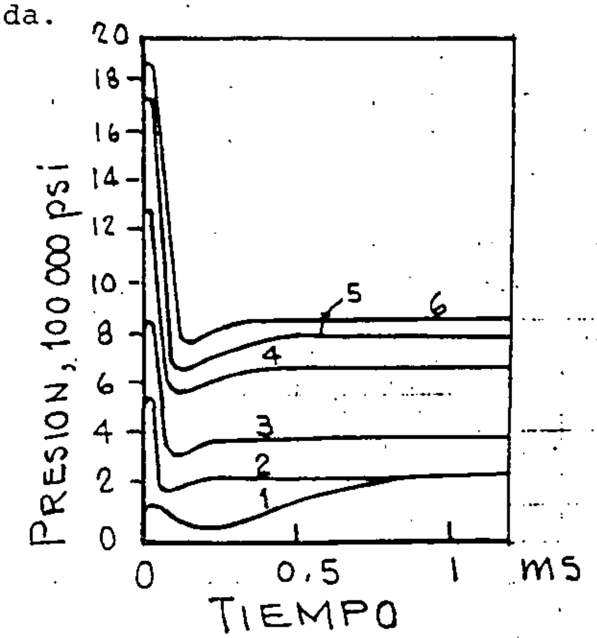
en donde:

P presión de detonación, en Kbar
1 Kbar = 14 504 lb/pulg^2
D densidad
C velocidad de detonación, en pies/seg

Hay que distinguir entre presión de detonación y presión de ignición o de explosión.

La presión de ignición o explosión es la que produce el choque o impacto y tiene un valor del doble de la presión de detonación. Esta presión de choque o ignición se caracteriza por una onda muy puntiaguda frente a la cual toda la materia es ionizada y pulverizada.

1.- ANFO-94/6 Granulado
2.- ANFO-94/6 Fino
3.- AN-Dinamita 60%
4.- NG-Dinamita 60%
5.- TNT-AN-H O-20/65/15
6.- AN-GELATINA, 75%



CURVAS DE PRESION CALCULADA BAJO CONFINAMIENTO PERFECTO

CALIDAD DE GASES

La detonación ideal de los explosivos comerciales es que deben producir vapor de agua, bióxido de carbono y nitrógeno. Sin embargo, gases venenosos como el monóxido de carbono y óxidos de nitrógeno (gases nitrosos), se forman muchas veces.

En excavaciones a cielo abierto los gases venenosos no son importantes, por el contrario, en excavaciones subterráneas hay que tener cuidado con ellos.

CRITERIOS PARA SELECCION DE UN EXPLOSIVO

Para cada sitio habrá un explosivo que proporcione los mejores resultados.

La selección del tipo más adecuado está en función de las propiedades geomecánicas de la roca, como son: estructura, dureza, densidad, resistencia, humedad, ventilación, etc., y de la fragmentación obtenida: altura y proyección del banco.

En rocas duras y densas, como la Taconita y los Granitos, un explosivo de alta velocidad tendrá buenos resultados; sin embargo, posiblemente el ANFO también diera buen resultado y es más económico.

En rocas blandas deben usarse explosivos de bajas velocidades, por ejemplo: caliches y basaltos vesiculares.

En general, la velocidad de detonación debe ser igual a la velocidad sónica del macizo rocoso (velocidad de las ondas P de compresión o primarias).

PROPIEDADES DE DINAMITAS PURAS DE NITROGLICERINA

PORCIENTO EN PESO	DENSIDAD	VELOCIDAD CONFINADA pies/seg	RESISTENCIA DEL AGUA	CALIDAD DE GASES
60	1.3	19,000	Buena	Pobre
50	1.4	17,000	Regular	Pobre
40	1.4	14,000	Regular	Pobre
30	1.4	11,000	Pobre	Pobre
20	1.4	9,000	Pobre	Pobre

COMPOSICION DE LAS DINAMITAS PURAS DE NITROGLICERINA

COMPONENTES	PORCENTAJE EN PESO				
	20	30	40	50	60
Nitroglicerina	20.2	29.0	39.0	49.0	56.8
Nitrato de sodio	59.3	53.3	45.5	34.4	22.6
Aceite vegetal	15.4	13.7	13.8	14.6	18.2
Azufre	2.9	2.0	-	-	-
Antiácido	1.3	1.0	0.8	1.1	1.2
Humedad	0.9	1.0	0.9	0.9	1.2

PROPIEDADES DE DINAMITAS DE AMONIO DE ALTA DENSIDAD

PORCIENTO EN PESO	DENSIDAD	VELOCIDAD CONFINADA pies/seg	RESISTENCIA DEL AGUA	CALIDAD DE GASES
60	1.3	12,500	Regular	Buena
50	1.3	11,500	Regular	Buena
40	1.3	10,500	Regular	Buena
30	1.3	9,000	Regular	Buena
20	1.3	8,000	Regular	Buena

COMPOSICION DE LAS DINAMITAS DE AMONIO DE ALTA DENSIDAD

COMPONENTES	PORCENTAJE EN PESO				
	20	30	40	50	60
Nitroglicerina	12.0	12.6	16.5	16.7	22.5
Nitrato de sodio	57.3	46.2	37.5	25.1	15.2
Nitrato de amonio	11.8	25.1	31.4	43.1	50.3
Aceite vegetal	10.2	8.8	9.2	10.0	8.6
Azufre	6.7	5.4	3.6	3.4	1.6
Antiácido	1.2	1.1	1.1	0.8	1.1
Humedad	0.8	0.8	0.7	0.9	0.7

EXPLO

SISTEMA BULK
(PRODUCTOS EXPLOSIVOS A GRANEL)

PROPIEDADES FISICAS (APROXIMADAS)

PRODUCTO	DENSIDAD G/M³	VELOCIDAD DE DETONACION M/SEG.	RESISTENCIA AL AGUA	POTENCIA RELATIVA AL PESO RWS	POTENCIA RELATIVA AL VOLUMEN RBS.
ANFO	0.85	4700	NULA	100	100
EXPLO BULK 20/80	1.01	> 5000	POBRE	96	120
EXPLO BULK 30/70	1.14	> 5000	POBRE	94	130
EXPLO BULK 40/60	1.34	> 5000	BUENA	92	141
EXPLO BULK 50/50	1.43	> 5000	BUENA	91	146

NOTAS:

1.- Potencia relativa al peso y potencia relativa al volumen, con un valor de 100 para un ANFO estandard perfectamente bien mezclado.

2.- Las propiedades que aquí se muestran son para diámetros de 12 1/4"

3.- Barrenos desaguados.

PARA MAYOR INFORMACION DIRIGIRSE A NUESTRO DEPARTAMENTO DE ASESORIA TECNICA

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GODYNE
HIDROGEL (ALTO EXPLOSIVO)

PROPIEDADES	GODYNE DIAMETRO PEQUEÑO	GODYNE DIAMETRO INTERMEDIO	GODYNE DIAMETRO GRANDE
— Densidad g/cc	1.2	1.2	1.2
— Velocidad de detonación en Km./seg.	4500	4600	5000
— Presión de detonación K bars	61	63	75
— R W S	115	96	96
— R B S	164	137	137
— A S V (Kj/100g)	433	363	362
— Sensitividad a la cápsula No. 6	SI	SI	SI
— Sensibilidad	- 0°C	- 0°C	- 0°C
— Resistencia al agua	Excelente	Excelente	Excelente
— Vida útil (en condiciones normales de almacenamiento)	6 meses	6 meses	6 meses
— Clasificación de gases	Clase No. 1	Clase No. 1	Clase No. 1

EMPAQUE	MEDIDA	CARTUCHOS/CAJA	PESO/CARTUCHO	PESO/CAJA
EMPAQUE DIAMETRO PEQUEÑO	1" X 8"	210 ± 5	118 ± 2 gr	25 Kg
	1 1/4" X 8"	136 ± 3	183 ± 4 gr	
	1 1/2" X 8"	90 ± 2	276 ± 5 gr	
DIAMETRO INTERMEDIO	2" X 16"	25 ± 1	925 - 1000 gr	25 Kg
	2 1/2" X 16"	18 ± 1	1380 ± 20 gr	
	3" X 16"	12	2000 ± 100 gr	
DIAMETRO GRANDE	4" X 26"	4	6.250 Kgs	25 Kg
	5" X 37"	2	12.5 Kgs	
	8"	2	12.5 Kgs	

INERTE AL IMPACTO DE BALA CALIBRE 30 06. INERTE A LA PRUEBA DE FUEGO DIRECTO. NO CAUSA MALESTAR FISICO EN LA DETONACION NI EN SU MANEJO.

POLVORINES VENTILADOS Y SECOS. TEMPERATURAS OPTIMAS MENORES DE 25°C.

ESTIBAR NO MAS DE 10 CAJAS. ROTACION DE PRODUCTO.

PARA MAYOR INFORMACION DIRIGIRSE A NUESTRO DEPARTAMENTO DE ASESORIA TECNICA

ICI EXPLOSIVOS - MEXICO
ATLAS DE MEXICO, S.A. DE C.V.

San Lorenzo No. 1009 Col. del Valle
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EXPLO
EMULSIONES SENSITIVAS
(ALTO EXPLOSIVO)

PROPIEDADES	EMULSION DIAMETRO PEQUEÑO	EMULSION DIAMETRO INTERMEDIO	EMULSION DIAMETRO GRANDE
— Densidad g/cc	1.10	1.18	1.18
— Velocidad de detonación en km/seg	5000	5200	5400
— Presión de detonación K bars	69	80	86
— R W S	81	81	81
— R B S	106	114	114
— A S V (Kj/100g)	305	305	305
— Sensitividad a la cápsula No. 6	SI	SI	SI
— Sensibilidad	— 5°C	— 5°C	— 5°C
— Resistencia al agua	Excelente	Excelente	Excelente
— Vida útil (en condiciones normales de almacenamiento)	6 meses	6 meses	6 meses
— Clasificación de gases	Clase No. 1	Clase No. 1	Clase No. 1

EMPAQUE	MEDIDA	CARTUCHOS/ CAJA	PESO/ CARTUCHO	PESO/ CAJA
DIAMETRO PEQUEÑO	1" X 8" 1 1/2" X 8"	227 ± 5	100 ± 2 gr	25 kg
DIAMETRO INTERMEDIO	2" X 16" 2 1/2" X 16" 3" X 16"	25 ± 1 13 ± 1 12	925 a 1000 gr 1380 ± 20 gr 2000 ± 100 gr	25 kg
DIAMETRO GRANDE	4" X 16" 5" X 17" 5" X 37"	6 4 2	4.166 kg 6.250 kg 12.5 kg	25 kg

INERTE AL IMPACTO DE BALA CALIBRE 30 06. INERTE A LA PRUEBA DE FUEGO DIRECTO. NO CAUSA MALESTAR FISICO EN LA DETONACION NI EN SU MANEJO.

POLVORINES VENTILADOS Y SECOS, TEMPERATURAS OPTIMAS MENORES DE 25°C.

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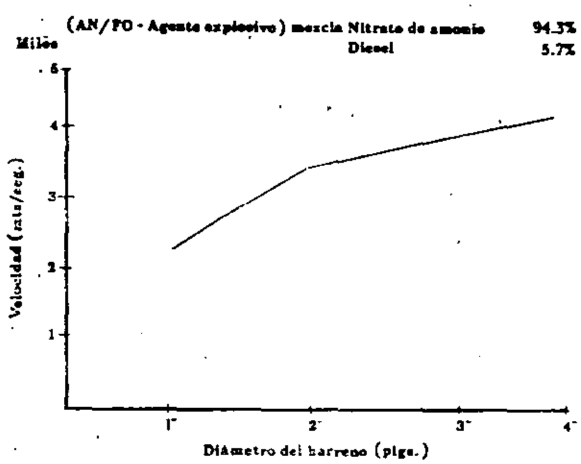
ANFOMEX "X"
(AGENTE EXPLOSIVO)

• Densidad g/cc	0.85
• Velocidad de detonación	3200 m/seg.
• Presión de detonación	13.7
• R W S	100
• R B S	100
• A S V (Kj /100 g)	377
• Sensitividad a cápsulas No. 6	NO
• Sensibilidad	--
• Resistencia al agua	Nula
• Empaque	25 Kgs/saco



Influencia del diámetro en la velocidad de detonación

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AN/FO B.D.
(AGENTE EXPLOSIVO BAJA DENSIDAD)

PROPIEDADES

Densidad g/cc	0.65
Velocidad de detonación	2800 m/seg.
Presión de detonación	12.5
R W S	96
R B S	74
A S V (Kj/100g)	361
Sensitividad a la cápsula No. 6	NO
Sensibilidad	--
Resistencia al agua	NULA
Empaque	25 Kgs./saco

POLVORINES VENTILADOS Y SECOS, TEMPERATURAS OPTIMAS MENORES DE 25°C.

ESTIBAR NO MAS DE 10 SACOS. ROTACION DE PRODUCTO.

PARA MAYOR INFORMACION DIRIGIRSE A NUESTRO DEPARTAMENTO DE ASESORIA TECNICA

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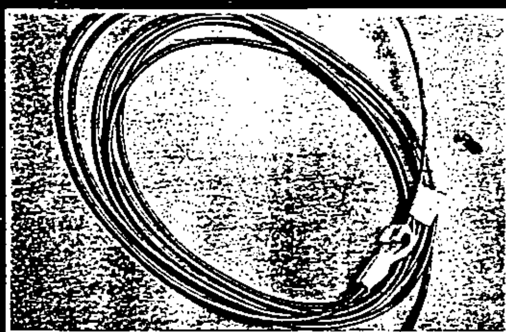


4) **Etiqueta**

El último componente del sistema es una etiqueta con un código de colores, la cual indica: El tipo de retardo "MS" o "LP", el número de período de retardo, así como el tiempo nominal de disparo.

Ventajas:

Seguridad: No requiere de modificaciones tales como: cortes, uniones o separaciones. El sistema Nonel Primadet; deberá usarse tal como llega de fábrica, evitando así la generación involuntaria de condiciones de riesgo.

El tubo Nonel no puede ser iniciado por transmisores de radio de alta frecuencia, electricidad estática o extraña, flama, fricción o impactos encontrados en condiciones normales de minado.

De uso sencillo y flexible: La conexión del sistema es bastante simple ya que los componentes se surten totalmente ensamblados. Además, ningún tipo de conocimientos sobre conexión de circuitos eléctricos es necesario.

Sistema No-Eléctrico: No es necesario entrenar y re-entrenar personal en el manejo de complejos circuitos eléctricos. Este es el sistema más sencillo que se dispone para aplicaciones donde una ilimitada secuencia de retardos es requerida.

Sin ruido. El sistema de iniciación no-eléctrico Nonel Primadet disminuye los niveles de ruido, ya que su transmisión es muda.

Tiempos de retardo disponibles.

Tabla 1

Iniciadores con retardo no eléctricos		**Iniciadores con retardo no eléctricos**	
"MS" Nonel Primadet Tubo Color Naranja		***"LP" Nonel Primadet*** Tubo Color Amarillo	
Periodo	**Retardo**	**Periodo**	**Retardo**
1	- 25 Milisegundos	½	- 0.1 Segundos
2	- 50 "	1	- 0.2 "
3	- 75 "	1½	- 0.3 "
4	- 100 "	2	- 0.4 "
5	- 125 "	2½	- 0.5 "
6	- 150 "	3	- 0.6 "
7	- 175 "	4	- 1.0 "
8	- 200 "	5	- 1.4 "
9	- 250 "	6	- 1.8 "
10	- 300 "	7	- 2.4 "
11	- 350 "	8	- 3.0 "
12	- 400 "	9	- 3.8 "
13	- 450 "	10	- 4.6 "
14	- 500 "	11	- 5.5 "
15	- 600 "	12	- 6.4 "
		13	- 7.4 "
		14	- 8.5 "
		15	- 9.6 "

ACCESORIOS

Cápsulas de detonación o estopines

Los estopines eléctricos son los accesorios más utilizados para iniciar o detonar los explosivos potentes. La cápsula puede insertarse directamente en el cartucho o sujetarse fuertemente al cordón detonante.

Una cápsula eléctrica consiste de dos alambres aislados insertados en una cápsula de metal que están conectados por un delgado filamento de alambre que forma un puente. Este alambre de puente a veces se pinta con una mezcla de fósforo que produce flama con los cerillos.

Cuando se le aplica la corriente eléctrica a los alambres el filamento de puente se calienta e inicia una carga instantáneamente de un explosivo altamente sensible al calor. La explosión del alambre detona una primera carga, la cual a su vez detona una carga de un explosivo potente en el fondo de la cápsula, tal como PETN o RDX.

Esta carga de fondo tiene potencia suficiente para detonar una cápsula-explosiva sensitiva o cebo (primer), o bien, un cordón detonante.

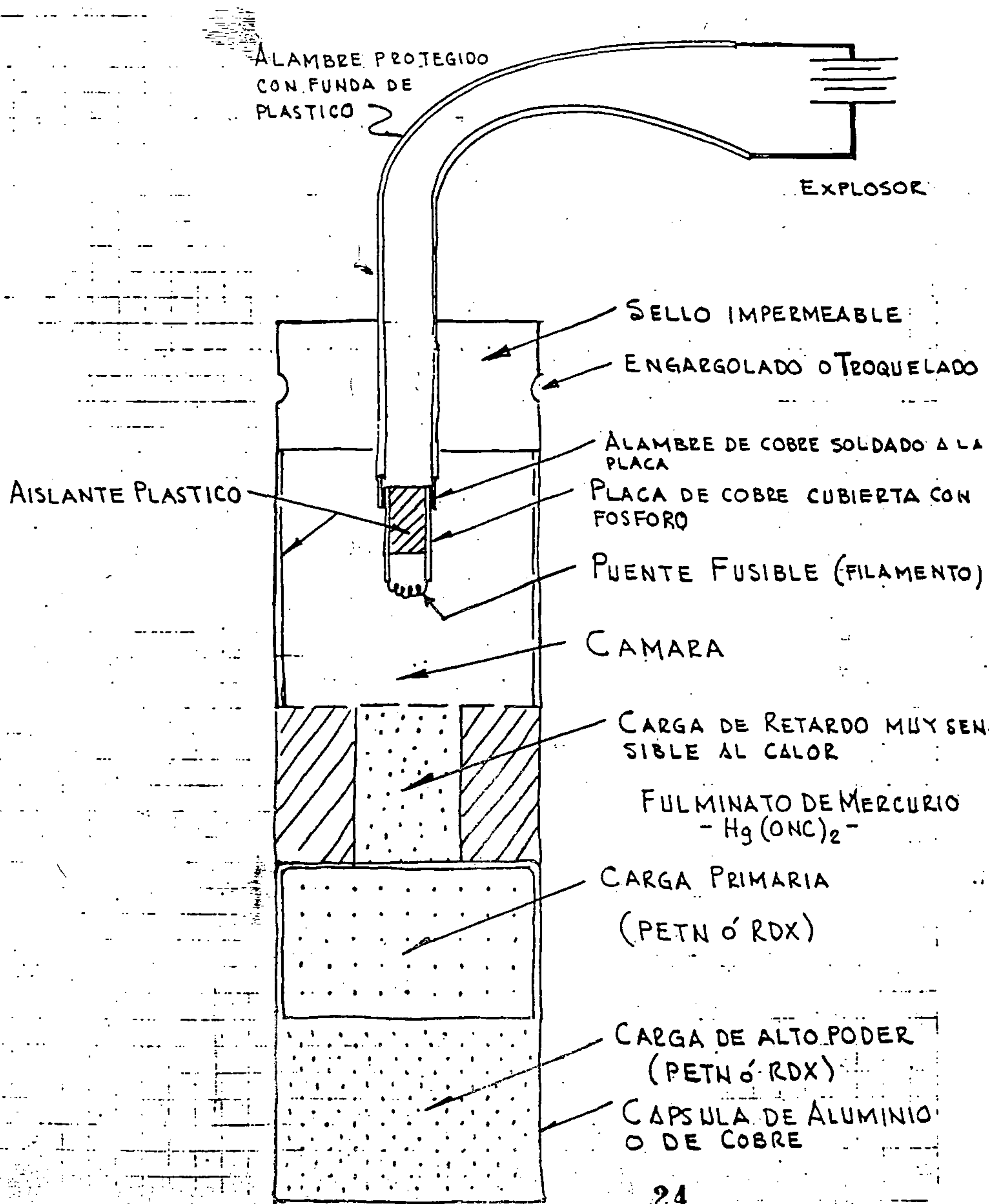
En las cápsulas eléctricas de retardos, un elemento retardante de explosivo en polvo se deposita entre filamento de puente y la carga potente del fondo. Este elemento de retardo está finamente calibrado para dar un intervalo de tiempo específico entre la aplicación de la corriente eléctrica y la detonación de la carga de fondo.

Hay dos series básicas de retardos disponibles: de retardos cortos o milisegundos con incrementos de retardo de 25 m en el intervalo inferior y 50 m en el intervalo superior y, retardos largos a menudo llamados retardos lentos o simplemente retardos, con incrementos de retrdo de 0.5 seg y 1 seg.

Con los estopines de milisegundos se produce mejor fragmentación y se reduce la presión de aire y las vibraciones del terreno.

Los estopines de retardo se usan en lumbreras o túneles para dar tiempo suficiente al movimiento de la roca. Probablemente se produce fragmentación más gruesa que la obtenida con milisegundos.

CAPSULA ELECTRICA O ESTOPIN



Cordón detonante

El cordón detonante consiste de un tubo de plástico resistente al agua, que se protege con una cubierta o forro fabricado con una combinación de textiles, plástico y alambre a prueba de agua. Las cubiertas tienen diferentes grados de resistencia a la tensión, abrasión y flexibilidad.

Dentro del tubo de plástico está el núcleo o corazón constituido por un alto explosivo, usualmente PETN. La cantidad de PETN varía entre 1 gramo/pie a 400 gramos/pie, y se produce en diferentes potencias.

Todas las potencias de PETN pueden detonarse con una cápsula eléctrica y su velocidad de detonación es de 21 000 pies/seg.

Su notable insensibilidad contra impacto y fricción es ideal para su uso en la línea de encendido y líneas troncales.

Como los estopines eléctricos se sujetan al cordón detonante hasta el final justamente antes de la voladura, la mayor parte de una falla aleatoria por detonación se elimina.

Usualmente se usa el cordón de 25 gramos/pie y el de 50 gr/pie se usa en trabjos especiales.

El cordón detonante es un explosivo de alta potencia que explota con una gran producción de aire. Hay que tener cuidado con este efecto.

Un cordón detonante de 25 a 50 gramos/pie detona cualquier cápsula-sensitiva (primer o cebo y cápsulas de alta potencia, como son los boosters).

Cordón detonante Non-electric (NONEL)

Este es un cordón detonante muy útil para voladuras subterráneas, pues se eliminan las fallas por electricidad estática. También se usa en voladuras a cielo aiberto para evitar vibraciones detonando barreno por barreno al igual que el cordón detonante y en zonas altas donde se generen tormentas eléctricas.

El NONEL detona en una sola dirección, por lo que hay que tener cuidado en su acoplamiento.

También existen conectores especiales de retardo constituidos por el mismo tubo de NONEL en longitudes de 2 pies con terminales de plástico.

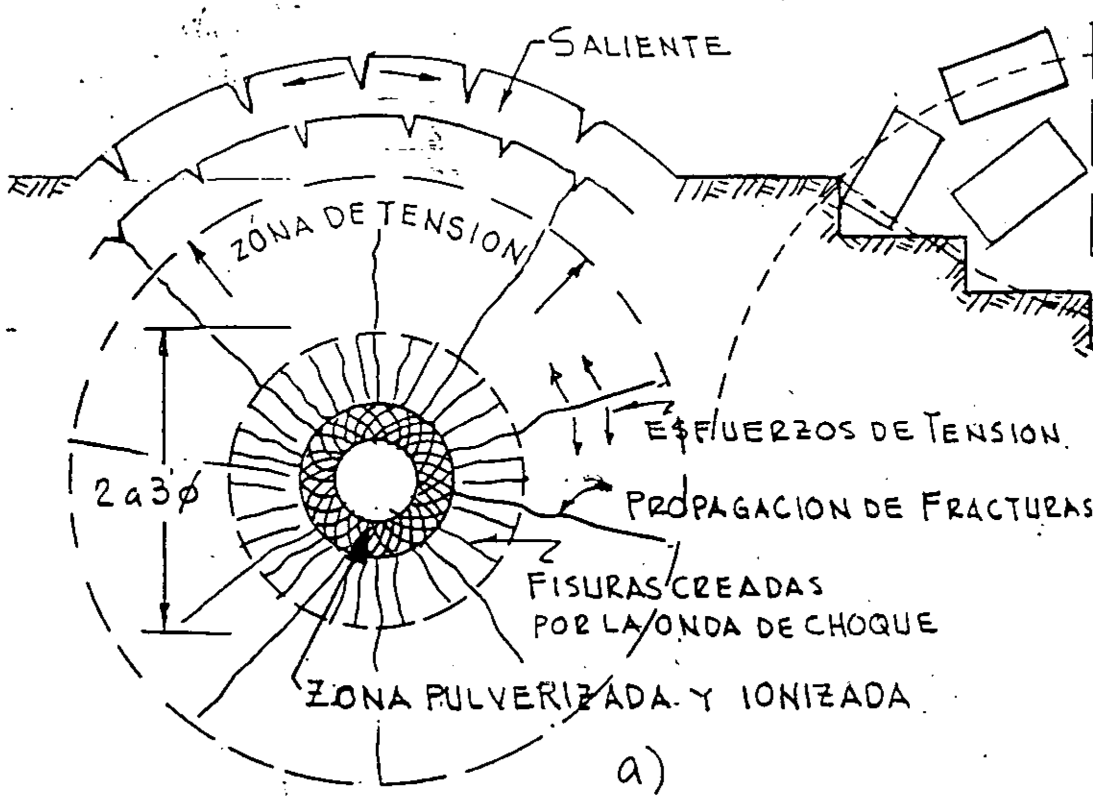
El NONEL tiene una gran resistencia al agua, ya que un extremo está sellado contra la cápsula de detonación y el otro está sellado contra una terminal de plástico.

El NONEL no explota, pudiendo sostenerse perfectamente con las manos.

El NONEL tiene una velocidad de 9000 pies/seg

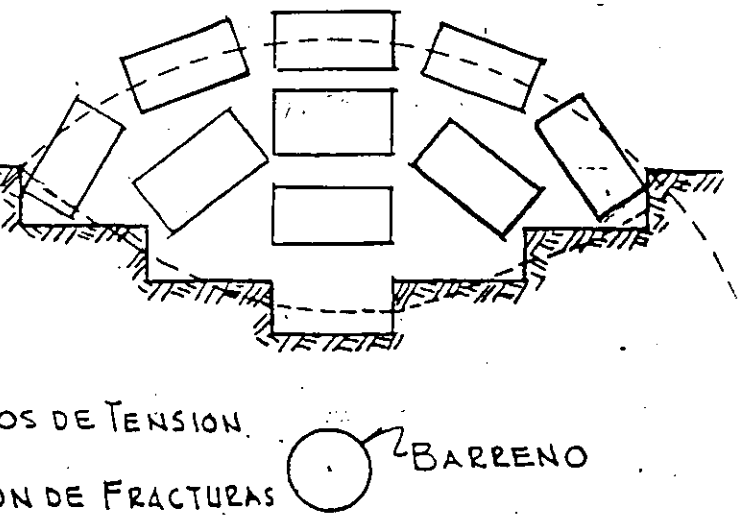
Su composición química es:

MECANISMO DE FRAGMENTACION



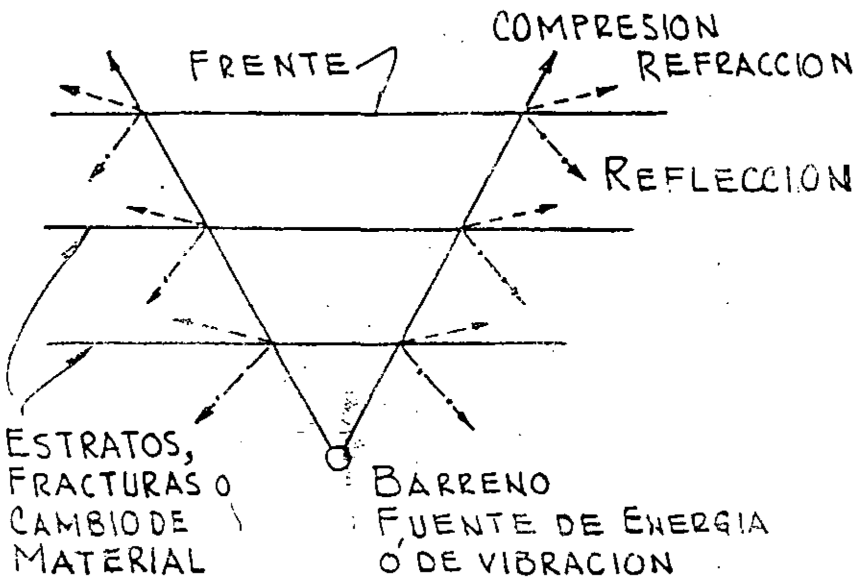
a)

FRACTURAS RADIALES



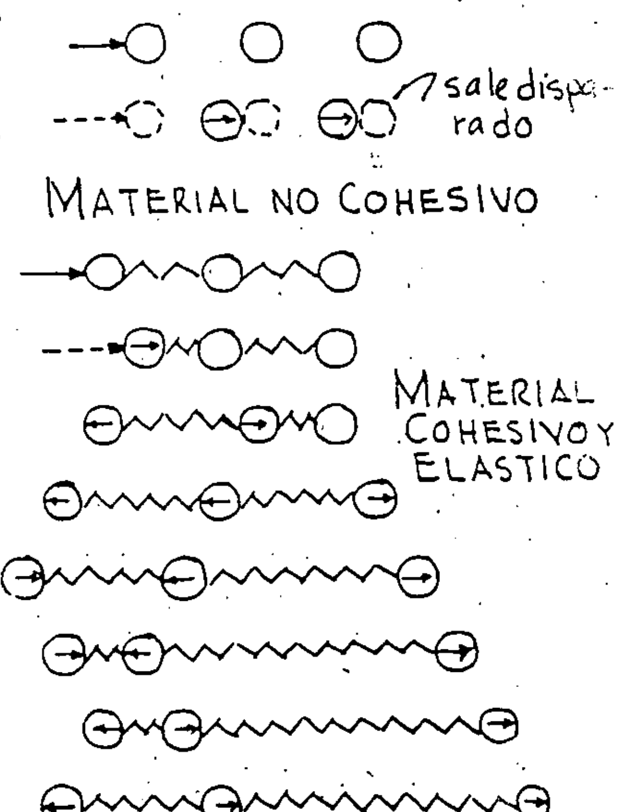
b)

SECUENCIA EN LA FORMACION DEL CRATER

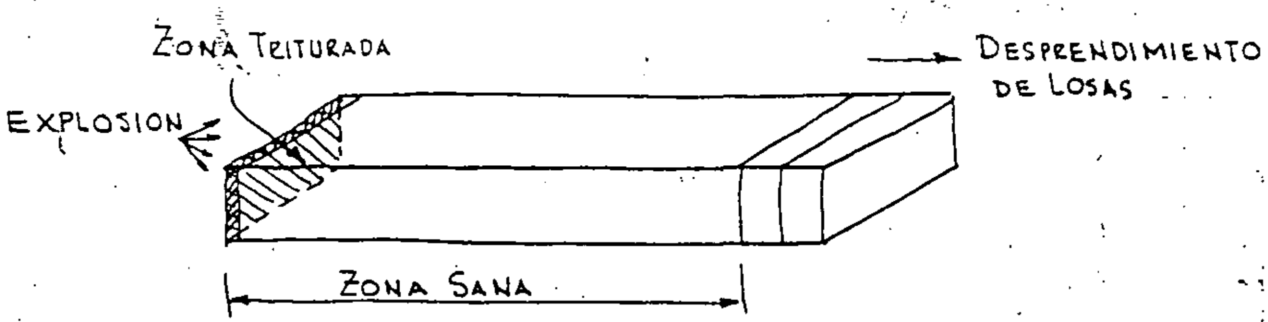


COMPONENTES DE LA ENERGIA POR IMPACTO DEL EXPLOSIVO

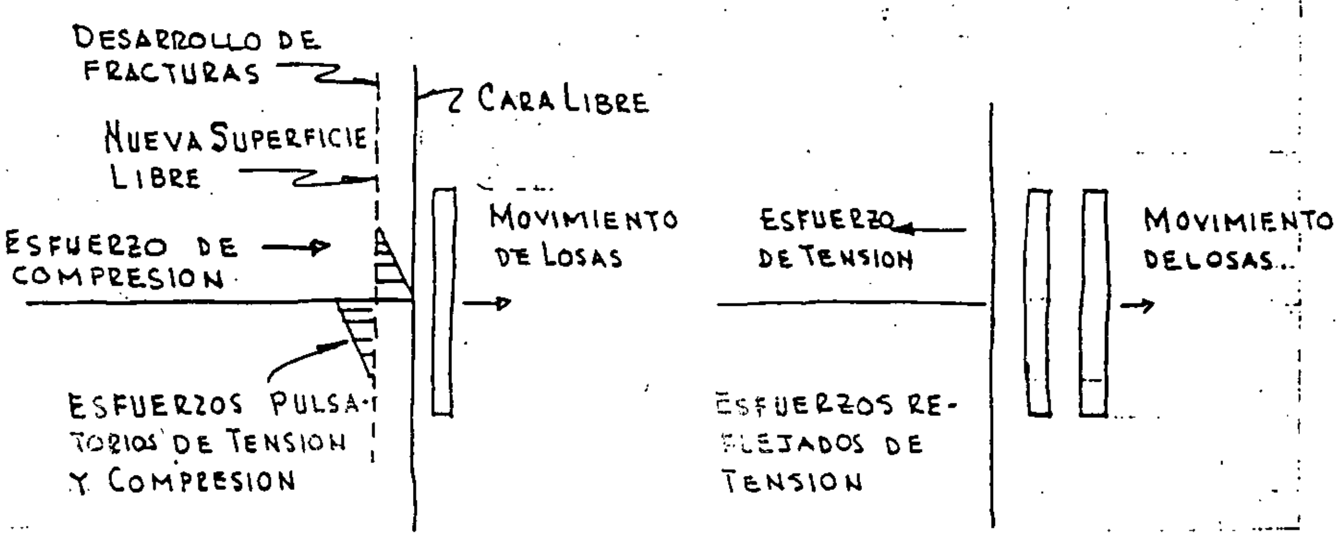
FRAGMENTACION Y DESPLAZAMIENTO = FUERZA - DISTANCIA = TRABAJO



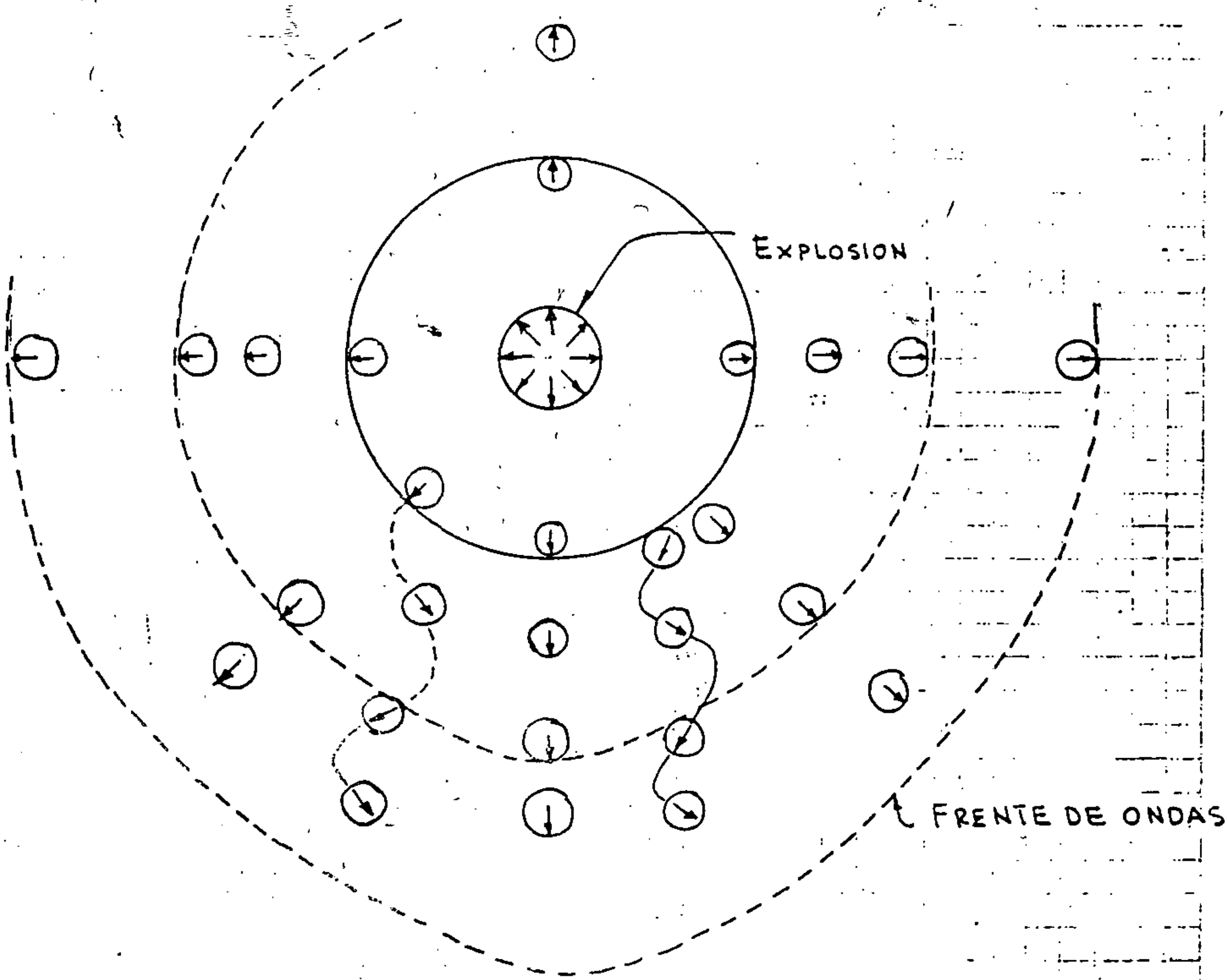
TRANSMISION DE ENERGIA EN MATERIALES
... CARGAS DE PULSACION



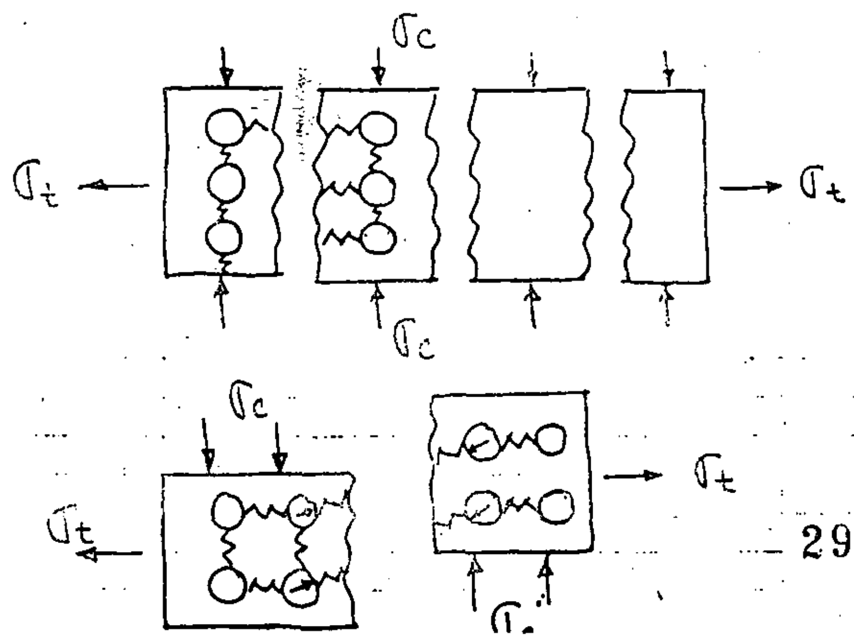
BARRA EXPERIMENTAL DE CONCRETO



FRACTURAMIENTO DE TENSION POR REFLEXION DE IMPULSOS.

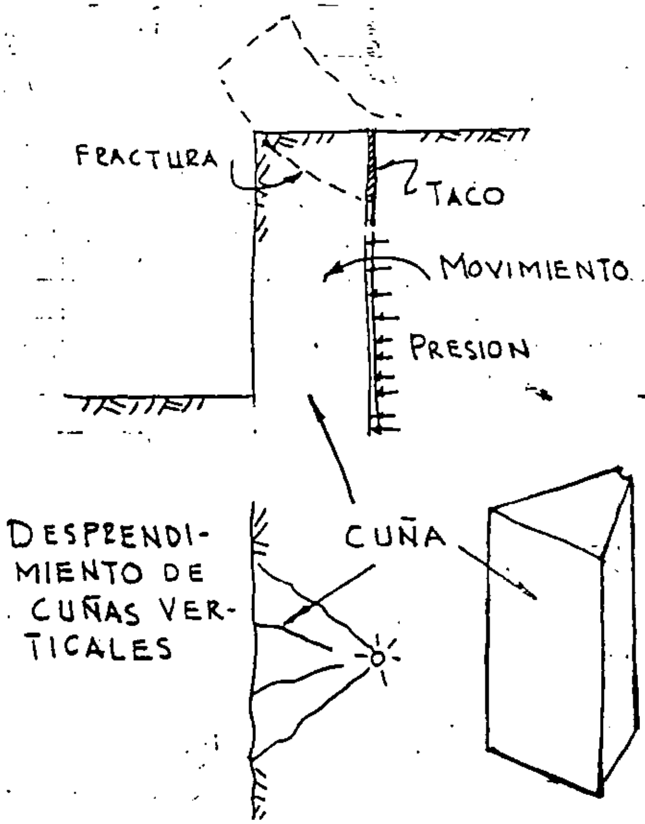


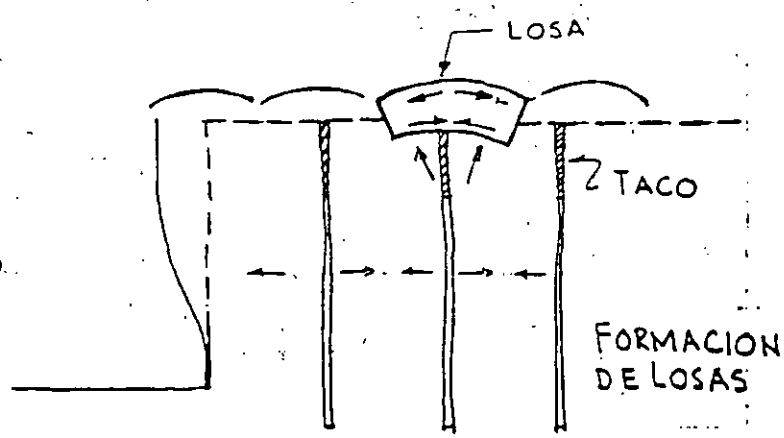
INFLUENCIA DEL PATRON ESTRUCTURAL DE LAS PARTICULAS

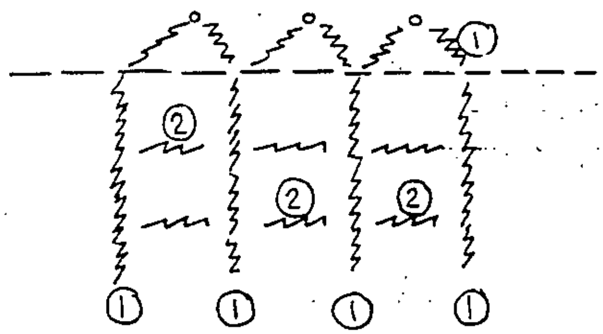


FRACTURAMIENTO POR COMPRESION O TENSION

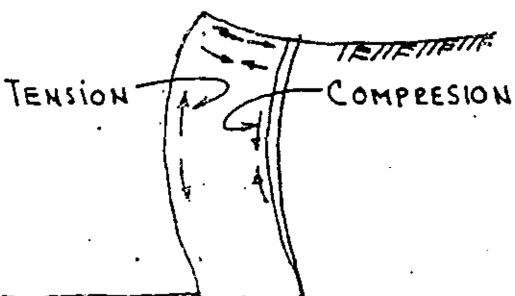
FRACTURAMIENTO POR CORTANTE



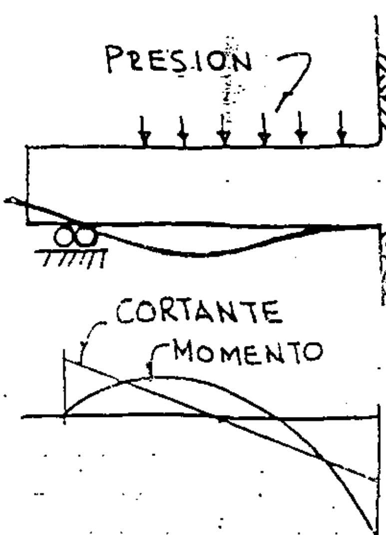




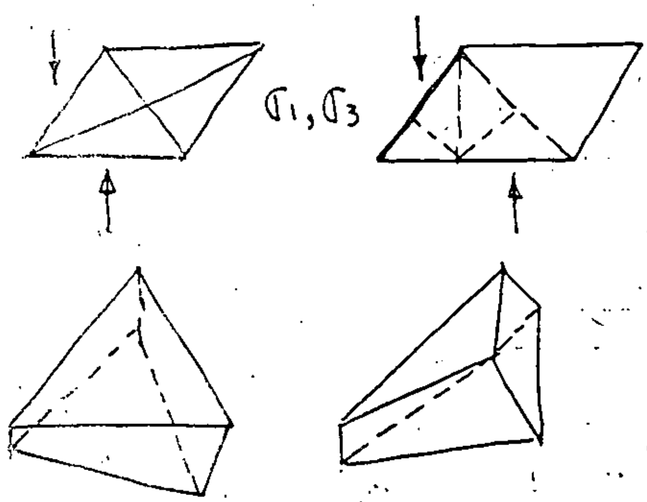
1. FRACTURAS VERTICALES COMO CUÑAS
2. FRACTURAS HORIZONTALES
3. SALIDA DE GASES



EFECTO DE VIGA



ANALOGIA DE LA VIGA



FRAGMENTOS EN FORMA DE TRIANGULOS POR EFECTO DE CORTANTE (TORSION) Y FLEXION

MECANISMO DE FRAGMENTACION

Las rocas normalmente son más resistentes en compresión y trituración que por tensión, por ejemplo: algunas calizas tienen resistencias a compresión entre 250 y 1500 kg/cm^2 y resistencias en tensión tan bajas como 35 a 150 kg/cm^2.

Por otro lado, los explosivos y agentes explosivos utilizados producen presiones muy altas que reaccionan con velocidades entre 2500 a 8000 m/seg (5300 a 17 000 mph).

La presión desarrollada súbitamente dentro del barreno alcanza valores desde 18 000 hasta 15 000 kg/cm^2 dependiendo del tipo de explosivo y de las condiciones de confinamiento.

El efecto del explosivo que reacciona contra la roca produce un impacto, o impulso, desde un golpe aplicado rápidamente, de extremadamente alta intensidad.

Cuando el explosivo está dentro de un barreno circular, se ejerce igual presión en todas direcciones a lo largo de todo el perímetro del agujero. La roca en toda esa región es comprimida y pulverizada hasta una distancia limitada del orden de $\phi/4$.

La aplicación súbita del impacto es seguida por la producción de alta presión que introduce ondas de esfuerzos compresionales que rápidamente penetran en forma de abanico a través del macizo rocoso como ondas elásticas. Esta acción se produce aún cuando las rocas son más bien frágiles, pero son algo elásticas. La velocidad con que viajan las ondas de choque a través de la roca, es función de la densidad del medio. Las rocas densas dan lugar a altas velocidades y las rocas blandas porosas o ligeras, a bajas velocidades.

Parte de la energía transmitida a través de las ondas compresionales es reflejada y refractada (flexionada) por cambios de densidad o discontinuidades de la estructura. Cualquier frente libre o cambio en el tipo de roca produce este efecto. El resto de la energía tiende a mantener su dirección original de viaje.

Los ángulos de reflexión son iguales a los que van hacia las fronteras. Los ángulos de refracción dependen de las características de los dos materiales. Esto es, que en cada cambio de densidad se produce relfexión y refracción de los impulsos de la energía, al equilibrarse la energía sigue viajando en su dirección original.

Si un golpe es ejercido a una partícula, la energía es transmitida en la dirección de aplicación del golpe hacia las partículas adyacentes, hasta que la energía es consumida como resultado del trabajo realizado y por efectos como fricción, amortiguamiento, fragmentación, etc.,

Los suelos granulares no tienen cohesión, de modo que tienen poca o ninguna atracción entre partículas, aún cuando cada partícula pueda tener un poco de elasticidad por sí misma.

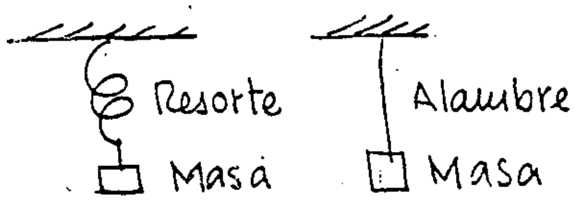
La mayor parte de las rocas es cohesiva y algo elástica, teniendo diferentes efectos que los producidos en fragmentos sueltos.

VIBRACIONES

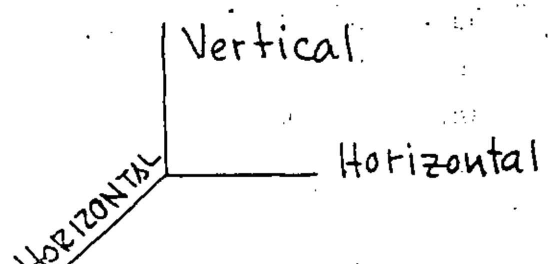
Las vibraciones del terreno pueden medirse mediante los desplazamientos que se produzcan a una masa sujeta a un resorte o a un alambre. Los impulsos pueden ser proyectados en una pantalla de un osciloscopio, en el cual puede determinarse la velocidad de la partícula, su aceleración y la amplitud de su desplazamiento.

Generalmente la masa viene a ser el núcleo de un pequeño transformador lineal en el cual al desplazarse el núcleo, se producen cambios de voltaje y amperaje en el transformador pequeño que significan los desplazamientos de la masa.

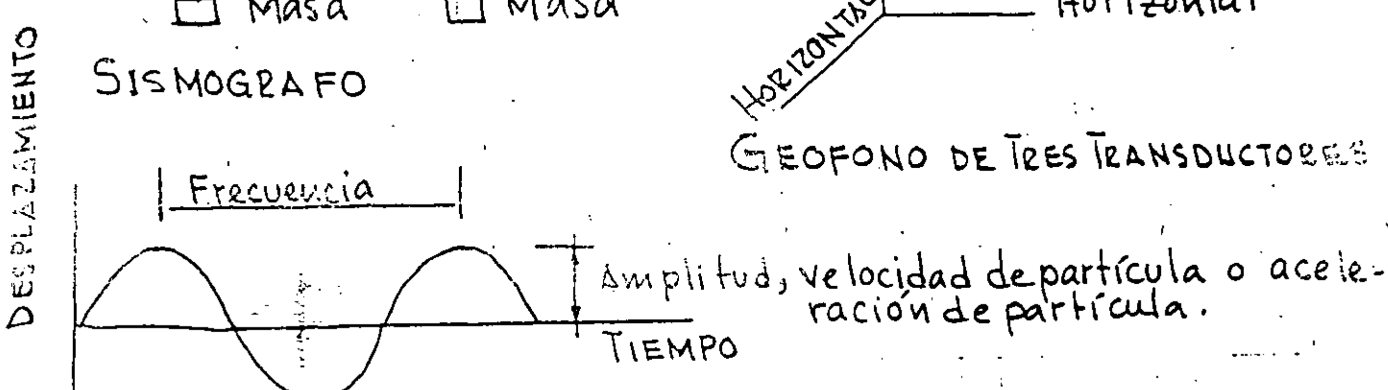
Estos transformadores (LVDT) constituyen los geófonos y pueden instalarse en tres direcciones dentro de un geófono.



SISMOGRAFO



GEOFONO DE TRES TRANSDUCTORES



ONDAS SISMICAS

ONDAS DE CUERPO:

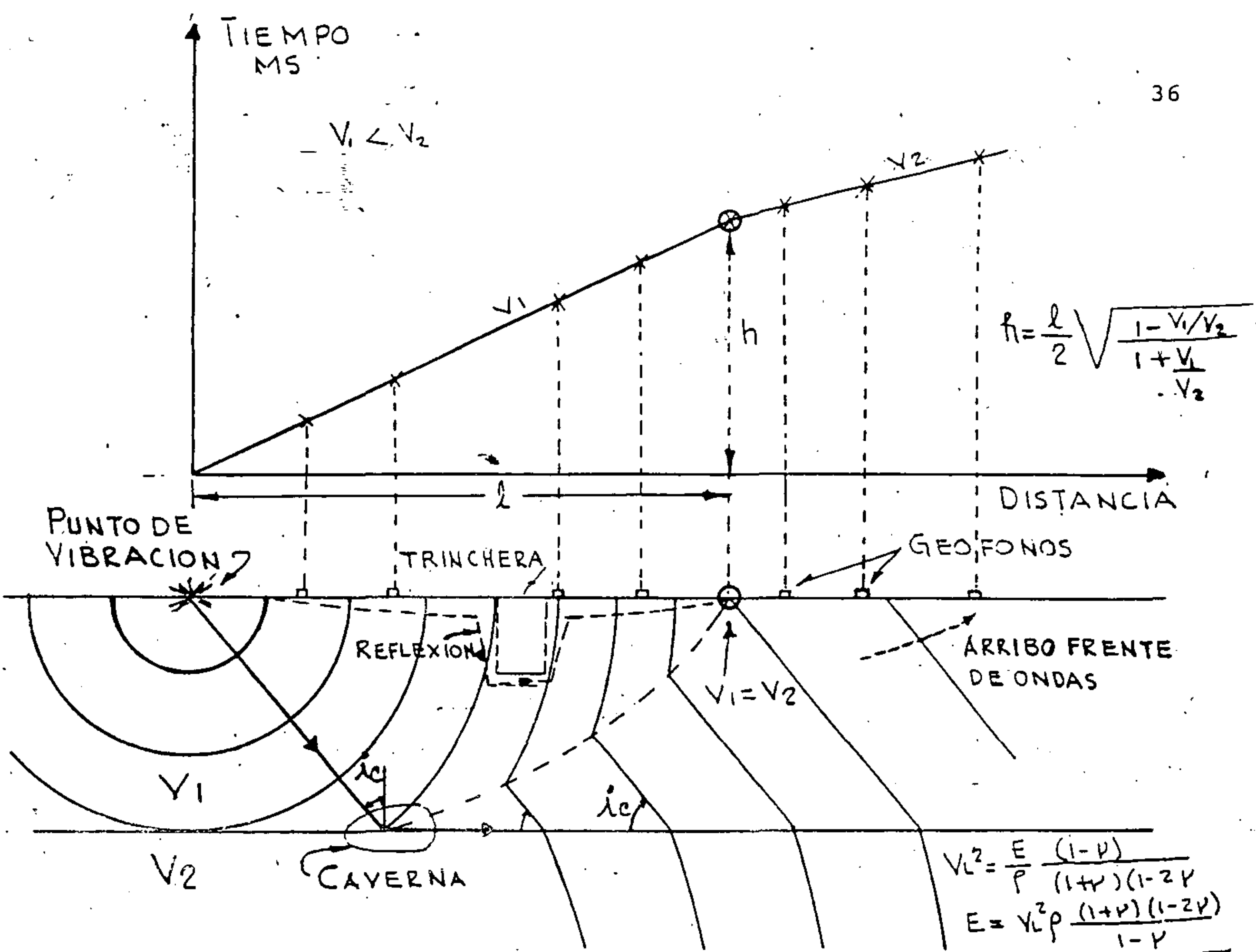
1. Compresional
   Longitudinal
   Primaria - P
   De empuje

2. Corte
   Onda transversal
   Onda secundaria - S

ONDAS DE SUPERFICIE:

3. Love
   Rayleigh — Igual de peligrosas que las P y S

En la figura 1 se presenta la transmisión de ondas de compresión por reflexión y refracción sísmica.



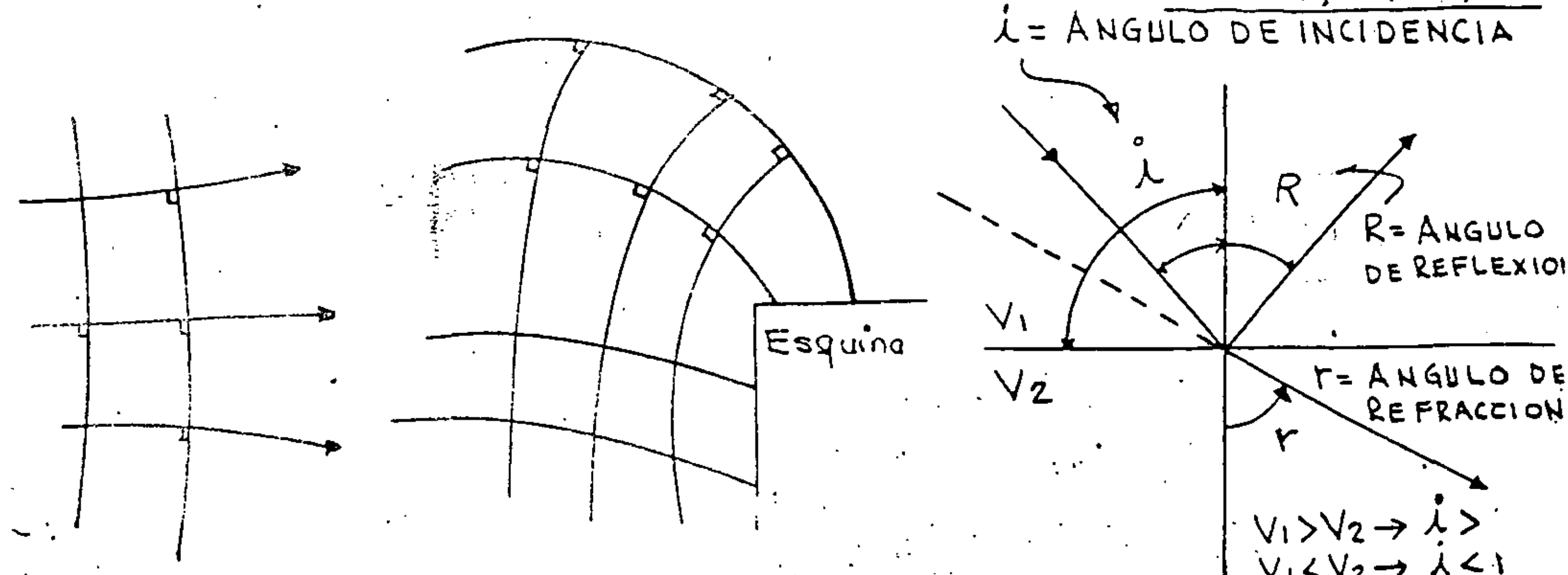
$$V_L^2 = \frac{E}{\rho}\,\frac{(1-\nu)}{(1+\nu)(1-2\nu)}$$

$$E = V_L^2\rho\,\frac{(1+\nu)(1-2\nu)}{1-\nu}$$

REFLEXION Y REFRACCION DE ONDAS

$$V_T^2 = \frac{E}{\rho}\,\frac{1}{2(1+\nu)}$$

$$E = V_T^2\,\rho \cdot 2(1+\nu)$$



PRINCIPIOS DE REFLEXION

GRADUATION OF DAMAGE RISK ON NORMAL RESIDENTIAL HOUSING UNITS IN RELATION TO VIBRATION VELOCITY v WITH RESPECT TO THE FOUNDATION OF THE BUILDINGS

WAVE VELOCITY c m/sec	1000-1500 Sand, gravel clay under groundwater level	2000-3000 Moraine, slate soft limestone	4500-6000 Granite, gneiss, hard limestone, quartzite sandstone, diabas	RESULT IN normal housing units	LEVEL AT c = 4500-6000 m/sec.
Vibration velocity, v mm/sec.	9	18	35	Not noticeable	0.008
	13 20	25	50		0.015
	18	35	70	cracking	0.03
	30 40	55	100	Fine cracks and fall of plaster (threshold value)	0.06
	40	80	150	Cracking	0.12
	60	115	225	Severe cracking	0.25
					0.50

The accuracy of the empirical values in the table are confirmed by hundreds of thousands of readings during thirty years.

In connection with electrical installations the acceleration parameter also has to be considered to prevent disturbances on the sensitive equipment. In the same way, it can be mentioned that appraisal has been made concerning permissible values in blasting work in the neighbourhood of following installations:

Telephone stations: v = 50 mm/sec. and a = 3.0 g
level: 0.015

Television stations: v = 35 mm/sec. and a = 3.0 g
level: 0.008

Computer: a = 0.25 g (under certain conditions)

The values for computers imply that normal blasting is made more difficult and under certain conditions impossible. To enable blasting operations in such cases Nitro Consult has developed a special method for damping of computer installations.

It is important to obtain experience from measuring methods and units to reduce risk of damage.

When planning blasting operations where ground vibration problems occur, it is important to be aware of the relationship between distance/charging and ground vibrations. We have used the Langefors' relationship for various charging levels:

Level: $\frac{Q}{R^{3/2}}$

Q = instantaneously detonating charge in kg

R = distance in m

The vibration velocity can be calculated from the relationship:

$$v = k\sqrt{\frac{Q}{R^{3/2}}}$$

where v = vibration velocity in mm/sec.
k = constant depending on the homogeneous of the rock and the occurence of faults and cracks, (approx. 400 for hard homogeneous Swedish granite)

In Sweden there are two kinds of vibration measurement instruments:

Mechanical and Electrical

The mechanical instruments are:

Ampligraphs
Combigraphs.

The AMPLIGRAPH® (see fig. 1) is a vibrograph designed for long periods of recording which has been developed for continuous registration and mapping of ground vibrations caused by blasting, pile-driving or other vibrations of a similar type. Because of the long periods of continuous recording; the speed of registration is low and this means that only the amplitude of the vibrations and the times at which they occur are noted. The AMPLIGRAPH may be used for either vertical or horizontal measurements and runs for up to eight days without attention.

When more detailed investigations are being made in critical cases, supplementary measurements must be made with e.g. a COMBIGRAPH® (see fig. 2) on which both the frequency and the velocity of vibration can be calculated.

The COMBIGRAPH is a vibrograph of the combination type. It consists of a lowspeed registration section for long periods of recording of the same type as that in the AMPLIGRAPH and also has a high-speed registration section. The low-speed registration section is for continuous registration and mapping of ground vibrations caused by blasting and pile-driving. The high-speed registration section is for the measurement of vibrations during critical blasting operations and registers the appearance of the complete vibrations from which amplitude and frequency can be calculated as well as the rate and acceleration of oscillation.

The high-speed registration section of the COMBIGRAPH has the drive motor located in the cover of the instrument. The recording paper - known as a combigram - is located on the inside of the cover.

Distancia (m)	Carga, kg (detonación instantánea)						
	Grupo A	B	C	D	E	F	G
0.5				0.02	0.04	0.08	0.16
1	0.008	0.0015	0.03	0.06	0.12	0.25	0.50
2	0.025	0.05	0.09	0.2	0.4	0.7	1.4
3	0.040	0.08	0.16	0.33	0.65	1.3	2.6
4	0.06	0.12	0.25	0.5	1.0	2.0	4.0
5	0.09	0.18	0.36	0.73	1.4	2.8	5.6
6	0.12	0.23	0.47	0.95	1.9	3.8	7.2
7	0.14	0.27	0.57	1.15	2.3	4.6	9.2
8	0.18	0.36	0.72	1.45	2.9	5.8	11.6
9	0.2	0.42	0.85	1.70	3.4	6.8	13.6
10	0.25	0.5	1.0	2.0	4.0	8.0	16.0
12	0.3	0.6	1.3	2.5	5.2	10.5	21
14	0.4	0.8	1.6	3.2	6.4	13.0	26
16	0.5	1.0	2.0	3.9	7.8	15.5	31
18	0.6	1.2	2.4	4.7	9.4	19	38
20	0.7	1.4	2.8	5.6	11	22	44
	1.0	2.0	4.0	8.0	16	32	64
30	1.3	2.6	5.2	10.4	21	42	84
35	1.6	3.2	6.5	13	26	52	104
40	2.0	4.0	8.0	16	32	64	128
45	2.4	4.8	9.5	19	38	76	152
50	2.8	5.5	11	22	44	88	176
55	3.3	6.5	13	26	52	104	208
60	3.8	7.5	15	30	60	120	240
65	4.3	8.5	17	34	68	136	272
70	4.8	9.5	19	38	76	152	304
75	5.3	10.5	21	42	84	168	336
80	5.8	11.5	23	46	92	184	368
85	6.4	12.8	25.5	51	102	204	408
90	7.0	14.0	28	56	112	224	448
95	7.6	15.2	30	61	122	244	488
100	8.5	16.5	33	66	132	264	528
110	9.3	18.5	37	74	148	296	592
120	10.5	21.0	42	84	168	336	672
130	11.7	23.5	47	94	188	376	752
140	13.2	26.3	52.5	105	210	420	840
150	14.5	29.0	58	116	232	464	928
160	16.0	32.0	64	128	256	512	1024
170	17.5	35.0	70	140	280	560	1120
180	19.0	38.3	76.5	153	306	612	1224
190	20.7	41.5	83	166	332	664	1328
200	22.5	45.0	90	180	360	720	1440

Los grupos A-G dependen de la vibración de suelo permisible para construcción, instalación, etc.

El C es el grupo normal

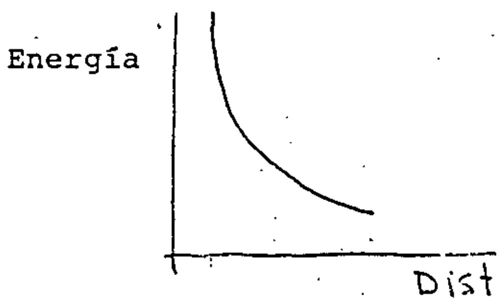
PROPAGACION DE ENERGIA

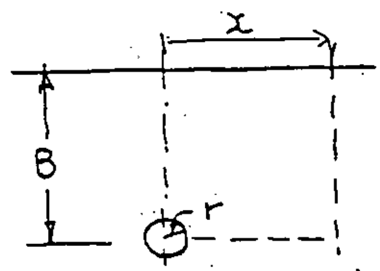
La energía se propaga disminuyendo con la distancia es directamente proporcional con la presión de detonación e inversamente proporcional al cuadrado de la distancia el Bordo:

$$B = K \left(\frac{Pe}{\sigma_t}\right)^{\frac{1}{2}}$$

en donde:

- B bordo
- K constante
- P presión de detonación
- σ_t resistencia a tensión





$$\sigma_t = \frac{-4p\,(x^2+r^2-B^2)}{(x^2-r^2+B^2)} \qquad x=(B^2-r^2)^{\frac{1}{2}}$$

El valor más significativo de la energía es la velocidad de la partícula. El Bureau of Mines usa la siguiente expresión:

$$V = H \left(\frac{D}{W^{\frac{1}{2}}}\right)^{-\beta}$$

Esta expresión puede graficarse en escala logarítmica como se presenta en la figura 2, en la cual la distancia escalada

$$SD = \frac{D}{W^{\frac{1}{2}}} \text{ o sea } W = \left(\frac{R}{SD}\right)^{\frac{1}{2}}$$

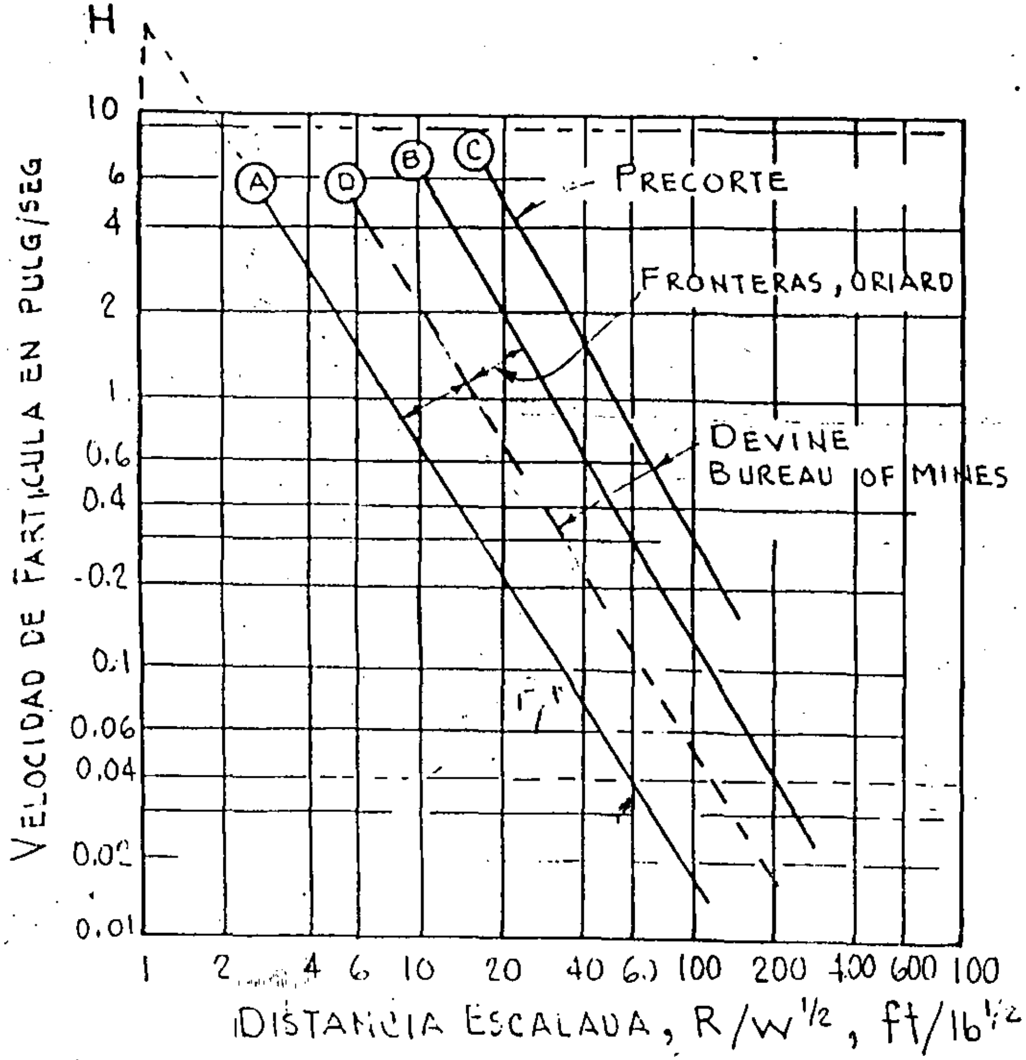
en donde

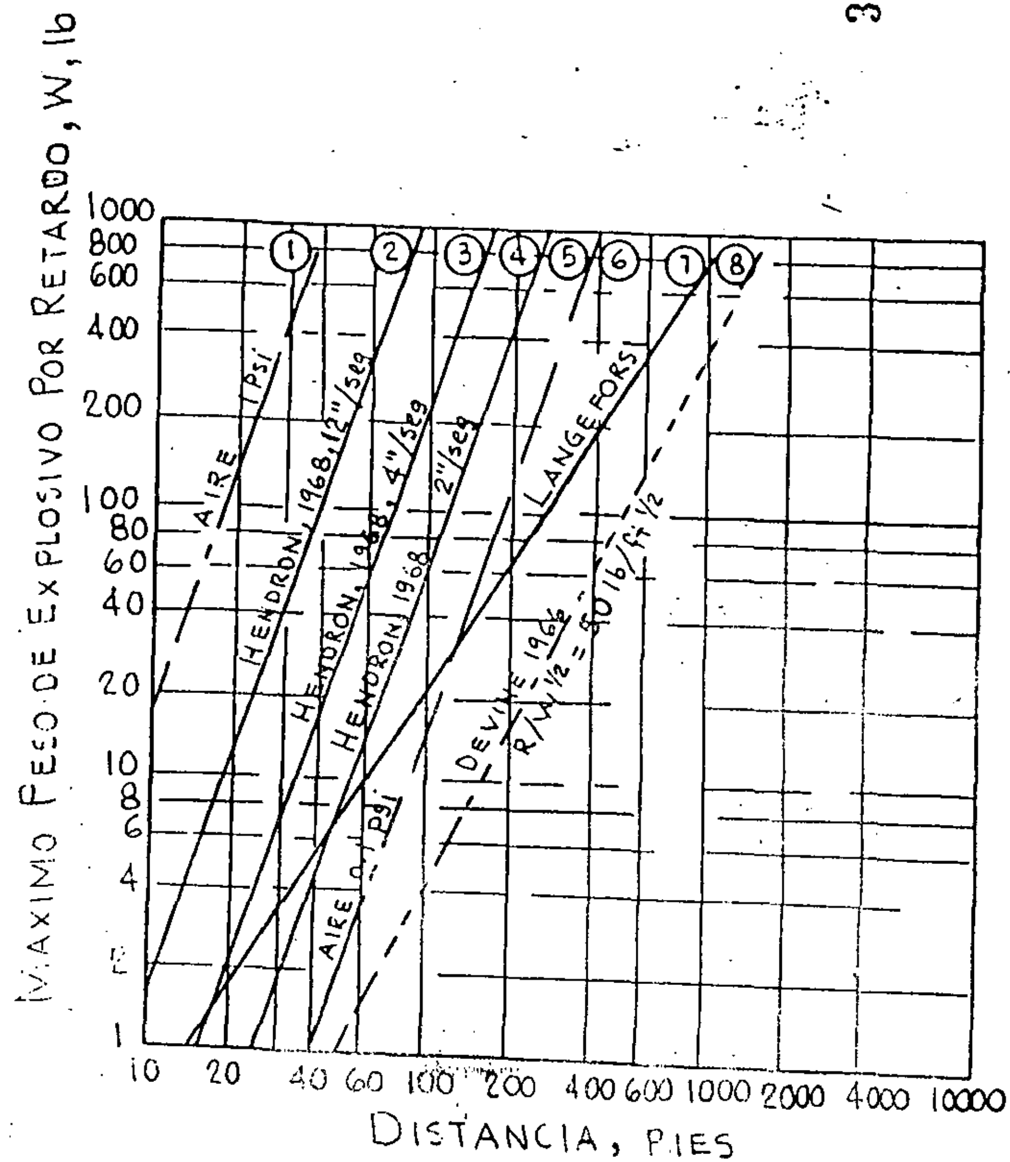
W   máxima carga por retardo

La velocidad de partícula permisible es de 2"/seg.

En la tabla I se presentan las cargas de explosivo máximas permisibles por retardo en Suecia.

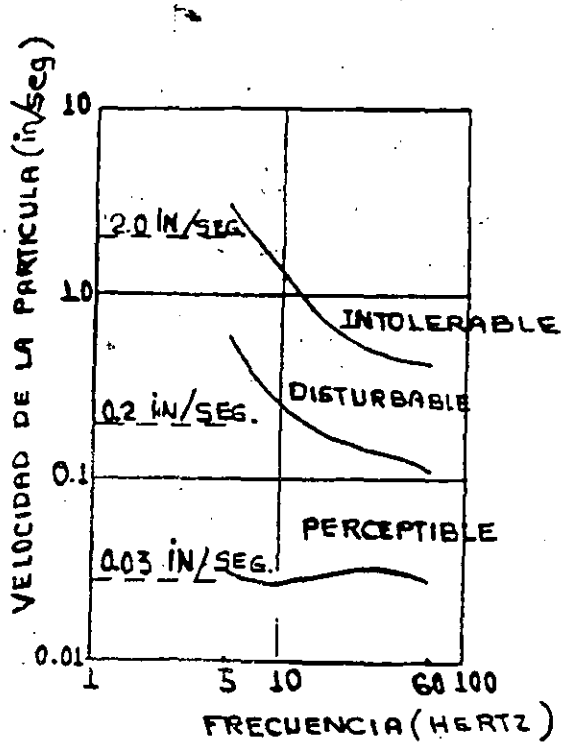
En la figura 3 se presentan los efectos de la velocidad de la partícula con la respuesta humana.





$$v = H\left(\frac{D}{W^{1/2}}\right)^{-\beta}$$

$$SD = \frac{R}{W^{1/2}} \; ; \; W = \left(\frac{R}{SD}\right)^2$$



RESPUESTA HUMANA A LA VIBRACION (SEGUN RATHBONE)

HUMAN AND STRUCTURAL RESPONSE TO SOUND PRESSURE LEVEL

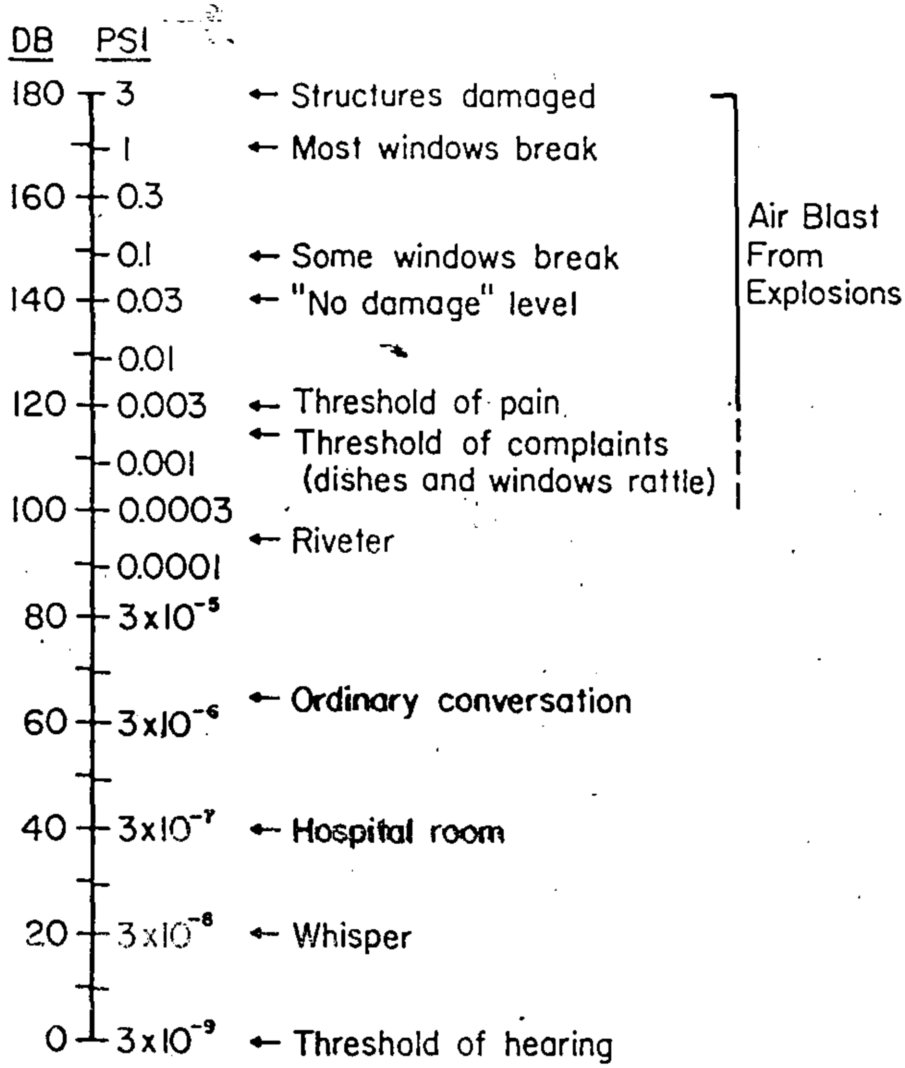


Figure V-3.

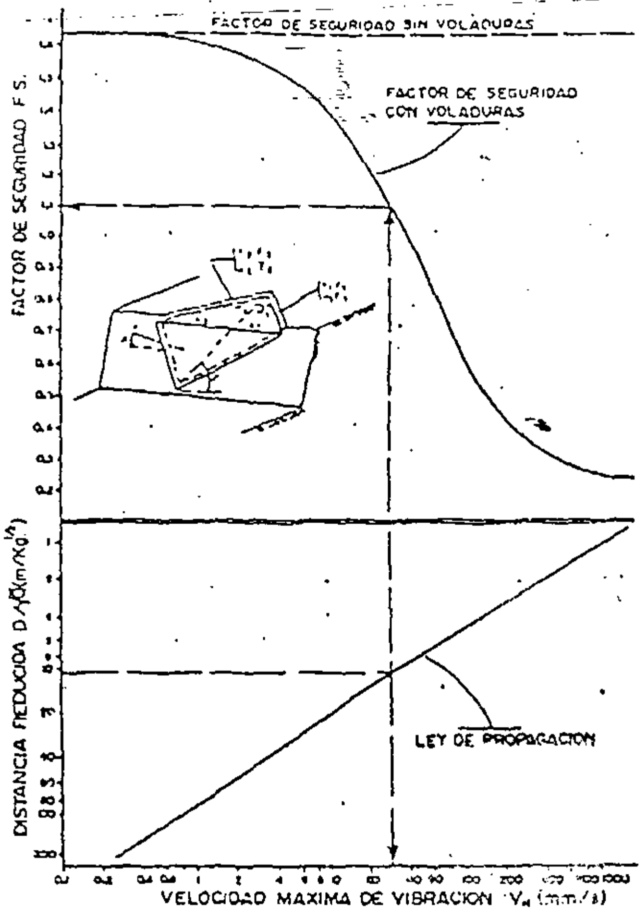


Figura 33.44. Variación del Factor de Seguridad para un bloque de una rotura en cuña en función de la Distancia Reducida.

Hormigón en masa o de relleno

$Q = 38{,}20 \times 10^{-3} \times DS^{\text{[illegible]}} \times K$ (DS en m y Q en kg)

siendo:

K = 1,0 para t = 0 – 4 horas
K = 0,16 » t = 4 – 24 horas
K = 0,3 » t = 1 – 3 días
K = 0,7 » t = 3 – 7 días
K = 2,3 » t = 7 – 10 días
K = 5,5 » t = + 10 días

Hormigón armado o estructural

$Q = 14{,}55 \times 10^{-3} \times DS^{\text{[illegible]}} \times K$ (DS en m y Q en kg)

Otros factores a tener en cuenta son las frecuencias características de las vibraciones, condiciones externas de fraguado, superficies de contacto roca-hormigón, etc.

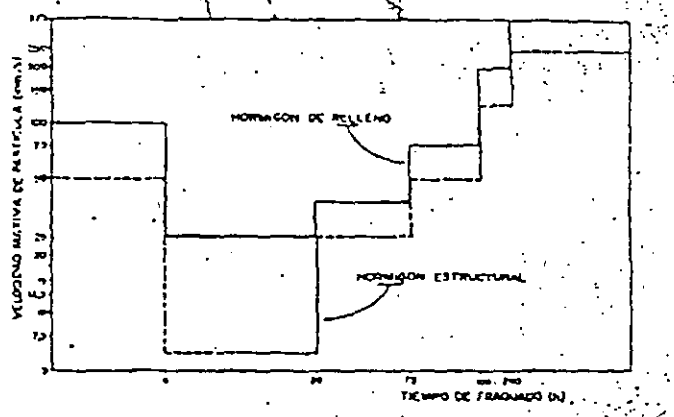
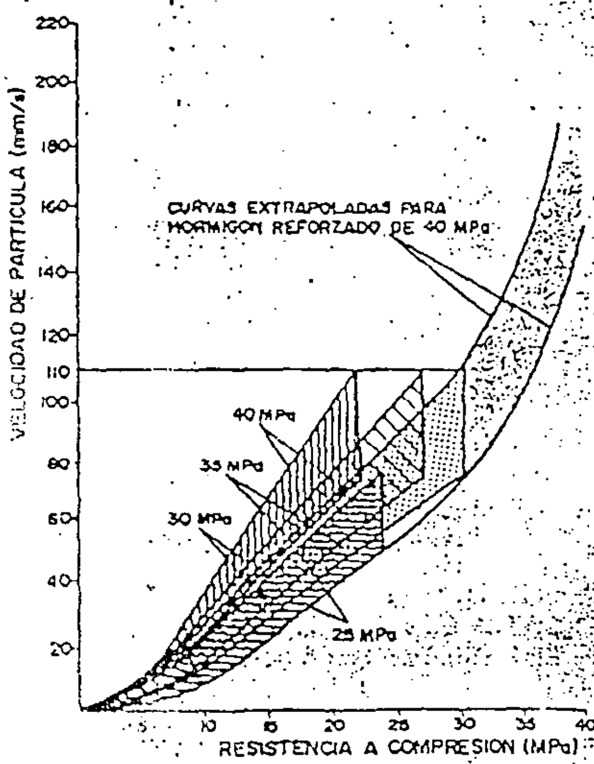


Figura 33.45. Velocidades máximas de partícula en función del tiempo de fraguado.

Por otro lado, Isaac y Bubb (1981) resumen todas sus experiencias y las de investigadores escandinavos en un gráfico donde según la resistencia adquirida por el hormigón se determina el nivel máximo de vibración.



INTERRUPT

TABLE 13-1 Blast-Produced Cracking in Rock

Explosive	Rock Type	Radius of Damage (hole radii)	Crushed Zone (hole radii)	Hole Diameter (mm)
C-4	Granite	18–20	—	50–125
60% dynamite	Shale	42–55	—	
AN-FO	Shale	15–22	—	
60% dynamite	Tuffaceous and pyroclastic	20–30	3	
	Soft rock	26–29	—	
	Hard rock	20–23	—	
Nuclear (TNT)	Granite	4.9	1.9	
	Granite	—	3.0–3.5	
C-4	Granite	—	2.3	
AN-FO	Granite	14	—	165

Source: After Siskind and Fumanti (1974).

postblast cores. Cracking was assumed to extend to the point at which the DPV diverged. In addition, strains were measured within the rock mass by grouting solid-state transducers on cores in a manner similar to that employed in the Tart study. See Chapter 18 for a conservative design guide for rock cracking.

CURING CONCRETE

One of the concerns on any construction project is the strength of the cast-in-place concrete, and most engineers assume that its resistance to vibration is proportional to its strength, which is very small before it cures. Therefore, blasting is usually greatly restricted near setting or curing concrete. Recently, several experiments have been performed to investigate the validity of the assumption of vibratory sensitivity and reveal surprising results.

The simplest, yet most direct experiment was conducted by Esteves (1978). He formed 0.85 × 0.2 × 0.2 m (2.8 × 0.65 × 0.65 ft) prisms of concrete and imposed longitudinal wave tensile stresses on the concrete as it cured with hammer blows as shown in Figure 13.4a. A plot of the resulting peak particle velocities required to fracture different prisms at various times after curing is shown in Figure 13-4b. If the longitudinal wave propagation velocity of fully cured concrete is assumed to be 3000 m/s (10,000 ft/sec), the plane wave strain associated with the minimum velocity for cracking is

$$\epsilon = \frac{\dot{u}}{c} = \frac{150 \text{ mm/s}}{3000 \text{ m/s}} = 50\mu$$

Other studies have shown that during curing, the modulus, and thus the compressive w... elocity, are much lower than the final value. Thus the strain [illegible]

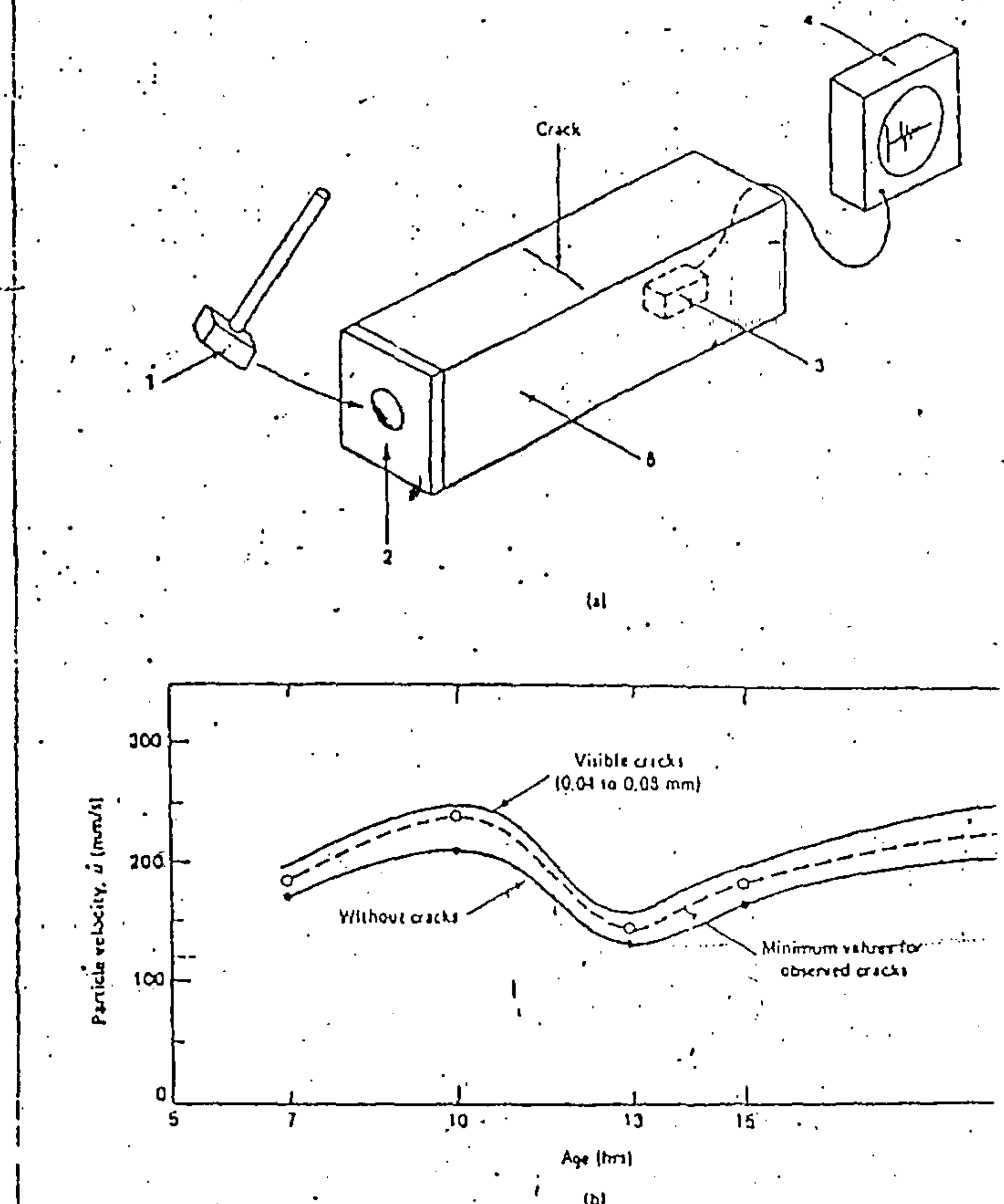


Figure 13.4 Vibratory cracking of curing concrete. (a) Scheme of the test: 1, hammer, 6 kg; 2, iron plate; 3, geophones; 4, recorder; 5, concrete block with 0.20 × 0.20 × 0.85 m. (b) Critical velocity level. (From Esteves, 1978.)

REDUCCION DE EFECTOS DE VIBRACIONES

Para reducir los efectos de las vibraciones deberán tomarse las siguientes acciones:

1. Seccionar la voladura (dividir el banco)

2. Reducir la carga por retardo

3. Cerrar el patrón de barrenación, y

4. Utilizar dos cargas por barreno.

ABBREVIATIONS AND SYMBOLS FOR EXPLOSIVE ENGINEERING DESIGN

a - Constant

A - Blast area, sf

Ap - Amplitude of incident compressive wave, ft.

AS - Time used in adding steels for one drill cycle, min.

b - Constant

B - Optimum burden for charge, ft.

B' - Minimum burden for charge, ft.

B" - Maximum burden for charge, ft.

c - Unit cohesion of material, psi

C - Total weight of primary explosive charge, lb.

C_d - Bucket capacity of power shovel, cy (broken material)

d - Any distance, ft.

d_e - Loading density of explosive, ppf

d_p - Distance between primers, ft.

d_p' - Minimum distance from collar to initial primer, ft.

d_p'' - Maximum distance between any two primers fired simultaneously, ft.

d_r - Density of rock, pcf (solid)

D - Time consumed in penetrating for one drill cycle, min.

D_c - Critical diameter of an explosive, in.

D_e - Diameter of explosive charge, in.

D_h - Diameter of blasthole, in.

D_p - Diameter of primer charge, in.

DFR - Drill footage rate, fpm

DPR - Drill penetration rate, fpm

e - Voids ratio

E - Total weight of blasthole explosive charge, lb.

E_r - Modulus of elasticity for rock, psi

f - Frequency, cps

F - Total footage drilled for blast round, ft.

g - Acceleration due to gravity, fps^2

G_r - Modulus of rigidity, psi

H - Length of blasthole, ft.

H_w - Depth of water in blasthole, ft.

H' - Minimum length of blasthole with single primer, ft.

H" - Maximum length of blasthole with single primer, ft.

J - Depth of subdrilling, ft.

KE - Kinetic energy, ft-lb

K_B - Burden ratio or constant

K_i - Refractive index of a material

K_m - Bulk modulus of a material, psi

K_r - Blastability coefficient for a rock

K_S - Spacing ratio

K_V - Velocity ratio

L - Length of open face parallel with blasthole axis, ft.

L_e - Length of explosive cartridge, ft.

L_d - Maximum dumping height for power shovel, ft.

L_m - Maximum cutting height for power shovel, ft.

M - Time used for moving and setting-up for one drill cycle, min.

n - Numer of blastholes per row

N - Total number of blastholes

p - Number of full passes (loading on both sides) for power shovel

P - Weight of primer charge, lb.

PC - Total length of explosive column in blasthole, ft.

PF - Powder factor for single charge, tons/lb or lb/cy

P_d - Detonation pressure, psi

P_e - Explosion pressure, psi

q - Number of explosive cartridges required to build out of water in blasthole.

PR - Production rate, tph or cy/hr (Solid material)

Q_e - Heat of explosion from an explosive reaction, Kcal/kg

Q_p - Heat of formation for explosive products, Kcal/kg

Q_r - Heat of formation for explosive reactants, Kcal/kg

r - Number of blasthole rows

R_c - Maximum radius of clean-up of floor level for power shovel, ft.

R_d - Maximum dumping radius for power shovel, ft.

R_m - Maximum cutting radius for power shovel, ft.

R_r - Modulus of rupture of a material (tension or flexure), psi

RE - Relative energy of explosive, ft-lb

RS - Time for removing steels for one drill cycle, min.

S - Spacing distance between blastholes within a row, ft.

SC - Stick count of an explosive

S_f - Swell factor of a material

SG_e - Specific gravity of an explosive

SG_r - Specific gravity of a rock

t - Time, min.

U - Total strain energy, ft-lb

T - Length of stemming in a blasthole, ft.

V - Volume, cf

v - Linear velocity, fps

v_b - Bar velocity of a material, fps

v_e - Explosive's reaction velocity, fps

v_i - Particle velocity, fps

v_p - P-wave propagation velocity of a material, fps

v_s - S-wave propagation velocity of a material, fps

w - Width of cut of blast round, ft (measured horizontally and perpendicular to y)

w' - Minimum cut width for loading by power shovel on one side only, ft.

W - Total weight of rock, tons

x - Constant factor

y - Depth of cut or blast round, ft. (measured horizontally and perpendicular to w)

z - Minimum horizontal spread of brokern material from vertical bench, ft.

Z - Accoustical impedance, lb-sec/cf

Σ - Total or sum of

α - Angle of incident and reflected P-wave, deg

β - Angle of generated reflected S-wave, deg

γ - Natural angle of repose, deg

η - Porosity

ϕ - Angle of internal friction, deg

θ - Angle between normal stress and principal stress direction of no shear (x axis), deg

ρ - Mass density, lb-sec^2/ft^4

ε - Strain, in./in.

σ_c - Compressive stress, or strength, psi

σ_d - Stress at any distance, d, psi

σ_e - Stress introduced into material from explosive, psi

σ_n - Normal stress, psi

σ_p - Compressive stress from P-wave, psi

σ_t - Tensile stress, or strength, psi

τ_s - Shear stress, or strength, psi

μ - Poisson's ratio

Ψ - Failure angle, deg

v_f - Crack or fracture propagation velocity, fps

v_m - Rock movement velocity, fps

t_m - Time for rock to move out from solid, sec.

t_f - Time for fracture to propagate, sec.

EXPLOSIVES ENGINEERING DESIGN RELATIONSHIPS

I. Explosives' Properties

1. $SG_e = 141/SC$
2. $d_e = 48D_e^2/SC$, lb/ft
3. $d_e = 0.34D_e^2(SG_e)$, lb/ft
4. (a) $P_d = [(6.06 \times 10^{-3})v_e^2(SG_e)]/[1+0.80(SG_e)]$, psi

   (b) $P_e \cong = P_d(\text{max.})/2$, psi
5. $\rho_e = 62.4\ (SG_e)/g$, lb-sec²/ft⁴
6. $z_e = \rho_e v_e$, lb-sec/cf

II. Material's Properties

7. $d_r = 62.4\ (SG_r)$, pcf
8. (a) $\rho_r = d_r/g$, lb-sec²/ft⁴

   (b) $\rho_r = 1.941\ (SG_e)$, lb-sec²/ft⁴
9. $E_r = \sigma_c/\varepsilon_c$, psi
10. (a) $G_r = E_r/[2(1+\mu)]$, psi

    (b) $G_r = \tau_s/\varepsilon_s$, psi
11. $K_m = E_r/[3(1-2\mu)]$, psi
12. $Z_r = \rho_r v_p$, lb-sec/cf
13. $K_r = \sigma_c/\sigma_t$
14. (a) $v_p = [2G_r(1-\mu)/\rho_r(1-2\mu)^{1/2}$, fps

    (b) $v_p = [E_r(1-\mu)/\rho_r\ (1+\mu)\ (1-2\mu)^{1/2}$, fps
15. (a) $v_s = (G_r/\rho_r)^{1/2}$, fps

    (b) $v_s = [E_r/2\rho_r(1+\mu)]^{1/2}$, fps
16. (a) $v_b = (E_r/\rho_r)^{1/2}$, fps

    (b) $v_b = [\sigma_c/(\varepsilon_c\rho_r)]^{1/2}$, fps

17. (a) $v_p = v_s\ [(2 - 2\mu)/(1 - 2\mu)]^{½}$, fps

    (b) $v_p = v_b\ [(1 - \mu)/(1 + \mu)\ (1 - 2\mu)]^{½}$, fps

18. (a) $K_i = v_p/V_s$

    (b) $K_i = [2(1 - \mu)/(1 - 2\mu)]^{½}$

19. $v_f = v_p/3$, fps

20. $\sigma_n = (\sigma_c/2)\ (1 - \sin\phi)$, psi

21. $\sigma_t = \sigma_c[(1 - \sin\phi)/(1 + \sin\phi)]$, psi

22. (a) $\tau_s = c + (\sigma_c/2)\ (1 - \sin\phi)\ \tan\phi$, psi

    (b) $\tau_s = (\sigma_c/2)\cos\phi$, psi

23. $c = (\sigma_c/2)\ [\cos\phi - (1 - \sin\phi)\ \tan\phi]$, psi

24. (a) $v_i = 2\pi f A_p$, fps

    (b) $v_i = v_p\varepsilon_c$, fps (for plane wave)

25. (a) $\sigma_p = Z_r V_i$, psi

    (b) $\sigma_p = \rho_r v_p^{\,2}\varepsilon_c$, psi (for plane wave)

26. $Z = (1/2S_f^{\,x})\ [L\cot\alpha + 2y\ (1 - S_f^{\,x})]$, ft

27. $e = \eta/(1 - \eta)$

28. $\Psi = (\phi + 90°)/2$, deg

III. Blasting Design

29. $E = d_e(PC)$, lb

30. $H = L + J$, ft

31. $PC = H - T$, ft

32. $J \cong B/3$, ft. (massive rock)

33. $T \cong 2B/3$, ft. (massive rock)

34. $B = K_B D_e 12$, ft

35. $K_B = 2.5(160/d_r)^{1/3}(SG_e/1.3)^{1/3}[v_e^{\,2}/(12{,}000)^2]^{1/3}$ f

36. $K_v = v_e/v_p$

37. (a) $d_p' = 1.61B$, ft

(b) $d_p'' = 6K_vB$, ft

38. (a) $B' = 3L/(6K_v + 1)$, ft. (single collar primer)

(b) $B' = 3L/(18K_v + 1)$, ft. (single center-of-column pri

(c) $B' = 3L/(9K_v + 2)$, ft. (single floor-level primer)

39. $B'' = 0.62L$, ft. (single floor-level primer)

40. $q = (H_wD_h^2)/(D_h^2 - D_e^2)\ L_e$

41. $A = wy$, sf

42. $W = wyLd_r/2000$, tons

43. (a) $PF = W/EN$, tons/lb

(b) $PF = 27EN/WyL$, lb/cy

44. $DFR = H/(D + AS + RS + M)$, fpm

45. $DPR = H/D$, fpm

46. $F = HN$, ft.

47. (a) $S = (BH)^{1/2}$, ft. (Simultaneous and short delay, $B<H<4B$)

(b) $S = 2B$, ft. (Simultaneous and short delay, $H \geq 4B$)

(c) $S = 1.41\ B$, ft. (Long delay, 90 deg crater)

(d) $S = 1.15\ B$, ft. (Long delay, 60-120 deg, crater)

Loading Shovel Specifications

48. (a) $L_m = L/S_f^x$, ft.

(b) $L_m = 1.2C_d + 30$, ft.

49. $L_d = 0.6C_d + 20$ ft.

50. $R_c = 2C_d + 19$, ft.

51. $R_m = 3C_d + 29$, ft.

52. $R_d = R_m - 5$, ft.

53. $w' = 2.5C_d + 24$, ft.

54. (a) $w = R_c\ (2p - 1) + R_m$, ft. (Rectangular box wt.)

(b) $w = S_f^x\{R_c[1 + 2(p - 1)] + R_m - 0.5L_m\}$, ft. (corner cu

Note: Use w for frontal loading, y for parallel loading.

55. (a) $PR = 1.35\ C_d S_f d_r$, tph (50-min. hour, 30-sec cycle)

(b) $PR = 100\ C_d S_f$, cy/hr (50-min hour, 30-sec cycle)

Note: For $\Upsilon = 45$ deg and $S_f^x = 0.85$,

$L = C_d + 26$, ft, and $Z = 0.59(L + 0.3y)$, ft.

FACULTAD DE INGENIERIA U.N.A.M.
DIVISION DE EDUCACION CONTINUA

CURSOS ABIERTOS

VI CURSO INTERNACIONAL DE INGENIERIA GEOLOGICA APLICADA A OBRAS SUPERFICIALES Y SUBTERRANEAS

MODULO IV: TECNOLOGIA SOBRE EL USO DE EXPLOSIVOS

NOTES ON DETONATION PHYSICS

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NOTES ON DETONATION PHYSICS

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CHAPTER 1

THE DETONATION PROCESS

1.1 Introduction

According to Persson[1] steady state detonation along a cylindrical charge can be regarded as a self propagating process in which the axial compressive effect of the shock front discontinuity changes the state of the explosive so that exothermic reaction sets in with the requisite velocity.

This reaction in homogeneous liquid explosives such as nitroglycerin is completed in a time interval of the order of 10^{-12}seconds[1]. In high explosives, such as RDX and PETN it is completed in about 1μsec . In composite explosives containing AN the reaction times are considerably longer. The significance of this will be demonstrated later.

1.2 Shock waves

Compressional waves of small intensity are propagated in gases at the velocity of the sound. Let us suppose that a column of gas is set in motion by a piston which is accelerated into it. Let us also consider that the velocity of the piston is a staircase function of time. Each step transmits a small compressional wave which advances through the gas already set in forward motion and heated by the previous waves. Since the velocity of the wave is larger at elevated temperatures, the new wave overtakes the previous[2]. Therefore the velocity, pressure and temperature gradients in the front of the wave grow steeper with time. If there is no dissipative mechanism (e.g. heat diffusion) the gradients become infinite[2].

This type of wave, in which a discontinuity has developed is known as a shock wave. The area of pressure rise is called the shock front. The front advances with a speed higher than the sound speed. The shock velocity depends on the conditions behind. If the pistons continues accelerating so does the front. If the piston maintains a constant velocity, the front maintains a constant velocity as well. If the piston decelerates a wave of rarefaction is formed ahead of it. Finally this wave overtakes and weakens the shock front.

It follows that the velocity of the front is determined by the conditions behind the front. The wave does not maintain itself. Rather it depends on the support provided by the piston.

1.3 Detonation waves

However from our experience we know that steady detonation waves exist. In this case the role of the piston is played by the reaction taking place in the detonation wave.

Let us consider a plane detonation wave which has been established in an explosive (Figure 1). The wave front advances into the unconsumed explosive with a constant velocity D and it is followed by the reaction zone. If an observer is moving with the velocity D of such a front, the wave will appear to him/her as in Figure 1. Undetonated explosive flows into the shock front AA' with constant velocity $U_0 = -D$. Its pressure, temperature and density and internal energy per unit mass are P_1, T_1, ρ_1, E_1 at all points to the right of AA'. The wave front is considered to be a discontinuity in comparison to the changes occurring behind it. Therefore at AA' these values change to values P_2, T_2, ρ_2, E_2. These values change at some later stage.

The apparent velocity of the mass leaving the front is -(D-U_p) where U_p is the particle velocity (mass velocity) in the zone between AA', BB', relative to the fixed coordinates.

If we consider a region of flow surrounded by a tube of unit sectional area and two planes, one just before the detonation front and one right after it, the mass flowing in must equal the mass flowing out (conservation of mass). The mass flowing in per unit time is $\rho_1 D\,dt$. The mass flowing out is $\rho_2(D-U_p)dt$. Therefore :

$$\rho_1 D = \rho_2(D-U_p) \qquad (1)$$

Furthermore the difference in momentum should be equal to the impulse of the net force. Thus:

$$\rho_1 Ddt\,D - \rho_1 Ddt(D-U_p) = (P_2-P_1)dt$$

or $$P_2-P_1 = \rho_1 DU_p \qquad (2)$$

P_1 is very small compared to the detonation pressure. Therefore it can be ignored and equation (2) can be written as :

$$P_2 = \rho_1 DU_p \qquad (3)$$

From equation (1), one can obtain:

$$U_p = (1-\rho_1/\rho_2)D \qquad (4)$$

According to Cook[3] U_p/D and ρ_1/ρ_2 are slowly variable functions of the original density. Thus:

$$U_p = f(\rho_1)D \qquad (5)$$

where $f(\rho_1) = 1 - \dfrac{\rho_1}{\rho_2}$

Therefore equation (3) can be written as:

$$P_2 = \rho_1 f(\rho_1)D^2 \qquad (6)$$

For most cases (explosives having a density between 0.9 -

1.4g/cc) it is sufficiently accurate to assume $f(\rho_1) = 4.0$. Under this approximation, the detonation pressure in atmospheres when the velocity of detonation is given in meters per second, is given by the following equation[8]:

$$P_2 = 0.00987 \rho D^2/4 \quad (7)$$

This is a relationship of great practical value. It allows the estimation of the detonation pressure when only the detonation velocity and the initial density are known. It is worth mentioning that the detonation velocity can be measured accurately in the laboratory.

Apart from equations (1) and (2) other equations are used in the theory of detonation. Many of these fall outside the area of interest of these notes. They are mentioned in the following to assist the reader in further studies.

The conservation of energy is expressed by the following equation:

$$E_2 - E_1 = \frac{1}{2} (P_2 + P_1)(V_2 - V_1) \quad (8)$$

This is known as the Rankine-Hugoniot equation.

A fourth equation is the equation of state of the reaction products of the explosive.

The above four basic equations are not enough to calculate the five unknown quantities behind the detonation front (energy, density, detonation velocity, pressure and particle velocity). A fifth condition is necessary. This is the Chapman- Jouguet hypothesis stating that the detonation velocity equals the local sound speed plus the particle velocity at the detonation state. Therefore:

$$D = C + U_p \quad (9)$$

Equations (1),(2),(8),(9) and the equation of state of the detonation products are essential for the calculation of the detonation parameters in the thermohydrodynamic codes.

1.4 The Detonation Head Model[3,4]

Practical explosives are used normally in the form of cylindrical charges. Cook's detonation head model illustrates the sequence of events taking place. Figure 2 shows the detonation head formation in a cylindrical unconfined charge. With strong priming a detonation wave travels out from the primer and along the charge. This is responsible for the promotion of the necessary exothermic detonation reactions within the explosive charge. At the back of the primer the high pressure gases expand into the surrounding air. As this expansion takes place it permits a release wave or a rarefaction wave to travel down the charge behind the detonation front. This always lags the detonation front for reasons which were explained earlier. In a similar manner at the sides of the charge immediately after the detonation wave the gases expand into the atmosphere. Again two release waves are travelling into the charge. The detonation front, rear release wave and side release waves define a region called the detonation head. The detonation head is a region associated with high pressure and high density. The shape of the detonation head depends on the geometry of the charge and changes as it travels out from the initiation source. This is due to the approximately constant relationship between the release wave velocity and the detonation velocity. Initially the shape is that of a section of a truncated cone with curved front and rear surfaces. Further away from the initiation the length of the detonation head grows so that it is controlled from the side release waves which meet on the axis of the charge forming a cone. It has been found (X ray radiography) that the length of the cone when the detonation is fully developed is approximately equal to the diameter of the charge. The density inside the detonation head is constant and approximately equal to 4/3 ρ_1 where ρ_1 is the initial density of the explosive. The distance from the initiator to the point where the full head is formed is approximately equal to 3 1/2 charge diameters for unconfined charges. As the explosive enters the detonation head it reacts. If it is in a granular form (e.g ANFO prills) the reaction starts at the surface and proceeds radially towards the centre of the prill. As it was mentioned in the previous the energy liberated supports the detonation. If the reaction is not completed inside the head the energy liberated is less than the maximum available and the detonation velocity is less than the maximum. This is what is normally known as non-ideal detonation. It is worth mentioning that non ideal detonations can be stable; indeed a great number of commercial explosives used by the mining industry today detonate at non ideal velocities at the diameters at which they are used.

The detonation velocity is the most important parameter of the detonating explosive. It is well known that the velocity of detonation is a constant characteristic of a particular explosive when the other parameters are kept constant. It was explained that the knowledge of the detonation velocity can lead to fairly accurate estimates of the detonation pressure which is of particular importance and cannot be measured directly. In the next chapter the parameters influencing the detonation velocity will be discussed.

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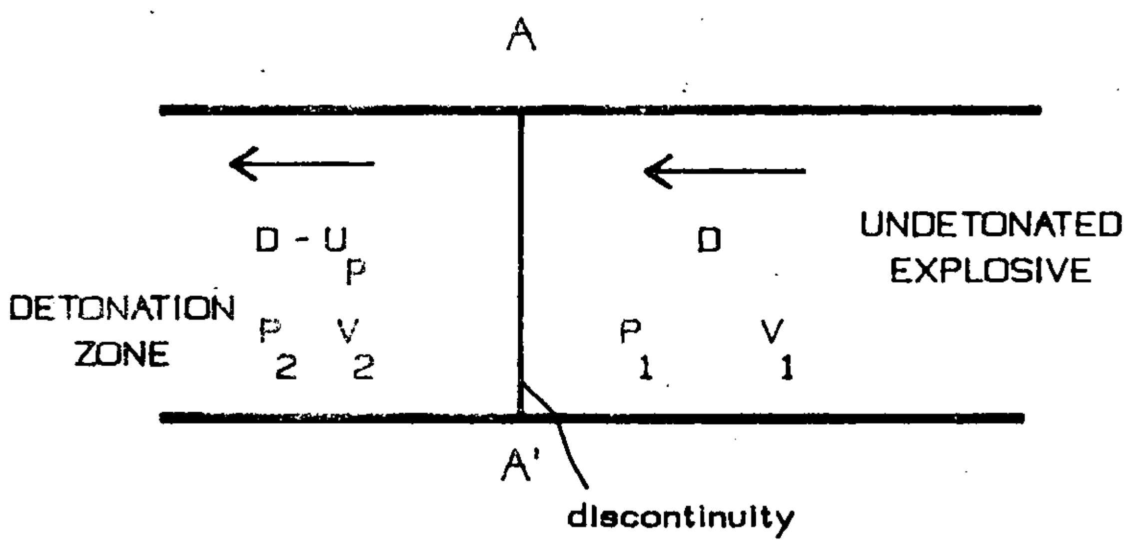
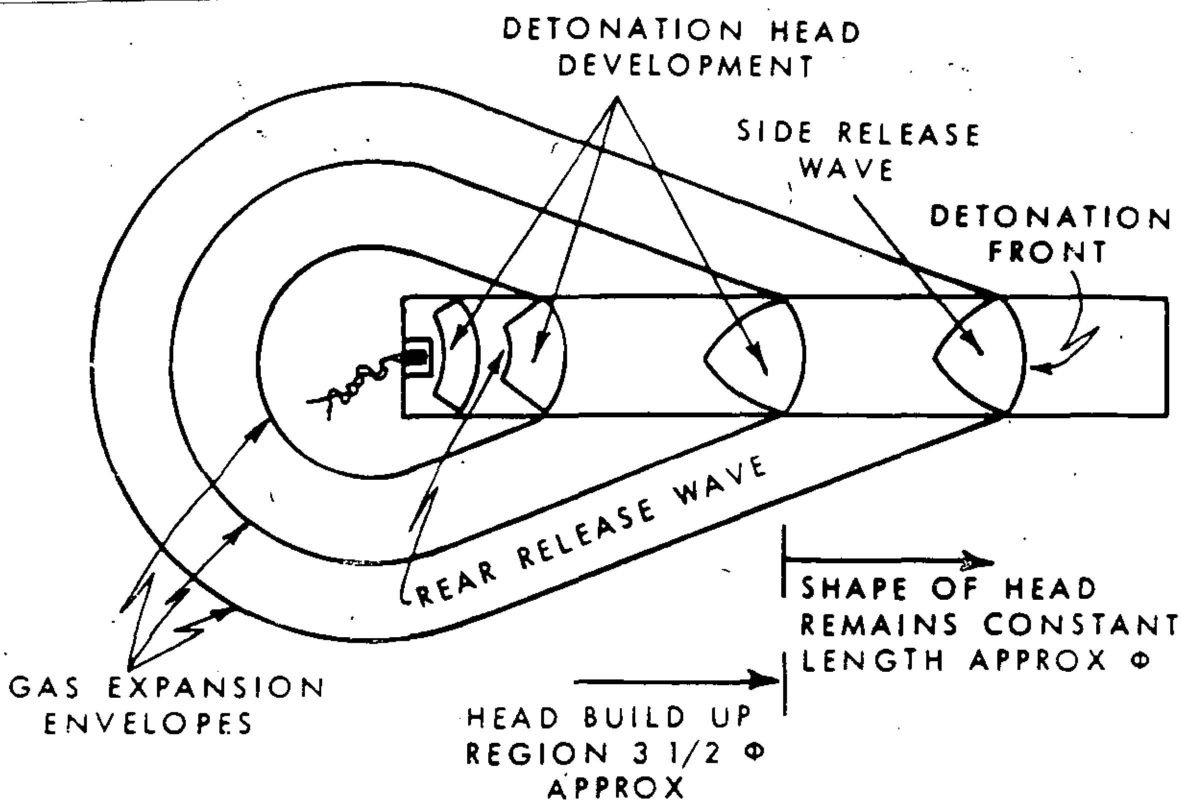


FIGURE 1: SECTIONAL DIAGRAM OF A DETONATION WAVE

Observer moves to right at wave velocity D.

The discontinuity is at rest



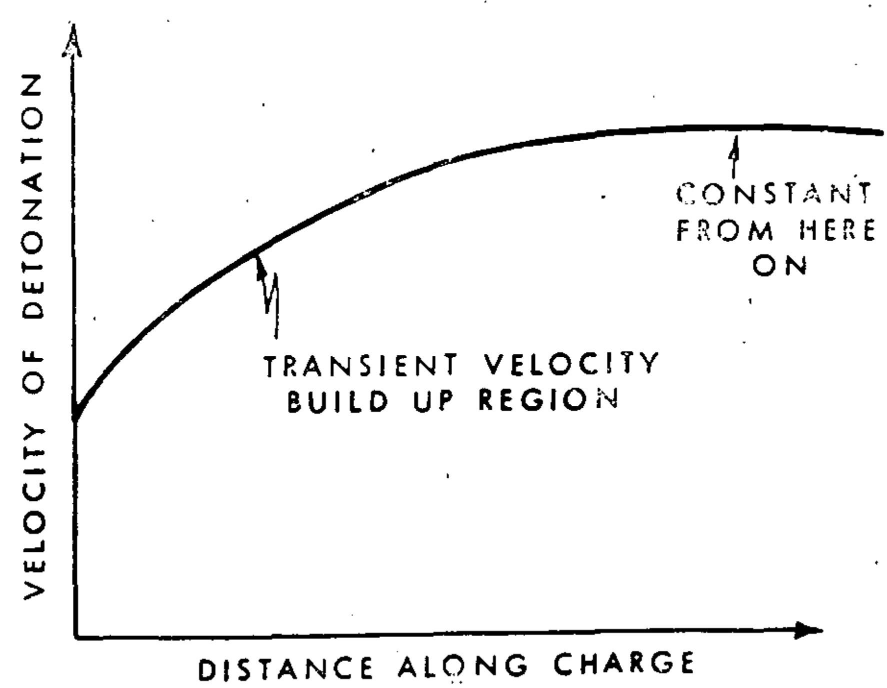


FIGURE 2: DETONATION HEAD FORMATION

(AFTER COOK AND BAUER)

RCB

EQUATIONS OF STATE

An equation of state is normally a pressure - volume - temperature relationship. Ideal gases have an equation of state expressed as:

$$PV = nRT$$

where P is the pressure
T is the temperature
n is the number of moles of gas
R is the universal gas constant and
V is the volume.

However real gases do not always behave according to the previous equation. It is obvious that a real gas cannot be cooled to zero volume. Under certain conditions gases turn into liquids or solids.

The origin of the deviations from ideality is the interaction between particles. Molecules excercise attractive forces when they are separated by some distance and repulsive forces when they are very close together.

Repulsive forces are short term interactions while attractive forces have a relatively long range. Figure 1 provides a plot of the compression factor $Z = PV/RT$ against pressure applied on the gas. One can obtain an indication of the imperfection at different pressures. For a perfect gas $Z \doteq 1$ under all conditions. For a real gas the case is somewhat different. At very low pressures all gases behave almost ideally ($Z = 1$). At high pressures the repulsive forces dominate and $Z > 1$, while at moderate pressures $Z < 1$ due to the attractive forces. Obviously an equation of state for the detonation products has to reproduce this behaviour of real gases.

EQUATIONS OF STATE FOR DETONATION PRODUCTS.

The equations of state used for detonation calculations are of two types: those which do not treat chemistry explicitly and those which do. The latter contain individual equations of state for the component molecules and a mixture rule for combining them to give an equation of state for any composition. The composition of the detonation products is calculated by assuming chemical equilibrium.

At this point it is worth mentioning that much of the work involving the development of an equation of state has been employed in an inverted form. Experimental values are used to calibrate an assumed form of an equation of state. Attempts to develop a general, completely theoretical equation of state have failed to produce a good result.

The most common equations of state for detonation products are:

1. The Abel Equation of State.

The Abel equation of state is a form of the Van der Waal's equation of state. It can be expressed as:

$$P(V-\alpha) = nRT$$

where α is a constant.

It was found that this form did not produce acceptable results for many cases of condensed explosives. Cook[1] provided a modification expressing α as a function of the volume of the detonation products without considering their chemical composition. He showed that the empirical values of the covolume fall in a common $\omega(V)$ curve.

2. The Becker - Kistiakowsky - Wilson Equation of State.

The most popular equation of state is the BKW equation. The equation has the following form:

$$\frac{PV}{RT} = 1 + xe^{\beta x}$$

where $x = \dfrac{K}{V(T+\theta)^{\alpha}}$

and $K = \kappa \Sigma k_i x_i$

with $\alpha, \beta, \kappa, \theta$ and k_i empirical constants. The constants k_i of each molecular species are the covolumes. For the mixture each k_i is multiplied by x_i, the mole fraction of species i, and summed to find the effective covolume.

According to a parameter study performed by the Los Alamos Laboratory, one may adjust the BKW parameters α, β, κ and θ and the covolumes of the detonation products. Cowan and Fickett[2] have shown that for a given α and β one may adjust κ to obtain the experimental velocity of detonation. The slope of the curve relating detonation velocity and density can be changed by changing β.

By using one explosive as a standard it was possible to obtain a set of parameters which can be used for a variety of explosives. BKW has been calibrated for RDX and TNT. The most common parameters used today are shown in Table 1[(3,4)]. It has been found that the RDX parameters result in realistic values of the detonation parameters (pressure and velocity of detonation). The parameters which have been developed based on TNT as the standard produce reliable results for very oxygen deficient systems which produce large amounts of carbon in the detonation products.

The best fit for RDX parameters should not be used in predictions of the detonation state parameters. This set was developed in order to have $(dP/dT)_V > 0$ at pressures of the order of 0.5 Mbar. It has been found that this set of parameters results in poorer predictions than the RDX set.

3. Other Equations of State

Other equations of state have been developed by Fickett and by Jacobs, Cowperthwaite and Zwisler[(4)].

These equations are similar and they are based on statistical mechanics. They use the Lennard-Jones potentials to describe the interactions between the molecules. The general form of the intermolecular potential energy is shown in Figure 2. When the molecules are squeezed together, the nuclear and electronic repulsions dominate the attractive forces. The repulsions increase steeply with decreasing separations. One approximation is the the hard sphere potential where it is assumed that the potential energy rises abruptly to infinity as soon as the particles come within some separation distance σ (collision diameter).

Normally the intermolecular potential is written as:

$$V = {}^{C_n}/_{R^n} - {}^{C_6}/_{R^6}$$

This is the Lennard-Jones (n,6) potential. Often the (12,6) potential is written in the form:

$$V = 4\varepsilon[(\sigma/R)^{12} - (\sigma/R)^{6}]$$

where ε is the depth of the potential well and
σ is the separation distance at which V=0.

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FIGURE 1: COMPRESSION FACTOR VS PRESSURE

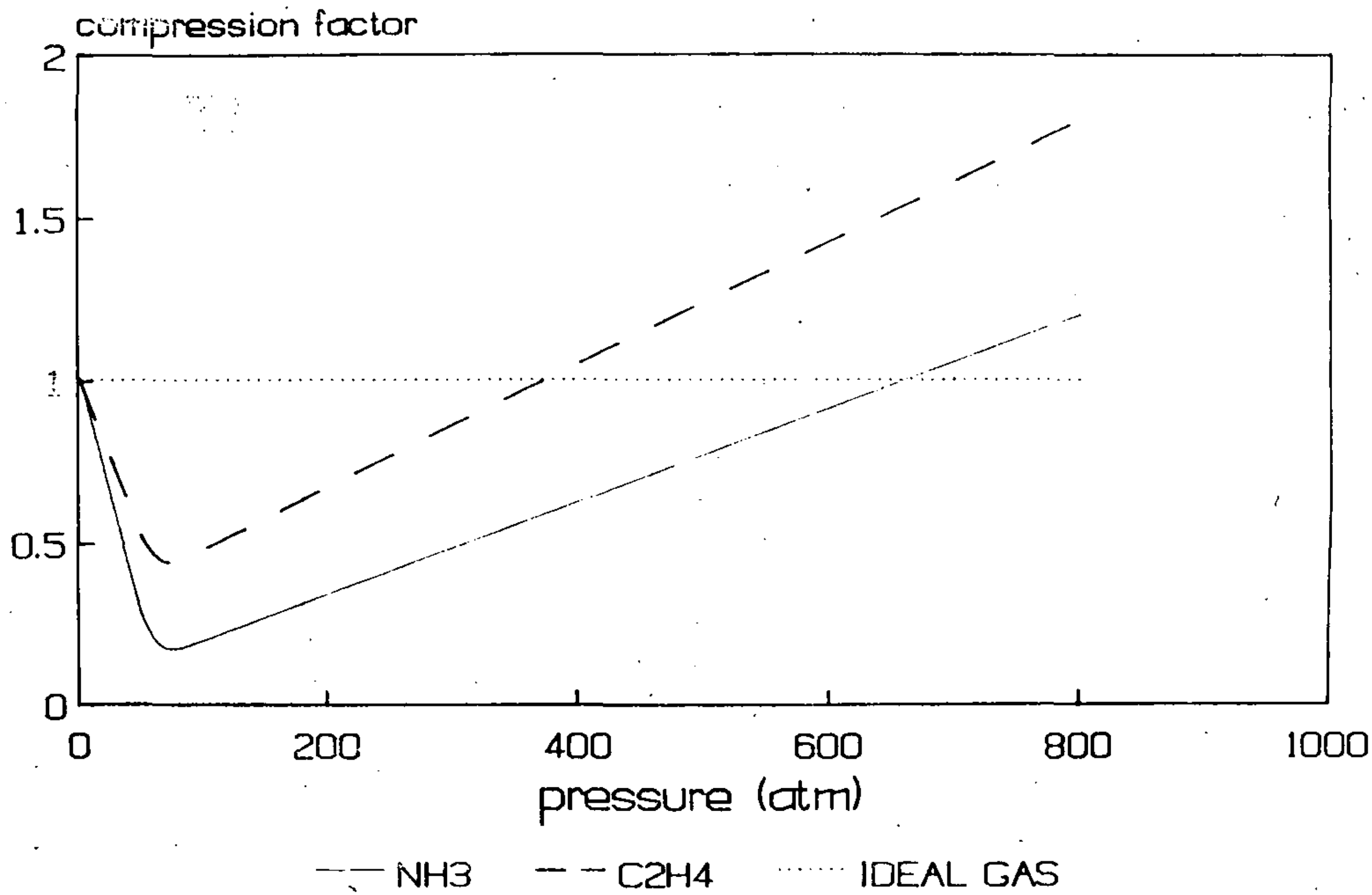

FIGURE 2: POTENTIAL ENERGY BETWEEN MOLECULES

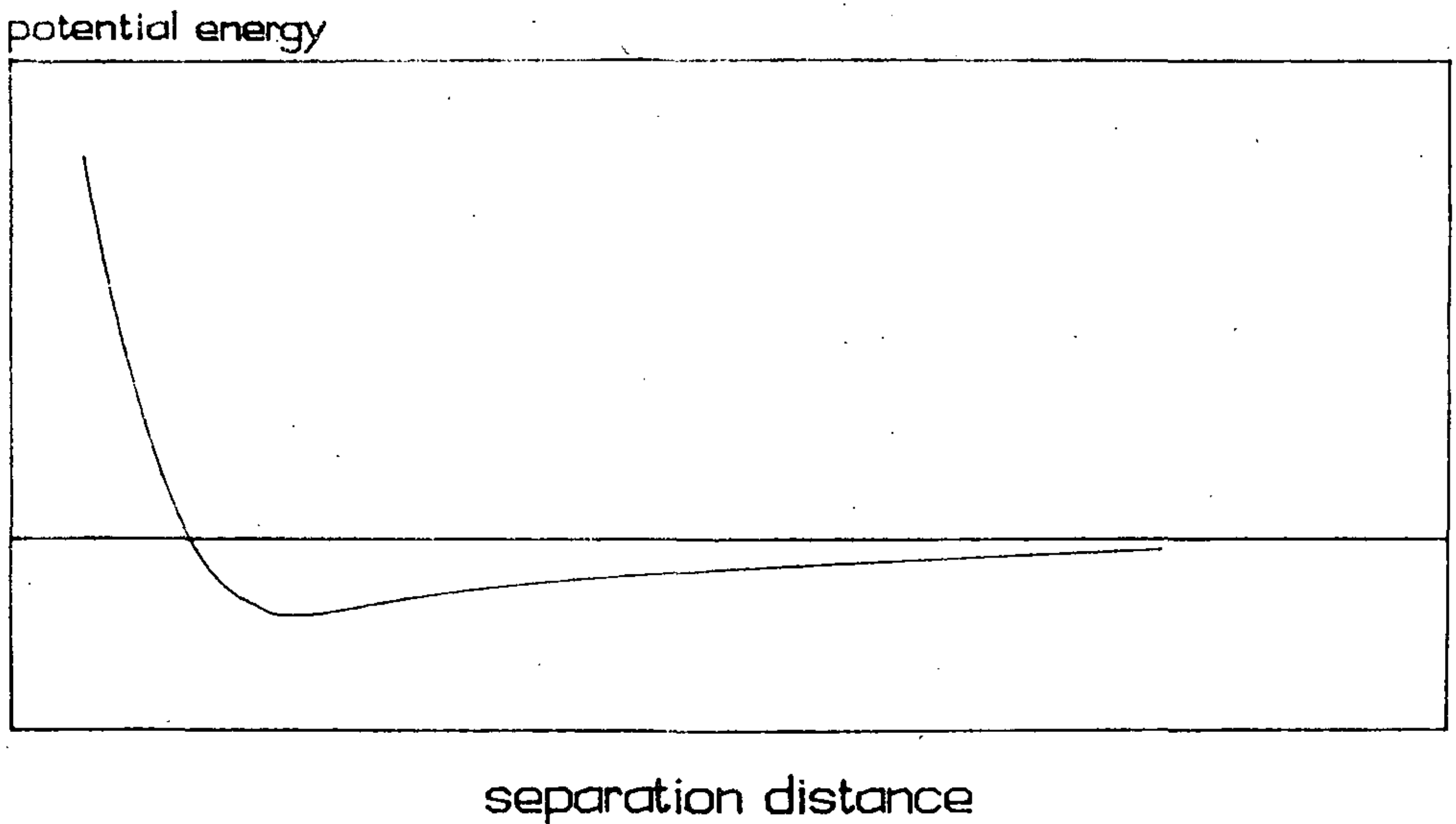

TABLE 1

COMMONLY USED BKW PARAMETERS FOR HIGH DENSITY EXPLOSIVES

NO.	PARAMETER SET	β	κ	α	θ
1	Fitting RDX	0.181	14.15	0.54	400
2	Fitting TNT	0.09585	12.685	0.50	400
3	Best fit for RDX with $(\partial P/\partial T)_V > 0$	0.16	10.91	0.50	400
4	Default parameters	0.10	11.85	0.50	400

CHAPTER 3

EXPLOSIVE PROPERTIES

3.1 Introduction

A variety of factors influence the explosives selection process. This chapter discusses the most important of them and the parameters which influence them.

3.2 Velocity of Detonation

The velocity of detonation is the velocity at which the detonation wave travels through an explosive charge. The detonation wave travels at speeds above the normal sound speed of the unreacted material. Typical detonation velocities for commercial explosives range from 2500 to 7000 m/sec. The detonation velocity is the most important property of the explosive. It can be measured easily and accurately and it can be used for the calculation of the detonation and borehole pressures which are of importance in explosive applications. The velocity of detonation of a particular explosive depends on factors such as charge diameter, confinement, density and particle size.

3.1.1 *The effect of charge Diameter*

Let us consider a typical velocity of detonation - diameter curve as shown in Figure 1[2]. If the diameter is too small the explosive fails to detonate. At some minimum diameter stable detonation occurs. This minimum diameter is called the critical diameter of the explosive.

As the charge diameter is increased the detonation velocity is increased as well. However when a certain diameter is reached, further increase in diameter does not result in an increase of the detonation velocity. At this point a maximum detonation velocity of the explosive is reached. This velocity is called the ideal detonation velocity of the explosive and is the value predicted by thermohydrodynamic codes.

The detonation head model as developed by Cook(1) can be useful in explaining the shape of the observed detonation velocity - diameter curves. Figure 1 illustrates the length of the established detonation heads in charges of various diameters and indicates what happens when a solid particle of explosive enters the detonation head. For the small diameters, the degree of reaction is small and the energy liberated is not enough to support a detonation. As the diameter is increased the detonation head length is increased and for the same size of particle the degree of reaction increases. At the critical diameter the degree of reaction is sufficient to support stable detonation. If the diameter is increased further a larger amount of explosive reacts in the detonation head. When the ideal detonation occurs, the full amount of explosive reacts in the detonation head.

3.2.2 *Effect of Confinement*

The effect of confinement is to lower the rate of expansion of the gases off the side of the charge(2). This in turn slows down the rate at which the lateral rarefaction travels into the reaction region. As a result it takes longer for the side release waves to meet on the charge axis. The length of the detonation head is thus increased. This is shown in Figure 2(1), where the development of the detonation head is outlined for both the confined and the unconfined cases. Therefore, if the explosive was not reacting fully at a particular charge diameter, the effect of confinement would be to increase the degree of reaction and consequently the detonation velocity at this diameter. Similarly, confinement will reduce the critical charge diameter (Figure 3)[2].

However confinement cannot be quantified. Steel, glass, various kinds of rock and soil will produce a different effect. For this reason most of the tests are done with the explosive charge unconfined.

3.2.3 *Effect of Particle Size*

If the size of the explosive particles is reduced at a given charge diameter in the non ideal velocity region, the degree of reaction is enhanced because of the increase of the surface area. Furthermore since the grains are smaller, they are consumed faster in the detonation head. As a result the critical diameter is decreased and the explosive reaches ideal detonation at a smaller diameter (Figure 4)[2].

3.2.4 *Effect of Density*

If the density is increased, the specific energy is increased; as a result the ideal detonation velocity is increased. It has been found that the detonation velocity and the density are related linearly. Figure 5[3] shows the detonation velocity density relationship for various explosives.

However if the density is increased beyond a critical point, steady state detonation is not possible. The phenomenon is called dead packing and a qualitative explanation can be given by the fact that the volume of the entrapped air is insufficient to provide enough hot spots for the reaction to proceed[2].

The relationship between critical diameter and density is shown in Figure 6[5]. It is obvious that apart from the density in which the material is dead packed there is a critical density below which the explosive will not shoot.

3.2.5 *Effect of Temperature*

The initial temperature of the explosive has a small influence on the velocity of detonation at diameters well above the critical. However the critical diameter is dependant on the initial temperature. Figure 7 shows the effect of the temperature on the critical diameter powdered TNT[4].

In the case of commercial liquid explosives the effect is more pronounced. Figure 8 shows the effect of low temperatures on the critical diameter of typical slurry explosives[5]. The effect on solid explosives is almost negligible.

3.2.6 *Effect of Water*

Generally dynamites are not affected by the presence of water inside boreholes. Ammonium nitrate mixed with fuel oil has no water resistance. The product absorbs water and soon becomes desensitized. Generally performance drops drastically as the weight of water in the composition is increased.

3.3 Detonation Pressure

The detonation pressure is a very important parameter. It is an indicator of the ability of the explosive to produce the desired fragmentation in the rock. However, due to its high magnitude the detonation pressure cannot be measured directly. For this reason the experimental determination is difficult.

The detonation pressure is related to the square of the detonation velocity. Parameters which influence the detonation velocity have a very significant effect on the detonation pressure.

3.4 Detonation Temperature

The detonation temperature is the parameter about which the least amount of information is available(6). The detonation temperature is measured from the brightness of the detonation front as it is observed by a sensor. However it is not known how much radiation is absorbed from the partially decomposed material between the sensor and the front. Furthermore, any gas bubbles in the material will flash brightly when they are impacted by the detonation wave. This, obviously, will affect the measurement.

3.5 Fumes

It must be assumed that in all cases explosive fumes are to some degree toxic. Excess oxygen causes the formation of nitrogen oxides while oxygen deficiency causes the formation of carbon monoxide.

In the United States the fumes of any explosive are classified after detonating the explosive in a Bichel bomb and analyzing its fumes. The following classes exist(7):

A. Permitted explosives (USBM)

Fume class	Toxic Gas ft^3/lb	Toxic Gas l/kg
A	< 1.25	< 78
B	1.25 - 2.50	78 - 156
C	2.50 - 3.75	156 - 234

B. Rock blasting explosives

Fume class	Toxic Gas ft^3/lb	Toxic Gas l/kg
1	< 0.16	10
2	0.16 - 0.33	10 - 21
3	0.33 - 0.67	21 - 42

Canada uses the same standards. However explosives of class 2 or 3 cannot be used in underground mines unless special application has been made to and permission is received from the authorities (EMR).

It is worth mentioning here that the relative toxicity of the fumes is important and this is not shown in the above tables. NO_2 is much more toxic than CO (about 6 times as much)[8].

It has been found that the fumes depend on[2]:

1. The oxygen balance
2. Marginal priming
3. Water attack
4. Critical diameter
5. Gaps in loading
6. Deflagrations.

3.6 Energy of Explosives

Explosives are substances that rapidly liberate their chemical energy as heat to form gaseous and solid decomposition products at high temperature and pressure. The hot and dense detonation products produce shock waves in the surrounding medium and upon expansion impart kinetic energy to the surrounding medium. The energy released in the detonation process is given by the following formula:

$$Q = \Delta H_f(\text{products}) - \Delta H_f(\text{reactants})$$

where ΔH_f is the heat of formation.

The energy per unit weight is called the weight strength of the explosive.

The energy per unit volume is called the bulk strength of the explosive.

Sometimes it is useful to express the weight and the bulk strengths as relative values obtained by dividing the strength (weight or bulk) to the corresponding strength of a standard explosive. The commercial industry normally uses AN/FO as the standard explosive.

3.7 Shelf Life

The shelf life of an explosive determines the maximum time period the explosive can be in storage. Various explosives age and their use is unsafe or they cannot be detonated reliably.

3.8 Pressure Desensitization

Commercial explosives can be susceptible to hydrostatic heads. Hydrostatic heads can compress the explosive to high densities and "dead packing" can result.

3.9 Measurement of the Detonation Properties

3.9.1 *Detonation Velocity*

There are various methods of measuring detonation velocities. These are outlined in the following:

i The continuous probe method.

The system consists of the explosive charge, along the central axis of which a uniform resistance probe is inserted, a constant current source, a triggering source and an oscilloscope.

The resistance probe consists of a resistance wire inserted into a small diameter brass tube. The resistance wire is a nichrome wire having an accurately known linear resistance.

The oscilloscope is connected in parallel to both the current source and the probe (Figure 9)[5]. At detonation the wire resistance probe is consumed. However the circuit remains closed due to the fact that the detonation wave is sufficiently ionized. The circuit follows Ohm's law. Therefore, since current is constant, the voltage change with time shown on the oscilloscope, is proportional to the resistance. Knowing the full voltage drop across the probe and the length of the probe, the voltage drop can be converted to distance along the charge. Therefore the velocity of detonation can be calculated by interpreting the voltage drop - time record provided by the oscilloscope.

ii. Start-stop method

Two probes are placed at a known distance apart in the explosive. Each probe consists of two wires placed in close proximity. When the detonation wave contacts each probe it shortens the circuit by bringing the two wires in contact. By measuring the signals obtained by either a counter or an oscilloscope one can measure the detonation velocity.

iii. Streak camera method

The method is shown in Figure 10[9]. The streak camera uses a mirror which rotates at the centre of the drum. The film is placed on the drum. The field of view of the camera lens is masked except for a narrow slit. The charge is aligned so that its axis is parallel to the slit of the camera. The light generated by the detonation front enters through the slit and after being reflected on the rotating mirror, leaves a mark on the film. Thus the streak camera trace is essentially a time distance record. The slope of the trace made by the luminous wave provides the velocity of detonation. A typical streak camera record is shown in Figure 11[10].

iv. D'Autriche Method

This is the least sophisticated method. It is outlined in Figure 12[9]. The method uses a detonating cord both ends of which are inserted in the explosive at a known distance apart. A metal witness plate is placed close to the middle of the detonating cord. The detonation wave in the charge initiates the detonating cord at both ends. When the detonation waves travelling in opposite directions in the detonating cord collide, they leave a dent in the witness plate. This helps to find the position in the detonating cord at which the collision took place. Thus, the distance, and therefore the time, each wave travelled in the detonating cord can be found. The difference in the times the two waves travelled in the cord provides the time it took the detonation wave in the test charge to travel the distance l.

3.9.2 *Detonation Pressure*

The measurement of the detonation pressure is normally based on photographic techniques. These techniques require a streak camera and accurate experiments (aquarium technique). In the aquarium technique, a transparent liquid serves as a pressure gauge for measuring transient pressures. The transparent liquid has to be selected in such a way that the reflected wave at the gauge-liquid interface is either a weak shock or a very weak rarefaction. The technique, as described by Cook[8] consists of the following two stages:

i. Initially the Hugoniot of the liquid which serves as a gauge is determined. The experimental set up is shown in Figure 13. The method consists of the simultaneous measurement of the shock velocity at the free surface and the free surface velocity as the shock emerges from the transparent medium. Observations of the shock velocity and the free surface velocity are made by using a streak camera. By changing the height (h) of the liquid inside the container, one changes the shock velocity and the free surface velocity. By assuming that the particle velocity of the liquid at the interface is half of the free surface velocity the relationship between shock velocity and the particle velocity in the liquid (Hugoniot) is obtained.

ii. The experimental set up for the second part of the technique is shown in Figure 14. In this experiment, the velocity of detonation in the explosive charge and the initial transmitted shock velocity in the liquid are measured. From the transmitted shock velocity in the liquid and the known Hugoniot of the liquid, the initial pressure in the liquid can be calculated. The corresponding pressure in the detonation head is calculated by using the following relationship:

$$P_d = P_{il}[(\rho U_s)_{il} + \rho_{1e}U_{se}]/(2(\rho U_s)_{il})$$

where

P_d is the detonation velocity

ρ_{ie} is the initial density of the explosive

U_{se} is the detonation velocity

$(\rho U_s)_{il}$ is the initial impedance of the liquid and

P_{il} is the initial pressure in the liquid.

The initial pressure in the liquid is calculated by the well known relationship

$$P_{il} = \rho_1 U_{sl} U_{pl}$$

where P_{il} is the pressure in the liquid

U_{sl} is the shock velocity

U_{pl} is the particle velocity and

ρ_1 is the initial density of the liquid.

Because of the difficulty in measuring detonation pressures it is often necessary to calculate the detonation pressure from the detonation velocity by using the approximate formula:

$$P = \frac{\rho D^2}{4}$$

where P is the detonation pressure

ρ is the initial density of the explosive and

D is the measured detonation velocity.

3.10 References

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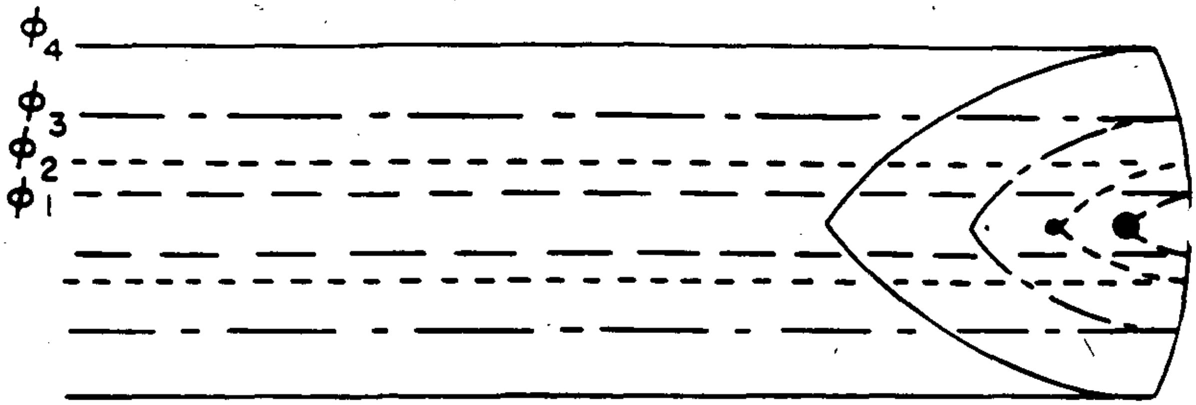


DETONATION HEAD IN UNCONFINED CHARGES OF INCREASING DIAMETER AND THE REACTION OF A SOLID PARTICLE OF EXPLOSIVE

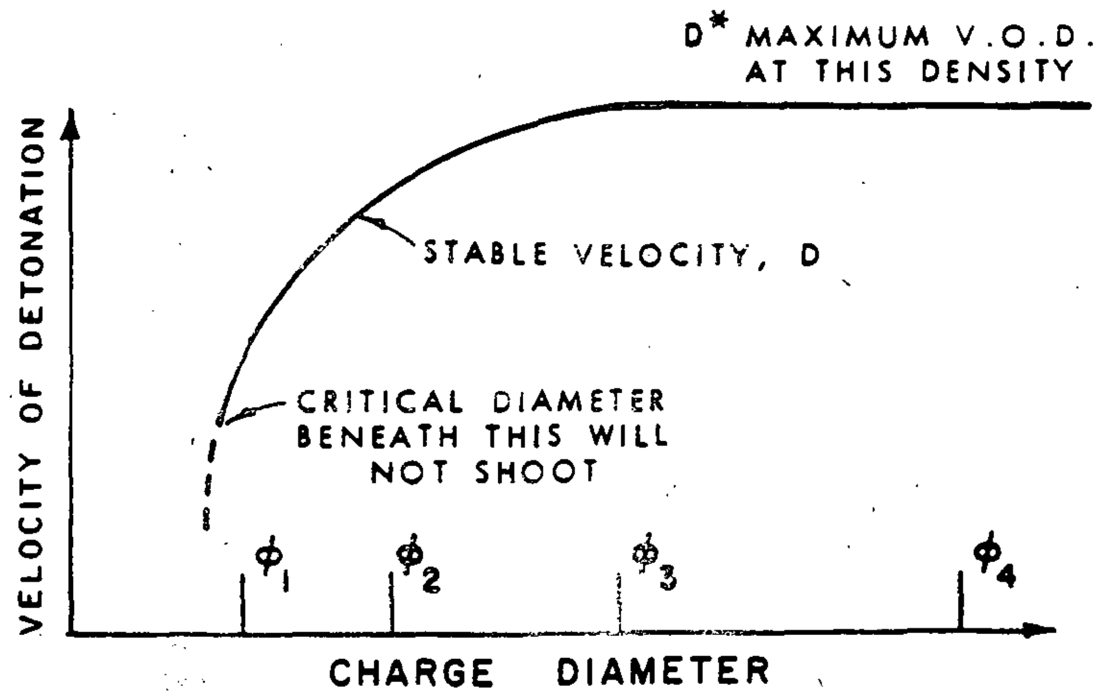


FIGURE 1: TYPICAL VELOCITY OF DETONATION CHARGE DIAMETER CURVE FOR A GRANULAR EXPLOSIVE (AFTER BAUER)

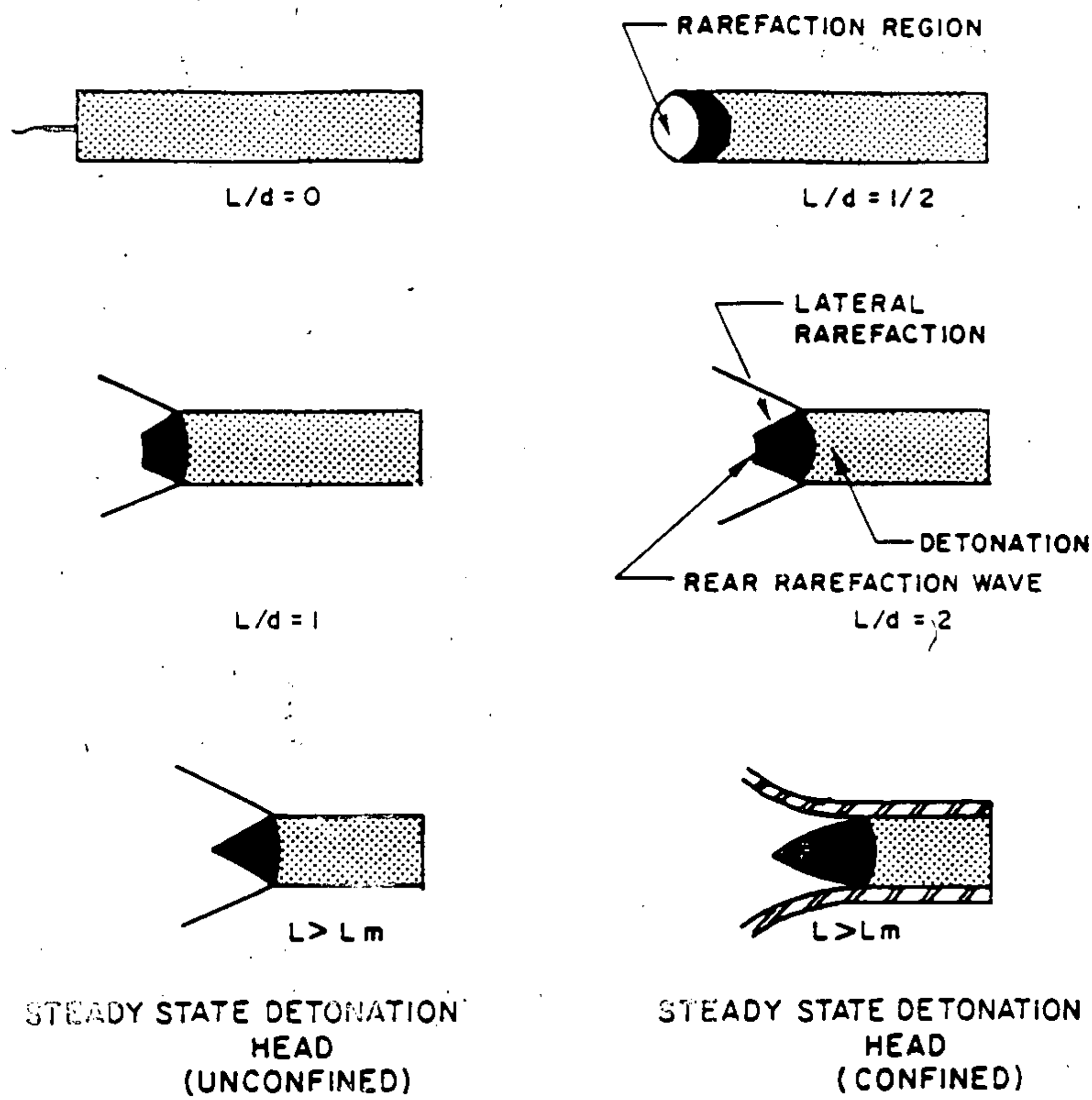
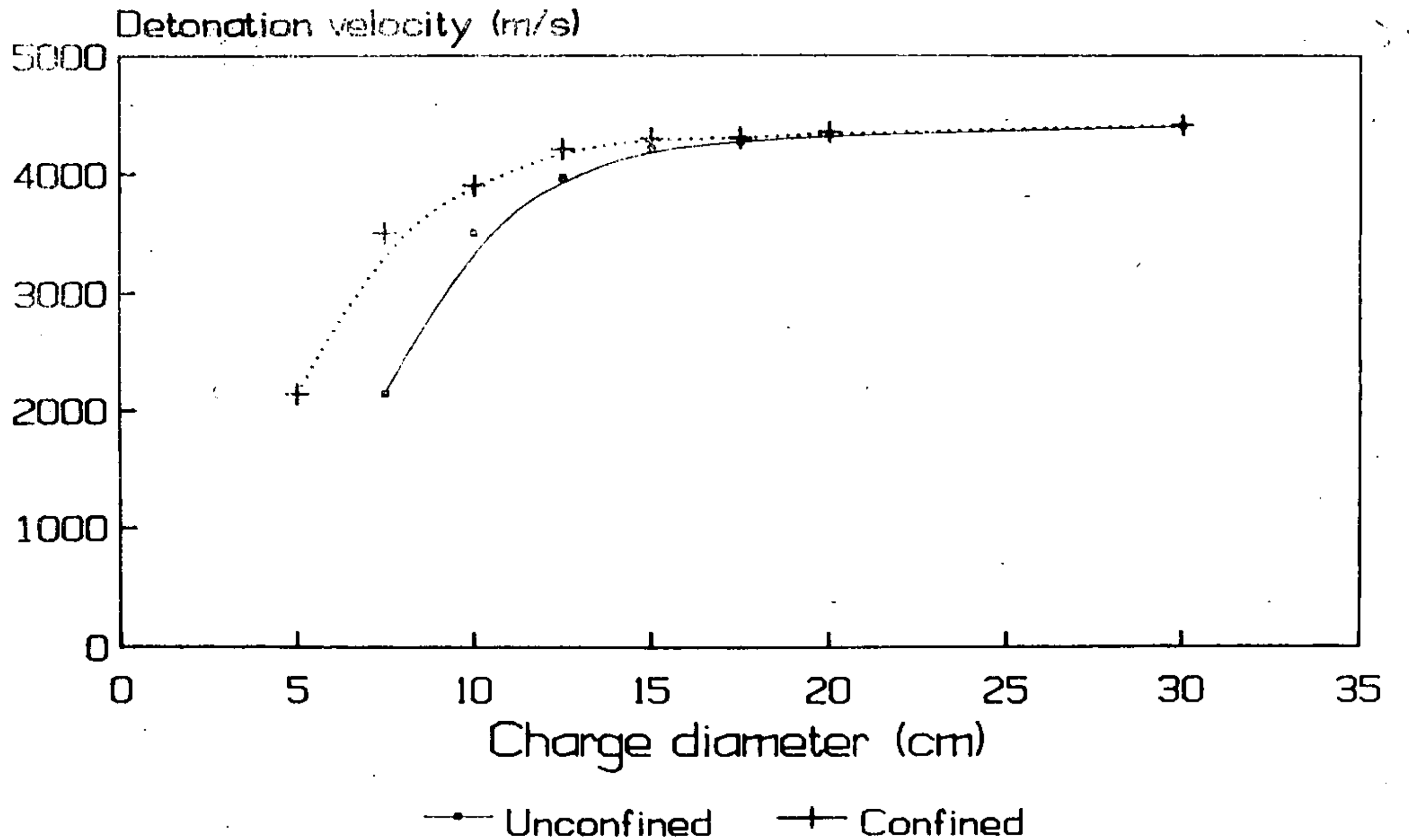


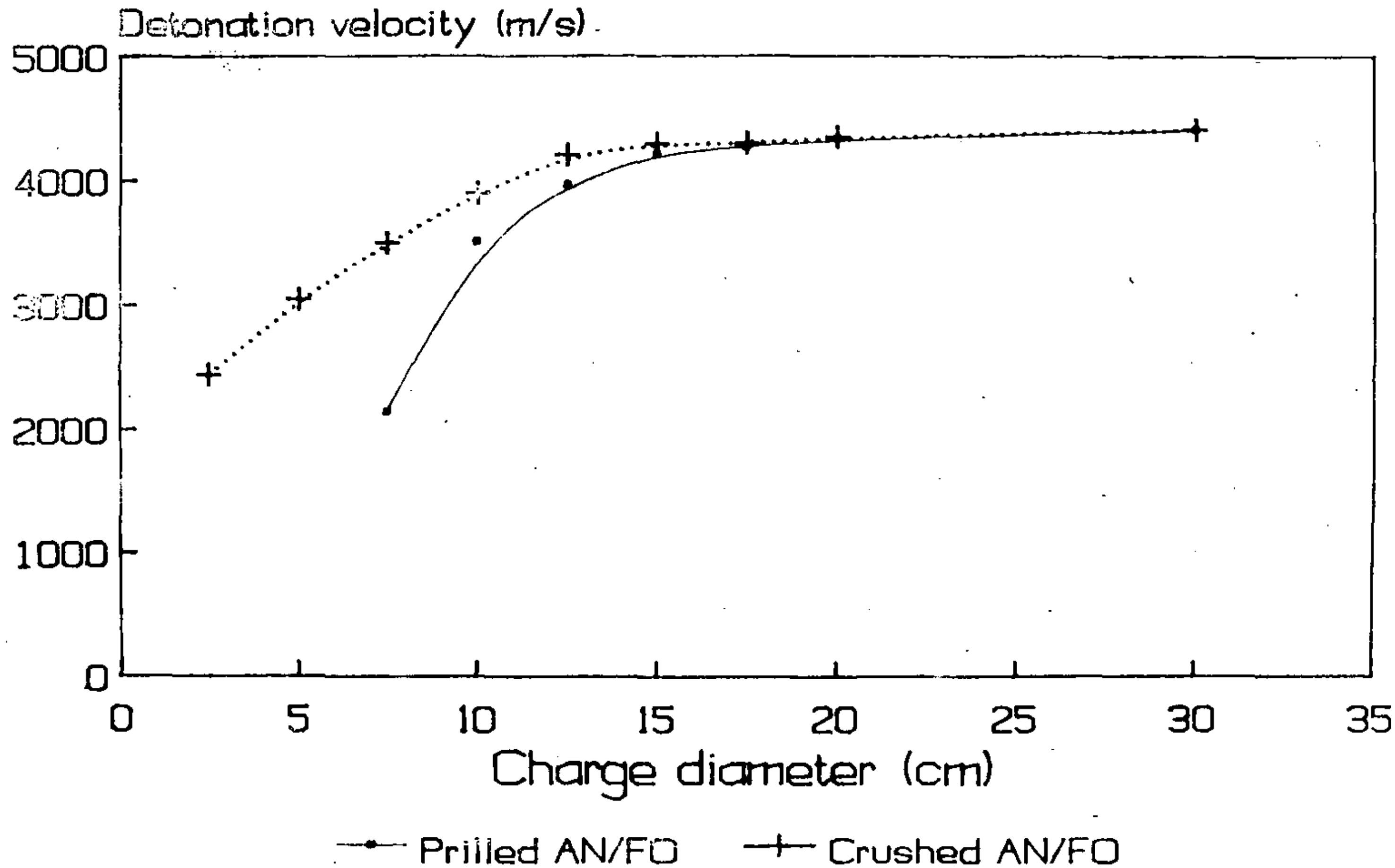
FIGURE 2: DEVELOPMENT OF THE DETONATION HEAD (AFTER COOK, 1958)

FIGURE 3: VOD - CHARGE DIAMETER CURVES
FOR CONFINED AND UNCONFINED ANFO



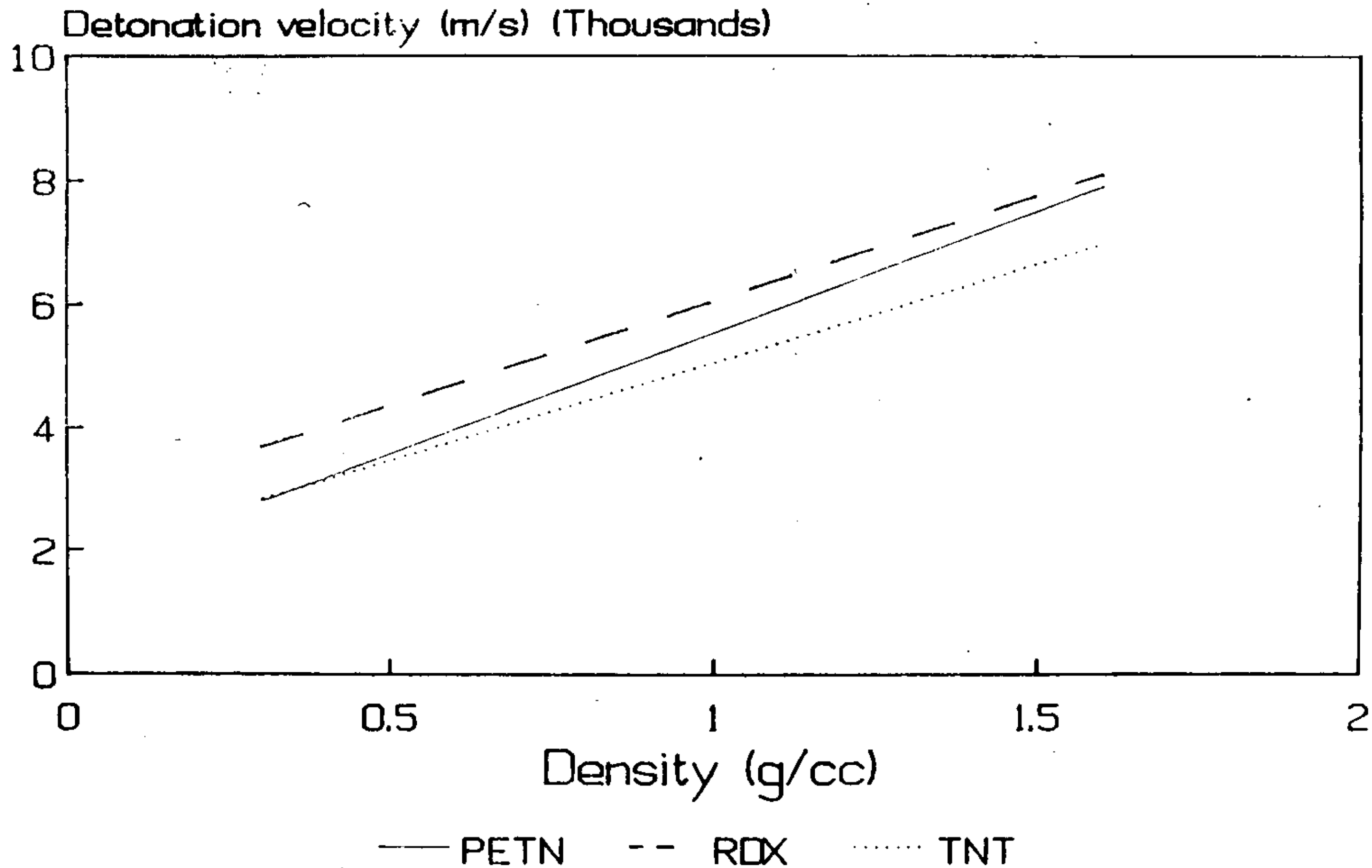
density = 0.85g/cc

FIGURE 4: EFFECT OF THE PARTICLE SIZE ON THE VELOCITY - DIAMETER CURVE OF AN/FO



density = 0.85g/cc

FIGURE 5: DETONATION VELOCITY - DENSITY RELATIONSHIPS



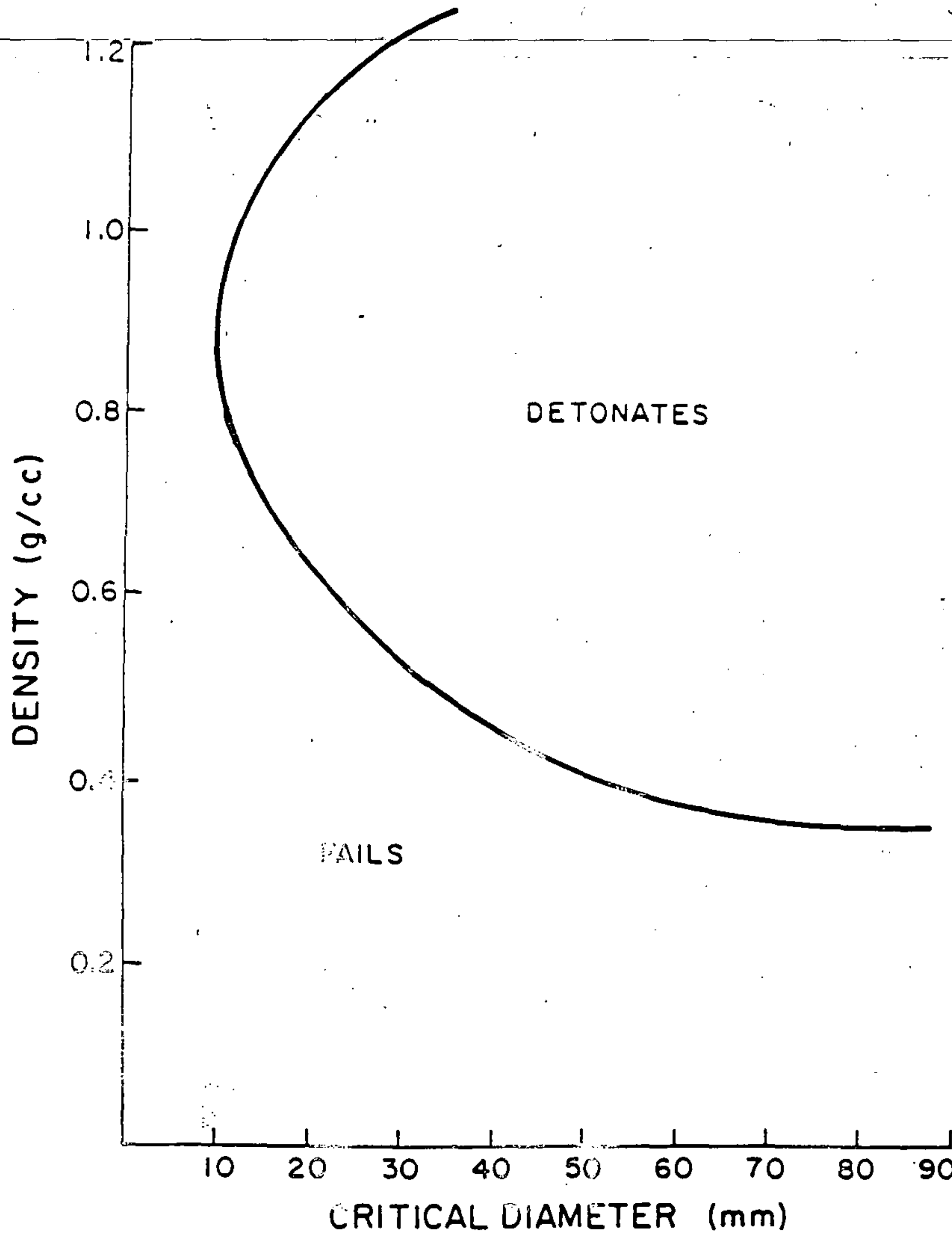


FIGURE 6: EFFECT OF THE DENSITY OF A TYPICAL EMULSION ON THE UNCONFINED CRITICAL DIAMETER

FIGURE 7: EFFECT OF TEMPERATURE ON THE CRITICAL DIAMETER OF TNT

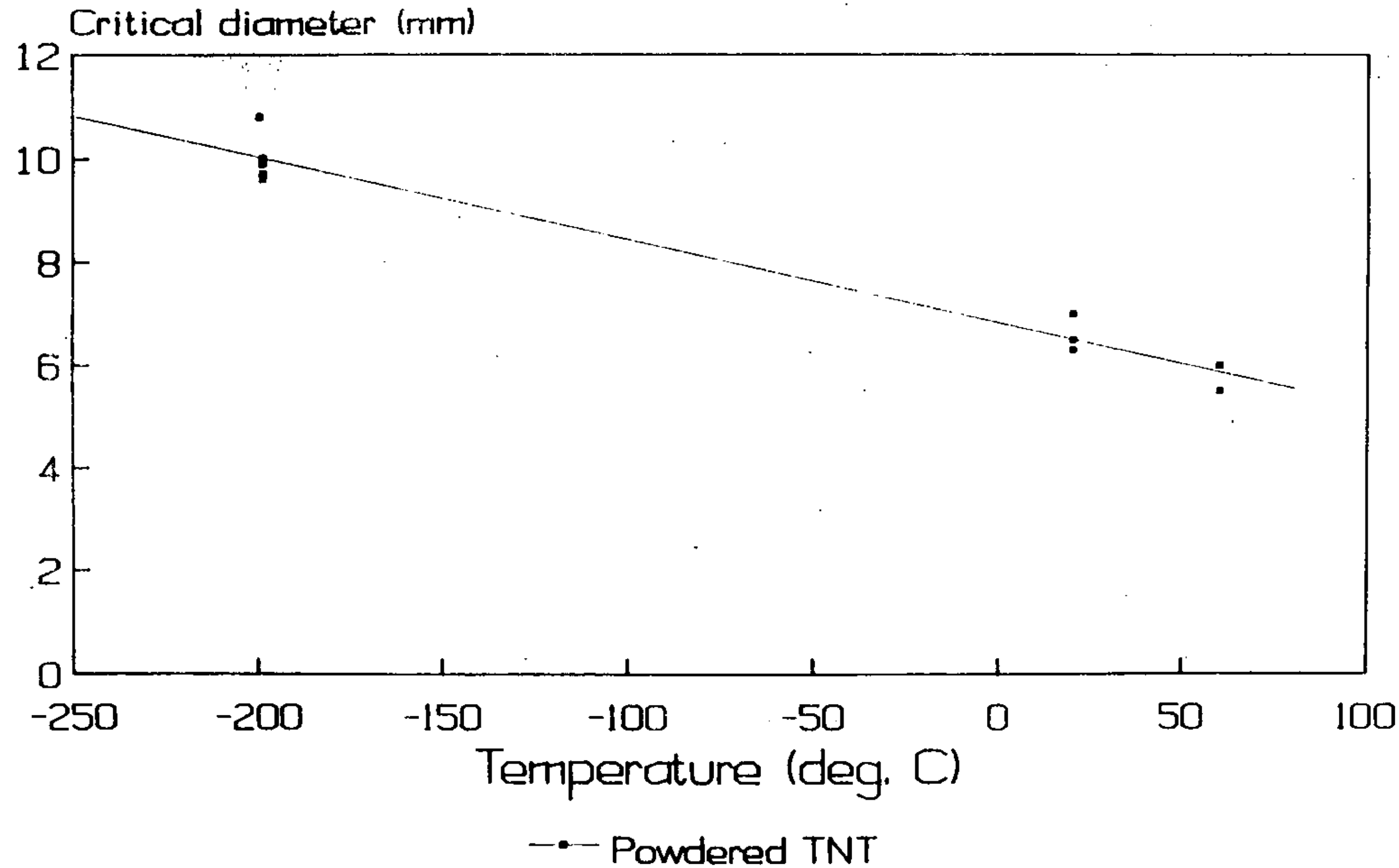
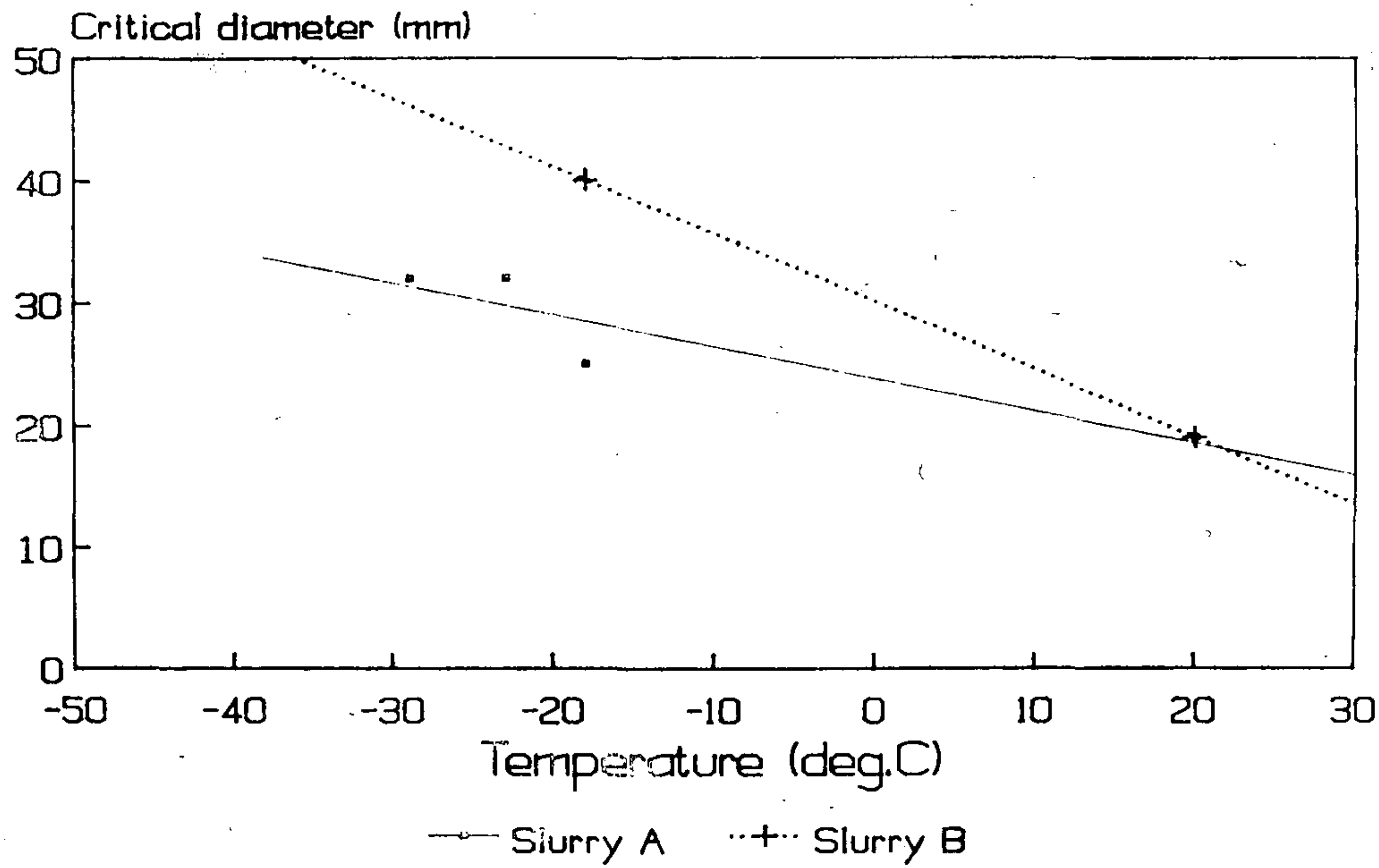


FIGURE 8: EFFECT OF TEMPERATURE ON THE CRITICAL DIAMETER OF SLURRIES



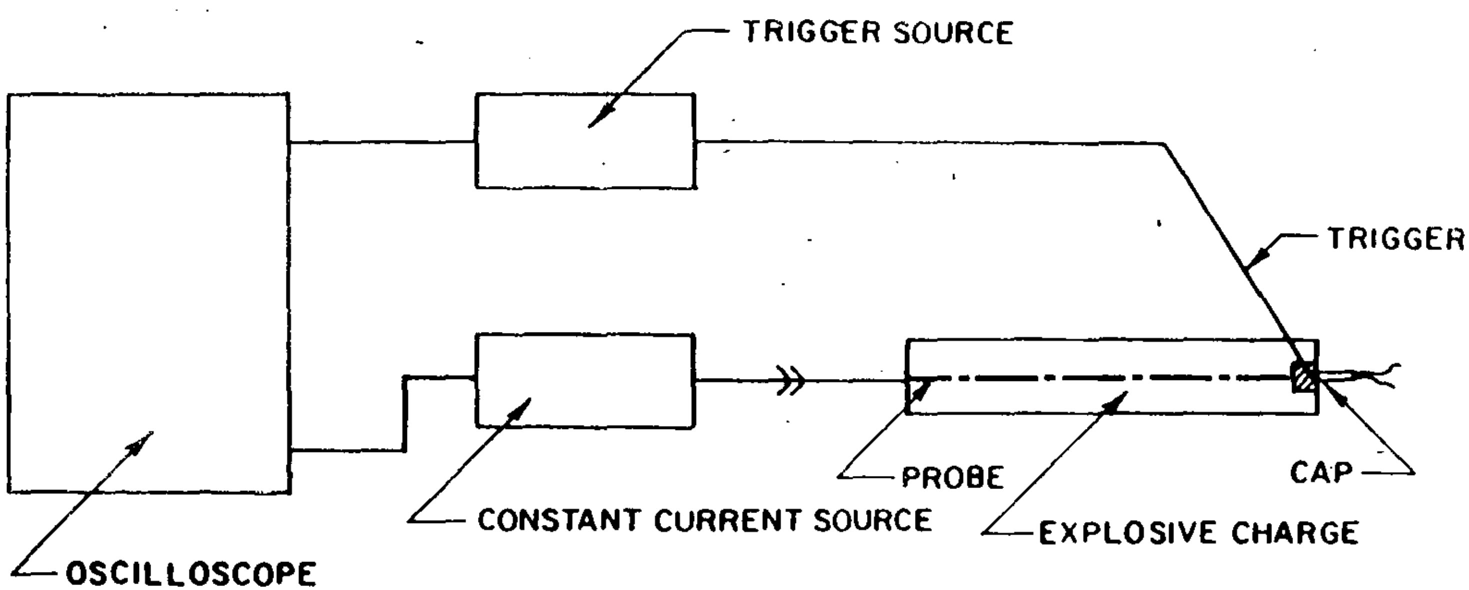


FIGURE 9: SCHEMATIC REPRESENTATION OF THE CONTINUOUS VELOCITY SYSTEM FOR THE MEASUREMENT OF THE VELOCITY OF DETONATION

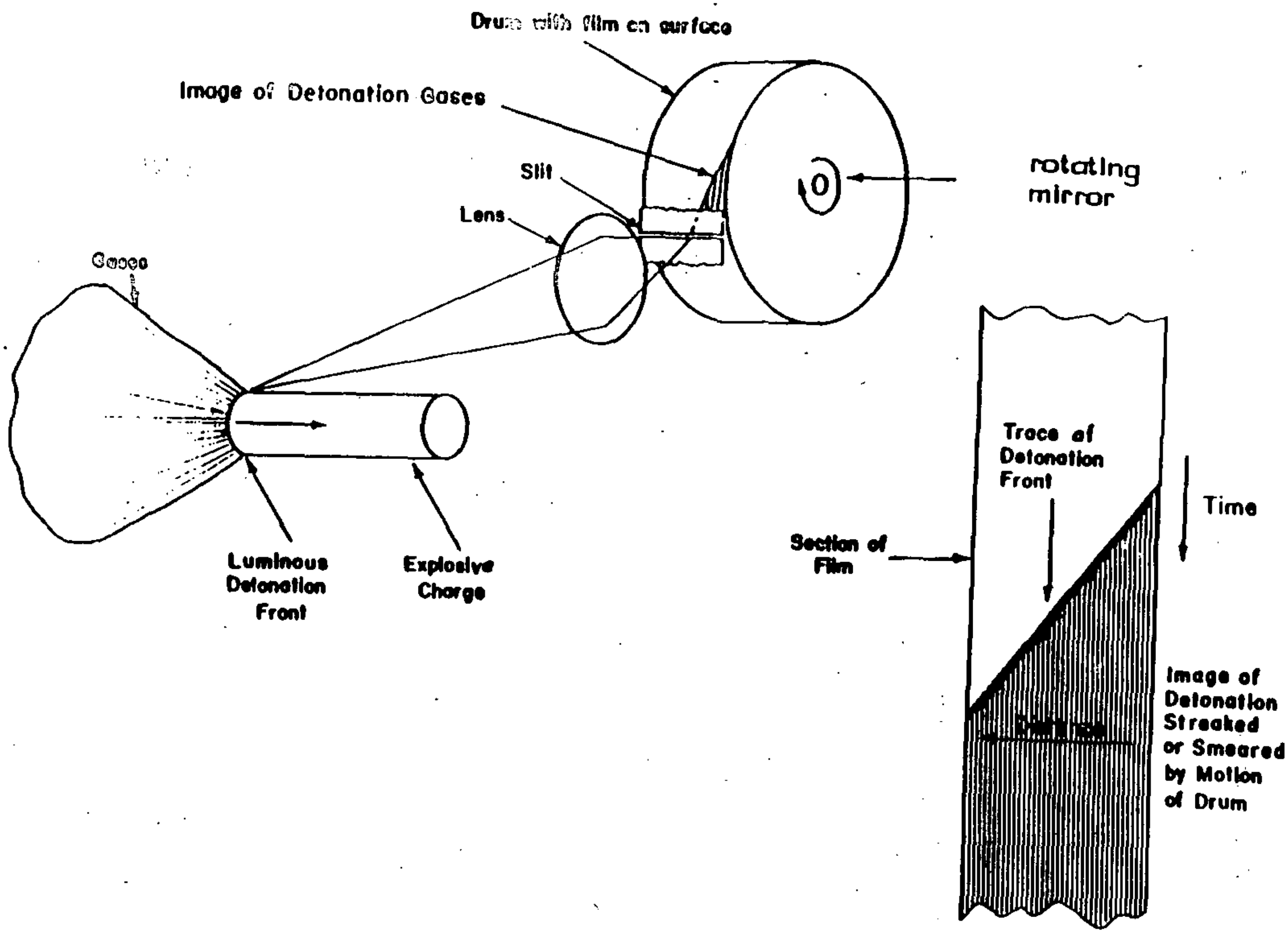


FIGURE 10: MEASUREMENT OF THE VELOCITY OF DETONATION
BY USING A STREAK CAMERA (ref. 9)

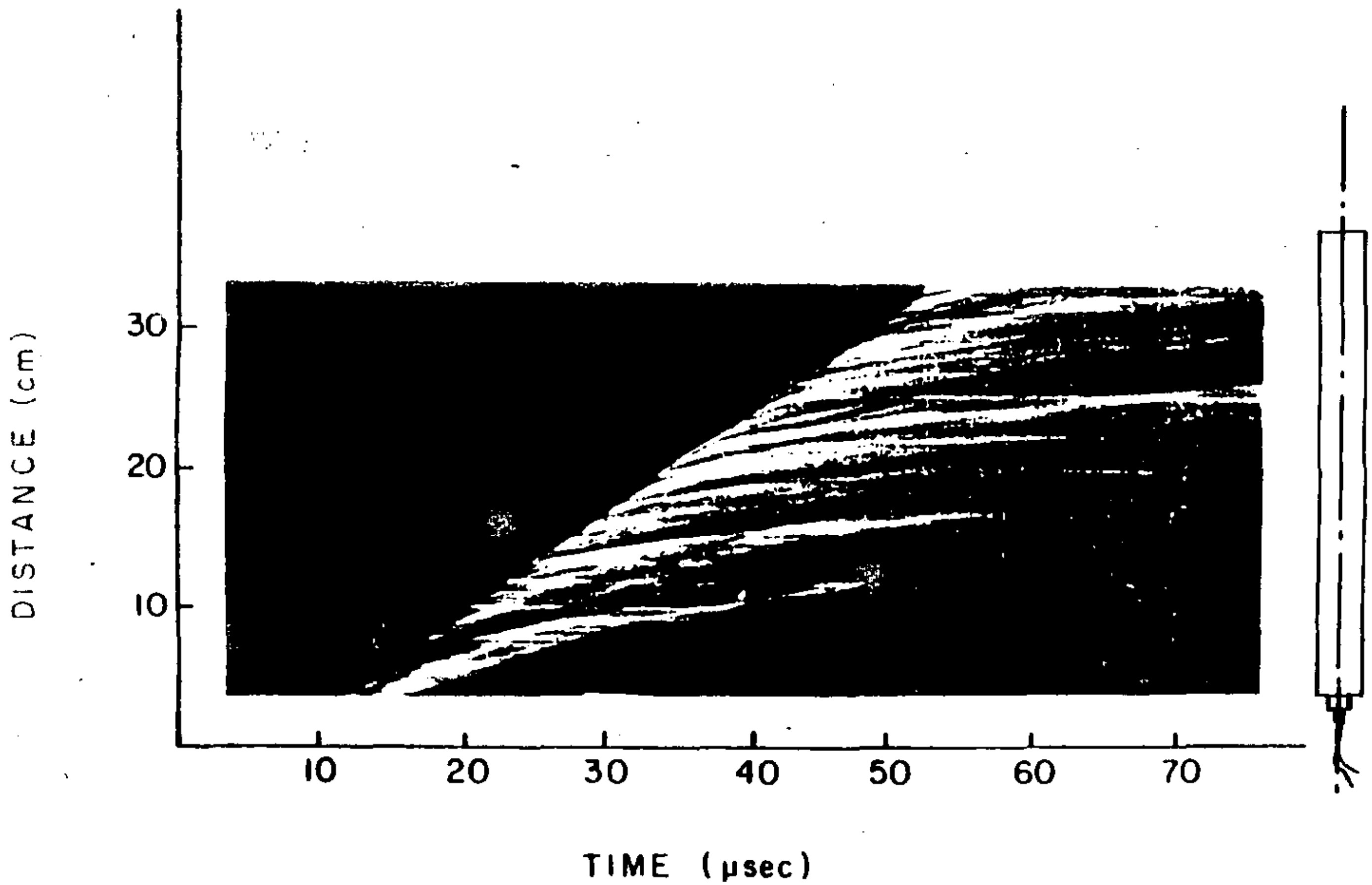


FIGURE 11: TYPICAL STREAK CAMERA RECORD FOR THE MEASUREMENT OF THE VELOCITY OF DETONATION OF PENTOLITE

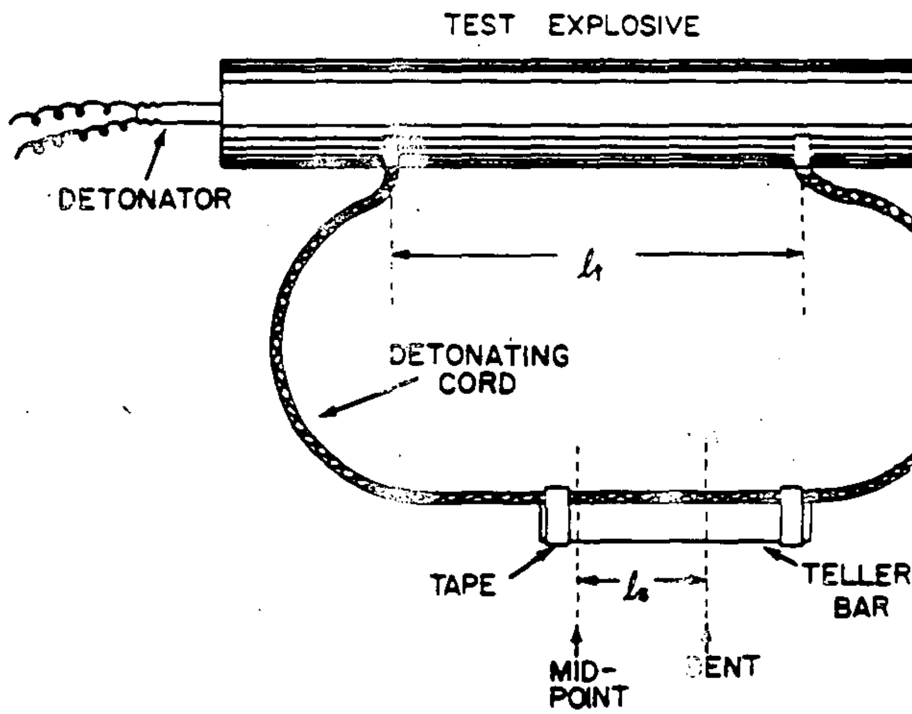


FIGURE 12: D'AUTRICHE METHOD FOR THE MEASUREMENT OF THE VELOCITY OF DETONATION

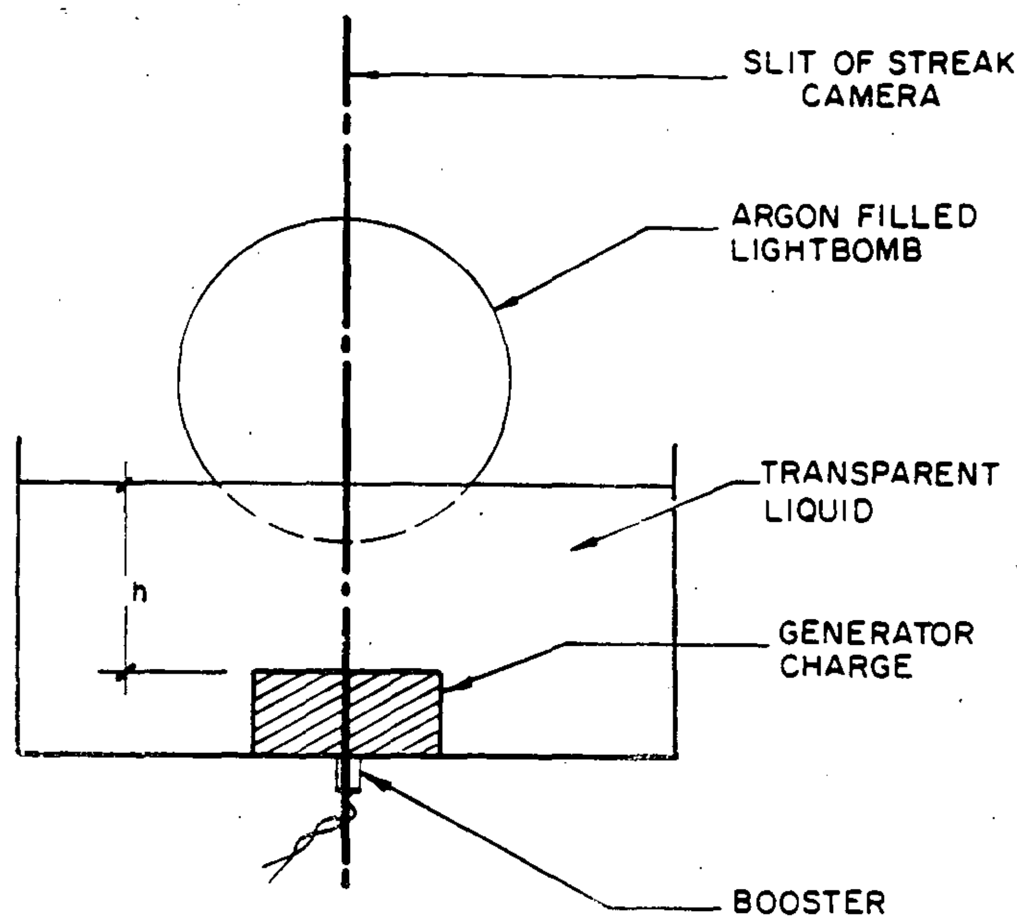


FIGURE 13: EXPERIMENTAL SET UP FOR DETERMINING THE HUGONIOT OF THE TRANSPARENT LIQUID

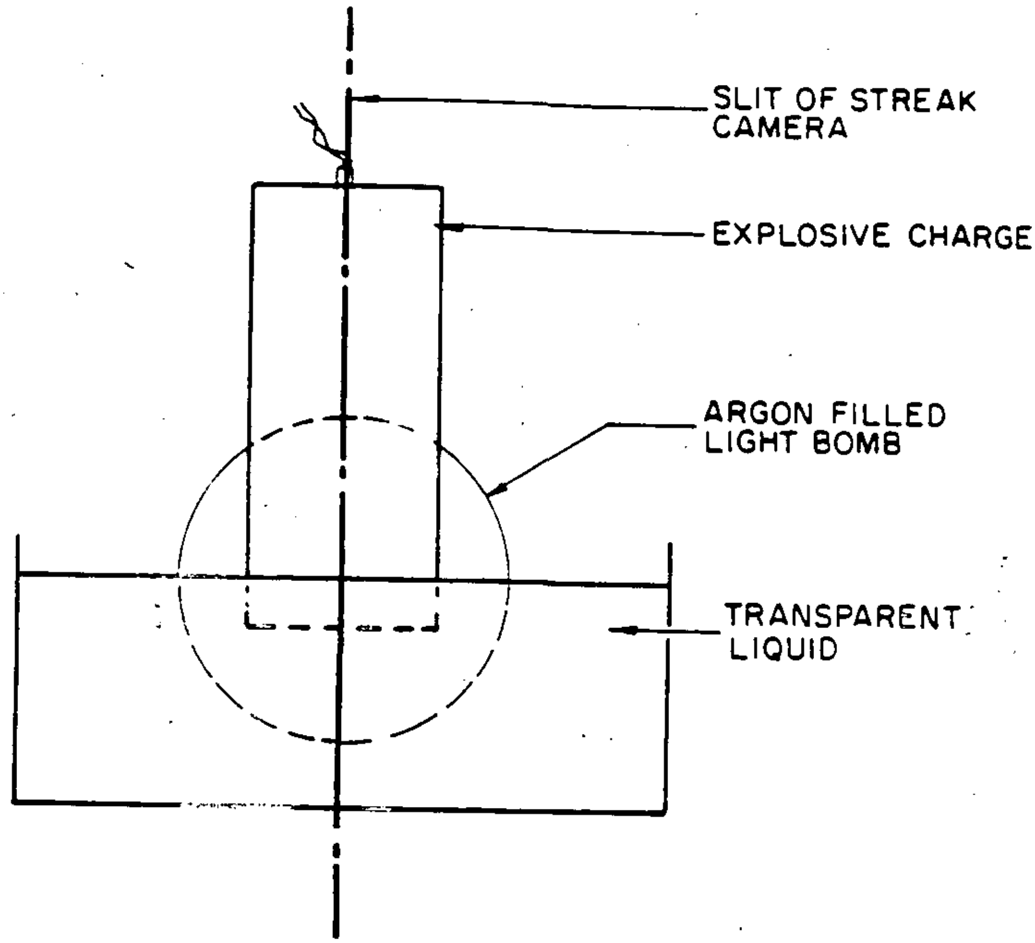


FIGURE 14: EXPERIMENTAL SET UP FOR THE MEASUREMENT OF THE DETONATION VELOCITY AND THE INITIAL SHOCK VELOCITY IN THE TRANSPARENT LIQUID

RCB

CHAPTER 4

GAP AND FRICTION SENSITIVITY OF EXPLOSIVES

4.1 Introduction

The gap sensitivity of explosive represents its ability to propagate through barriers. The gap sensitivity of an explosive is an important property to be considered in blasting operations. If the sensitivity is low, the detonation in the borehole can be interrupted because of obstacles (rocks) or air gaps. On the contrary, an explosive which is very sensitive can be dangerous to handle and can detonate sympathetically in the boreholes. Cross propagation of adjacent holes is very undesirable since this eliminates the effects of delays and results in excessive vibrations and poor fragmentation.

However one has to differentiate between solid gap and air gap sensitivity because the phenomena involved in each case are considerably different.

The friction sensitivity determines the safe handling of explosive charges. Charges can be subjected to friction forces when loaded in blastholes. These can be of a significant magnitude especially where pneumatic loaders are used.

4.2 Underdriven and Overdriven Detonations

The detonation state (C-J state) represents a dynamic stable condition. If the detonation wave encounters a small gap in the explosive charge, it will weaken temporarily and will come back to the original stable condition once the perturbation is passed.

The same will happen if the detonation wave encounters a part of the explosive which has greater energy. Temporarily it will strengthen but later it will reach the stable condition.

Consider the situation shown in Figure 1 a. A detonation is transmitted from a donor explosive to an acceptor explosive. In this case there are three possibilities; the shock wave transmitted in the acceptor can be stronger than the detonation wave in the acceptor, the shock wave can be of equal magnitude to the detonation wave in the acceptor or the shock wave can be of a smaller magnitude than the detonation wave in the acceptor. The first case is called overdriven and the last case underdriven detonation. It has been found that in the case of an overdriven wave the strength always decays until the C-J condition is reached. In the case of the underdriven wave the detonation builds up to the C-J value. However, there is a limiting strength below which the wave decays and detonation does not propagate. This limiting strength is of importance since it determines the conditions required for safe handling and reliable initiation of explosive materials.

4.3 The Gap Test

Experimentally a simple way to determine the sensitivity of an explosive to initiation is represented in the gap test. The gap test is shown in Figure 1 b. The experiment consists of a donor charge, an attenuator and an acceptor charge. By varying the attenuator thickness, different underdriven waves are transmitted to the acceptor. The thickness of the attenuator at which 50% of the times the acceptor detonates is called critical gap thickness. At that thickness the shock wave in the acceptor has a limiting value above which the acceptor has a high probability of detonation. The gap material is normally a standard solid material. Air gaps are not desirable because hot decomposition products of the donor explosive will impinge directly on the acceptor.

The result of the gap test depends on the geometry of the donor and acceptor charges as well as the attenuator material and the donor explosive. For this purpose various laboratories standardize gap tests by using the same donor and the same attenuator material. Thus the results of the tests are indicative of the explosives shock sensitivity.

Typical gap tests are shown in Figures 2 and 3.

The following factors affect the result of a standard gap test:

1. Density. The effect of density is shown in Figure 4[2] where the critical gap pressure is plotted against the percent of the theoretical maximum density. It is obvious that the explosive becomes less sensitive as the theoretical maximum density is approached. This is a general trend obtained in a variety of explosive compositions[2].

2. Temperature. The effect of temperature is shown in Figure 5. This is a general trend for any material in which the reaction rate increases with temperature[2].

3. Composition. It is obvious that the result of the gap test is composition dependant. It has been found that if wax is added to RDX or TNT, the shock sensitivity is decreased. However if wax is added to ammonium nitrate, the sensitivity is drastically increased. This happens because of the combination of an oxidizer

with a fuel and the dominant factor is the oxidation-reduction reaction. Figure 6 is typical of this phenomenon[2].

4. Acceptor diameter. Initiation is controlled not only from the magnitude of the impacting shock wave but from its duration as well. The reduction of the diameter of the acceptor has changed the duration of the shock wave. It is recommended that the charges are tested at a diameter above the minimum diameter for ideal detonation, where this is possible. According to Price the critical initiating pressure - diameter relationship should follow a curve as in Figure 7[5]. Experimental results by Moulard indicate the same trend for Composition B[6].

5. Confinement. Price has found that confinement of the acceptor in the test prevents the lateral rarefaction from producing a large disturbance. The confinement gives a result which is comparable to that which would be obtained for a very much larger diameter unconfined charge. The result may approach that which would be obtained in the one dimensional flow[2]. In Figure 8 the critical gap pressures for confined charges are compared to the critical cap pressures of unconfined charges. It is obvious that confinement increases the sensitivity of explosives.

4.4 Air Gap Sensitivity

This term denotes the initiation of an explosive charge without a priming device by the detonation of another charge in the neighbourhood. The transmission mechanism is complex. The important parameters are the shock wave, the hot reaction products of the donor and the flying parts from the casing of the donor charge. Various tests are conducted to determine the air gap sensitivity of explosives. In Europe the smallest diameter of manufacture is used in the test charges which are tested unconfined[3]. This will provide the largest gap below which detonation will always be observed. Confinement however affects the result. For this purpose coal mining explosives are tested in pipes which simulate boreholes. It is recommended that gap tests simulating the conditions of application are performed to determine the gap sensitivity of a particular product.

4.5 Initiation by Friction

The mechanism of heating by friction has been investigated by Bowden and co-workers. When solid bodies are pressed against each other contact will occur only at the summits of the surface irregularities. The total area of contact is a small fraction of the total surface area[4]. When the bodies are sliding against each other heat is developed at the regions of contact. Hot spots are created at the points of contact and their temperature depends on the pressure, sliding velocity and heat conductivity of the sliding material. The contact material with the lowest melting point determines the hot spot temperature. When melting occurs its supporting capacity is taken over by other points[4]. According to Bowden if the melting point of the slider is below the critical hot spot temperature for the explosive, detonation does not occur.

Several friction tests have been developed. The Swedish[4] developed a friction test in which the explosive is subjected to stresses similar to those when the explosive is charged in boreholes. The test consists of a block of granite which has a semi-cylindrical groove. A thin layer of explosive is placed in the groove and a slider moves on top. Various loads are put on the slider. The slider moves at a constant speed and the result is recorded as a function of the load.

In Germany a sample is placed on a roughened porcelain plate[3]. The sample is put on top of it and a porcelain cylinder is placed on top with various loads. The plate moves at a certain speed and the result is recorded as a function of the load.

Similar tests have been developed in other countries.

4.6 References

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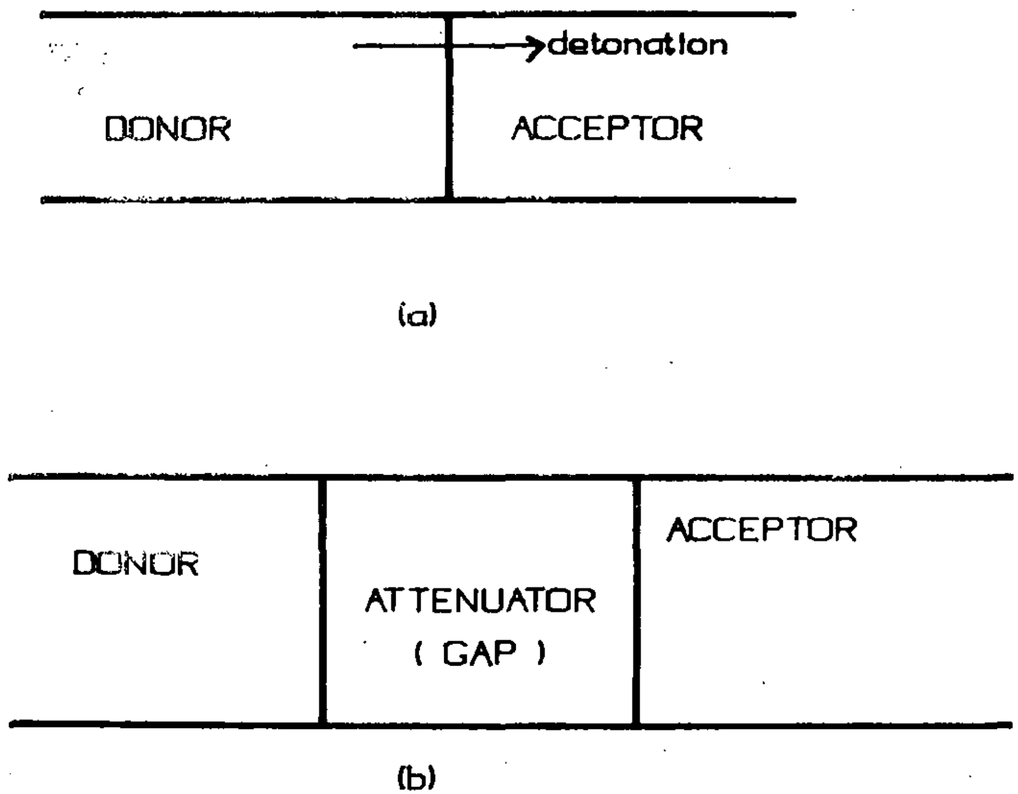


FIGURE 1: TYPICAL GAP TEST CONFIGURATION

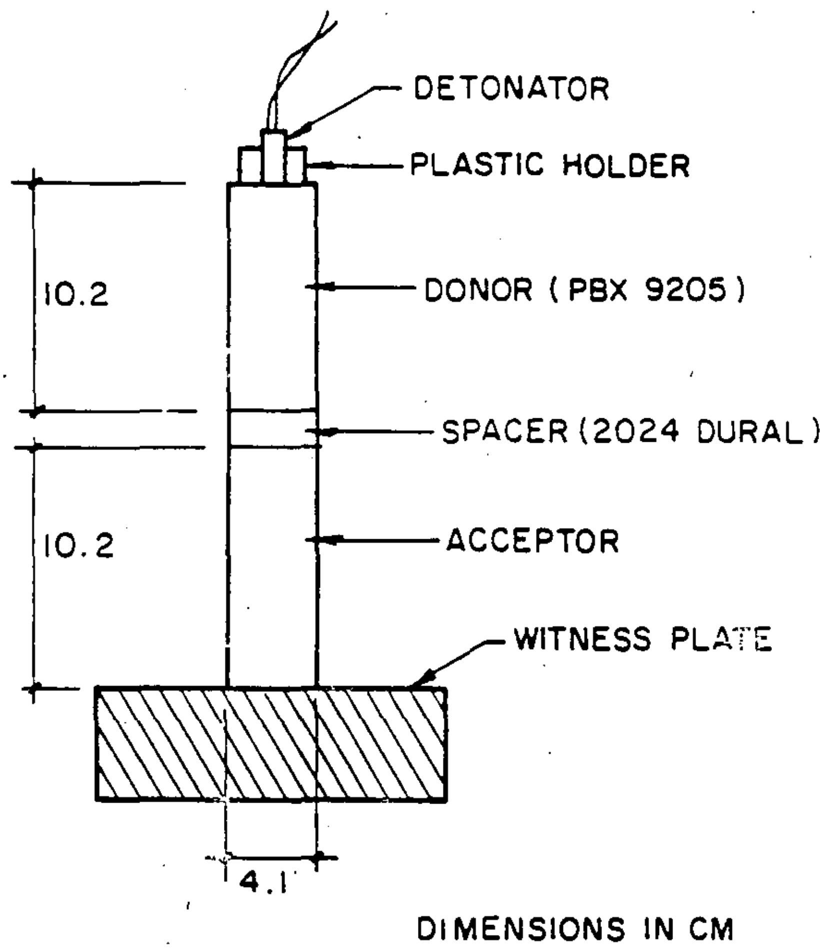


FIGURE 2: THE LOS ALAMOS LARGE GAP TEST

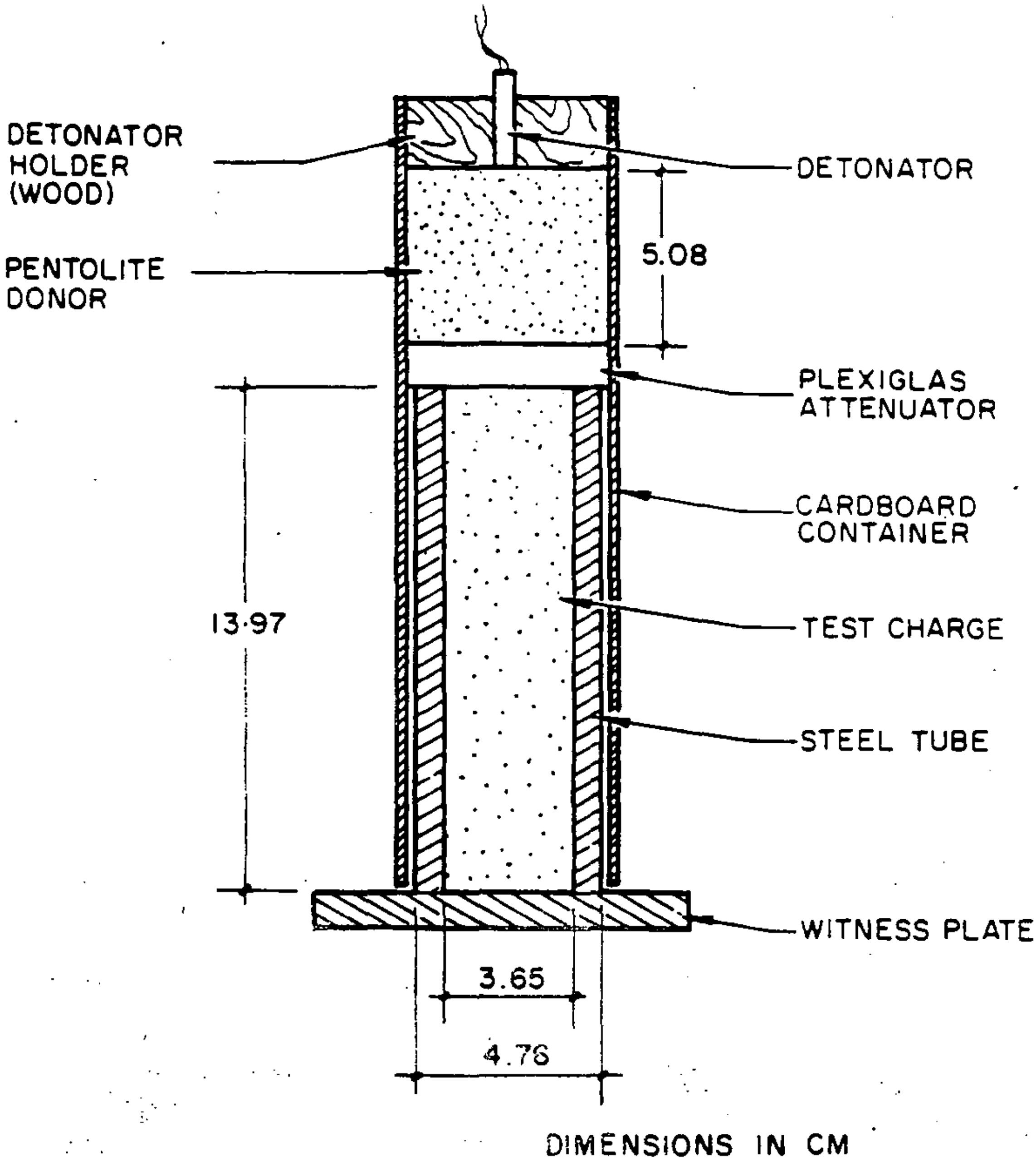


FIGURE 3: THE NOL GAP TEST

FIGURE 4: EFFECT OF DENSITY ON CRITICAL GAP PRESSURE

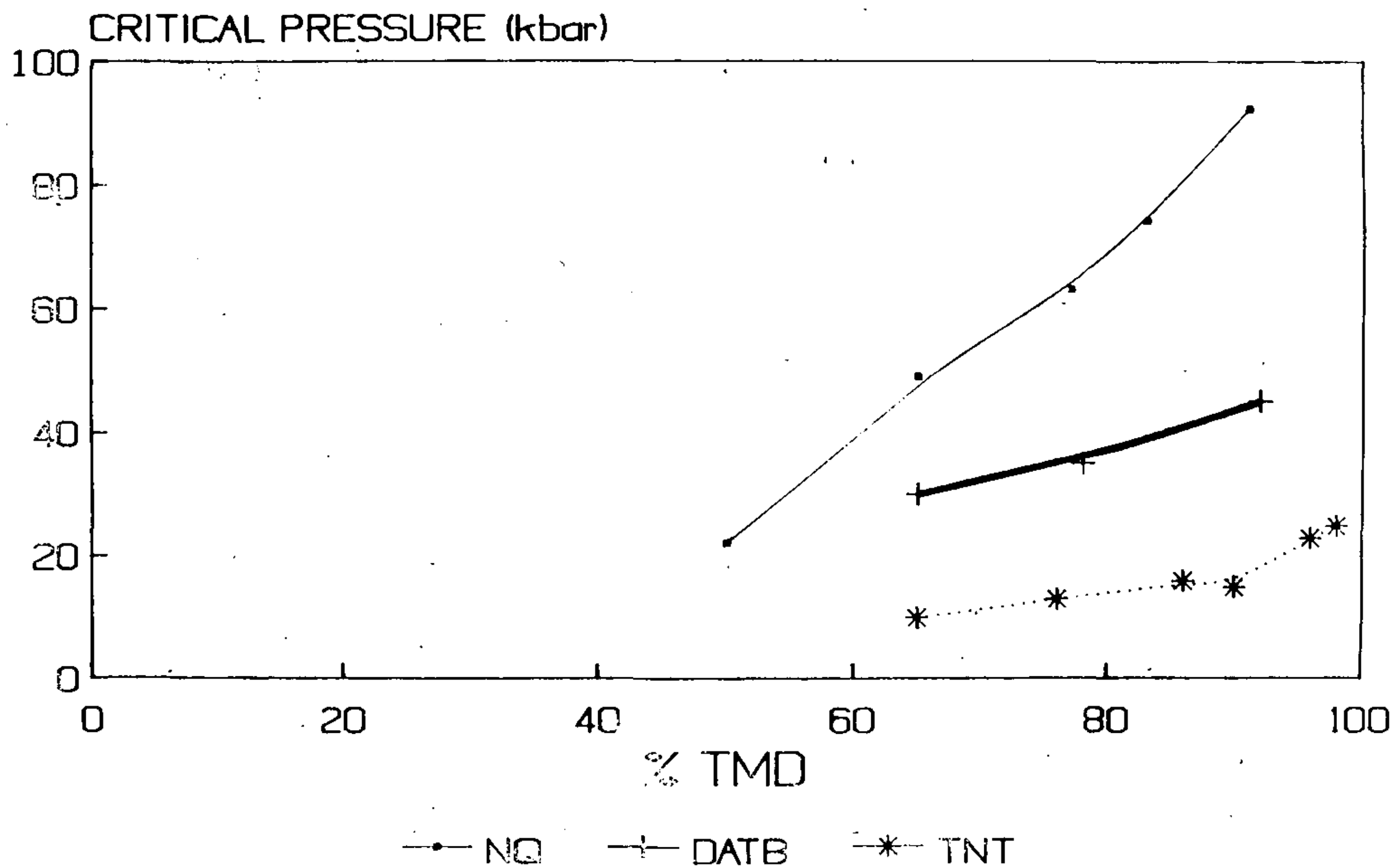


FIGURE 5: EFFECT OF THE COMPOSITION ON CRITICAL GAP PRESSURE

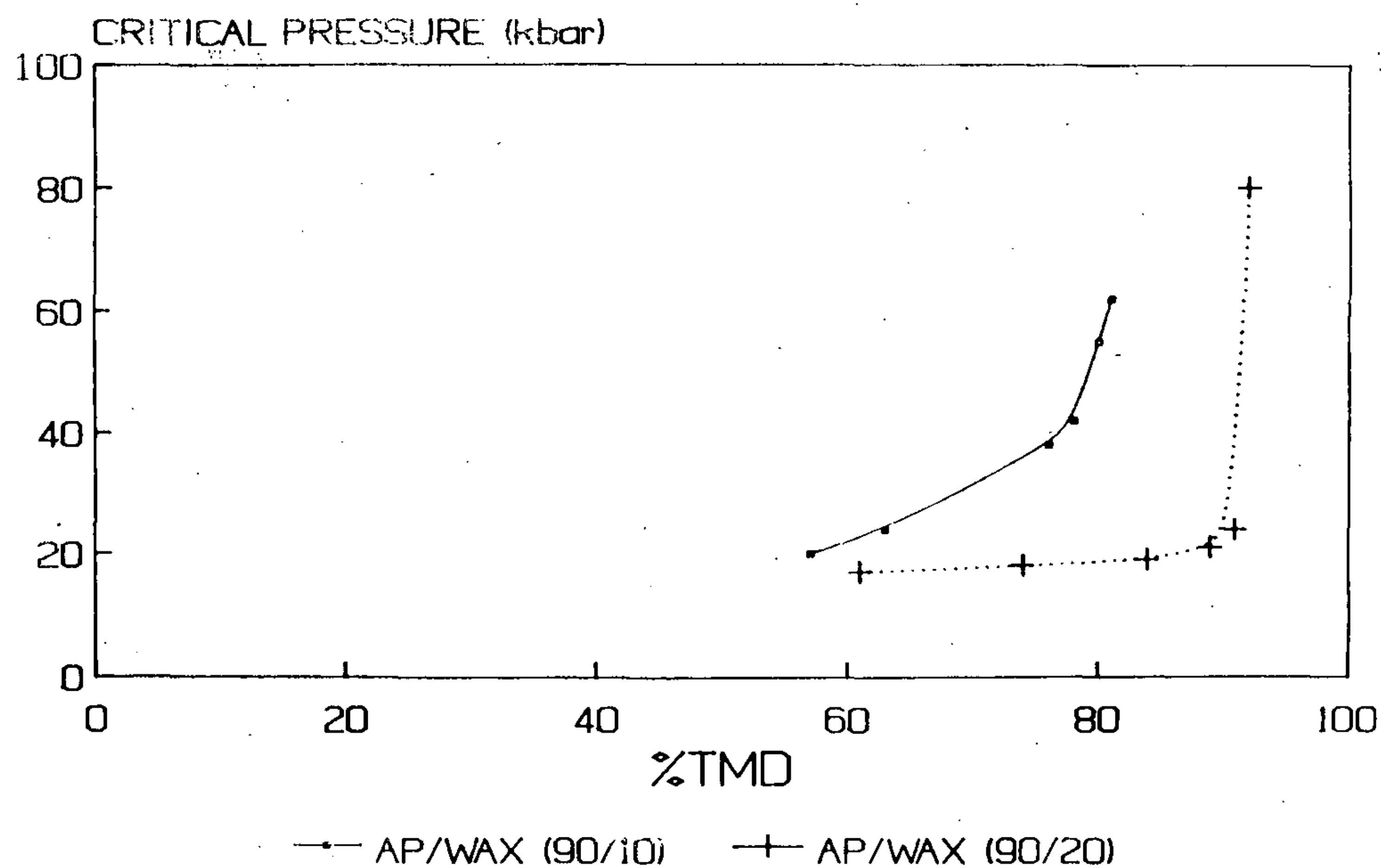
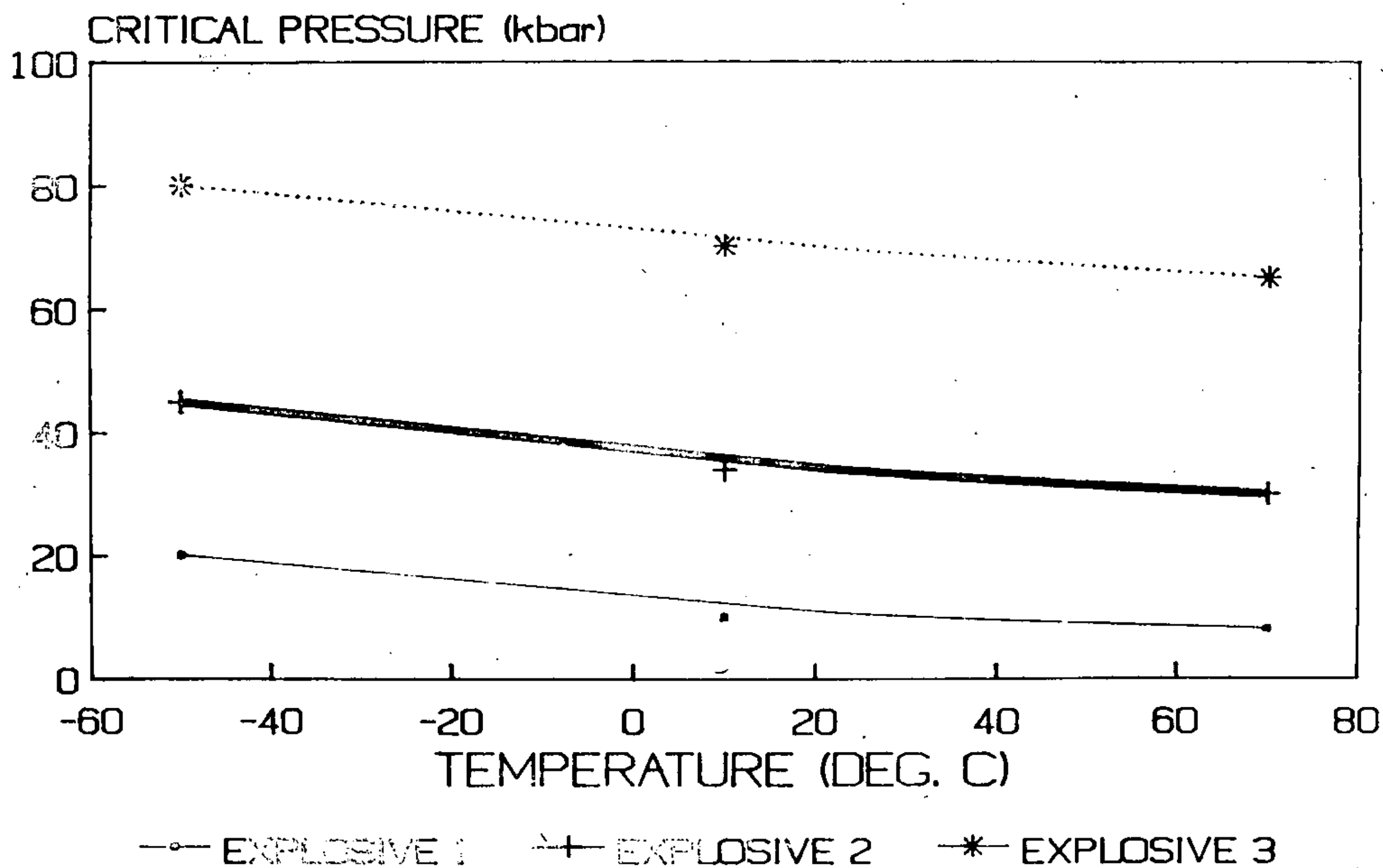


FIGURE 6: EFFECT OF TEMPERATURE ON THE CRITICAL GAP PRESSURE



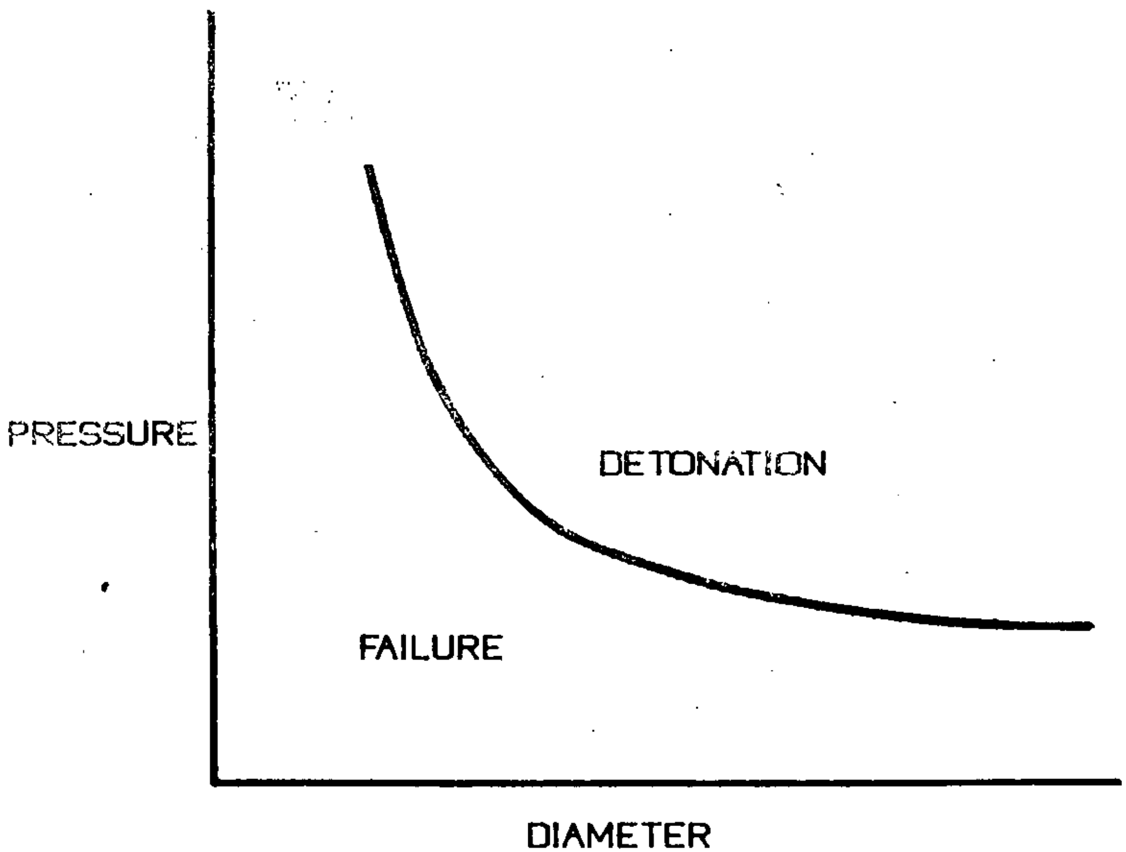
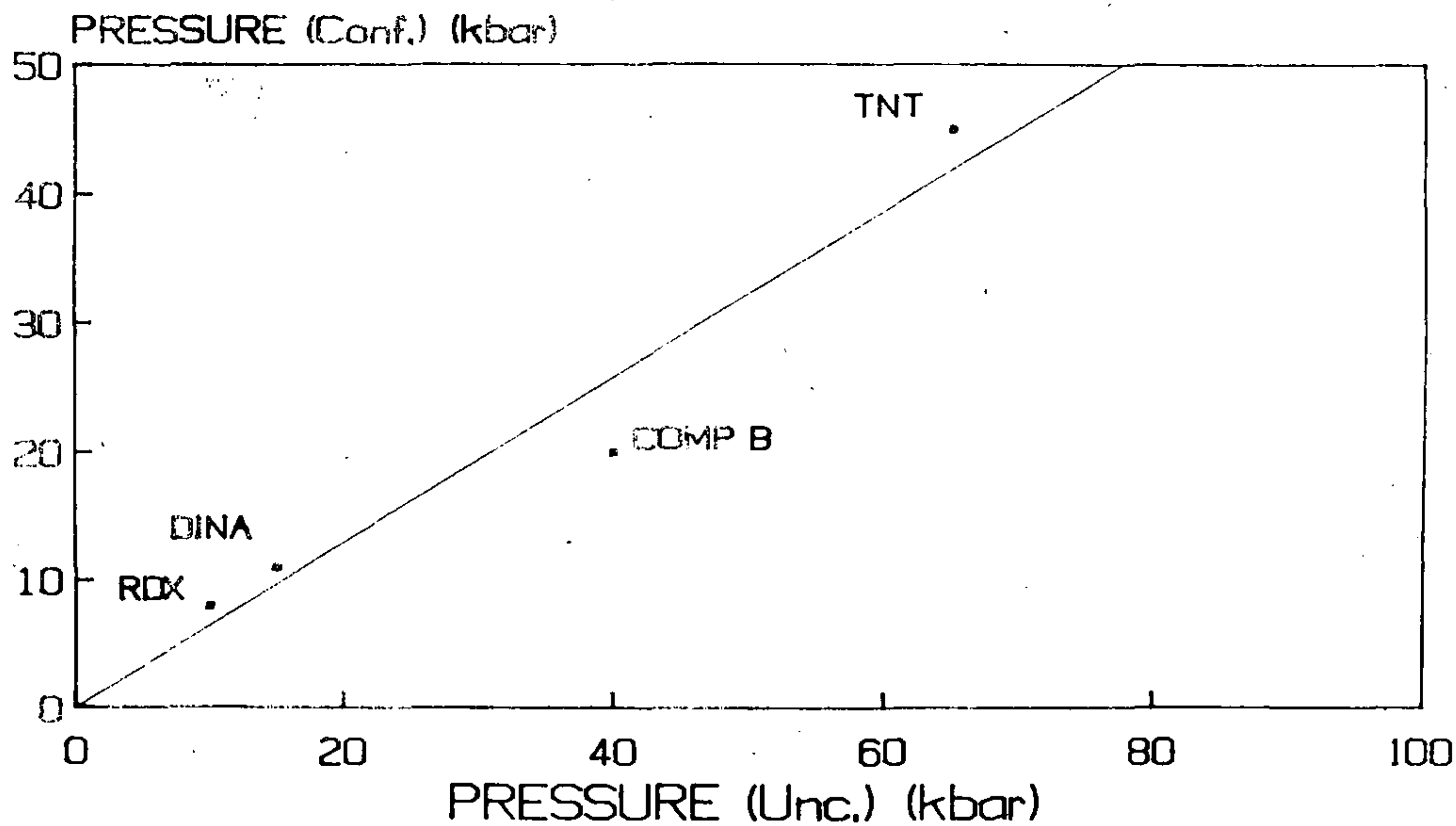


FIGURE 7: VARIATION OF CRITICAL PRESSURE WITH DIAMETER

FIGURE 8: EFFECT OF CONFINEMENT ON CRITICAL GAP PRESSURE



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VI CURSO INTERNACIONAL DE INGENIERIA GEOLOGICA APLICADA A OBRAS SUPERFICIALES Y SUBTERRANEAS

MODULO IV: TECNOLOGIA SOBRE EL USO DE EXPLOSIVOS

CHAPTER 11 BLASTING THEORY

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CHAPTER 11
BLASTING THEORY

by R. Frank Chiappetta

1. INTRODUCTION

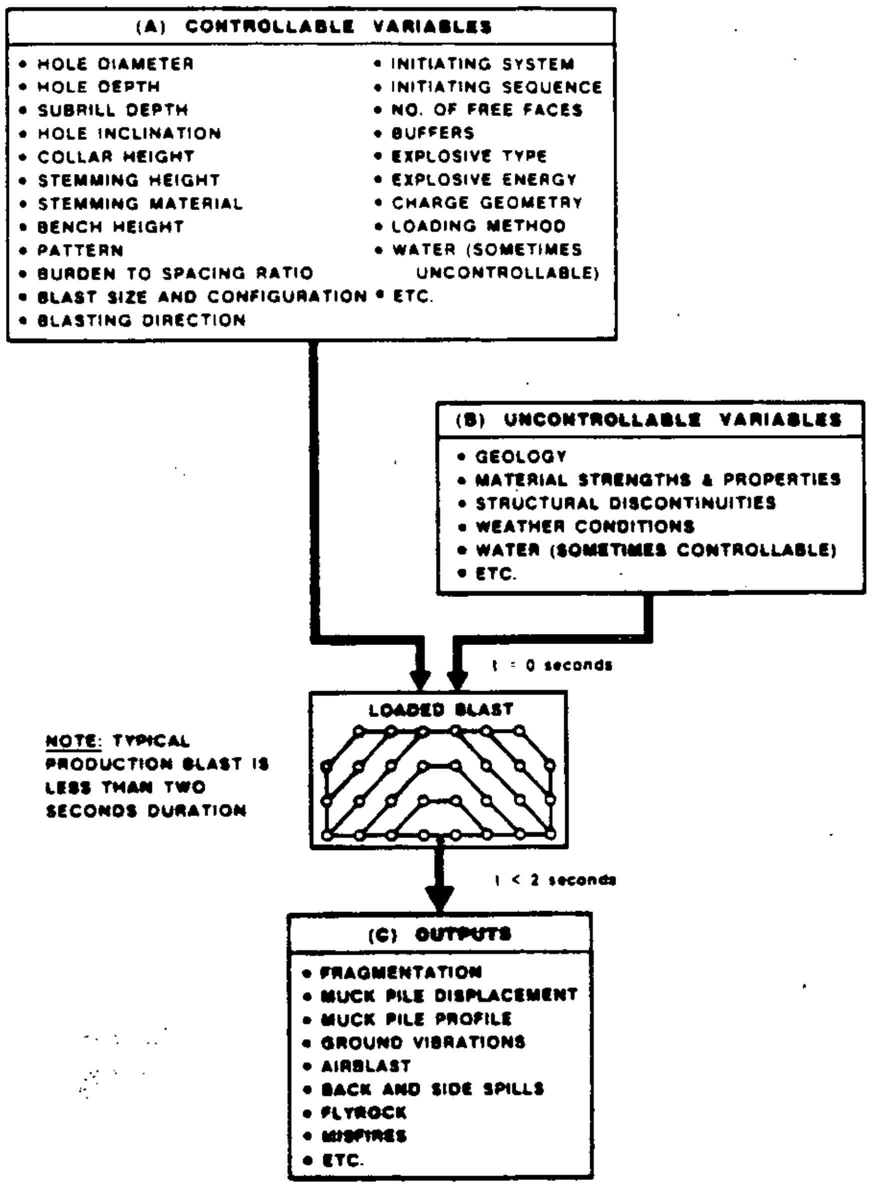
Blasting theory is perhaps one of the most interesting, thought provoking, challenging and controversial areas of our industry. It encompasses many areas in the science of chemistry, physics, thermodynamics, shock wave interactions, and rock mechanics. In broad terms, rock breakage by explosives involves the action of an explosive and the response of the surrounding rock mass within the realms of energy, time and mass. Past, current and new blasting theories are presented along with the factors affecting fragmentation and general blast design criteria. The chapter content has been carefully selected to emphasize the concepts associated with each blasting theory rather than a rigorous mathematical, physical, or chemical treatment through formulae. Where formulae are introduced, they are merely to enhance the concepts presented.

In spite of the tremendous amount of research conducted in the last few decades, no single blasting theory has been developed and accepted that adequately explains the mechanisms of rock breakage in all blasting conditions and material types. Given specific test environments, conditions and assumptions, individual researchers have contributed valuable information and insight as inputs into blasting theories, although a simple "plug-in" type formula for predicting "optimum fragmentation" is still largely unresolved. There is as yet no consistent and widely applicable theory of blasting, but only a number of limited and disconnected theories, many of which are empirical in nature and based on ideal blasting conditions. Blasting theories have been formulated and based on pure speculation, years of blasting experience on a trial and error approach, laboratory testing, field investigations, and mathematical and physical models adapted from other disciplines of science.

Primary breakage mechanisms have been based upon:

- Compressional and tensile strain wave energy
- Shock wave reflections at a free face
- Gas pressurization on the surrounding rock mass
- Flexural rupture
- Shear waves
- Release-of-load
- Nucleation of cracks at flaws and discontinuties
- In-flight collisions

Since so many schools of thought surround blasting theory, one must be prepared to investigate not only the theories, but the overall field input variables that are inherent in any blast design to have any practical meaning. Given the diverse nature of field conditions encountered and the overwhelming number of blast design variables to select from, blast results may not always be easily predicted as is outlined in Figure 11-1. Where one theory is successful in one specific environment or application, it may not be as predictive in another.



FIELD MODEL ILLUSTRATING BLAST DESIGN INPUTS AND OUTPUTS

FIGURE 11.1

Often more than one theory is needed to clarify or explain certain results. Parallel this approach to the physicist trying to explain light with only one theory – that is, the wave theory. With the passage of time it became apparent that everything associated with light could not always be adequately explained with this theory alone and hence, another theory, the particle or "packets of energy" theory was developed to explain the phenomena of light in which the first theory failed. With both theories, the physicist could now explain many of the mysteries surrounding light which eventually led to new developments such as the laser. Similarly, in trying to define the mechanisms of rock breakage by explosives, more than one theory or explanation is often needed. In any case, a blasting theory should not only attempt to explain and predict the breaking process, but more importantly, it should suggest and allow new methods and techniques to improve on current blasting practices.

2. TIME EVENTS FOR THE BREAKING PROCESS

There are basically four time frames designated as T1 to T4 in which breakage and displacement of material occur during and after complete detonation of a confined charge.

The time frames are defined as follows:

T1 — Detonation
T2 — Shock or Stress Wave Propagation
T3 — Gas Pressure Expansion
T4 — Mass Movement

Each time frame is first discussed separately, and then discussed in conjunction with blasting theories for an overall, more detailed explanation and meshing of events. Although these are treated as discrete events, it should be emphasized that in a typical shot hole or production blast, one event phase can occur simultaneously with another at specific time intervals.

a. T1 — DETONATION

Detonation is the beginning phase of the fragmentation process. The ingredients of an explosive consisting of a fuel and oxidizer combination; upon detonation, are immediately converted to high pressure, high temperature gases. Pressures just behind the detonation front are in the order of 9 Kbars to 275 Kbars, while temperatures range from approximately 3000° to 7000°F.[2]

Detonation pressure is generally expressed as a function of the velocity of detonation and density of the explosives as,

$$P_d = (2.325 \times 10^{-7}) \times \rho \times VOD^2$$

Where P_d = detonation pressure in Kbars
ρ = density in g/cc
VOD = velocity of detonation in ft/sec.

To change detonation pressure from Kbars to lb/in², multiply Kbars by 14,700. Generally, explosives yielding higher detonation pressures are required to fracture materials which are massive, fine grained, hard, tightly bonded and strongly consolidated with heavy burdens. Typical values of detonation pressure for selected explosives are presented in Table 11-1.

TABLE 11.1
DETONATION PRESSURES FOR SELECTED EXPLOSIVES

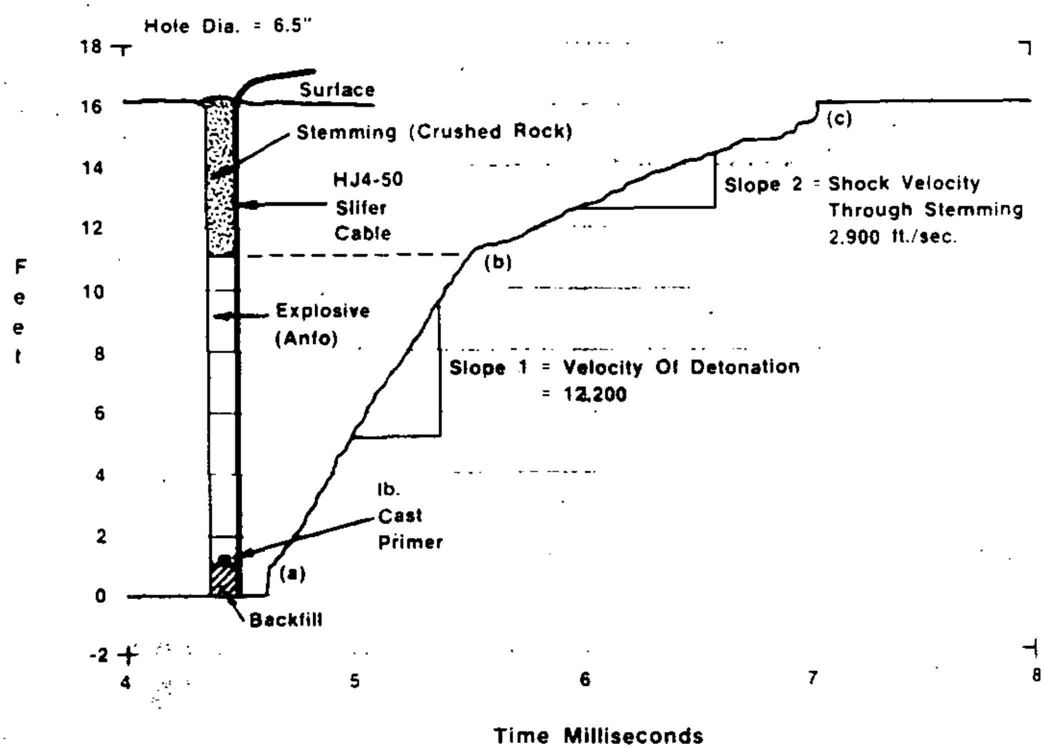
Explosive	Density (g/cc)	VOD (ft/sec)	Detonation Pressure (Kbars*)	Pressure (psi)
ANFO	0.81	12,000	27.00	396,900
POWERMAX 420	1.19	19,000	100.00	1,470,000
HI-PRIME	1.40	20,000	130.00	1,911,000
"G" BOOSTER	1.60	26,000	251.00	3,689,700

*1 Kbar = 14,700 PSI

The detonation wave starts at the point of primer initiation in the explosive column and travels at supersonic speeds. Supersonic refers to velocities which are faster than the speed of sound in the explosive. Typical velocities of detonation for commercial explosives range from 8,000 to 26,000 ft/sec. This velocity, sometimes referred to as the steady-state velocity, remains fairly constant for a given explosive, but varies from one explosive to another, depending primarily on the composition, particle size and density of the explosive. To a lesser extent, the steady state velocity is also affected by the degree of confinement and explosive diameter.

Since the velocity of detonation is greater than the velocity of sound in the explosive, the explosive material directly in front of the detonation head is totally unaffected until the detonation head passes through it. In a typical 30 foot explosive column loaded with an explosive having a characteristic velocity of detonation of 10,000 ft/sec, complete detonation and energy release within the entire column would occur in about 3 milliseconds. For an explosive with a velocity of detonation of 20,000 ft/sec, detonation and energy release would be complete in 1.5 milliseconds. Detonations of this kind are self-sustaining due to the inertia of the explosive itself that provides confinement necessary to maintain conditions for fast chemical reaction rates.

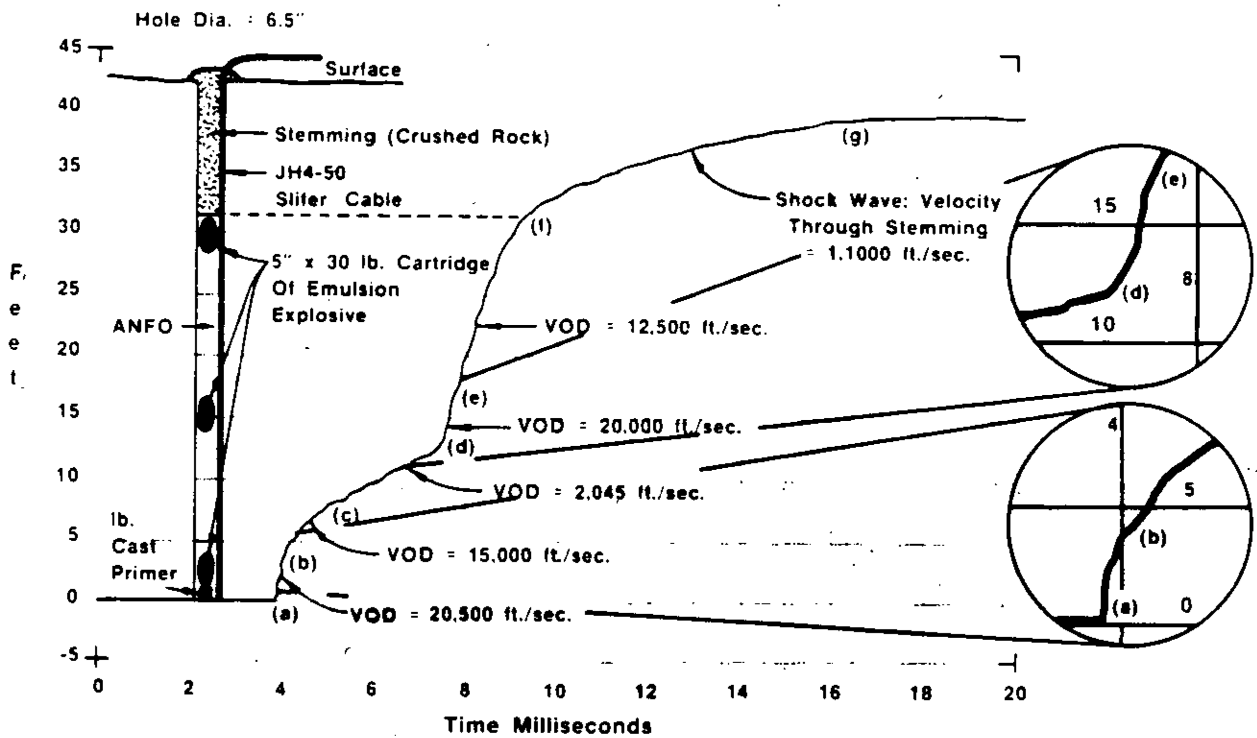
Figure 11-2 and 11-3 illustrate two typical hole load configurations. Velocity of detonation within the explosive column was measured with the SLIFER System developed at SANDIA NATIONAL LABORATORIES. For a continuous 11 foot column of cartridged ANFO, the velocity of detonation was measured to be 12,200 ft/sec as indicated by the slope of the straight line segment between point (a) and (b) in Figure 11-2. The straight line is indicative of a consistent explosive composition, constant density and a stable velocity of detonation. As detonation progresses along the column, not only is a



VELOCITY OF DETONATION MEASUREMENT USING THE SLIFER SYSTEM DEVELOPED AT SANDIA NATIONAL LABORATORIES

FIGURE 11.2

shock wave imparted into the surrounding medium adjacent to the borehole wall, but is also imparted into the stemming as indicated by the slope of the straight line segment between points (b) and (c). In this case, the shock wave velocity through the stemming was measured to be 2.900 ft/sec, or approximately ¼ that of the velocity of detonation.



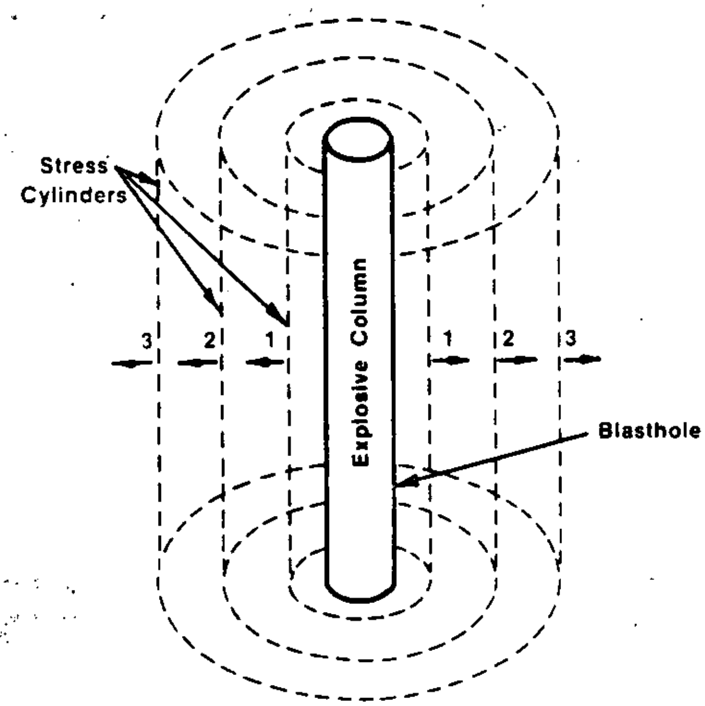
VELOCITY OF DETONATION MEASUREMENT USING THE SLIFER SYSTEM DEVELOPED AT SANDIA NATIONAL LABORATORIES.

FIGURE 11.3

In Figure 11-3, results are shown using ALTERNATE VELOCITY techniques with a hole loaded with ANFO as the main charge, with cartridges of APEX 260 emulsion spaced 11-12 feet along the column. Without direct measurements of the continuous velocity of detonation, much of the information would not have been discernable in the field by direct observation. Many important points are noteworthy in the results. Between points (a) and (b), the velocity of detonation for the 3 foot length of emulsion cartridge is 20.500 ft/sec. Between (b) and (c) the velocity of detonation is reduced from 20.500 ft/sec to 2.045 ft/sec within the ANFO and the detonation is sustained at the lower velocity until point (d) is reached. At point (d) the detonation head encounters another emulsion cartridge, which when detonated, at 20.000 ft/sec between points (d) and (e), brings ANFO back up to its normal velocity of detonation of 12.500 ft/sec. Thus, even a low order ANFO detonation can act as a very effective primer for the emulsion cartridge. The decrease in velocity between points (b) and (c) is attributed to water trickling into the bottom part of the hole from the surrounding rock mass. Although ANFO can tolerate up to a 10% water saturation level, it does so at the cost of blasting efficiency. If the center emulsion cartridge was not present, one of two things would have occured. It may have sustained a low order ANFO detonation with a velocity of 2,045 ft/sec throughout the remaining explosive column, or it would have soon failed. It has been demonstrated in field trials that where an explosive of higher velocity of detonation is embedded sparingly within the column of a main explosive with a lower velocity of detonation, that better results are generally achieved. The greater the difference in detonation velocities and the harder the material to be blasted, the more pronounced are the results.

b. **T2 — SHOCK AND STRAIN WAVE PROPAGATION**

The second phase, immediately following detonation or in conjunction with the detonation phase of T1, is the shock and strain wave propagations throughout the rock mass. This disturbance or emitted

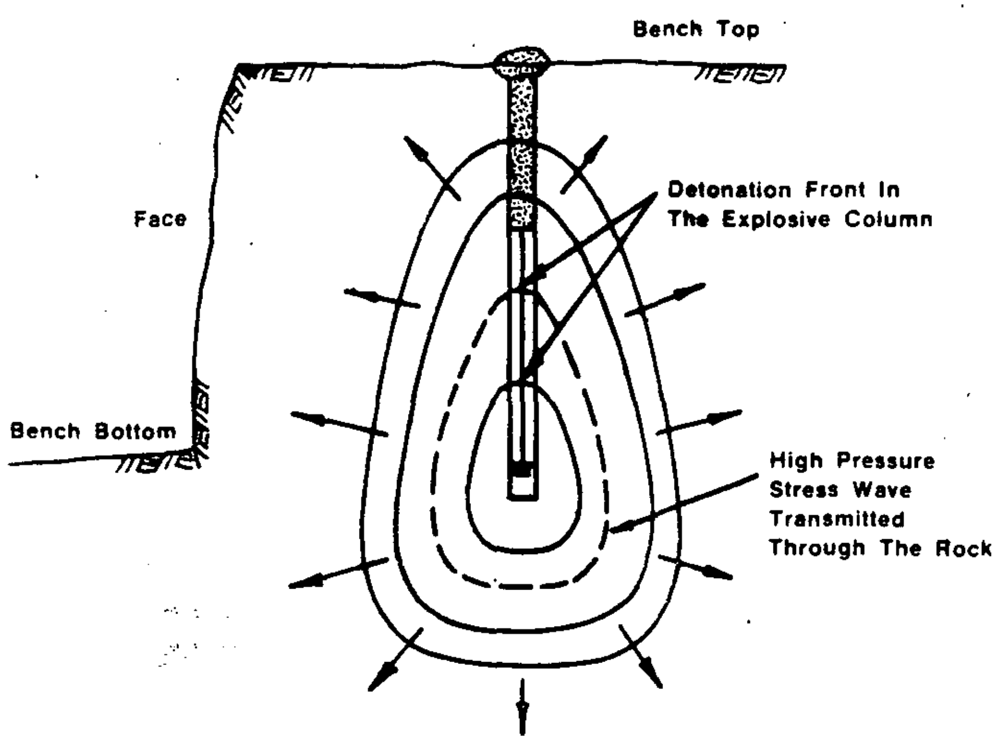


1,2,3 Successive Positions Of Stress Wave

THEORETICAL POSITIONS OF THE OUTBOUND DISTURBANCE FROM A COLUMN CHARGE

FIGURE 11.4

pressure wave(s) emitted into the rock mass results, in part, from the rapidly expanding high-pressure gas impacting the borehole wall. The geometry of dispersion depends primarily on the shape of the charge. If the charge is shot, with a length to diameter ratio of less than or equal to 6:1, then the disturbance is propagated in the form of an expanding sphere. If the charge is long, with a length of diameter ratio of greater than 6:1, then the disturbance is propagated in the form of an expanding cylinder. (Figure 11-4). However, in a typical, bottom primed, cylindrical shot hole normally encountered in bench blasting, the strain waves originally formed near the point of initiation are already in progress and propagating into the surrounding medium, while the detonation is still progressing within the explosive column. Thus, close to the shot hole, strain wave propagation is neither perfectly spherical nor cylindrical but more like that shown in Figure 11-5.



SECTION THROUGH THE FACE DURING DETONATION SHOWING EXPANDING STRESS WAVE FRONT

FIGURE 11.5

The pressure next to the borehole wall will rise instantaneously to its peak and then rapidly decay exponentially. The quick decay is due to cavity expansion of the borehole and increased gas cooling. Cavity expansion around the borehole can occur through crushing, pulverization, and/or displacement of material and can range anywhere from about one to three hole diameters depending on the medium and explosive used. Generally, extensive compressive, shear and tensile failure occur as a region of pulverized material since the wave energy is at its maximum near the borehole wall.

As the strain wave front proceeds outward, it has a tendency to compress the material at the wave front through a volume change. At right angles to this compressive front, there exists another component referred to as the tangential or "hoop" stress. The tangential stress, if large enough, can cause tensile failures at right angles to the direction of propagation. The largest tensile failures are expected to occur close to the borehole where the tangential stress is high enough for failure to occur. Both the compressive and tensile components of the wave front decay with distance from the borehole.

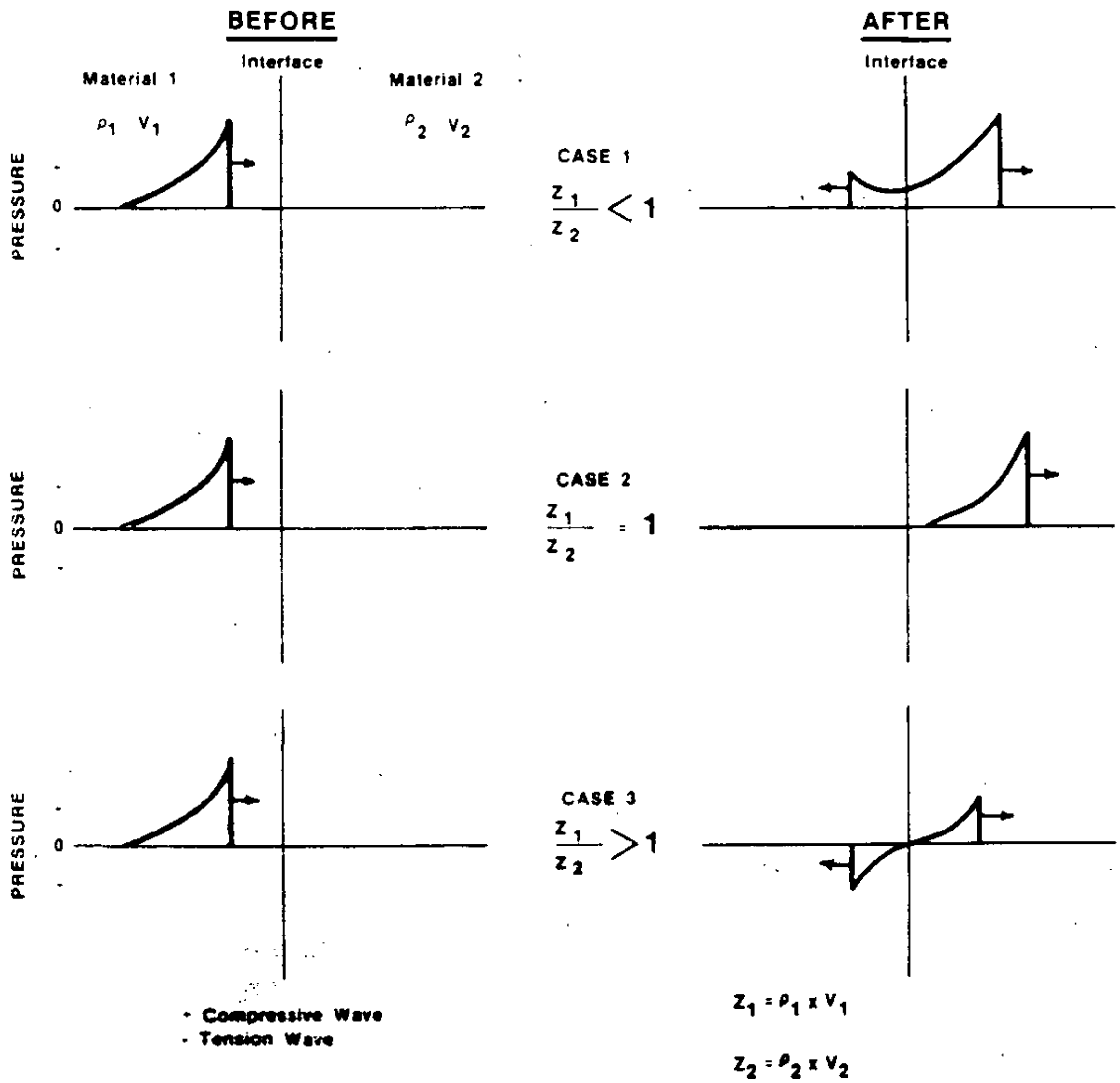
When the compressive wave front encounters a discontinuity or interface, some of the energy is transferred across the discontinuity and some reflected back to its point of origin.[4] For the most part, the partitioning of energy depends on the ratio of the acoustic impedance of the materials on either side of the interface, as illustrated in Figure 11.6. Acoustic impedance, Z, for any material is defined as:

$$Z = \rho \times V_p$$

where: Z = acoustic impedance
ρ = density of material
V_p = sonic velocity of material

In reference to Figure 11-6, where the ratio of the acoustic impedance of material 1 to material 2 is less than one, some of the wave energy is transferred into material 2 and some reflected back, but both waves remain compressional. When the acoustic impedance ratio is 1, all of the energy is transferred into material 2 and no reflected wave occurs. When the impedance ratio is greater than 1, then some of the energy gets transferred into material 2 as a compressive wave and the remaining energy gets reflected at the interface as a tensile wave. When a compressive wave travelling through rock encounters an interface such as a free face, nearly all of the energy will be reflected back as a tensile wave. If the burden distance between the free face and explosive column is relatively small in contrast to normal burdens for a chosen explosive, then most of the energy is consumed in spalling at the free face.

The interaction of stress waves in the outgoing compressive and reflected tensile modes around discontinuities and flaws within the rock mass is an area of intense research and is considered to be quite important in some of the newer blasting theories.



INTERACTION OF STRESS WAVES AT AN INTERFACE

FIGURE 11.6

c. T3 — GAS PRESSURE

During and/or after strain wave propagation, the high pressure, high temperature gases impart a stress field around the blasthole that can expand the original borehole, extend radial cracks and jet into any discontinuity. It is during this phase where some controversy exists as to the main mechanism of fragmentation. Some believe that the fracture network throughout the rock mass is completed while others believe that the major fracturing process is just beginning. In any case, it is the gases that have jetted into discontinuities and the fracture network that is either fully developed or being developed, which are responsible for the displacement of broken material.

It is not clear as to the exact travel paths that gases take within the rock mass, although it is agreed that they will always take the path of least resistance. This means that gases will first migrate into existing cracks, joints, faults, and discontinuities, in addition to seams of material which exhibit low cohesion or bonding at interfaces. If a discontinuity or seam between the borehole and free face is sufficiently large, the high pressure gases will immediately vent to the atmosphere, rapidly reducing the total confinement pressures, and results in reduced displacement of broken and fragmented material.

The confinement time of gas pressures within a rock mass vary significantly depending on the amount and type of explosive, material type and structure, fracture network, amount and type of stemming, and burden. ATLAS studies, with the use of high-speed photography in full scale bench blasts, have shown that gas confinement times before the onset of movement can vary from a few milliseconds to tens of milliseconds.[3] To date, confinement times have been measured to range from 5 to 110 milliseconds for a variety of materials, explosives and burdens. Generally, but not always, confinement times can be decreased by employing higher energy explosives, decreasing the burden, or a combination of both. This applies equally to material at the bench face or at the bench top, as in the case of stemming blowouts or cratering. Refer to Figures 12.35 and 12.36 Vibration/Airblast for specific examples of gas confinement times for stemming blowouts. It is evident that only suitably burdened and well stemmed charges can deliver their full potential of additional gas extension fracturing and mass movement.

d. T4 — MASS MOVEMENT

Mass movement of material is the last stage in the breaking process. The majority of fragmentation has already been completed through compressional and tensile stress waves; gas pressurization or a combination of both. However, some degree of fragmentation, although slight, occurs through in-flight collisions and also when the material impacts the ground. Generally, the higher the bench height, the greater is this type of breakage owing to increased impact velocities of individual fragments when falling onto the bench floor. Similarly, material ejected from opposite rows of a "V-shot" design upon head-on collisions can result in increased fragmentation. This phenomenon was evidenced and documented with the use of high-speed photography of bench blasts.

Mass burden movement of fragmented material is shown in Figure 11-7 for a number of typical face conditions encountered in bench blasting operations. Face profiles and velocities are based on the results of high-speed photographic analysis performed at the ATLAS POWDER COMPANY. Where no subdrilling is utilized, (a and b), two types of face movement may be encountered. In 11-7a the entire length of face burden, directly in front of the explosive column, moves out similar to a plane wave and the face velocity at any point is constant. This behavior is usually encountered where material is very competent, quite brittle, and structured with well defined, largely spaced joints, much greater than the spacings or burdens employed in blast designs. When the material is soft, highly fissured, and/or closely jointed as might be found in coal and some sedimentary deposits, face profiles resembling that of flexural rupture is more likely. In this case, the greatest displacement and velocity occur adjacent to the center of the explosive column with the least amount of movement occuring at the toe and crest. When identical conditions in 11-7b are assumed and when subdrilling is employed, face movement results in much the same way except that the toe burden is displaced upwards faster and at a greater angle to the horizontal.

The first three cases assumed a relatively straight face between the crest and toe, however, in many bench blasting operations, the condition is more like that illustrated in Figure 11-7d, where toe burden is considerably greater than the crest burden. The toe burden is too great for the explosive selected, hence, very little movement occurs at the toe while the greatest displacement results in the upper half of the bench.

Three options are available to increase toe movement:

- Employ angle drilling in an attempt to maintain constant burdens from the crest to the toe
- Use a higher energy bottom charge in the current vertical drill holes
- Decrease the burden with the current vertical drill holes

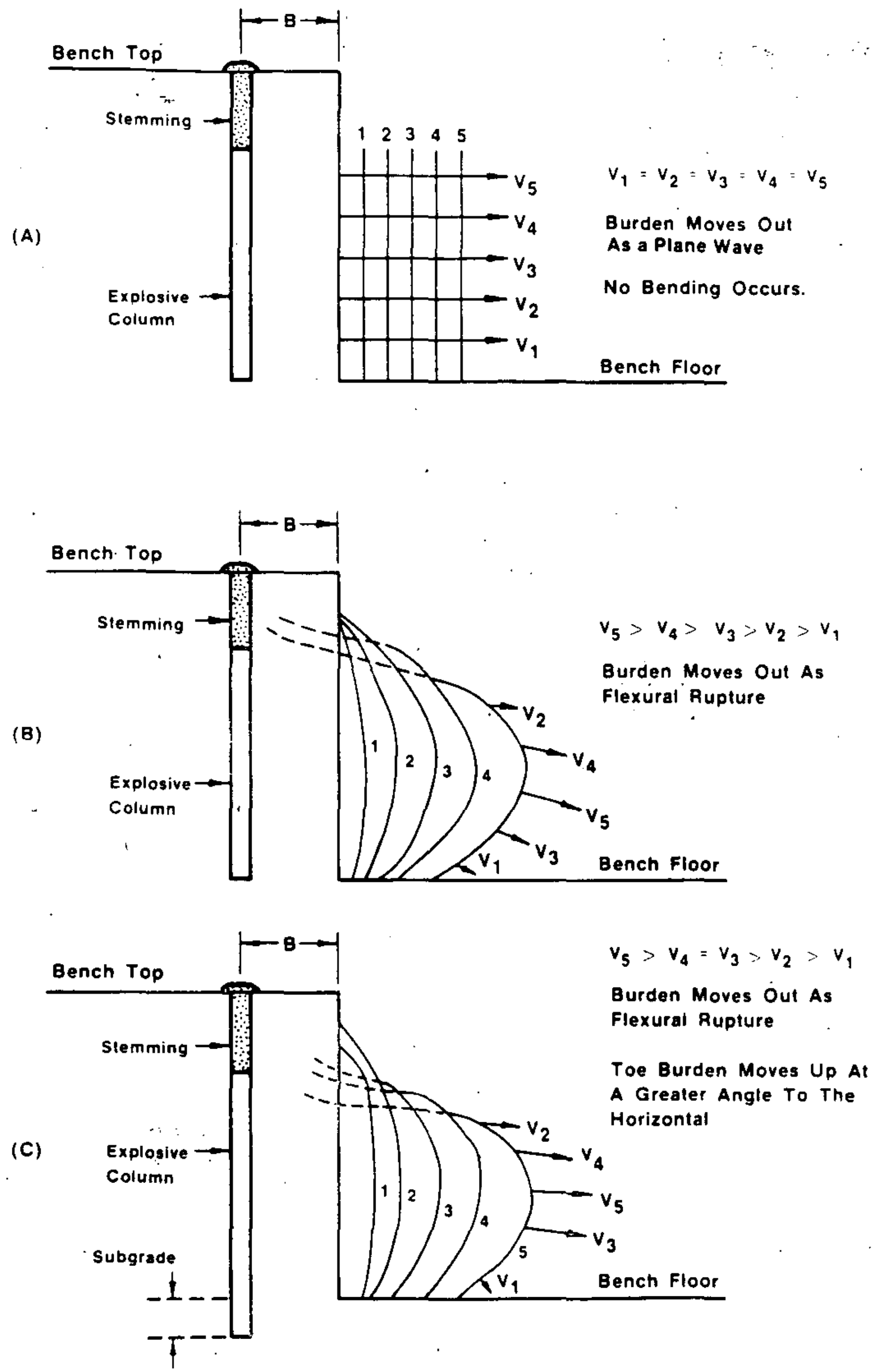
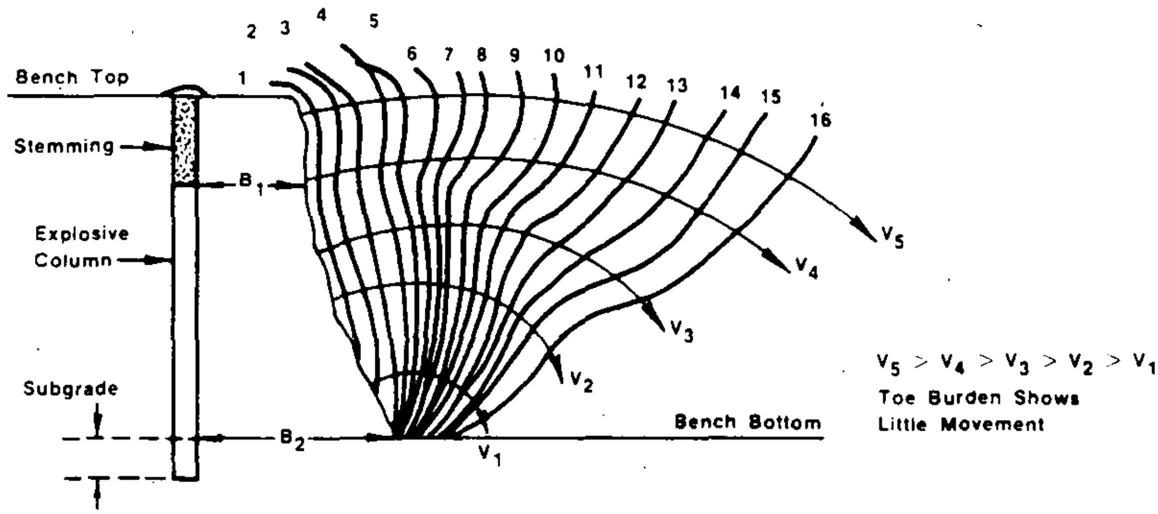

B
Bench Top
Stemming
1 2 3 4 5
V5
V4
V3
V2
V1
V1 = V2 = V3 = V4 = V5
Burden Moves Out
As a Plane Wave
No Bending Occurs.
(A)
Explosive
Column
Bench Floor
B
Bench Top
Stemming
V5 > V4 > V3 > V2 > V1
Burden Moves Out As
Flexural Rupture
(B)
1
2
3
4
V2
V4
V5
V3
V1
Explosive
Column
Bench Floor
B
V5 > V4 = V3 > V2 > V1
Bench Top
Burden Moves Out As
Flexural Rupture
Stemming
Toe Burden Moves Up At
A Greater Angle To The
Horizontal
(C)
Explosive
Column
V2
V4
V5
V3
1
2
3
4
5
Subgrade
V1
Bench Floor

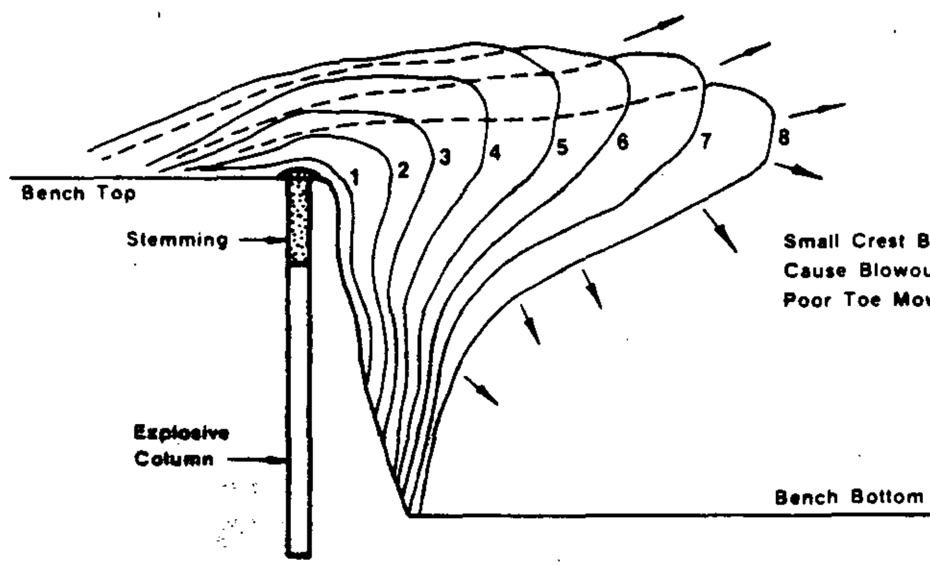
FIGURE 11.7

(D)



$V_5 > V_4 > V_3 > V_2 > V_1$
Toe Burden Shows Little Movement

(E)



Small Crest Burdens Cause Blowouts And Poor Toe Movement

FIGURE 11.7 (Cont'd)

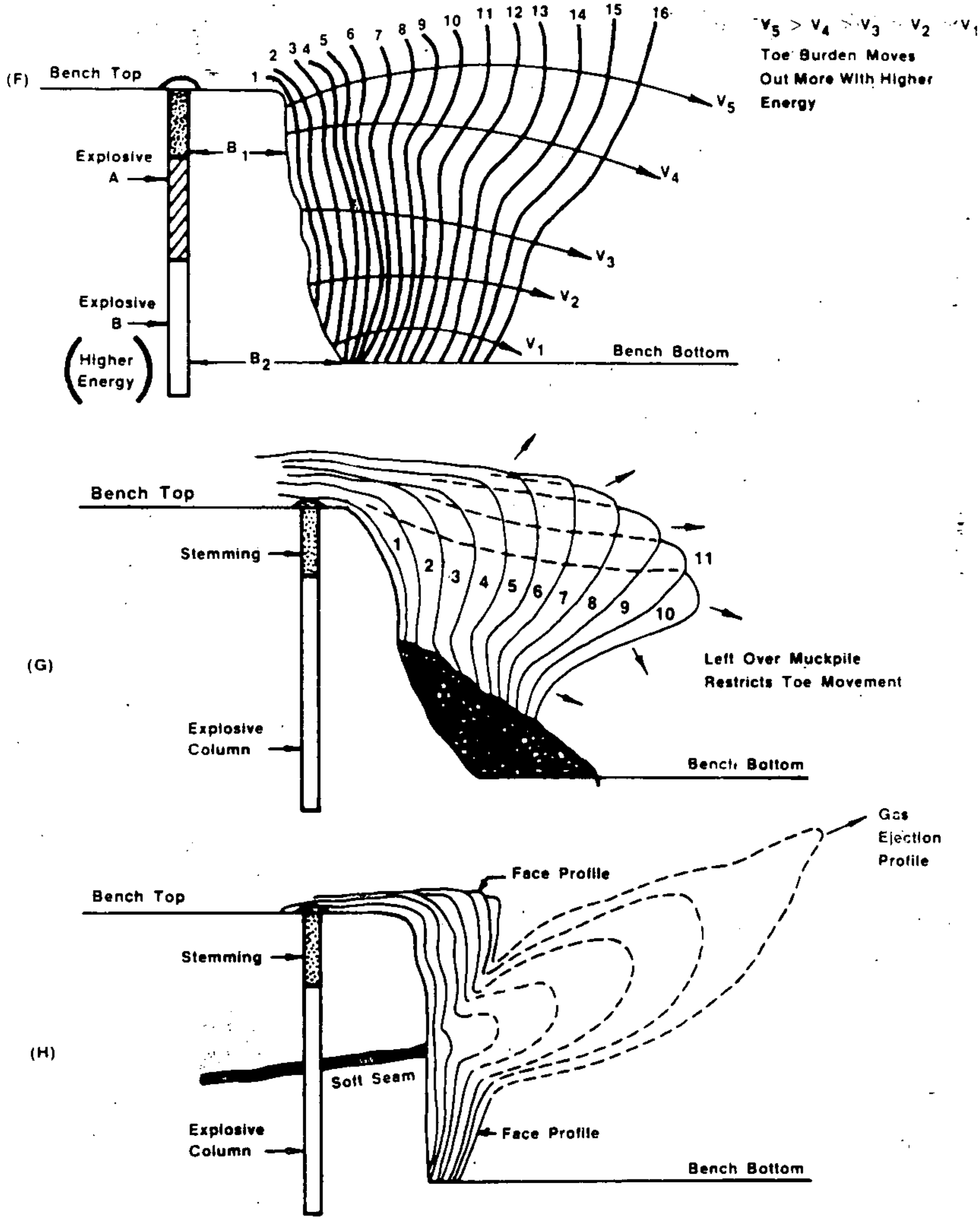


FIGURE 11.7 (Cont'd)

In selecting the latter, care should be exercised so as not to decrease the burden to the point of obtaining the condition shown in Figure 11-7e. The toe burden is now correct for the explosive selected, but the crest burden is substantially reduced. This may bring about many adverse conditions near the crest burden such as flyrock, blowouts, and increased airblast complaints. Because confinement pressures are released near the crest (in this case, a path of least resistance relative to the toe burden), restricted toe movement will result. It is better to use the same burden, but with a higher energy bottom charge near the toe. This load configuration as shown in Figure 11-7f tends to pressurize more of the burden mass for longer periods without adverse effects, and adequate toe movement generally results.

Where large leftover muckpiles are left against the face, Figure 11-7g, toe movement will be restricted and increased ground vibration levels are likely. Unless the situation requires a buffer, such as when blasting in the vicinity of mining equipment or to avoid dilution of an ore blast adjacent to a waste muckpile, it should be avoided.

Where seams are encountered in a blast, Figure 11-7h, tremendous gas ejections with velocities up to 600 ft/sec can occur. When such gas venting occurs, it will adversely affect other parts of the burden to displace adequately and inevitably leads to poor overall blasting results. A stemming deck immediately adjacent to the seam will give better results.

e. TIME EVENTS T1-T4 COMBINED

Up to this point, time events T1 to T4 have been discussed more or less as separate isolated events. However, in a real blasting environment, more than one event can occur at the same time.

Consider a single vertical hole in a quarry face with the primer located near the bottom of the hole as is illustrated in Figure 11-8. Assume the explosive used is 40 feet of ANFO with a velocity of detonation equal to 13,000 ft/sec, the material blasted is limestone with a sonic wave velocity of 15,000 ft/sec and a density of 2.3 g/cc. Upon initiation of the primer, it takes only a few microseconds and a distance of 2 to 6 hole diameters up the column to form a full detonation head. When a full detonation head is formed, it travels up the explosive column with a velocity characteristic of the steady-state velocity, (in this case 13,000 ft/sec). It takes approximately 3.0 ms for the 40 foot column of ANFO to be completely detonated.

Within this 3.0 ms, many other things have occurred. Starting at the bottom of the hole and progressing up the column, borehole

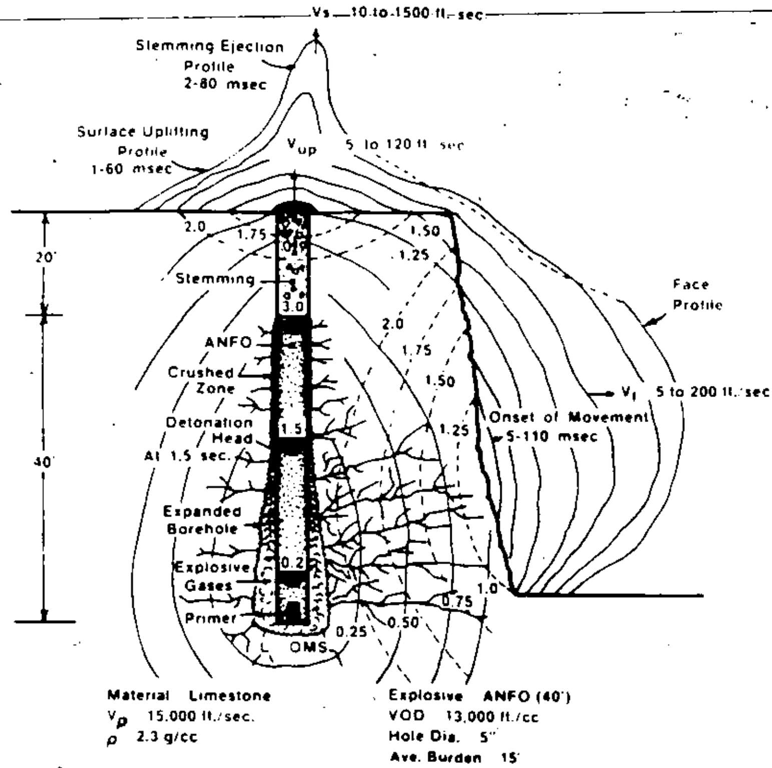


ILLUSTRATION SHOWING THE INTERACTION OF TIME EVENTS T1 TO T4 IN A TYPICAL QUARRY BENCH
FIGURE 11.8

expansion through crushing of the borehole walls has taken place. This produces compressive stress waves with tangential components emanating from the borehole walls and progressing outward in every direction with a velocity characteristic of the sonic wave velocity of limestone. It takes approximately 1.0 msec for the compressive strain wave to transverse 15 feet of burden to the free face. Behind the strain wave propagation some radial cracks start to develop in the crushed zone region of the borehole with a velocity ranging from 25 to 50% of the P-wave velocity for limestone. If the intensity of the compressive strain pulse is high enough, new cracks and/or extensions of pre-existing cracks and flaws can be initiated anywhere between the crushed zone next to the borehole and the free face. The greatest number of cracks are generally found closest to the borehole.

When the compressive wave strikes a free face, it is immediately converted to a tensile strain wave which starts at the free face and travels back through the rock mass towards the borehole. Owing to the new fractures created from the outgoing compressive strain wave, the tensile strain wave will take somewhat longer to travel the same burden distance of 15 feet. If the burden is small enough and the intensity of the reflected strain wave is large enough, then some spalling at the free face or bench top is expected, although no significant mass movement will occur.

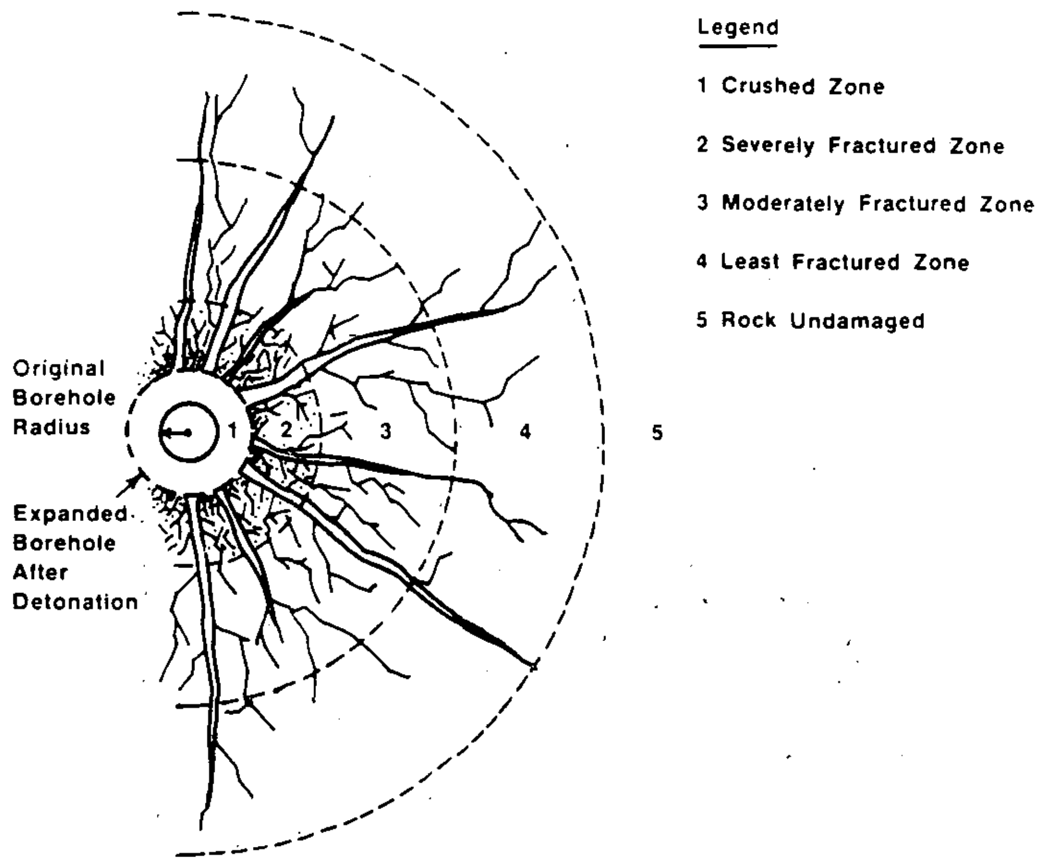
At 3 ms after detonation and complete reaction of ANFO, the original high temperature, high pressure gases have reached a new equilibrium due to borehole expansion. Both temperature and pressure have dropped significantly resulting in an energy reduction ranging from 25 to 60% of the theoretical energy originally available. This remaining energy acts on the surrounding "preconditioned" rock mass to displace it in the direction of least resistance. Further fragmentation can occur at this stage from gases entering and extending preexisting cracks or discontinuities. It is at this stage where some blasting theories are contradictory. Some believe that the major fracture network is completed within about 3 ms due to the interaction of stress waves on the surrounding material, while others believe that the major fracture network is just beginning.

Regardless of which time frame is responsible for the development of a fracture network, mass movement and displacement of material at the bench top or face occurs much later in time due to the confinement of gas pressure within the rock mass. The onset of mass movement depends on the material response in conjunction with the strain and gas pressure stimulus generated from the explosive. For typical stemming and burdens encountered in the field, bench top swelling occurs between 1 to 60 ms, stemming ejections between 2 to 80 ms and bench burdens between 5 to 110 ms. Surface uplifting velocities around the collar region of a hole occur between 5 and 120 ft/sec, stemming ejections between 10 to 1500 ft/sec and burden velocities between 5 to 200 ft/sec. Gas ejection velocities at discontinuities have been recorded as high as 700 ft/sec and often occur in less than 5 ms.

3. RUPTURE RADIUS

The degree of damage and fracturing around a borehole can be characterized by four zones as illustrated in Figure 11-9. In the crushed zone immediately around the borehole, the explosive induced pressures and stresses exceed the dynamic compressive strength of the rock by factors ranging from 40 to 400. These high pressures acting against the borehole wall will crush, pulverize and shatter the surrounding rock mass, causing intense damage. This zone is also referred to as the hydrodynamic zone in which the elastic rigidity of the rock becomes insignificant. (6)

Next to the crushed zone is a region defined by a severely fractured zone referred to as the non-linear zone. Here fracturing can range from severe crushing through partial fracturing, to plastic deformation. Extension



ZONES OF RUPTURE RADIUS
FIGURE 11.9

of cracks can occur from previously formed cracks by the tangential component (hoop stress) of the shock wave, infiltration of gas pressure and at flaw sites.

In zones 3 and 4 (elastic zones) tensile failures and crack extensions occur in a less intense mode because the stress wave amplitude has attenuated significantly. Much of the original energy from the detonation has been consumed in the form of heat, friction, and fracturing in zones 1 and 2. The peak amplitude of the compressive stress is now much smaller than the compressive strength of the rock so no new fractures are likely in this wave type. However, the tangential stress component of the wave is still substan tially larger than the tensile strength of the rock. Since the tensile strength of rock is about 1/15 to 1/10 of the compressive strength, the tangential stress of the wave is large enough to cause radial fractures. These new fractures are formed from the extension of cracks in the non-linear zone (zone 2) or from cracks initiated from microfractures and flaws inherent in a typical rock mass.

Once the tangential stress has attenuated below the critical tensile strength of the rock, no further breakage occurs beyond this point as illustrated in zone 5 (Figure 11-9). Once the wave or disturbance passes into and through this zone, the individual particles of the medium will oscillate and vibrate about their rest positions within the elastic limits of the rock and so no permanent damage results. It is this region where seismic waves are carried considerable distances and are responsible for ground vibrations.

Table 11-2 gives an idea of the degree of maximum damage found around the crushed and fractured zones in terms of charge radii for a number of conditions. Results are based on the works of many researchers, conducted in a number of different materials with varying explosives. For a given explosive, the rupture radius is greater in soft rock than hard rock. Given the same rock, the rupture radius is greater for higher strength explosives than lower strength ones. Thus, the degree of radial rupture is influenced by the explosive, material properties and structure.

TABLE 2
DEGREE OF DAMAGE AROUND A BOREHOLE IN TERMS OF CHARGE RADII

SOURCE	EXPLOSIVE	EXPLOSIVE AMOUNT	CHARGE SHAPE	MATERIAL OR ROCK TYPE	CRUSHED ZONE IN CHARGE RADII (MAX)	RADIUS OF DAMAGE IN CHARGE RADII (MAX)	COMMENTS
Olsen (7)	C4	0.25 kg	S	Granite	—	18	
		2.00 kg	S	Granite	—	20	
Siskind (8)	60% Dynamite	—	C	Shale	—	45-55	
	ANFO	—	C	Shale	—	15-22	
Cattermole (9)	60% Dynamite	—	C	Tuffaceous Pyroclastic	3.0	20-30	
Colorado (10) School of Mines	—		-	Soft Rock		26-29	
			-	Hard Rock	-	20-23	
Oerlich (11)	Nuclear (TNT)		—	Granite	1.9	4.9	
Atchison (12)	—	3.6 kg (max)	C	Granite	3-4.5	—	
D'Andrea (13)	C4	0.00 216 kg to 0.487 kg	S	Granite	2.3	—	
Siskind (14)	ANFO	—	C	Granite	—	14	
Kutter et al (6)	Underwater Spark Discharge	—	S	Plexiglass & Rock	—	6	Theoretically Calculated
			C	Plexiglass & Rock	—	9	Theoretically Calculated
Vovk et al (15)	—	—	—	Granite, Limestone & Concrete	8-12	30-50	
Roig (16)	Nuclear			Competent	2.7-3.5	-	Horizontal Fracturing Below Shot Point
					2.0	-	

4. BLASTING THEORIES (Past & Present)

In this section, blasting theories of the past and present are discussed in concept form. Table 11-3 is a list of some of the more common thoughts regarding breakage mechanisms and the researchers responsible for their introduction. This list is by no means complete, but it does illustrate how certain thoughts on blasting theory started with the simple reflection theory after World War II and progressed to the more complex nuclei or stress-wave flaw theory of the present.

Since each theory has inherent strengths and weaknesses, the main concepts of each theory are best explained with a brief description. Blasting theories discussed are.

a) Reflection Theory (Reflected Stress Waves)
b) Gas Expansion Theory
c) Flexural Rupture
d) Stress Waves & Gas Expansion Theory
e) Stress Waves, Gas Expansion & Stress-Wave/Flaw Theory
f) Nuclei or Stress-Wave/Flaw Theory
g) Torque Theory
h) Cratering Theory
i) Cratering Mechanisms

a. REFLECTION THEORY (Reflected Stress Waves) (17, 18, 19, 20)

One of the first attempts to explain, analytically, how rock breaks when a concentrated explosive charge is detonated in a borehole near a free surface was with the reflection theory. The concept was simple, straight forward, and based strictly on the well known fact that rock is always less resistant in tension than in compression. A compressive strain pulse is generated by the detonation of an explosive charge, moves through the rock in all directions with a decaying amplitude, and is reflected only at a free surface. At the free surface, the compressive strain pulse is converted into a tensile strain pulse that progresses back to its point of origin. (See Figure 11-10). Since rock is weakest in tension, it is easily pulled apart by the reflected tensile strain pulse and damage at the face appears in the form of spalling. The high pressure, expanding gases, are not deemed directly responsible for the major degree of fracturing that occurs.

A more detailed explanation follows: Detonation of an explosive charge in rock generates a large quantity of high temperature, high pressure gas in a very short time. Typically, this occurs in a few microseconds for small cylindrical charges and in a few milliseconds

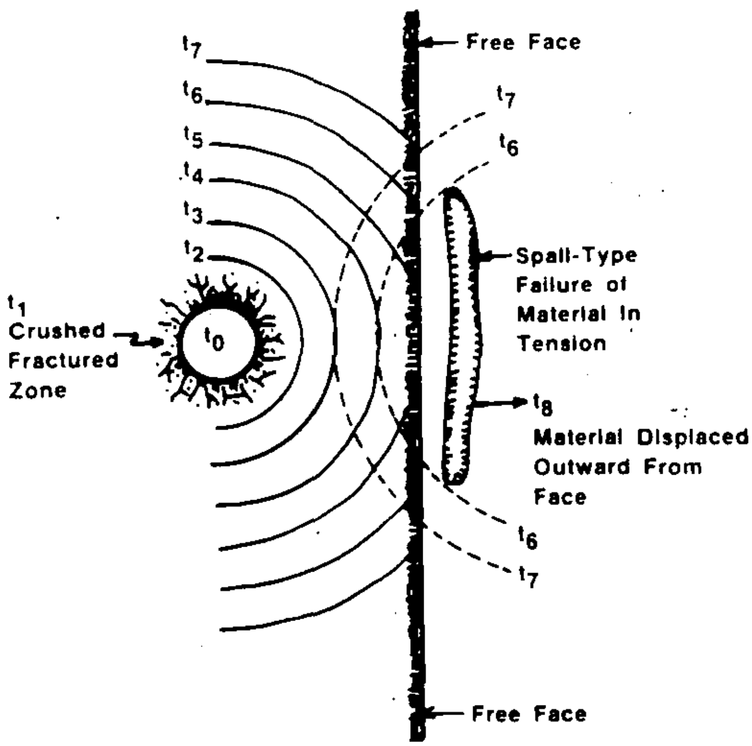
TABLE 11.3
BLASTING THEORIES AND
THEIR BREAKAGE MECHANISM

DATE	RESEARCHER(S)	BREAKAGE MECHANISMS				
		TENSILE REFLECTED WAVES	COMPRES-SIONAL STRAIN WAVES	GAS PRESSURE	FLEXURAL RUPTURE	NUCLEI STRESS-FLAW
1949	Obert, Duvall (17) (18)	1				
1956	Hind (19)	1				
1957	Duvall, Atchison (20)	1				
1958	Rinehart (21)	1				
1963	Langfors, Kihlstrom (22)		2	1		
1966	Starfield (23)	1				
1970	Porter, Fairhurst (24)		2	1		
1970	Persson, Lunborg, Johansson (25)		1			
1971	Kutter, Fairhurst (6)		1	1		
1971	Field, Ladegarrd - Pederson (26)		1	1		
1972	Johansson, Persson (27)	2		1		
1972	Lang, Faureau (28)	4	2	1		3
1973	Ash (29)			1	1	
1974	Hagan (30) (31)		1			
1978	Barker, Fourney, Dally (32) (33) (34)					1
1983	Winzer, Anderson, Ritter (35)					1
1983	Adams, Margolin (36) (37)					1
1983	McHugh (38)					1

for long cylindrical charges found in normal bench blasting. This gas pressure acting against the borehole wall generates a compressive strain or stress pulse of high amplitude which will crush and/or fracture rock next to the borehole. This stress pulse travels radially outward in all directions from the shot point at speeds equal to or greater than the velocity of sound in the medium. Due to wave divergence and energy absorption by the rock, the pulse amplitude decreases very rapidly. Thus, the extent of the crushed zone immediately next to the borehole is relatively small.

When a longitudinal compressive stress strikes a free surface, two reflected pulses are generated, a tensile and shear pulse. The amount of energy imparted to each depends on the angle of incidence of the compressional stress pulse. Of the two reflected pulses, the tensile one predominates in breaking rock as it moves back into the rock.

The effective transfer of detonation pressure to stress in the rock depends on the impedance match of the explosive-to-rock. A smaller explosive to rock impedance ratio was shown to provide a more effective transfer of this pressure to stress. The concept of reflection breakage is illustrated in Figure 11-10. The time order of key events are:



t_0 — detonation, generation of high pressure, high temperature gases

t_1 — borehole walls are crushed and slightly fractured due to high gas pressure, and borehole expands

t_2-t_4 compressional strain pulse propagates outward in all directions

t_5 — part of compressional strain pulse impinges on free surface

t_5-t_6 — part of pulse continues to travel outward and part of it is reflected at the free surface as a tensile strain pulse

— slab of rock begins to detach from free face and moves forward

t_7 — other compressive stress pulses arrive at the newly formed face and repeats breaking process

RELECTION THEORY
TENSILE FRACTURE BY RELECTION OF A COMPRESSIVE STRAIN PULSE AT A FREE SURFACE
FIGURE 11.10

Slabs broken off closer to the hole are displaced with lower velocities.

b. **GAS EXPANSION THEORY** (25) (39)

The pressure acting on the walls of an explosive filled hole, upon detonation, will be approximately one-half of the detonation pressure due to expansion of the borehole. This pressure will propagate out from the borehole into the rock as a shock wave. The material between the borehole and the shock front is compressed and flows elastically or plastically, depending on the pressure and strength of the rock. Some radial cracks form next to the borehole wall starting at about two hole radii out and then propagate radially inwards as well as outward. The greatest frequency of radial cracks are next to the borehole, but a few extend farther out. When no free face exists, a small number of these radial cracks become very much larger than the others.

By the time the shock wave reaches the free surface, radial crack lengths formed are less than one quarter of this distance. At this stage the longest of cracks have extended inwards and reached the borehole wall. Gas pressure is now capable of entering these cracks and if the pressure is high enough can reach out towards the crack tips, thus further elongating the cracks. This has the effect of aiding cracks that interact with the returning tensile wave and cause them to reach the free surface. Up to this point, acceleration of the rock mass between the hole and free face has been negligible. Only after the cracks have reached the free surface is the rock accelerated by the remaining gas pressure.

The key point of the gas expansion theory are:

- **Radial cracks are initiated not immediately next to the borehole but about two hole radii out and extend inwards toward the hole as well as outwards towards a free face.**
- **Rock displacement does not occur until pressurized radial cracks extend to the free surface.**

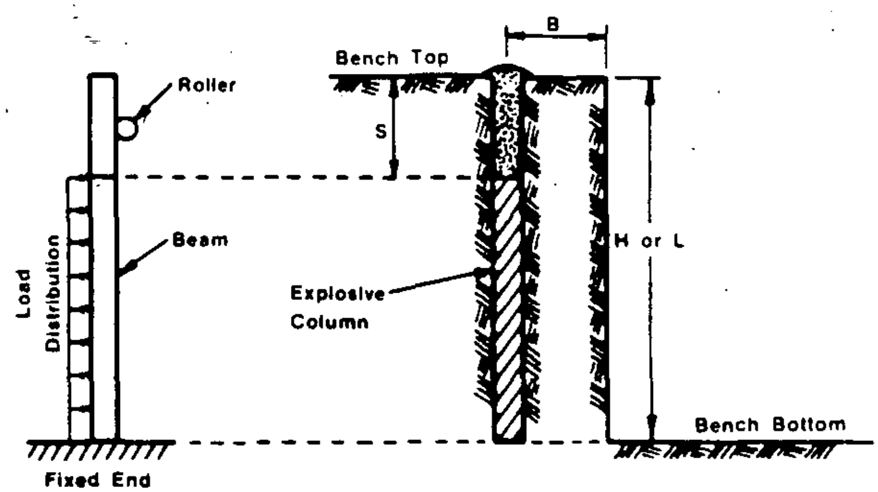
c. **FLEXURAL RUPTURE** (A Gas Expansion Theory) (29)

During detonation of an explosive confined in a borehole, two distinct pressures are formed: one from the detonation itself and the other from the highly heated gases acting on the borehole walls. In this theory, ninety percent of the total energy to break rock is in the latter. Detonation pressure acts only momentarily against any one part of the borehole's internal surface area, while gas pressure is sustained considerably longer until some form of cavity volume change occurs. Gas pressure, then, is the major component responsible for fragmentation and flexural rupture.

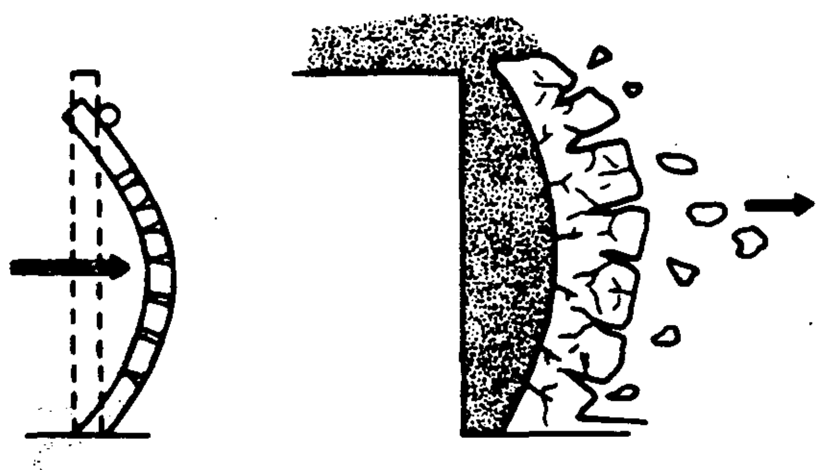
Radial cracks form only in planes parallel with the borehole axis. No cracks develop where the explosive is not in immediate contact, thus most cracks form adjacent to the borehole wall where tangential stresses are produced within the borehole's wall as the cavity is pressurized. Providing strain energies at crack tips are adequate, extension of fractures continue. Breakage by reflection of strain energy at a free face is considered negligible. Gas pressure drives the radially produced cracks through the burden to the free face and displaces rock through bending and in the direction of least resistance generally following naturally occuring planes of weakness. It is during this final stage where the major breakup of intact material takes place.

Breaking of rock by flexural rupture is analogous to bending and breaking a beam as illustrated in Figures 11-11 and 11-12. A rectangular beam is used to represent the field configuration of bench height, H, and burden, B, in the form of a modified cantilever beam model. The fixed end of the beam represents toe conditions while a roller, placed directly opposite the center of the stemming column, represents the stemming function. The roller allows the collar region to rotate and move longitudinally but does not allow deflection normal to the borehole axis. Although not shown for clarity of concept, the beam thickness in Figures 11-11 and 11-12 is actually equal to the burden. Borehole pressure is represented as a load distributed along the length of blasthole containing the explosive. Rock weight of the bench segment is considered negligible relative to the load resulting from the borehole gas pressure. Maximum contribution of total rock load acting at floor level is only at a ratio of about 1:100,000 or more compared to gas pressure.

The degree of fragmentation is controlled by the stiffness property of the burden-rock mass. This stiffness depends on existing restraints to movement, rock (Young's modulus), radially-cracked block's geometric shape as defined by its average thickness, width, and length. In terms of blast configuration, burden, spacing, and bench height are the controlling factors for any given rock.



BEAM BENDING MODEL BEFORE DETONATION
FIGURE 11.11



BEAM BENDING MODEL AFTER DETONATION
FIGURE 11.12

cant amount of rock breakage, but it does provide the basic conditioning for the last stage of the breakage process.

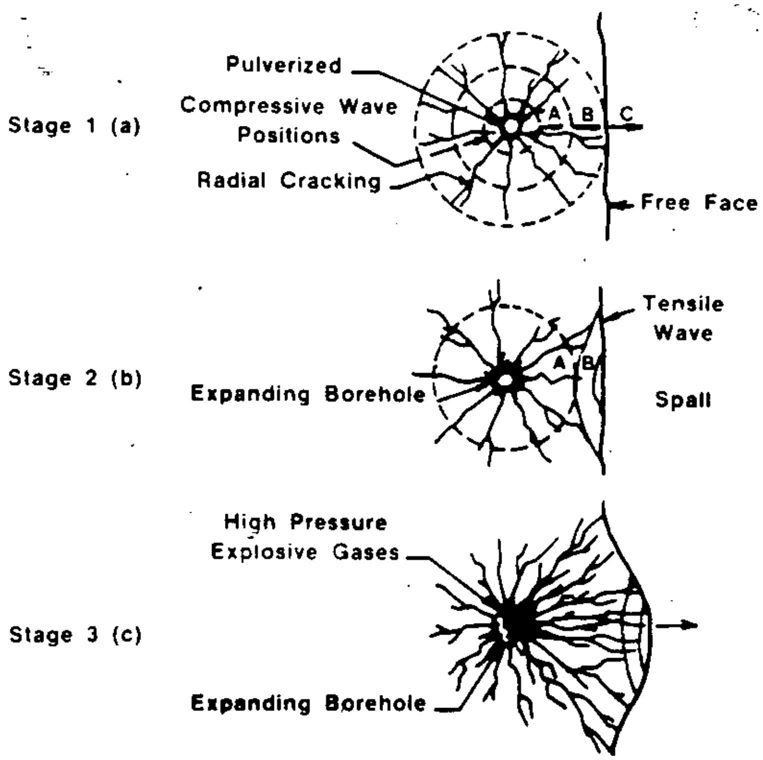
Stage 3—In this last stage the actual breakage of rock is a slower action. Under the influence of the exceedingly high pressure of the explosion gases, the primary radial cracks are enlarged rapidly by the combined effect of tensile stress induced by radial compression and by pneumatic wedging. When the mass in front of the borehole yields and moves forward, the high compressive stresses within the rock unload in much the same way as a compressed coil spring being suddenly released. The effect of unloading is to induce high tension stresses within the mass which complete the breakage process started in the second stage. The small fractures and threshold fracture conditions created in the second stage serve as zones of weakness to initiate the major fragmentation reactions. (Figure 11-13c)

f. NUCLEI OR STRESS WAVE—FLAW THEORY (32, 33, 34, 35, 37, 38)

This relatively new theory was formulated at the University of Maryland in the fracture mechanics laboratory. Laboratory tests were conducted in homolite-100 models, both unflawed and flawed, by simulating many of the geologic structures and discontinuities (joints, fractures, bedding planes) typically found in large scale bench blasting. Results showed that stress waves were quite important in the fragmentation process and caused a substantial amount of crack initiation at regions rather remote from the borehole. These regions consisted of small or large flaws, joints, bedding planes, and other discontinuities that acted as a nuclei for crack formation, development or extension. This new stress wave dominated mechanism of fragmentation is referred here as the nuclei theory.

The theory and actual mechanisms of stress wave propagation and interaction in a flawed medium are quite complex. They involve many phases such as: (40)

- **detonation and crack nucleation around borehole**
- **crushed zone extension**
- **dynamic crack stability**
- **activation of flaws**
- **coalescence of wave velocities and strains**
- **branching of cracks**
- **interaction of cracks and reflected wave systems**
- **instability of crack direction**
- **random progressive failure**



FRACTURES OPENED UP AND PROPAGATED BY GAS EXPANSION PRODUCING AN ISOLATED FRAGMENTED ROCK MASS OR CRATER
FIGURE 11.13

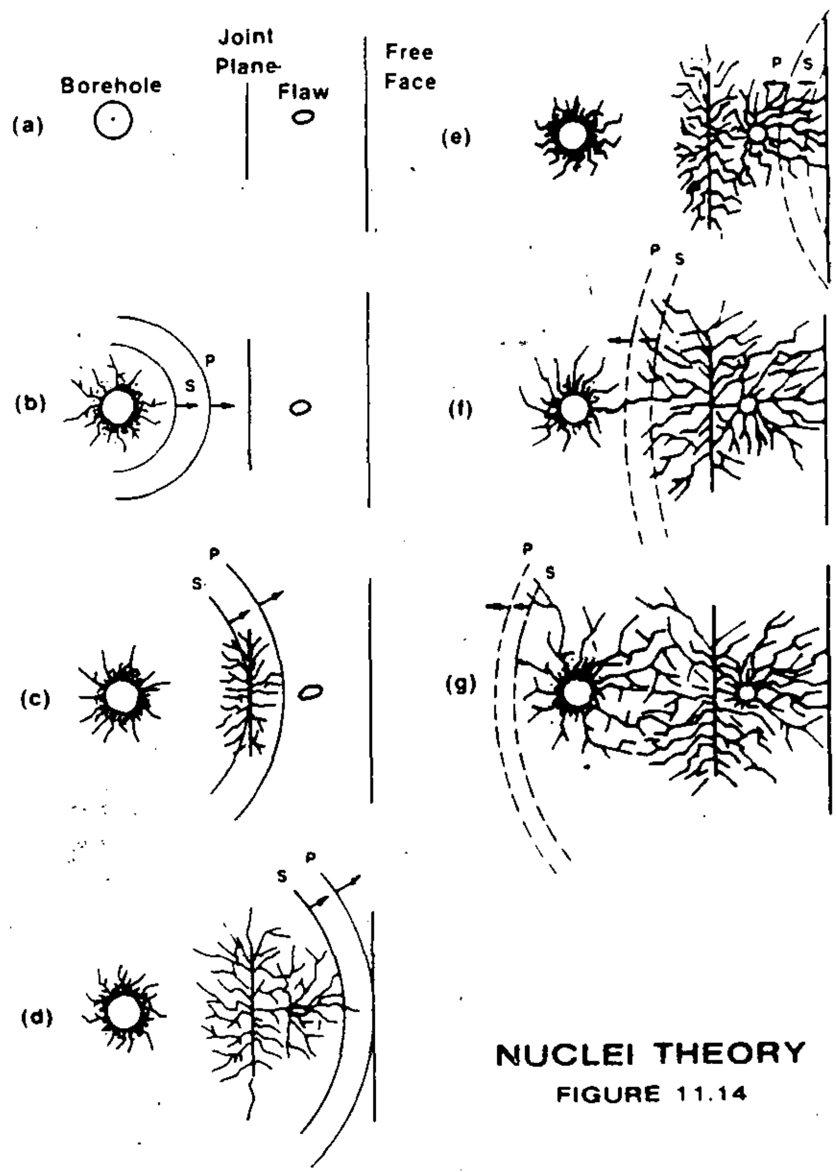
Stage 2—The pressure associated with the outgoing shock wave of the first stage is positive. If the shock wave reaches a free face it will reflect, but in so doing the pressure falls rapidly to negative values and a tension wave is created. This tension wave travels back into the rock and since this material is less resistant to tension than to compression, primary failure cracks will develop due to the tensile strength of this reflected wave. If these tensile stresses are sufficiently intense they may cause scabbing or spalling at the free face. (Figure 11-13b)

In rock breaking this spalling effect appears to be of secondary importance. It has been calculated that the explosive load must be in the order of 8 times the normal load to cause failure of the rock by reflected shock wave alone.

In the first and second stages, the function of the shock wave energy is to condition the rock by inducing numerous small fractures. In most explosives the shock wave energy theoretically amounts to only 5 to 15% of the total energy of the explosive. This strongly suggests that the shock wave is not directly responsible for any signifi-

In more simple terms, the important points of the theory are explained with the illustration in Figure 11-14. A borehole is located behind a free face with two discontinuities, a joint plane and a small flaw, located between the borehole and free face. Assume all other areas in the medium to be homogeneous and flaw free.

In unflawed material, only 8 to 12 dominant cracks emerge from a dense radial network around the borehole. These dominant cracks can travel significant distances and consequently form large pie shaped segments, that alone are not conducive for good fragmentation. Stress waves continuing away from the fractured zone around the borehole result in no further damage.



NUCLEI THEORY

FIGURE 11.14

In flawed material or sections of the material which contain flaws, fragmentation is quite different. Consider the P and S waves propagating away from the fracture network around the borehole in Figure 11-14b and 11-14c. Refer to Chapter 12—Vibration/AirBlast section for a discussion on Seismic Waves. No fracturing takes place until the flaw (joint plane) is initiated by the P wave tail and the leading front of the S wave. (Figure 11-14c). The remainder of the S wave has sufficient energy to keep the crack from arresting. A similar effect occurs as the P and S waves move past the small flaw between the joint plane and the free face, (Figure 11-14d). It is important to note that cracks are initiated at flaw sites remote from the borehole region by the combined action of the P wave tail and the S wave front. Flaws initiated in the immediate borehole vicinity of these waves have only a small effect. Note also, that the outward directed P and S waves can initiate flaws anywhere independent of the presence of a free surface.

When a P wave encounters a free face (Figure 11-14d and 11-14e), it is reflected and travels back into the medium as a tensile wave to meet the outcoming S wave. At this stage, constructive interference can occur which allows for further crack initiation or extension of cracks previously formed. New wave systems (PP, PS, SP, SS, PP, and S, PS, and S) will also form from the original outgoing wave system upon reflection at a free surface or discontinuity. These new wave systems can also contribute to crack extensions. Figure 11-14f and 11-14g illustrate further crack extensions when all wave systems have been reflected back towards the hole.

The important points of the nuclei or stress-wave flaw theory are:

- **the fracture network spreads with the speed of the P and S waves, which initiate fracture around flaws remote from the borehole**
- **in highly flawed material, fragmentation results from the nucleation of new cracks at flaws and reinitiation of old cracks from the reflected stress wave systems**
- **gas pressurization does not contribute significantly to the fragmentation process**

Computational models incorporating stress wave/flaw interaction as a mechanism of nucleating and extending cracks is growing in popularity. (32-38, 40) Although the models differ in approach and/or details, the main idea is that shock and/or stress waves fragment material and gas pressure acts to displace the broken material. Stress wave functions not only to initiate fractures at or near the borehole wall, but also initiate fractures throughout the rock mass being blasted.

Recent work in full scale production shots and in large blocks added further insight into this phenomena. (35) Stress wave induced fracturing at flaws and discontinuities removed from the borehole was found to be considerably greater than either spalling or borehole radial tensile failure documented by earlier works. Gas pressurized radial fracturing, in typical bench blasting operation, was found to be only a minor contributor to the overall fragmentation of the rock mass.

Some key points of Winzer's theory and observations are:

i) new fractures are seen to form at the face at about twice the time it takes for the P wave to traverse the burden distance

ii) old fractures are the loci of new fractures or are re-initiated themselves early in the event; they continue to be active for several tens of milliseconds after detonation of the explosive

iii) fragmentation continues in blocks of rock, following detachment from the main rock mass, by trapped stress waves

iv) the fracture pattern on the free face is well developed prior to the expected time of arrival of radial cracks from the borehole

v) in blasted faces from production-scale shots, fractures are observed to have initiated at, and propagated from, joint and bedding planes, suggesting the same operating mechanism(s) as those observed in homolite models at the University of Maryland

vi) gas venting occurs through already open cracks relatively late in the event, indicating that the majority of fractures observed on the free face are not gas pressurized

vii) in more massive rock stress waves are transmitted with higher velocity and less attenuation, but fewer fractures will form because there are few fracture sites. However, more radial fractures will form in massive rock, while fewer fractures form at a distance from the borehole

viii) large fragments will form early in the event, and as they move and fractures open, large segments of the rock mass will be effectively isolated from further stress energy

ix) in more heavily fractured rock, the stress wave velocity will be lower and attenuation higher, but there are more fractures to serve as initiation sites

x) the stress wave takes longer to penetrate the mass, and movement of the rock can be expected to be slower as more energy is absorbed by the rock mass

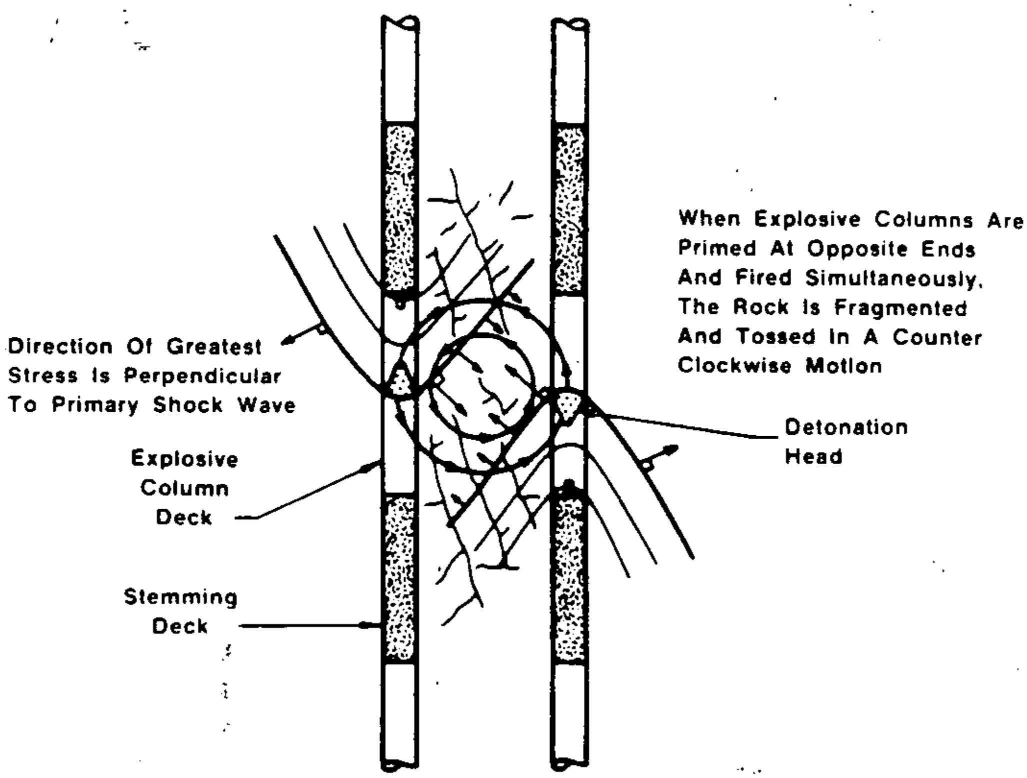
xi) cracks open more slowly, and smaller masses of rock are isolated early in the event, so that later arriving stress waves can continue to increase crack initiation and propagation

g. TORQUE THEORY

The success of this theory is totally dependent on the absolute, accurate timing of initiators. When two adjacent explosive columns are initiated simultaneously from opposite ends, a compressional shock wave from each column traveling parallel but n opposite directions is formed. (Figure 11-15) The greatest stress is always directed perpendicular to the primary shock front. This stress is also assumed to be greatest near the detonation head in the explosive and diminishes with distance away from the detonation head. An uneven stress distribution is formed between explosive columns when the columns are fired simultaneously and from opposite directions. This action tends to toss the fragmented rock between explosive columns in a counterclockwise motion. Reversing the primers of each explosive column will toss the material in a clockwise motion. This action is precisely what is needed to obtain uniform fragmentation and avoid tight muck piles such as in the case of in-situ retorting. For this theory to work, exact initiators are crucial; nothing less will do, especially when using explosives with very high velocity of detonation.

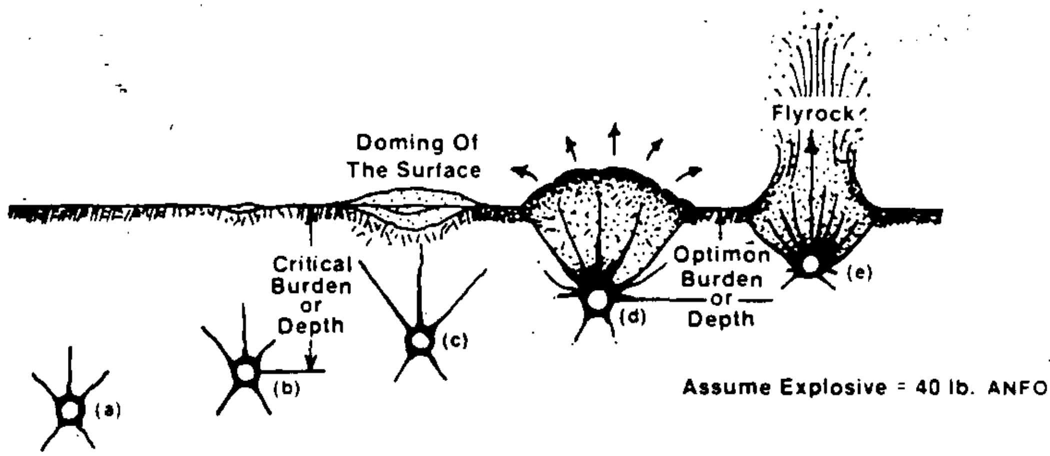
h. CRATERING THEORY (41-45)

The concept of cratering, its development, and resulting applications were originally proposed by C.W. Livingston and later modified by others such as Lang and Bauer. (41) (43) (44) It involves a spherical charge of length to diameter ratio of less than or equal to 6 to 1, detonated at an empiracally determined distance beneath the sur-



APPLICATION OF NEW BLASTING THEORY TO IN-SITU RETORTS BLASTING
FIGURE 11.15

face to optimize the greatest volume of permanently fragmented material between the charge and free surface. This implies that given a specific explosive and material, there exists a burden distance between the charge and free surface which yields the largest crater (Figure 11-16d). This burden is referred to as the optimum burden or depth. Similarly, there exists another burden distance referred to as the critical distance, which is too far below the surface to result in any crater or expulsion of material at the surface, other than minor radial cracks. This is the point where material at the surface just begins to show evidence of failure, (Figure 11-16b).



(a) B = 15'	(b) B = 12'	(c) B = 9'	(d) B = 6'	(e) B = 3'
Completely Contained. Only Failure is Pulverisation Near The Charge and Radial Tensile Failure Running Out From It.	Start of Surface Failure. Burden Not Broken. Some Doming of The Surface.	Surface and Subsurface Failure Almost Meet There Will Be A Shelf of Unbroken Rock Between The Two. Doming or Surface Bulging.	Full Crater. Burden Completely Broken Out. Surface And Subsurface Failures Run Through To The Surface.	Full Crater. Lower Volume Than Optimum Fine Fragmentation. Noise, Flyrock, Bowl Shaped Crater

SCHEMATIC OF THE EFFECT OF DECREASING THE BURDEN ON CHARGES FIRED IN ROCK
FIGURE 11.16

Livingston determined, experimentally and theoretically, that there was a constant factor between this critical burden distance and the cube root of the weight of explosive and expressed it as:

Strain Energy Equation

$$N = E \times W^{\frac{1}{3}}$$

where:

- N = critical distance in feet
- W = weight of explosive in pounds
- E = proportionality constant or the **strain energy factor which has no units and is constant** for one given explosive - rock combination

If a sufficient number of tests are performed as illustrated in Figure 11-16, then the strain energy factor could be calculated. For example if the critical burden was found to be 12 feet when using 40 pounds of ANFO, then

$$E = \frac{N}{W^{\frac{1}{3}}}$$

$$E = \frac{12}{(40)^{\frac{1}{3}}}$$

$$E = \frac{12}{3.42}$$

$$E = 3.51$$

Strain Energy Factor = 3.51

This strain energy factor, E, will differ if the same explosive is used in a different material or the same material is blasted with a different explosive. When rock gets more brittle, E increases and the optimum crater volume occurs at lower values of depth ratio. In softer material, E decreases and the optimum crater volume occurs at higher values of depth ratio.

The strain energy equation can be written in another form that relates the charge depth from surface to the depth ratio, strain energy and explosive weight as:

Upper Limit of Shock Range

$$d_C = \triangle \times E \times W^{\frac{1}{3}}$$

where:

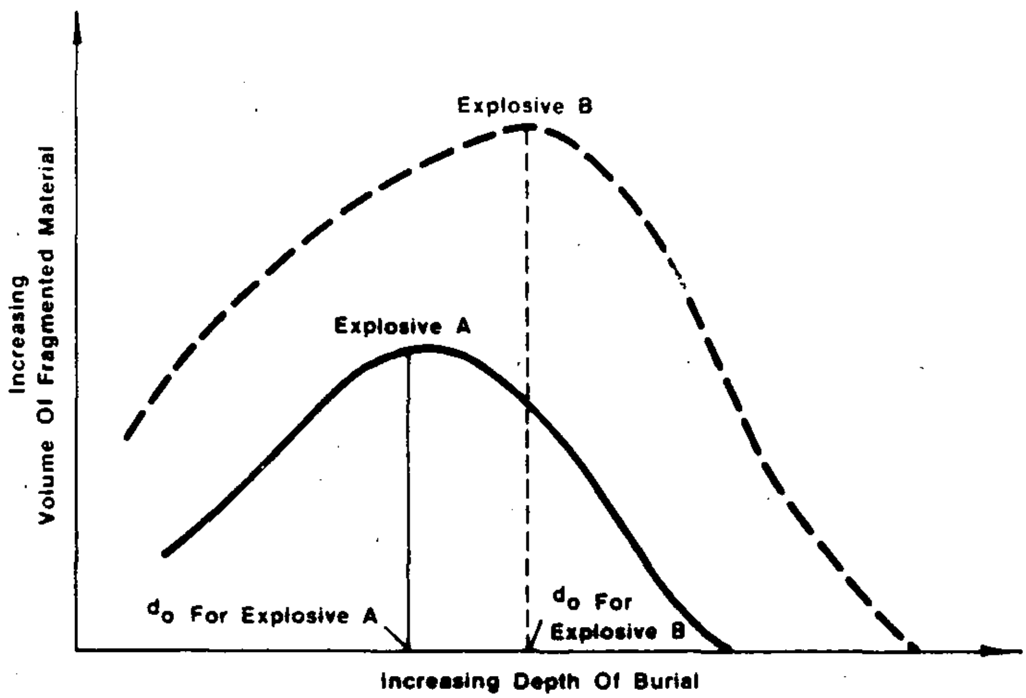
d_C = distance from surface to the center of gravity of the charge in feet

$$\triangle = \text{depth ratio} = \frac{\text{depth of burial}}{\text{critical depth}}$$

W = weight of explosive in pounds

If d_C is the optimum burden that yields the greatest volume of fragmented material, then it is referred to as d_O and the optimum depth ratio is referred to as $\triangle_o$.

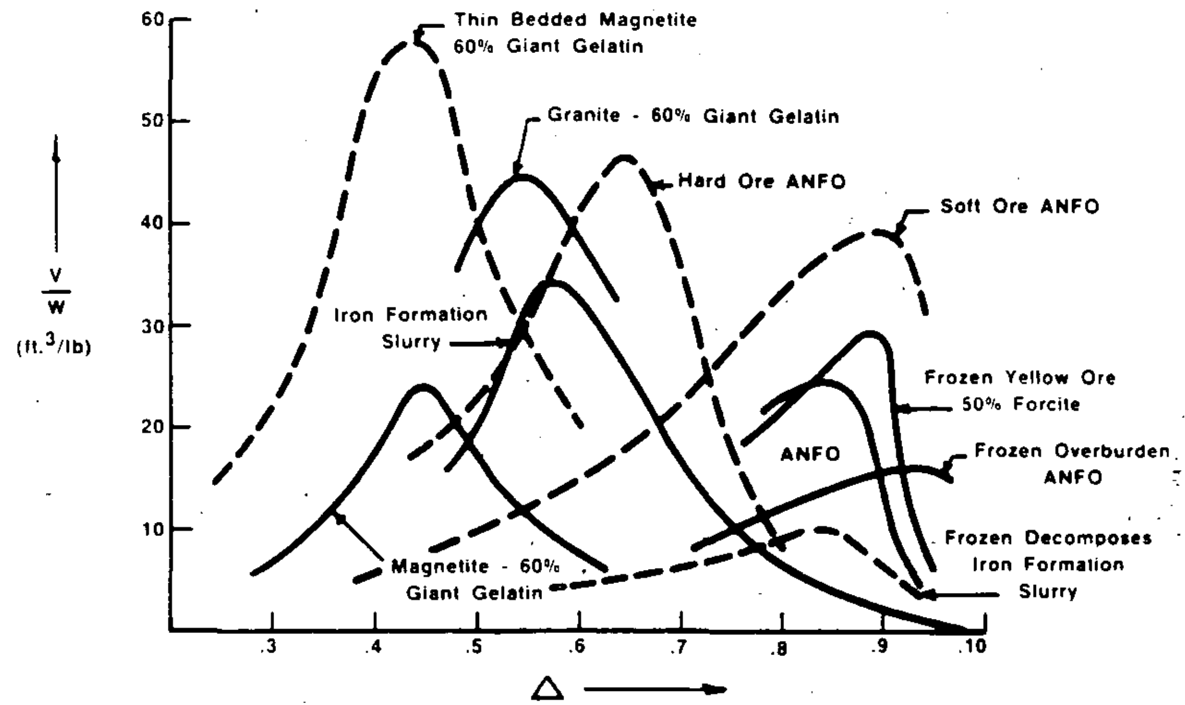
Crater data can be plotted in a number of different ways. Figure 11-17 illustrates the effect of two explosives, A and B on the amount of fragmented material that each is capable of achieving at different depths of burials. Note that the higher energy explosive always fragments a greater volume of material at the same depth of burial as explosive A, but that the optimum depth of burial differs for each explosive.



VOLUME OF FRAGMENTED MATERIAL VERSUS DEPTH OF BURIED FOR TWO EXPLOSIVES IN THE SAME MATERIAL

FIGURE 11.17

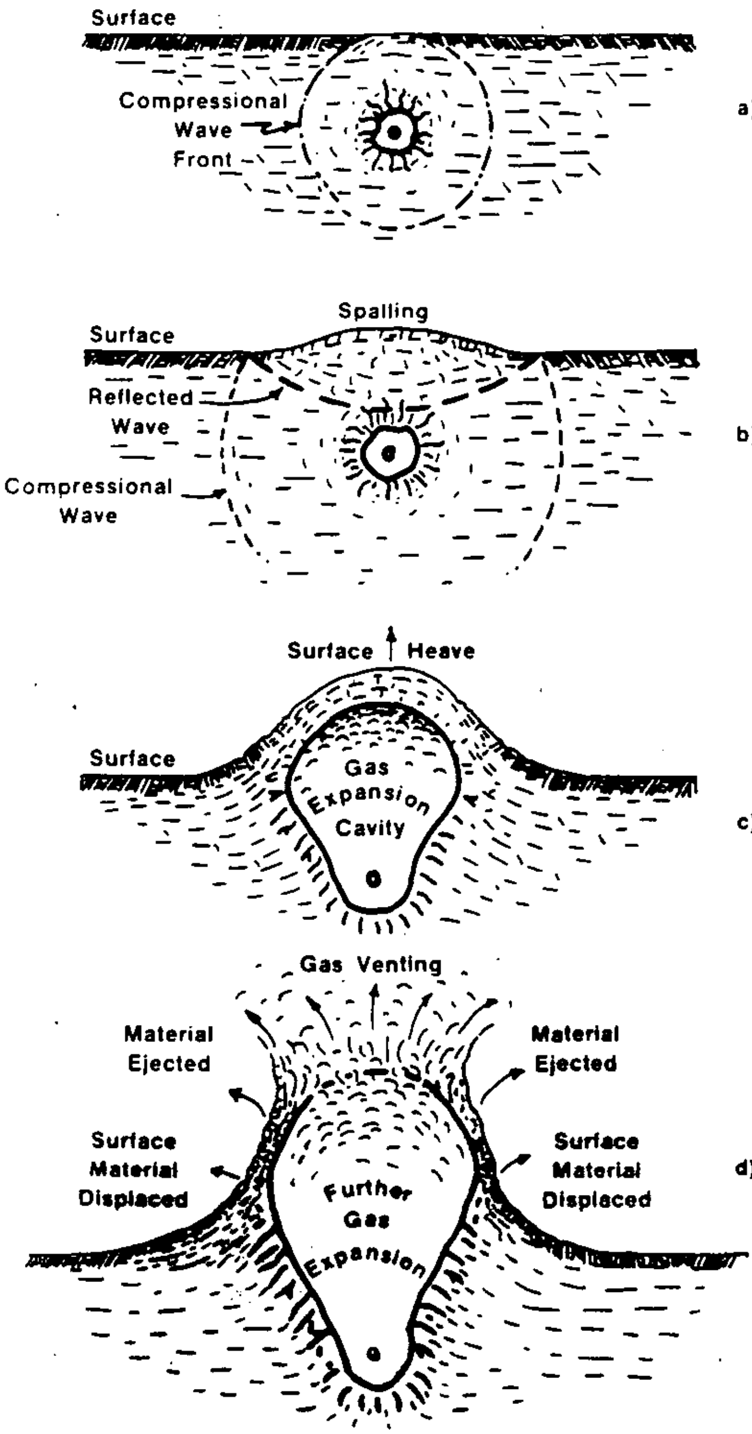
Another method of representing crater data on a common base is by plotting V/W on the y-axis and the depth ratio on the x-axis as shown in Figure 11-18. (44) V is the volume of broken material in cubic feet, W is the weight of explosive in pounds, and the depth ratio has been defined as the depth of burial divided by the critical depth. The important thing to note is that the optimum depth ratio, (Δ_o), varies with each explosive-rock combination. The advantage of performing such field experiments is that one would obtain crater data specifically suited to the user environment for a number of different explosives. Although the curves in Figure 11-18 are fitted as smooth curves, one should remember that some scatter of data is always present and it is important to take this into account for crucial applications of cratering.



ROCK REMOVED IN CU. FT. PER LB. OF EXPLOSIVE VS DEPTH RATIO
FIGURE 11.18 (44)

i. CRATERING MECHANISMS (4) (45)

As the high pressure explosive gases expand against the medium immediately surrounding the explosion, a spherical shock wave is generated causing crushing, compaction and plastic deformation. (Figure 11-19a) For commercial explosives the initial shock pressures are on the order of 100 to 200 thousand atmospheres (one atmosphere = 14.7 pounds per square inch). As the shock front moves outward in a spherically diverging shell, the medium behind the shock front is put into radial compression and tangential tension. This results in the formation of radial cracks directed outward from the cavity. The peak pressure in the shock front becomes reduced due to spherical divergence and the expenditure of energy in the medium. For shock pressures above the dynamic crushing strength



a) • Detonation
Crushing Around Charge
Stress Wave Reaches Surface

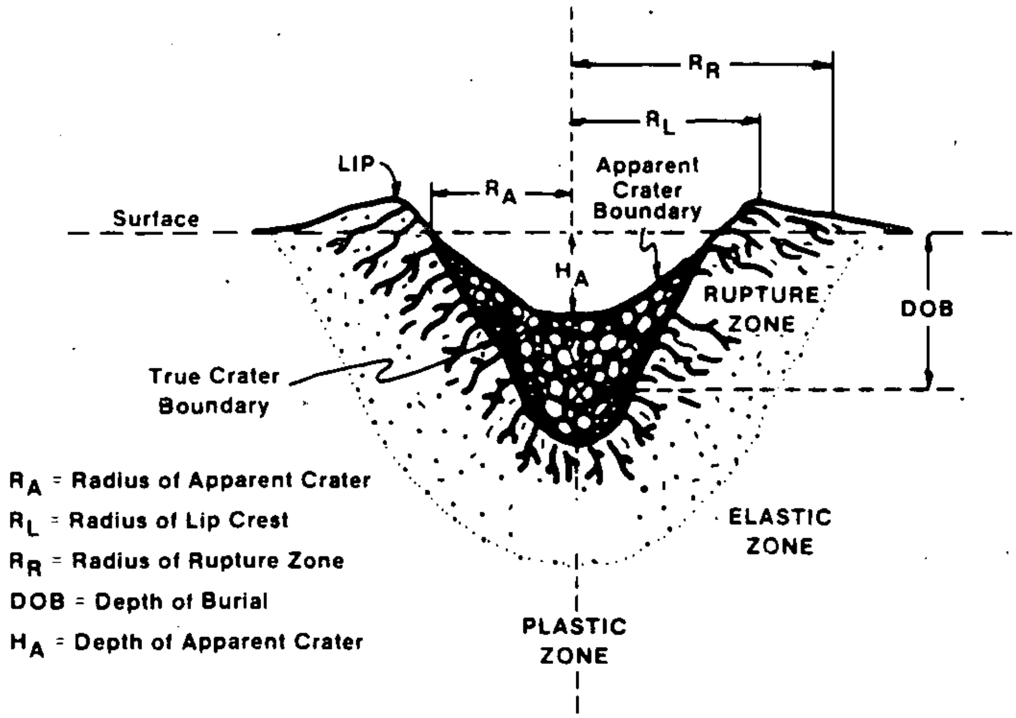
b) • Stress Wave Reflects
at Surface
Some Surface Spalling

c) • Gas Expansion And
Acceleration

d) • Gas Venting And
Material Ejection

CRATERING EVENTS AND MECHANISMS
FIGURE 11.19

of the medium, the material is crushed, heated and physically displaced, forming a cavity. In regions outside this limit the shock wave will produce permanent deformation by plastic flow, until the peak pressure in the shock front has decreased to a value equal to the plastic limit of the medium. This is the boundary between the plastic and elastic zones shown in Figure 11-20.



EMPLOYMENT OF ATOMIC DEMOLITION MUNITIONS
DEPARTMENT OF THE ARMY, WASHINGTON, D.C. AUG. 1971
FIGURE 11.20

When the compressive shock front encounters a free face, it must match the boundary condition that the normal stress or pressure be zero at all times. This results in the generation of negative stress, or rarefaction wave which propogates back into the medium(Figure 11-19b). Thus the medium which was originally under high compression is put into tension by the rarefaction wave. This phenomenon causes the medium to break up and fly upward with a velocity characteristic of the total momentum imparted to it. In a loose soil material, this spalling makes almost every particle fly into the air individually, while in a rock medium the thickness of the spalled material is generally determined by the presence of pre-existing fracture patterns and zones of weakness. As the distance from surface increases, the peak negative pressure decreases until it no longer exceeds the tensile strength of the medium. The velocity of spalled material also decreases in proportion to the peak pressure. This breakage mechanism is predominant only for charges placed at very shallow depths of burial.

The two mechanisms described so far are short term, lasting only a few milliseconds. The gas acceleration mechanism, however, is a much longer lasting process which imparts motion to the medium around the detonation by the expansion of gases trapped in the explosion-formed cavity. (Figure 11-19c and 11-19d) These gases are produced in the surrounding material by vaporization and chemical changes induced by the heat and pressure of the explosion. Venting occurs because the material is no longer cohesive enough to contain the explosion gases. As the gases are released, fragments assume free ballistic trajectories. At depths of burial at which crater dimensions are maximum, the gases produced will give appreciable acceleration to overlying material during its escape or venting through cracks extending from the cavity to the surface. At shallow depth of burials the spall velocities are so high that the gases are unable to exert any pressure before venting occurs. For very deep explosions the weight of the overburden precludes any significant gas acceleration of the overlying material. Gas acceleration is the dominant mechanism at optimum depth of burial. With a constant weight of explosive, the optimum depth of burial varies with the surrounding material.

At deep depths of burial, the mechanism of overburden collapse (subsidence) becomes dominant. This effect is closely linked to the crushing, compaction and plastic deformation mechanism which produces an underground cavity. At these depths of burial, spall and gas acceleration will not impart sufficient velocity to the overlying material to physically eject it from the crater. Most throwout returns to the crater as fallback material. In a rock medium the bulking action of the rock, when it is disoriented from its original fracture pattern, could produce a volume greater than the underground cavity. This could result in no crater or a mound above the ground rather than a crater.

At even deeper depths of burial, about twice or deeper of that of optimum, another type of subsidence occurs. In this case the spall and gas acceleration has no significant effect on the overlying material. Only an underground cavity is formed. When the pressure in the cavity decreases below overburden pressure, the roof of the cavity begins to collapse. In most media this collapse will continue upward forming a chimney of collapsed material. In soil, where the density of the material will not significantly change after it has fallen, the volume of the undergrcund cavity will be transmitted to the surface.

Figure 11-21 illustrates surface time profiles after detonation of a 40 pound equivalent charge of ANFO, buried 8.0 feet in an unconsolidated, sedimentary type material. (46) High-speed photography was

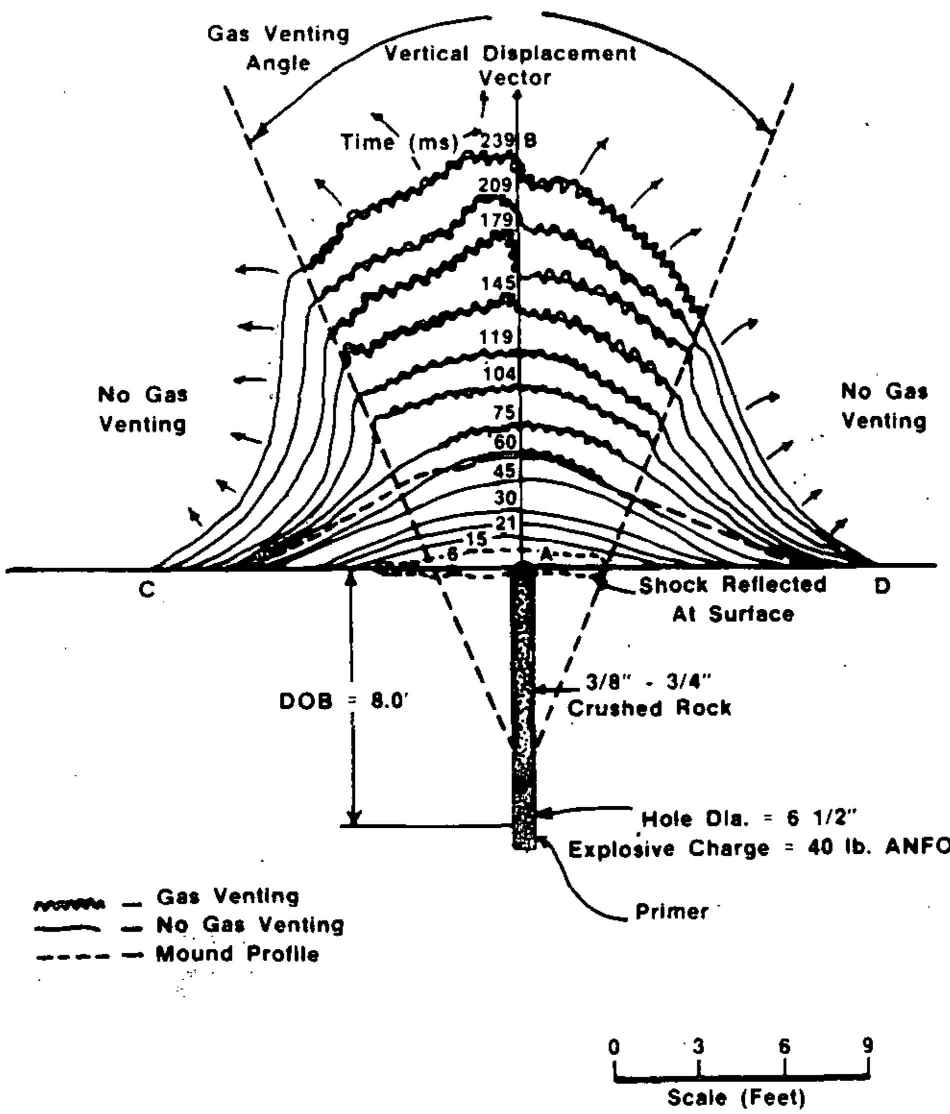
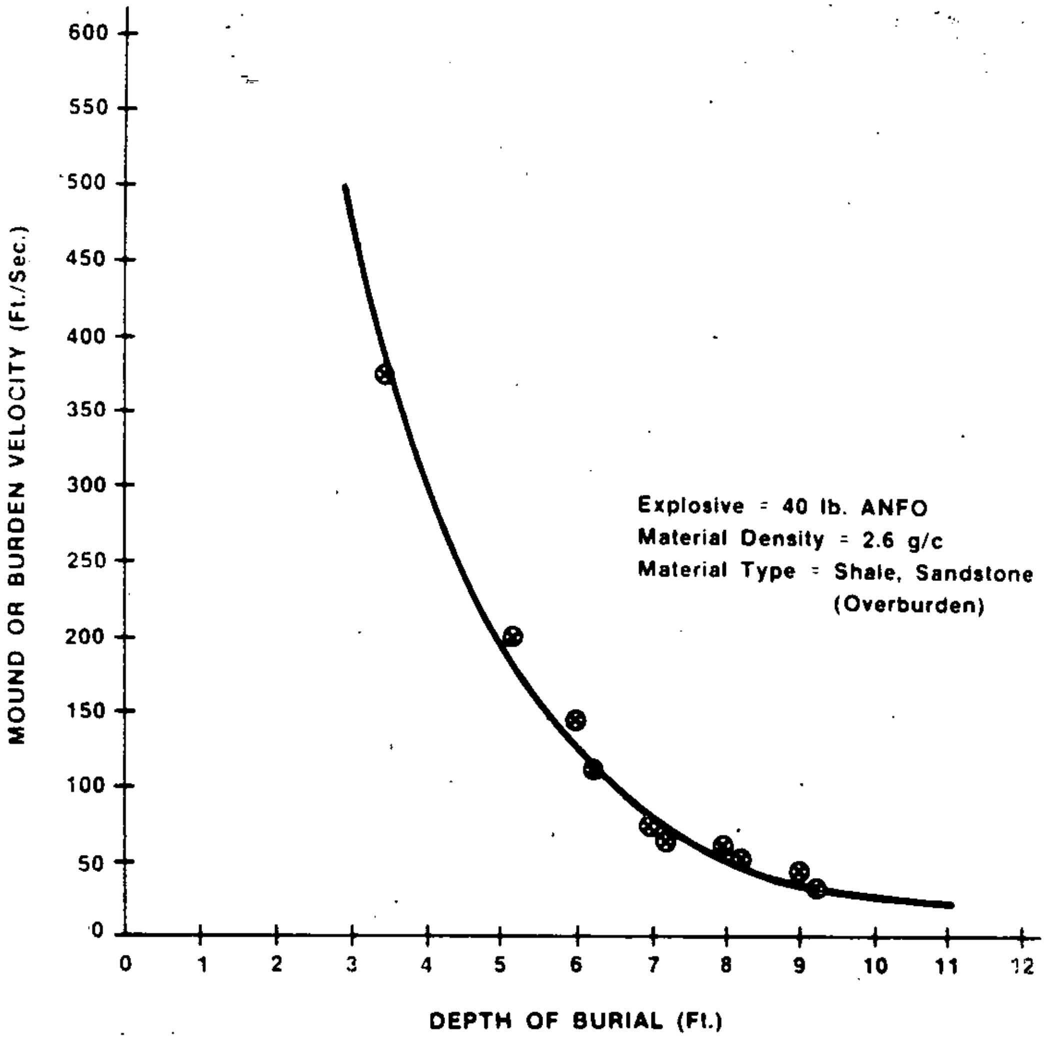


FIGURE 11.21

used to document the effects of shock and gas pressure. The first observation was that of brisance or the reflection of the compressive shock at the surface a few milliseconds after detonation. This is indicated by the dotted eclipse immediately above the charge hole or surface. With sufficient camera coverage and appropriate viewing angles, this shock ring can often be used to estimate, in rough, the degree of crater damage. In this case, sufficient viewing angles were not available and so only part of the total reflected shock could be resolved. Because the charge was placed at a depth significantly greater than the optimum depth of burial, no appreciable spalling occurred. Gas pressure was the dominant mechanism responsible for uplifting and ejecting material radially outward.

As gas expansion occurs around the charge cavity, the material above the charge is compacted and heaved upwards. Between 0 to 45 milliseconds after detonation, the uplifted material is resiliant and compacted enough to maintain sufficient cohesion to contain all gases resulting from expansion. At 60 ms gas venting begins to occur directly above the charge and continues to expand in a well defined arc with respect to time. If the gas venting contacts at each end of each time profile are connected with straight lines, the lines will most always point toward the top or the center of the charge. In this case, the gas venting angle was measured to be approximately 45 degrees. The gas venting angle is useful in determining how much of the top part of a cylindrical charge, as found in production holes, actually contributes to gas venting, cratering and/or lost energy through lack of stemming confinement. At either side of the gas venting angle, no gas venting occurs, but material fragments are displaced and/or ejected outwardly. Material fragments are also ejected from within the bounds of the gas venting angle. Owing to a charge depth beyond optimum, the final result is a mound rather than a crater. The mound is indicated by the shaded section underneath the 60 ms time profile.

The initial instantaneous uplifting velocity above the charge is generally high but diminishes to zero when the material has reached its highest displacement. In reference to Figure 11-21, the average initial velocity along the vertical displacement vector up to 45 ms is 68 ft/sec. The average velocity from 60 ms to 239 ms is 54 ft/sec. The difference in velocity is attributed to the effects of gas venting and expansion beyond 60 ms. These velocities are dependent on material type and structure, explosive and depth of burial. In general, the velocity will decrease exponentially with depth for a given explosive and material type as shown in Figure 11-22. (46)



MOUND OR BURDEN VELOCITY VS. DEPTH OF BURIAL FOR 40 POUND CHARGES OF ANFO
FIGURE 11.22

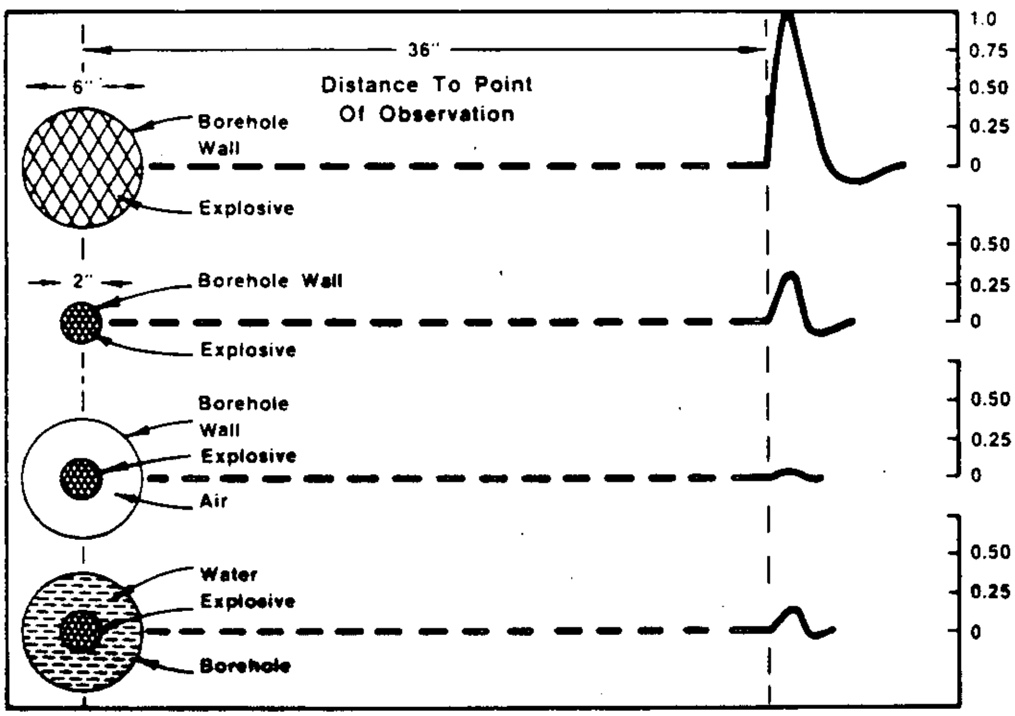
5. DECOUPLING

Decoupling is generally used as a control to reduce backbreak to the final planned excavation limit for pit wall slopes in open pit mines, shafts, drifts, ditches, road cuts and mine benches.

Since the borehole pressure is quite intense for a fully coupled borehole, exceeding many times that of the dynamic compressive strength of the rock, it must be reduced to avoid extensive damage. The three principal modes of rock failure occur by exceeding the dynamic compressive, shear or tensile strengths. Ideally, the borehole pressure should be somewhere between the compressive and tensile strength of the rock, so as to avoid extensive crushing at the borehole wall, yet provide enough pressure to extend a single predominant crack between any two perimeter holes in the control line of holes.

A good example of decoupling in air and water in relation to fully coupled holes is illustrated in Figure 11-23. (47) The pressure imparted in the rock mass at 36" away for the same explosive is shown for four conditions:

i) a 6" diameter explosive in a 6" hole
ii) a 2" diameter explosive in a 2" hole
iii) a 2" diameter explosive in a 6" hole (air decoupled)
iv) a 2" diameter explosive in a 6" hole (water decoupled)



EFFECT OF AIR AND WATER DECOUPLING VS FULLY COUPLED HOLES
FIGURE 11.23 (47)

All measured stress levels are compared relative to the 6" diameter explosive in a 6" diameter hole. A number of important points are immediately evident. The greatest stress level was achieved with a fully coupled explosive in a 6" diameter hole. The next highest stress level was achieved, again, with a fully coupled explosive, even though the hole diameter was reduced three-fold to a 2" diameter. Water decoupling followed next and air decoupling produced the smallest stress level. Thus, an air decoupled charge is the most effective means of reducing borehole pressure and consequently the peak stress level within the rock mass.

A reasonably reliable method of calculating the borehole pressure is with the following formula which takes into account two decoupling ratios. (48) (49) (50)

$$P_b = 1.69 \times 10^{-3} \times \rho \times VOD^2 \times \left[\sqrt{c} \times \frac{de}{dh}\right]^{2.6}$$

where:

P_b = Borehole pressure in PSI.

ρ = Density of explosive in g/cc

VOD = Velocity of detonation in ft/sec

c = Percentage of explosive column loaded expressed as a decimal

de = Explosive diameter (in.)

dh = Hole diameter (in.)

This formula is best suited for explosives which contain no metallic elements or relatively small amounts, since the addition of energizing metals lowers the detonation velocity of the explosive and hence, the borehole pressure as calculated by this equation. Computer codes such as TIGER and EXPLODE are used to calculate borehole pressures from explosives containing metallic elements.

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VI CURSO INTERNACIONAL DE INGENIERIA GEOLOGICA APLICADA A OBRAS SUPERFICIALES Y SUBTERRANEAS

MODULO IV: TECNOLOGIA SOBRE EL USO DE EXPLOSIVOS

THE. MECHANICS OF TOCK BREAKAGE

EXPONENTE: ING. RAUL CUELLAR BORJA

The Mechanics of
ROCK BREAKAGE

By RICHARD L. ASH, P.E.

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The Mechanics of

Part I

IN quarrying, the profitability of an operation is directly controlled by the blasting, because it is at the face that the production cycle begins. Poor blast results invariably will lead to economic difficulties. In addition, the frequent changes and complexity of operating conditions force operators to struggle continually with their problems, often without reaching satisfactory solutions. The usual trial-and-error approach as such is expensive and often hazardous, and it rarely leads to complete success because of a lack in flexibility of application. Also information that is generally available on blasting is not usually applicable from the practical viewpoint.

For these reasons certain basic standards have been developed to assist producers in the design and evaluation of their blasting. It is the purpose of this discussion, therefore, to describe those guidelines and show how they can be applied, in order that normal blasting difficulties might be reasonably avoided.

There are two fundamental effects from blasting that must be controlled: fragmentation and displacement. For the first effect, uniformity of particle-size distribution and the limits of actual sizing are the two important qualities. Usually reasonably uniform sizing is preferred, too many fines or too many slabs being undesirable. Similarly, for the second effect, rock movement, too little or too much displacement is not wanted for economic and safety considerations. The two effects always become problems if overbreak occurs. Air blast and objectionable ground vibration are also problems that can lead to serious difficulties if uncontrolled. Thus, to direct these effects properly and apply the basic standards successfully, one should first have a working knowledge of the blasting process itself.

THE MECHANICS OF ROCK BREAKAGE

Rocks are normally more resistant to failure by compression, or crushing, than they are to being separated by tension. For example, limestones as a group may have compressive strengths of 3,500 to 25,000 psi, but they may have tensile strengths as low as 500 to 2,500 psi. In addition, the ordinary high explosives and blasting agents normally used in blasting produce very high pressures at extremely rapid reaction velocities, which may be from 8,000 to

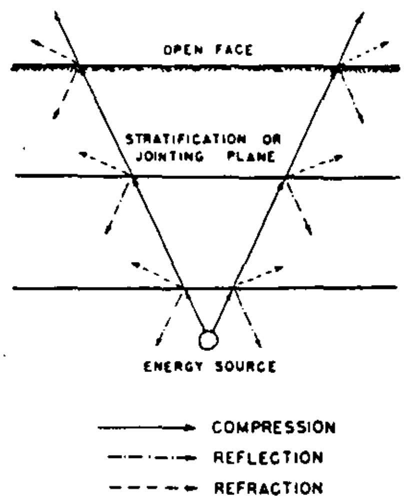

Figure 1—Energy reflection and refraction force components at density interfaces.

26,000 fps (5,300 to 17,000 mph). The rapidly developed pressures in blastholes may be as low as 250,000 psi or in excess of 2,000,000 psi, depending on the particular type of explosive and the conditions under which it is used. The effect of explosives reacting on rocks, then, is one of impact, or impulse, from a quickly applied blow of extremely high intensity.

When explosive charges are used in circular blastholes, the sudden application of high pressures into the surrounding rock is exerted equally in all directions along the blasthole perimeter. The rock in that region is quickly compressed, usually crushing the rock for a limited distance. The sudden application and following quick release of high pressure introduces a compressive stress-wave that quickly spreads throughout the rock mass as an elastic wave. This action results because most rocks are characterized by some brittleness and are therefore somewhat elastic. The particular speed at which the energy travels through the rock is a function of the rock's density, denser materials transmitting compressive-wave energy at high rates and the porous or lighter rocks at relatively low speeds.

For simplicity, one might visualize the wave effect as being similar to that achieved by dropping a pebble into a pond of water. As with the waves in water when they encounter a shoreline, some of the compressive-wave energy from the explosive transmitted through the rock is reflected and refracted (bent) at all changes of density or structural discontinuities (Figure 1). Any open face, change of rock type, etc., will produce this effect. The remainder of the energy, however, tries to continue along its original travel direction. The angle of travel

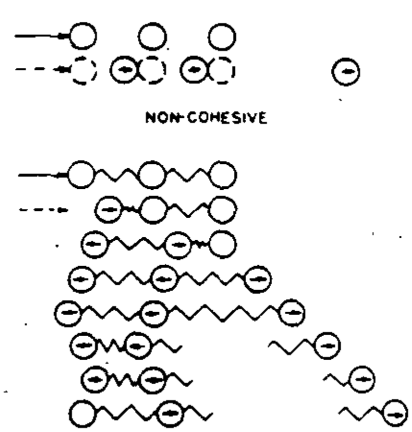

Figure 2—Energy transmission in materials from impulsive loads.

ROCK BREAKAGE

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direction of the reflected energy is the same in value but opposite to the direction of the energy imparted at the boundary, the direction of energy refracted into the next material being a function of the characteristics of both materials. Thus, at every change of density some of the impulsive energy is reflected and refracted, the balance continuing to travel in its initial direction through the second material.

The action of energy transmission is more easily understood if one first considers the material being blasted as being made of many small particles (Figure 2). If a blow is exerted on one particle, we could expect the energy to be transmitted in the direction of the applied blow to adjacent particles, until the energy is eventually consumed as a result of work-performing effects such as friction, dampening, fragmentation, etc. Particles in a pile of sand are noncohesive; so there is little or no attraction between the particles, even though each may have a certain amount of elasticity within itself. Most rocks, however, are cohesive as well as somewhat elastic, thus promoting a different effect from that occurring in loose materials.

For the noncohesive particles, the one on the outside of the pile, on receiving a blow from an adjacent one inside, would endeavor to keep traveling outward, since there are no particles remaining to impede its movement. The cohesive material, on the other hand, would have the outer particles held to adjacent ones, as if by springs. If the blow is sufficiently strong, the inertia of the outer particles will tend to keep them moving outward, once the energy has been applied to them, the springs then being placed in tension. If the tensile strength of the springs is exceeded, they will break. The sudden release of tension will in turn cause the adjacent particles toward the inside of the mass to rebound. As each particle is acted upon in this fashion, beginning at the open face, the springs will be broken in subsequent order back to the source of the initial blow, provided that there is enough energy remaining to exceed the tensile strength of all of the springs.

Thus, the stressing action of breaking rock begins at a free surface, or change in density, and moves back in toward the explosive charge. The problem for proper fragmentation, then, is to be certain there is sufficient applied energy to permit travel outward from the explosive charge and return, with sufficient strength to exceed the tensile strengths of the rocks along the entire path of travel.

Since blastholes are circular, the energy propagation will spread out in distance from the source, or as a fan. This action causes the energy travel in particles to move in different directions. In addition, stresses developed in the walls of blastholes will decrease rapidly as the energy pulses travel away from the charges. There will be only one direction, that perpendicular to a free face and usually called the burden, where energy will be the strongest and first to reach the boundary surface. Energy from the explosive charge will continually weaken and will reach outer particles along the face at later intervals in progressive order.

Fly rock velocity will be greatest at the center point, where the energy travel distance is least; on either side, particles will have less energy imparted to them and will have a progressively greater lateral action as distance is increased from the center. The appearance of the face assumes the shape of a large bubble opposite the charge, with the outermost edge stretched in lateral tension (Figure 3). As a result of this action a crater forms, caused by the combination of tensile effects *along* the energy travel paths from the charge outward *and* those between particles *laterally* because of the diverging action imposed by the differences in energy travel directions.

The outline of the excavation and fracture pattern within the cratered portion are influenced strongly by the structural planes of weakness in the rock mass, such as slips and joints. Whether or not there is enough energy to travel outward *and*

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The author of this article is a third-generation mining engineer, who received his formal education in this field at the Pennsylvania State University and his extensive experience in working for Atlas Chemical Industries, Inc. (Explosives Division) and later with firms in the construction, mining, quarrying, and seismic prospecting industries. He served as a naval engineering officer in the Pacific theatre from 1942 to 1946.

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Mr. Ash is a reserve officer with the United States Civil Engineers Corps and a member of the following: AGI, AIME, ASEE, MSPE, NSPE, SAME, SGE, and SME. He has published articles and presented addresses on equipment performance, explosives, and blasting applications.

return must be determined for each blasting situation. If the amount of initial explosive energy is inadequate for the total travel distance, so that the tensile strengths are not exceeded both outward and on return, one can expect to find the unbroken rock, or very coarse breakage, *inside* the broken rock pile, nearest the location of the blasthole.

Where excess energy is used, the broken rock will be thrown farther out from the face, and there may be some overbreak in back of holes and on the edges. On the other hand, if slabs or boulders are found on the *outside* of the pile after blasting, it is most likely because the ledge was cracked before the blast was made, from earlier overbreak, or because mud seams or similar density changes existed in the rock mass. Cracks, or density changes, serve to reflect and refract energy before it reaches the outer free face, with a subsequent reduction in energy levels passing through, the outside portions therefore being merely pushed out from the face.

For most field blasting, more than one free face will exist, i.e., a bench or ledge is present. The addition of a third free face, such as a corner, will alter the crater effect (Figure 4). Since the relative distances to open faces from a charge determines which face is stressed first, too large a difference in distances often gives humps, toes, or very coarse fragmentation in the area with the longest distance. Full cratering with overbreak will occur on the other side, where energy travel is the least, even though a corner may be present.

In that blastholes are much greater in length than they are in width, the effects from the explosive reaction along the blasthole must also be considered, cylindrical rather than spherical effects being the usual condition. Figure 5 illustrates blastholes in a ledge with pertinent terminology described, while Figure 6 gives wave forms in rock resulting from the cylindrical effect.

It is apparent from the sketches that the time when the compressive-energy wave in rock first arrives at an open face will be different for each blasting situation. The shape of the wave will vary from that of a sphere to a cone, the actual shape of which is a function of the explosive's reaction velocity (v_r) to that of en-

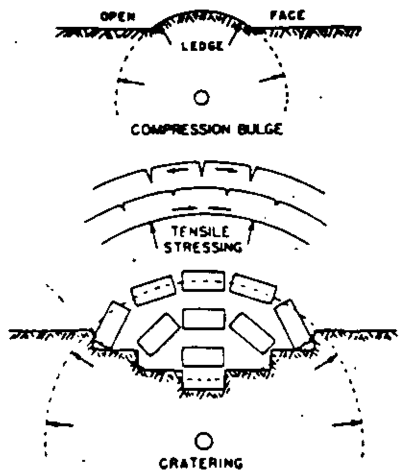


Figure 3—Sequence of actions in crater formation.

ergy travel in the rock (v_r), usually expressed as the K_v or velocity ratio.

The primer location will determine that portion of the ledge which will be stressed and displaced first. As hole depths increase, the difference in blast effects will become greater. Collar priming usually pro-

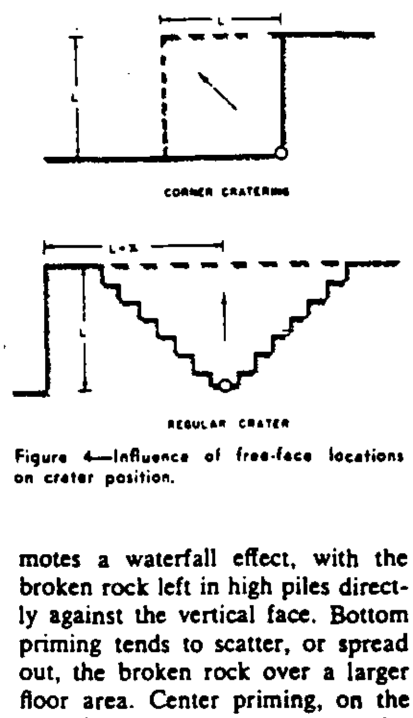


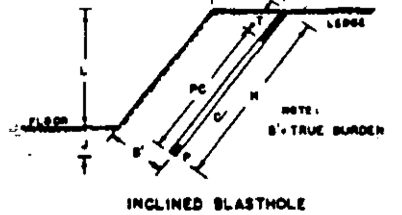
Figure 4—Influence of free-face locations on crater position.

motes a waterfall effect, with the broken rock left in high piles directly against the vertical face. Bottom priming tends to scatter, or spread out, the broken rock over a larger floor area. Center priming, on the other hand, produces a compromise effect. Collar and bottom priming, when used together in the same blasthole, will tend to increase the stressing in the ledge center, thereby intensifying the fragmentation and displacement actions.

The influence of gravity, or static loading, has little or no practical effect on fragmentation under most blasting conditions. However, for vertically drilled blastholes the higher the ledge, the proportionately greater the resistance to displacement of rock at ledge bottoms. Since the pressure waves produced in the rock from every point along an explosive column cannot reach the vertical and horizontal free faces at the same time, it is most often preferred that stressing begin at the base of the vertical free face. This is usually because of the need for adequate displacement to insure easy and safe digging.

Blastholes that are inclined (Figure 5) help to compensate for weight effects as well as to extend the effective area for stressing in the vicinities of hole collars and bottoms. Boulders most often come from those areas. It has been shown that the greater the angle of inclination the better geometrically proportioned becomes the stemming zone for cratering, thus reducing backbreak effects. But air blast and possible violence are more likely to occur since the volume of rock is appreciably reduced in the stemming region. Thus, less dense explosives would be preferred in collar areas. It should be noted, however, that stressing in portions of the ledge other than at the collar and floor level will be no different, regardless of the hole inclination, provided that the bench face parallels the charge column.

Figure 5—Blasthole terminology.



B = BURDEN, C = MAIN CHARGE, P = PRIMER
T = STEMMING, L = LEDGE HEIGHT, J = SUBDRILLING
PC = CHARGE LENGTH, H = HOLE DEPTH

The Mechanics of ROCK BREAKAGE

STANDARDS FOR BLASTING DESIGN

Part II of a Series

IT is not enough just to understand what happens during blasting. Probably the most important thing to the average person is to know how blast effects can be controlled to suit the requirements of his operation. In this respect there are available five basic standards upon which to evaluate blasts, all of which are unitless (dimensionless) ratios. They can be applied to both underground and surface blasting with equal success. For simplicity, however, their use will be discussed as applied to surface (open-pit) blasting conditions. The standards are defined as follows:

1. *Burden Ratio* (K_B)—the ratio of the burden distance in feet to the diameter of the explosive in inches, equal to 12 B/D.

2. *Hole-Depth Ratio* (K_H)—the ratio of the hole depth to the burden, both measured in feet, or H/B

3. *Subdrilling Ratio* (K_J)—the ratio of the subdrilling used to that of the burden, both expressed in feet, or J/B

4. *Stemming Ratio* (K_T)—the ratio of the stemming, or collar distance to that of the burden, both being in feet, or T/B

5. *Spacing Ratio* (K_S)—the ratio of the spacing dimension to that of the burden, both measured in feet, or S/B.

Burden Ratio The most critical and important dimension in blasting is that of the burden. There are two requirements necessary to define it properly. To cover all conditions, the burden should be considered as the distance from a charge measured perpendicular to the nearest free face *and* in the direction in which displacement will most likely occur. Its actual value will depend on a combination of variables, including the rock characteristics, the explosive used, etc. But when rock is completely fragmented but displaced little or not at all, one can assume the critical value has been approached. Usually, an amount slightly less than the critical value is preferred by most blasters.

There are many formulas that provide approximate burden values, but most require calculations that are bothersome or complex to the average man in the field. Many also require knowledge of various qualities of the rock and explosives, such as tensile strengths and detonation pressures, etc. As a rule, the necessary information is not readily available, nor is it understood.

A convenient guide that can be used for estimating the burden, however, is the K_B ratio. Experience shows that when K_B—30, the blaster can usually expect satisfactory results for average field conditions (Table 1). Thus, for a 3-in. diameter explosive, a 7½-ft. burden (30 x 3/12) would be a reasonable approximation. To provide greater throw, the K_B value could be reduced below 30, and subsequent finer sizing is also expected to result.

Light density explosives, such as field-mixed AN/FO mixtures, necessarily require the use of lower K_B ratios (20 to 25), while dense explosives, such as the slurries and gelatins, permit the use of a K_B near 40. The final value selected should be the result of adjustments made to suit not only the rock and explosive types and densities but also the degree of fragmentation and displacement desired.

To estimate the desired K_B value, one should know that densities for explosives are rarely greater than 1.6 or less than 0.8 gm/cc. Also, for most rocks requiring blasting, the density in gm/cc rarely exceeds 3.2, nor is it less than 2.2, with 2.7 (165 lb. per cu. ft. in the solid) by far the most common value. Thus, by first approximating the burden make simple estimations toward 20 at a K_B of 30, the blaster can then

Table 1—Standard Blasting Ratios for Vertical Blastholes
(All Types of Surface Blasting, 20 Different Rock Types, Hole Depths From 5 to 260 ft., and Hole Diameters From 1⅝ to 10⅝ in. for All Grades of Explosives)

All Operations				All Operations but Coal Strippings			
K_B		K_H		K_J*		K_T*	
Group	Frequency	Group	Frequency	Group	Frequency	Group	Frequency
						0.10-0.19	0
		0.0-0.9	0			0.20-0.29	6
10-13	0	1.0-1.9	43			0.30-0.39	12
14-17	5	2.0-2.9	70	0.00-0.09	15	0.40-0.49	18
18-21	13	3.0-3.9	56	0.10-0.19	18	0.50-0.59	18
22-25	51	4.0-4.9	45	0.20-0.29	27	0.60-0.69	25
26-29	74	5.0-5.9	22	0.30-0.39	26	0.70-0.79	19
30-33	66	6.0-6.9	22	0.40-0.49	25	0.80-0.89	13
34-37	44	7.0-7.9	11	0.50-0.59	2	0.90-0.99	6
38-41	20	8.0-8.9	4	0.60-0.69	6	1.00-1.09	14
42-45	7	9.0-9.9	2	0.70-0.79	2	1.10-1.19	7
46-49	4	10.0-10.9	8	0.80-0.89	0	1.20-1.29	7
50-53	0	11.0-11.9	0			1.30-1.39	3
		12.0-12.9	1			1.40-1.49	2
						1.50-1.59	2
Total	284	Total	284	Total	125	Total	152
Mean	30	Mean	4.0	Mean	0.28	Mean	0.74
Mode	38	Mode	2.6	Mode	0.24	Mode	0.65
Median	29	Median	3.4	Median	0.27	Median	0.67

*Note—Rf: Ash, R. L., and Pearse, T. E.—"Velocity, Hole Depth Related to Blasting Results," *Mining Engineering*—September, 1962, p. 75.

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(or 40) to suit the rock and explosive characteristics, densities for the latter exerting the greater influence.

Thus, for light explosives in dense rock, use $K_B = 20$; for heavy explosives in light rock, use $K_B = 40$; for light explosives in average rock, $K_B = 25$; for heavy explosives in average rock, $K_B = 35$, etc. Figure 7 illustrates the relationships between burdens and explosive diameters and can be used to approximate values for quick estimations. It should be noted, however, that the burden must be more carefully selected for small-diameter blastholes than for the larger charges, a fact well confirmed by field experience.

Hole-Depth Ratio As a rule, a blasthole should never be drilled to a depth *less* than the burden dimension, if overbreak and cratering are to be avoided. The primer location and the K_v ratio (Figure 6) have an important influence on the minimum required depth, in that the shape and direction of the wave form determines where and which face is stressed first. In practice, blastholes are generally drilled from 1½ to 4 times the burden dimension; and blasting is done most frequently with a K_H value of 2.6 (Table 1).

One could then presume that when using a 3-in. explosive of average density in normal rock with a 7½-ft. B, a hole depth from 10 to 30 ft. would normally give satisfactory results. As the depth increased beyond 30 ft., displacement problems could result, leaving toes or bootlegs (part of the hole left intact) because of the failure to pull the full ledge height. Inclined drilling will help to eliminate some of the difficulty. But a hole depth less than the burden, 8 ft., for example, could always be expected to be violent and to produce overbreak in back of holes.

Subdrilling Ratio The primary reason for drilling blastholes below floor level (or grade) is to insure that a full face will be removed. Uneven floors caused by humps and toes generally create problems for later blasting, as well as in loading and haulage operations. For most conditions, the required subdrilling (J) should never be less than 0.2 the burden dimension, a K_J of at least 0.3 being preferred for quite massive ledges (Table 1).

The amount of necessary overdrilling logically depends upon the structural and density characteristics of the ledge, but also on the direction of the blastholes, in that inclined holes require less subdrilling and horizontal holes no subdrilling whatsoever. Under certain conditions no subdrilling is required also for vertical holes, as would be the case for many coal strippings or rock quarries having a pronounced parting at floor level. However, for relatively massive rock drilling, at least

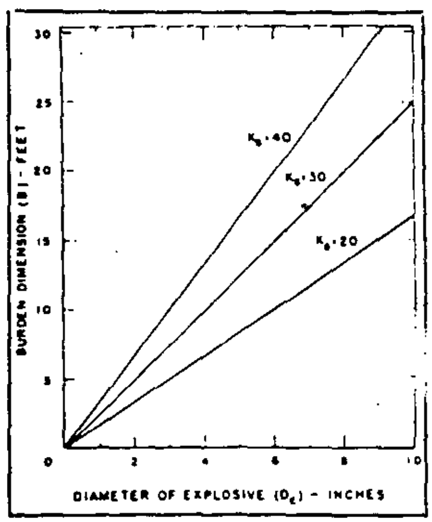
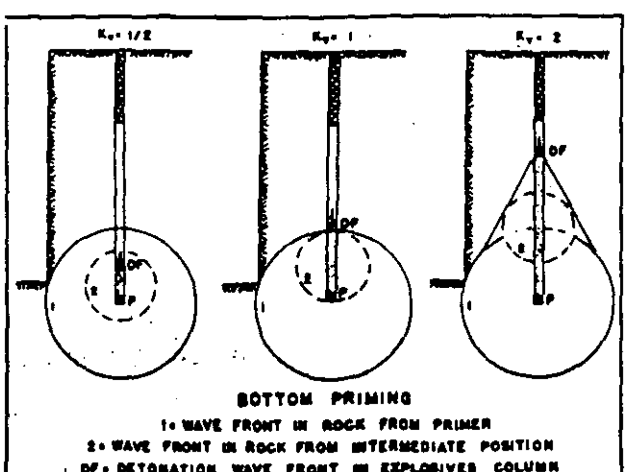
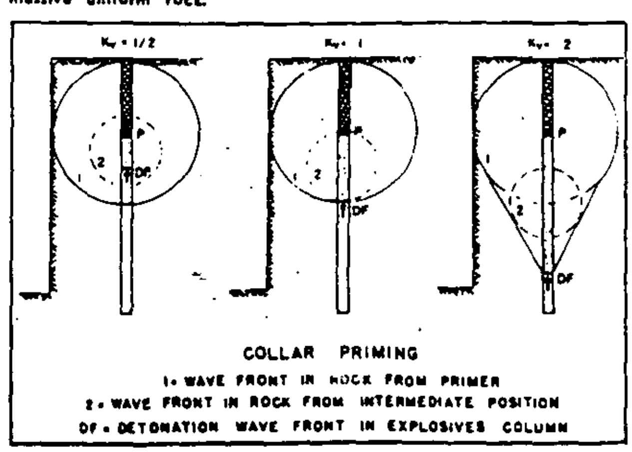


Figure 7—Relationships between burden dimension and explosive diameter.

Figure 6—Compressive stress wave-forms in massive uniform rock.





0.3 the burden below the floor will insure that full ledge heights are obtained, provided, of course, that a proper K_H value is also used. Thus, for the 3-in. explosive and 7½-ft. burden, the blasthole should be drilled at least 2½ ft. below floor level.

Stemming Ratio Collar and stemming are sometimes used to express the same thing. However, stemming refers to the filling of blastholes in the collar region with materials such as drill cuttings to confine the explosive gases. But stemming and amount of collar, the latter being the unloaded portion of a blasthole, perform other functions in addition to confining gases. Since an energy wave will travel much faster in solid rock than in the less dense unconsolidated stemming material, stressing will occur much earlier in the solid material than compaction of the stemming material could be accomplished. Thus, the amount of collar that is left (T), whether or not stemming is used, determines the degree of stress balance in that region. The use of stemming material then assists in confining the gases by a delayed action that should be long enough in time duration to permit their performing the necessary work before rock movement and stemming ejection can occur. For stress balance in bench-blasting of massive material, the value of T should equal the B dimension (Figures 5 and 6).

Usually a K_T value of less than 1 in solid rock will cause some cratering, with back break and possible violence, particularly for collar priming of charges. However, if there are structural discontinuities in the collar region, reflection and refraction of the energy waves reduce the effects in the direction of the charge length. Thus, the K_T value can be reduced under such circumstances, the amount depending upon the degree of energy reduction at the density or structural interfaces. Field experience shows that a K_T value of 0.7 is a reasonable approximation for the control of air blast and stress balance in the collar region (Table 1). Thus, for the 3-in. explosive using a 7½-ft. burden, 5 to 6 ft. of collar with suitable stemming is generally satisfactory.

Spacing Ratio Commercial blasting usually requires the use of multiple blastholes, making it necessary for blasters to know whether or not there are any mutual effects between charges. If adjacent charges are initiated separately (in sequence), with a time-delay interval of sufficient length to permit each charge to complete its entire blasting action, there will be no interaction between their energy waves (Figure 8).

However, if the time interval for initiating adjacent charges is reduced, complex effects will result. There might be reinforcement or cancellation of forces, depending upon the force magnitudes and directions at their point of interference. For charges initiated simultaneously, or at extremely short-delay intervals, the reinforcement action increases with larger angles of force collision. This action promotes greater ground vibration force-effects. However, as described earlier, the energy levels of stresses in the rock are reduced by the fan effect as distance from the source of energy increases. The mutual reinforcement action then tends partially to minimize the energy reduction because of fan effect reductions, thus permitting greater spacings to be used between blastholes initiated simultaneously than when delayed.

The manner in which the zone of rock between holes is broken depends then not only on the particular initiation-timing system used but also on the spacing dimension. Ideal energy balancing between charges is usually accomplished when the spacing dimension is nearly equal to double that of the burden ($K_S=2$) when charges are initiated simultaneously. For long-interval delays, the spacing should approximate the burden, or $K_S=1$. For short-period delays, the K_S value will vary from 1 to 2, depending upon the interval used. However, since structural planes of weakness such as joining, etc., are not actually perpendicular to one another, the exact value for K_S normally will vary from 1.2 to 1.8, the preferred value of which must be tailored to local conditions.

Most difficulties resulting from blasting can be attributed to the use of an unsuitable K_S relationship.

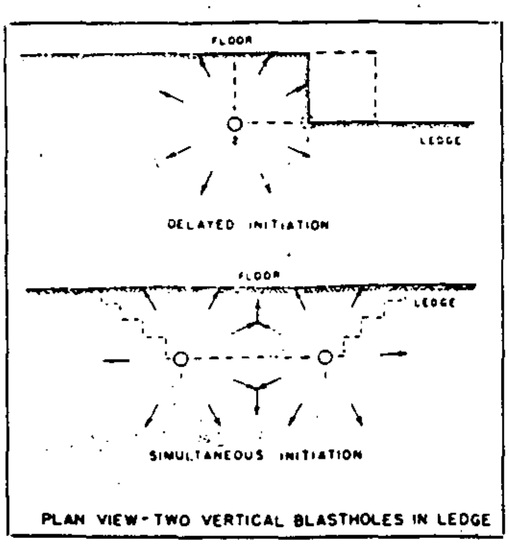
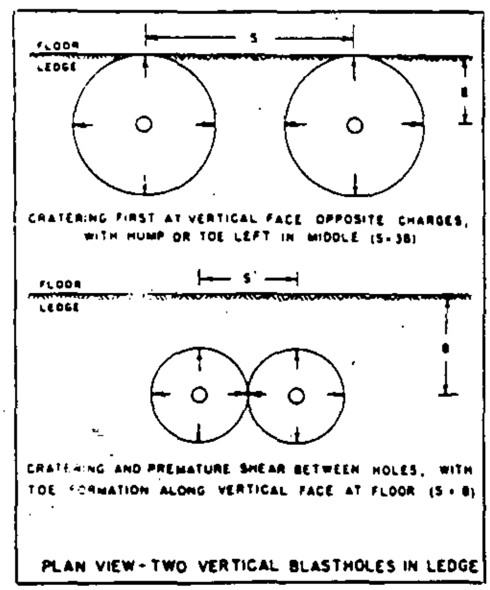


Figure 8 — Compressive-pulse force interactions for multiple-charge initiation (balanced K_S ratios).

Figure 9 — Compressive-pulse wave-positions at instant cratering begins. (Simultaneous initiation of multiple charges with unbalanced K_S ratios.)



For example, from Figure 9, illustrating compressive-pulse wave positions, one can see that when fracturing begins for simultaneous initiation, extended spacings (K_s greater than 2) always lead to horizontal cratering. The action always leaves humps at floor level between the blastholes. Too close a spacing, on the other hand, causes premature shearing between holes. This condition produces finely broken rock between holes, providing all the explosive reacts, but with boulders or slabs formed in the burden zone.

Premature shear and related loss of confinement further promotes volume changes, with subsequent pressure drops in the blasthole region, which for the relatively insensitive blasting agents may kill the reaction completely and result in a misfire. The action also usually loosens stemming too early and permits the release of gases out through the collar regions. Unless deliberate shearing is desired, as for pre-splitting when charge loads should be reduced and fairly sensitive explosives are used, normal blasts exhibit vertical cratering, overbreak, violent fly rock, nonuniform breakage, and toes at floor level.

It can be generally assumed that uniformity of sizing is a direct result of the K_s ratio. If on firing a single hole the rock is satisfactorily broken and cleanly removed without excessive displacement, it may be assumed the burden is satisfactory. Too often blasters reduce the burden rather than extend the spacing in their desire to eliminate boulders or to make rock sizing more uniform.

The basic principles for spacing selection apply to all multiple-charge blasts, as long as all holes are drilled parallel and in the same direction relative to one another. Figure 10 illustrates the basic drill patterns for most field conditions and may be summarized as follows: (1) for sequence delays in the same row, the K_s should be near 1; (2) for simultaneous initiation of holes in the same row, the preferred K_s is near 2; (3) for sequence timing in the same row and simultaneous initiation *laterally* between holes in adjacent rows, the entire blast should be drilled in a square arrangement in order to avoid stress unbalance; and (4) staggered drill patterns are preferred between rows within which all charges are initiated simultaneously.

It should be noted that the actual (or true) burden may be different from that normally considered for each separate blasting condition, if we take into account the fact it should be measured *in the direction* in which displacement occurs. For example, in Figure 5 the true burden for an inclined hole is not actually the horizontal distance, since stressing from wave travel will occur earliest at a point on a line perpendicular to the free face (B′). Thus, the normally considered horizontal burden can be extended by inclining the blasthole even though the true burden would be the same as that discussed previously (K_s=20 to 40).

Table 2—Normal Drill-Pattern Dimensions for Average Blasting Conditions (All Values in Feet Except for Explosive Diameter)

D_i: Inches	B	J	T	L (Max.)*	Equivalent Patterns: Staggered (Simultaneous Timing)	Equivalent Patterns: Square (Sequence Timing)
1	2½	1	2	10	2½ x 4	3 x 3
2	5	2	4	20	5 x 9	7 x 7
3	7½	2½	5	30	7½ x 13	10 x 10
4	10	3	6	40	10 x 18	13 x 14
5	12½	4	8	50	12½ x 22	16 x 16
6	15	5	10	60	15 x 27	20 x 20
7	17½	5½	12	70	17½ x 31	23 x 23
8	20	6	14	80	20 x 36	26 x 27
9	22	7	15	88	22 x 40	29 x 30
10	24	7½	16	96	24 x 43	32 x 32
11	26½	8	18	106	26½ x 48	35 x 36
12	29	9	20	116	29 x 52	38 x 39

*Note—Minimum L=B

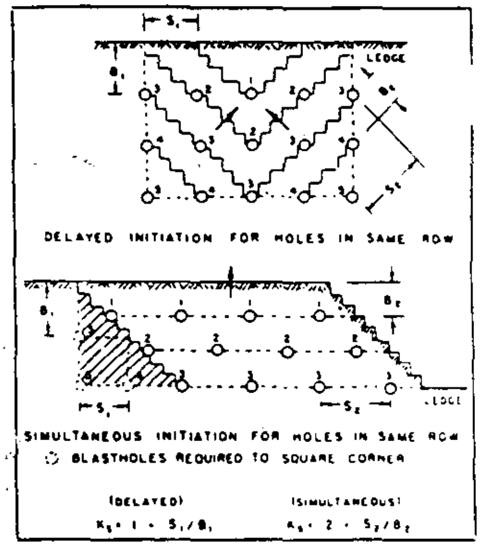


Figure 10—Basic drill-pattern relationships. (Ideal blasting conditions.)

From Figure 10 one can see that the preferred K_s never changes, regardless of conditions, with a K_s near 1 for sequence and near 2 for simultaneous initiation patterns. Because movement is about 45 degrees with the open face for sequence timing, when holes in adjacent rows measured laterally are initiated at the same time, their true actual burden must be considered as measured laterally since movement is perpendicular to that direction. Thus, for different drill patterns but using the same K_n value, the actual area (or volume) of rock blasted should not change.

This can be explained by the example of the 7½-ft. burden for a 3-in. explosive, where a 10 x 10-ft. square pattern is desirable for sequence timing in the same row; but a 7½ x 13-ft. staggered pattern would work equally well when all holes in the same row are fired together. A typical 8 x 12-ft. pattern often followed in the field is merely a compromise between the two more desirable arrangements. However, the pattern invariably gives nonuniform breakage, particularly in massive rock, no matter what timing system is used because of stress unbalance, and resulting overbreak in corners.

Under certain conditions the K_s ratio controls displacement to an advantage. If the timing system is properly selected to give a desired blast effect, slight adjustments can be made to the K_s ratio so as to place the broken rock in an other than normal position, but with some sacrifice in uniformity of rock sizing. For example, for a K_S of 0.7 to 0.9 (whereby the spacing actually becomes the burden) the use of sequence timing causes broken rock to move parallel or along the ledge face and not out onto the floor, as is the effect often desired in coal stripping. On the other hand, a K_S of 1.2 to 1.4 for delayed charges moves the broken rock farther away from the ledge.

SUMMARY

Most blasting difficulties occur because of a lack in understanding of how rock is broken and the use of improper charge-placement and initiation-timing practices. The clues as to what could be wrong are often revealed by observing how a blast performs: whether or not nonuniform breakage results, toes are left, overbreak and violence occur, and similar undesirable effects exist. Provided that the proper explosives are employed for the operating conditions, certain standards can be applied, to help in the evaluation of blasts. These standards can also assist in providing guidelines as to which direction adjustments should be made for correcting any difficulties. The standards are practical and simple to apply, being based on two fundamental, usually known qualities: explosive diameters and ledge height. The standards are as follows:

$K_B = 20$ to 40 (30 avg.)
$K_H = 1\frac{1}{2}$ to 4 (2.6 avg.)
$K_J = 0.3$ minimum
$K_T = 0.5$ to 1 (0.7 avg.)
$K_S = 1$ to 2

As a rule, the K_B relationship is the first standard to apply, since it provides the burden dimension. An exception to this is for blasting extremely low or very high ledges. In such cases the ratio must be adjusted to suit the ledge height. For normal conditions and using a 2-in. explosive, for example, the burden will average near 5 ft. for hole depths not less than 7½ nor more than 20 ft., with subdrilling of at least 1½ ft. and stemming near 3½ ft. The ledge height (L) could then be from 5 to 6 ft. up to about 18½ ft. Table 2 lists data for normal operating conditions. However, the spacing value for adjacent charges will depend entirely on the timing system used and on the rock structural features; but it will vary from 5 to 10 ft. for the example given.

For ledge heights less than the minimum, smaller-diameter explosives should be used; otherwise, overloading and possible violence will occur. For very high faces, the burdens must be reduced or the explosive diameters increased. The latter can be accomplished by drilling larger vertical holes, springing or enlarging holes at their bottoms, using additional snake, or horizontally drilled, holes in the toe region, inclining the drill holes, etc.

An additional problem often present in blasting is that of cap rock, or hard massive layers at the top of a ledge. Using less than normal stemming does nothing but promote violence, since this solution only aggravates vertical cratering, with subsequent overbreak. Instead, an additional short hole should be drilled in the block center, with part of the normal explosive charge for the deeper holes divided equally between a small deck charge, loaded near the collar of the deep hole but separated from the main charge by stemming, and a small charge placed in the extra short hole. In this manner the ledge height limitations are satisfied, with the cap rock and remainder of the ledge then being considered as two separate benches, even though they are blasted at the same time.

The standards will be found to be quite convenient and useful, after very little practice, not only for the initial design of blasts but also in providing guidelines upon which to correct normal blasting difficulties which invariably occur from time to time. However, one must realize that the standards in themselves are not cure-alls, since blasting as such depends heavily on cost and safety considerations as well as on the explosive grades used, the material's characteristics, and blasting techniques employed.

The Mechanics of

CHARACTERISTICS OF EXPLOSIVES

Part III of a Series

IN selecting an explosive upon which to base a particular set of blasting standards, the choice will depend largely on the cost and properties of the explosive and its adaptability to the materials to be blasted. Since blasting effectiveness from any explosive is controlled by its chemical composition and the effects produced by the field conditions under which it is used, the user should have a working knowledge of the various explosives products available and their particular properties. In this manner he is then better able to make a practical choice to suit his own operating conditions.

An explosive can be considered simply as a tool for performing work, designed to accomplish a specific job. The work performed is made possible by the gas pressures produced when the explosive reacts. The ideal explosive would be one in which only gases are formed from the original ingredients. However, if some solids are also produced by the reaction, the gas pressures would be correspondingly reduced, with the explosive then being capable of producing less work. Since there are many different field conditions with which to contend, manufacturers offer many different types and grades, many of which are nonideal and designed to have their own qualities that make them differ from one another. Part of the differences are chemical, part are physical. However, since explosives are chemical compounds, it is from their original composition that all basic qualities are first determined.

Ingredients and Composition Most commercial explosives are mixtures of compounds containing four basic elements: carbon, hydrogen, nitrogen, and oxygen. Other compounds with additional elements such as sodium, aluminum, calcium, etc., may also be included to produce certain desired effects. As a rule, manufacturers design their products to be nearly oxygen-balanced. This means that there is the correct amount of oxygen available in the mixture so that during the reaction all of the hydrogen reacts to form only steam (H_2O), the carbon reacts to form only carbon dioxide gas (CO_2), and the nitrogen released forms only free nitrogen gas (N_2).

If there are other than the basic four elements, e.g., sodium, solids would be expected to be produced, and for these there must be included sufficient additional oxygen to combine with them. When there is an excess of available oxygen, however, certain other compounds are produced, among which are the highly poisonous nitrous-oxide fumes (NO/NO_2). These particular fumes are easily detectable by their obnoxious odor and red-brown color. On the other hand, if there is an oxygen shortage, the deadly carbon-monoxide fume (CO) will be formed, as well as certain other compounds, depending on the ingredients. Unfortunately, carbon monoxide cannot be detected by odor or sight. In addition to the formation of poisonous fumes, an excess or deficiency of oxygen will yield a lower heat of explosion, with a subsequent reduction in pressures produced.

It should therefore be recognized that if one is to expect safe and efficient results from explosives, there should be a suitable initial chemical balance, with thorough mixing of ingredients to ensure that all materials are in intimate contact, maintenance of the desired mixture while in storage, and then proper use on the job. The following chemical equations may help to illustrate the effects from oxygen balancing, using an AN-FO blasting agent for an example:

(1) *Balanced for oxygen:*
$$3NH_4NO_3 + CH_2 \rightarrow 7H_2O + CO_2 + 3N_2$$

(2) *Excess oxygen:*
$$5NH_4NO_3 + CH_2 \rightarrow 11H_2O + CO_2 + 4N_2 + 2NO$$

(3) *Deficient oxygen:*
$$2NH_4NO_3 + CH_2 \rightarrow 5H_2O + 2N_2 + CO$$

It is not necessary for an explosive to contain nitroglycerin (NG) nitrostarch (NS), TNT, and similar explosive compounds. The individ-

Table 3—Some Ingredients of Explosives

Name	Chemical Symbol	Function
Nitroglycerin (NG)	$C_3H_5(NO_3)_3$	Explosive base
Trinitrotoluene (TNT)	$C_6H_2CH_3(NO_2)_3$	Explosive base
Dinitrotoluene (DNT)	$C_7N_2O_4H_6$	Explosive base
Ethylene glycol dinitrate (EGDN)	$C_2H_4(NO_3)_2$	Explosive base, antifreeze
Nitrocellulose	$C_6H_7(NO_3)_3O_2$	Explosive base, gelatinizing agent
Ammonium nitrate (AN)	NH_4NO_3	Explosive base and oxygen carrier
Potassium chlorate	$KClO_3$	Explosive base, oxygen carrier
Potassium perchlorate	$KClO_4$	Explosive base, oxygen carrier
Sodium nitrate (SN)	$NaNO_3$	Oxygen carrier, reduce freezing point
Potassium nitrate	KNO_3	Oxygen carrier
Wood pulp	$C_6H_{10}O_5$	Absorbent, combustible
Fuel oil	CH_2	Fuel
Paraffin	CH_2	Fuel
Lampblack	C	Fuel
Chalk	$CaCO_3$	Antiacid
Zinc oxide	ZnO	Antiacid
Aluminum metal	Al	Catalyzer
Magnesium metal	Mg	Catalyzer
Kieselguhr	SiO_2	Absorbent, anti-caking material
Liquid oxygen	O_2	Oxygen carrier
Sulphur	S	Fuel
Salt	NaCl	Flame depressant
Organic nitro compounds		Explosive base, but used primarily to sensitize, reduce freezing point, and as anti-caking material

ROCK BREAKAGE

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ual characteristics of each ingredient determine whether it may be desirable for use in a mixture. Table 3 gives a partial listing of the many ingredients that might be included in an explosive. It can be recognized that certain compounds may be highly explosive by themselves or may be normally inert; but when combined, the entire mix may form an explosive. For this reason the compounding of explosives should not be attempted by the average person.

Explosive Reactions To be an explosive, the change in form from liquid or solid, or a combination of both, to that of a gas, or gas and solid, must be an exothermic reaction, or one from which heat is released. For most explosives, the quantity of heat released is quite large (Table 4). The gases formed, in turn, quickly produce very high pressures, with the reaction being called either deflagration or detonation.

The distinction between the two types of reaction is that deflagration consists of a burning action at a high rate of speed, the chemical reaction of which causes gaseous formation and pressure expansion *along* with the burning. Thus, a heaving action from the pressures produced is experienced at nearly the same rate as that of the burning. This type of reaction is characteristic of low explosives, of which black powder is one particular type.

Detonation, on the other hand, consists of the propagation of a shock wave through the explosive, accompanied by a chemical reaction that furnishes energy to sustain the shock-wave propagation in a stable manner, with gaseous formation following shortly thereafter. The shock wave is characterized by a very sharp rise in pressure (Figure 11), in front of which there is a zone in which all immediate matter is ionized. The pressures developed by detonation (shock) are nearly double those produced by the gaseous expansion that follows. All high explosives are designed to detonate,

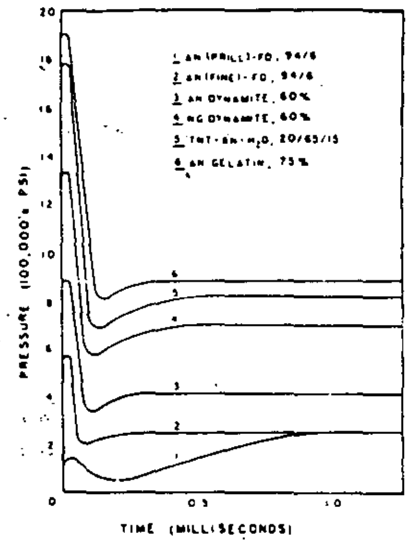

Figure 11—Curves of calculated pressure developed by some selected explosives under perfect confinement.

all low explosives will deflagrate; and blasting agents may exhibit one or the other type of reaction, according to their specifications and conditions of use. The important thing to remember about the reactions is that the effects of one type are very much different from those of the other, detonation producing higher energy and much higher velocities.

To accomplish a desired reaction, certain temperature and pressure conditions must be met, most explosives being designed for use under confinement, e.g., in blastholes. If the temperature required for a proper reaction is not present, no detonation may occur, with only burning or possible deflagration resulting. In practical terms, this means that even though the designed chemical composition calls for detonation, inadequate initial heat from an initiator or primer or a loss in confinement conditions can result in lower blast energy being developed from the explosive charge, or even in complete failure, causing a misfire.

For this reason, control over the confinement and the selection of primers with adequate heat energy and initiating power are particularly important. One should recognize then, which of the explosives need strong priming and which need very little heat for initiating their reactions, not only for reasons of blasting efficiency but for safety considerations as well. (*Turn page*)

Table 4—Available Heat Energies (Q) for Certain Selected Explosives

Explosive	SG	SC	Q (Cal gm)
Nitroglycerin (NG)	1.6	88	1420
PETN	1.6	88	1400
RDX	1.6	88	1320
Composition B	1.6	88	1140
Tetryl	1.6	88	1010
NG gelatin 40%	1.5	94	820
Slurry (TNT-AN-H_2O, 20/65/15)	1.5	94	770
NG gelatin 100%	1.4	101	1400
NG gelatin 75%	1.4	101	1150
AN gelatin 75%	1.4	101	990
NG dynamite 40%	1.4	101	930
AN gelatin 40%	1.4	101	800
NG dynamite 60%	1.3	109	990
PETN	1.2	118	1200
Semigelatin	1.2	118	940
Extra dynamite 60%	1.2	118	880
Amatol, 50/50	1.1	128	890
RDX	1.0	141	1280
DNT	1.0	141	960
TNT-AN, 50/50	1.0	141	900
TNT	1.0	141	870
AN-FO, 94/6	0.9	157	890
AN low-density dynamite	0.8	176	880
AN	0.8	176	350

To better understand the requirements just described, Table 5 illustrates the approximate temperature characteristics of two basic ingredients used in many commercial explosives. It should be noted that at a very low temperature NG begins to decompose, boiling occurring shortly thereafter. Flame from a fuse, heat released by blasting caps, a relatively warm blasthole (such as one just recently drilled), friction from metal objects, and similar effects can all provide quite easily the relatively low temperature needed to provide dangerous conditions. If the NG is confined, e.g., in a blasthole, the initial decomposition will be accelerated to result in detonation.

On the other hand, AN requires a fairly high temperature before it will begin to decompose and fume, necessitating a large amount of initial heat. However, once decomposition begins, detonation or deflagration will follow with a very small temperature rise. By combining the two ingredients, as is done in the ammonia dynamites, a compromise effect is achieved, the grades having the most NG being the easier to initiate.

Important Properties Of Explosives Most manufacturers supply catalogs and other information concerning the specifications of their products. However, certain properties are particularly important to quarry blasting. A review and explanation of their practical aspects should therefore be of special interest to the operator.

Water Resistance For all explosives, the presence of water in blastholes tends to promote chemical unbalance, as well as retard the heating reaction. Water supplies additional hydrogen and oxygen and requires additional heat to be vaporized into steam. If water is flowing through the ground, a leaching action can occur, whereby certain salts that may be easily dissolved could be removed from the explosive mixture. Explosives may be protected internally from water action by gelatinizing the mix or externally by cartridging. The ingredients added for gelatinizing are usually included in the chemical balance, as with the use of nitrocellulose in the gelatin grades.

Table 5—Comparison of Approximate Reaction Temperatures (°F) of NG and AN

	NG	AN
Detonate	420	460
Boil	290	—
Decompose	140	410
Freeze	50	340

Similarly, the paper, wood fiber, paraffin, or polyethylene used for external cartridging are generally included in the chemical balance. For this reason explosives that are made for use in cartridges should not be removed if preservation of the oxygen balance is to be maintained.

If an explosive is properly compounded initially, but detrimental effects occur from water, the action will be noticeable by the formation of brown nitrous-oxide fumes and a low blasting action. If these effects are observed, the explosive grade should be changed or other appropriate action taken. Primers must of necessity possess unlimited water resistance.

Fumes Most explosives are given a fume rating, the classification of which is based on the amounts of poisonous gases produced by the explosive reaction. Limits are set by many of the states, the U. S. Bureau of Mines, and certain other agencies. Where inadequate ventilation and exposure of personnel to toxic gases may exist, care must be taken to ensure that the explosives used give amounts below the established limits.

This property is particularly important for underground blasting; but for open-cut operations the problem could also be quite serious. Fumes may lie inside piles of broken rock. Such material, when stirred up by loading equipment, will release the fumes, to contaminate the air in which men are working. The problem may be aggravated by atmospheric conditions, deep cuts, and similar factors that hinder air circulation. Men will become ill and nauseated if this situation is present.

A person should understand the distinction between fumes and smoke, the latter of which is composed of liquid or solid particles suspended in the air. Usually when white smoke is observed from blasts, it is quite likely composed primarily of the steam from the reaction.

Sensitivity This property actually refers to two related characteristics. It defines the relative ease with which an explosive reaction can be initiated and the relative ease with which the reaction is propagated through an entire charge. Several tests are used to rate sensitivity, the most common of which is the minimum booster required for initiation. Usually the total number of No. 6 strength blasting caps required for initiation is used to classify sensitivity.

However, an explosive may initiate easily but in small diameters the reaction may not propagate and dies out. For this reason explosives may not be manufactured below specific diameters. A critical diameter, or that below which propagation of a reaction will not continue, exists for practically all commercial products. Some blasting agents have a large critical diameter; most high explosives have a small one. By definition, blasting agents cannot be sensitive to initiation by a single No. 6 blasting cap, while high explosives all are one-cap sensitive.

On the other hand, an explosive may be quite insensitive to initiation but propagate easily when above the critical diameter. For safety reasons this situation is the more desirable. it is a definite advantage offered by many of the blasting agents. However, adequate priming is mandatory for their use. If propagation is difficult or impossible through a column of explosives, boosters may be used to sustain the reaction. But it should be recognized that both boosters and primers must be sensitive to initiation.

The sensitivity of an explosive is a function of the ingredients, their particle sizing, the charge diameter, the degree of confinement, and certain other factors. For example, ammonium-nitrate explosives may become quite sensitive in time by particle degradation due to the process of cycling. AN has the characteristic whereby it will change its crystalline form with changes in temperature; two of the changes often encountered in normal field blasting are at 0 and 90 deg. F. Constant changes through those temperatures causes the particles to break into smaller sizes. The smaller particles offer more contact surfaces between ingredients, making it easier for particles to be consumed by the explosive reaction. The result is to permit easier initiation and subsequent more rapid propagation through a charge. Blasting agents that would normally be insensitive become quite sensitive to initiation by a single No. 6 blasting cap, similar to that expected of high explosives.

Larger charge diameters also propagate reactions more easily because of the greater surface area available. Confinement tends to concentrate the reaction's force along the charge length rather than permit the action to spread.

Certain hydrocarbons have an adverse effect on some types of explosives, principally those with free NG, as do the straight and extra grades of dynamites (Table 6). Since some of the blasting agents have liquid hydrocarbons as one of their ingredients, e.g., FO, one should be particularly cautious in his choice of primer explosive. Under certain conditions there could be an accumulation of the hydrocarbon in the blastholes, particularly at the bottoms, which in turn may lead to misfires when charges are bottom-primed. This situation can be avoided by using gelatins or simigelatins or high explosives containing no NG for priming. Furthermore, it is simply good practice to avoid the use of excessive FO in any blasting agent, to avoid upsetting the oxygen balance.

Table 6—Percent by Weight of Diesel FO Additive Where Detonation Fails

Explosive	Pct. Add.	Qt. FO/lb. of Expl.
Extra dynamite 40%	1.5	0.008
Extra dynamite 60%	2.5	0.014
Low-density dynamite (SC 120)	4.0	0.022
AN gelatin 60%	8.0*	0.05*
NG gelatin 60%	39.0*	0.21*

*Amounts applied, but detonation successful; no failures.

Density Explosives are manufactured and sold on a weight basis, the densest explosives usually being the strongest. The density, or weight per unit volume, of an explosive is therefore one of its most important properties. In industry this property may be specified in three ways: (a) by specific gravity (SG) expressed as a unitless number *or* in gm/cc; (b) by stick count (SC) or the number of 1¼ x 8-in. cartridges per 50-lb. box; and (c) by loading density (d_e) or the pounds of explosive per foot of charge length. The value for the loading density, however, is a function of the explosive's charge diameter (D_e), which should then also be specified easily for clarity.

The various measures for density can be calculated easily for rapid use in the field, provided that the charge diameter (D_e), expressed in inches; and one of the density values are known. The relationships are as follows:

$$d_e = 48D_e^2/SC \qquad (1)$$

$$d_e = 0.34D_e^2(SG) \qquad (2)$$

$$SG = 141/SC \qquad (3)$$

These formulas provide a very convenient means for estimating explosive quantities, in that most explosive manufacturers supply the SC or SG for their products. For example, if a free-flow AN-FO mixture with an SC of 176 were to be used in a 10-in. diameter blasthole, one would expect slightly in excess of 27 lb. per foot of hole (or $d_e = 48 \times 10^2$ divided by $176 = 27$ lb./ft.). (The relationships are illustrated graphically by Figure 12.)

It will be noted that an SC of 176 corresponds to an SG of 0.8, which could also be determined from Expression (3), above. Since the SG of water is 1 and its equivalent SC value is 141, any explosive with an SG *greater* than 1 or an SC *less* than 141 could be expected to sink in wet blastholes. It should be pointed out, however, that D_e is the diameter of the explosive, *not* that of the blasthole. These diameters are equal only in the case of free-flowing explosives or charges composed of cartridges that are thoroughly tamped.

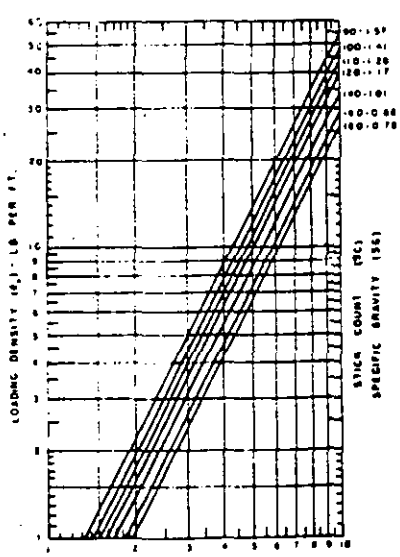

Figure 12—Relationships between densities of explosives.

Because certain ingredients may be included in explosives that do not contribute to the energy produced, there is no distinct relationship between density and pressures developed. In fact, some manufacturers make a 40 percent Extra type dynamite, for example, that is denser than the 60 percent of the same type of explosive. Similarly, a 90 percent gelatin is lighter than a 30 percent gelatin. But as a general rule it is reasonably approximate to relate the energy developed by explosives to their relative densities. This is because explosives are characterized by general density groups that correspond to their various types, e.g., gelatins, dynamites, etc. The denser types as a group produce more energy than the lighter ones, even though there may be exceptions to the rule between grades within the same type.

Velocity The rate, usually expressed in feet per second (fps), at which a reaction propagates through an explosive is considered by many as the most important quality of an explosive. It is often called the detonation velocity, but this is not always technically correct. Its importance can be better appreciated when it is understood that the energy produced by any explosive is a function of the *product* of its density and velocity characteristics. Since the initial reaction for most explosives used in commercial blasting is detonation with subsequent gaseous expansion, the action would be considered dynamic.

Thus, impulsive and momentive forces are produced as a result of the kinetic energy of the reaction, which can be expressed by the relationship $KE = \frac{1}{2}Mv_e^2$, where M is the mass and v_e is the velocity of the explosive's reaction. The rela-

(v_e) must be known, or approximated. Furthermore, to simplify calculations, one can assume blastholes would be filled across their entire diameter, or $D_e = D_H$. This condition ensures little or no energy losses, or dampening, for a complete energy transfer from the explosive's reaction into the surrounding rock to be blasted.

The relative energy (RE) and that exerted to the rock could then be expressed by a simplified kinetic-energy relationship, or $RE = a(SG)v_e^2$. The "a" is a conversion factor to permit the use of specific gravity instead of mass, and it assumes that the explosives will be used in the same diameter. For any set of similar field conditions the "a" will be a particular constant number, making it then possible to omit it from the relationship when explosives are compared under identical field conditions. Thus, the following expression can be used for comparing two or more explosives, based on their energies:

$$RE_2/RE_1 = (SG_2)(v_{e2})^2/(SG_1)(v_{e1})^2$$

If Explosive No. 1 represented the average explosive ($SG_1 = 1.2$ and $v_{e1} = 12{,}000$ fps) and Explosive No. 2 had $SG_2 = 1.5$ and $v_{e2} = 18{,}000$ fps, the relative energy of the second compared to the first according to Expression (4) would be as follows:

$$RE_2/1 = (1.5)(18{,}000)^2/(1.2)(12{,}000)^2 = 2.8$$

The RE value shows then that the second explosive has 2.8 times the energy potential of the standard explosive. Since the comparison is made between explosives used for blasting the same material, the comparative blast results in the rock would vary as the cube root of their relative energy value. The cube root is used rather than the direct ratio because of the spherical fan effect for energy propagation through homogenous materials. This relationship then tells us that the K_B ratios and therefore the burdens will vary in proportion to the cube root of the explosives' relative energies. To provide a simple formula for illustrating the relationship, the following may be used:

$$K_{B2} = K_{B1}(RE_2/RE_1)^{1/3}$$

If one assumes that average rock will be blasted, a K_B value of 30 would represent the average explosive (Figure 7). The burden used would be 7½ ft. for a 3-in. diameter explosive, since $K_{B1} = 30 = 12B_1/D_e$, which gives $B_1 = 30D_e/12 = 30 \times 3/12$ or 7½ ft.

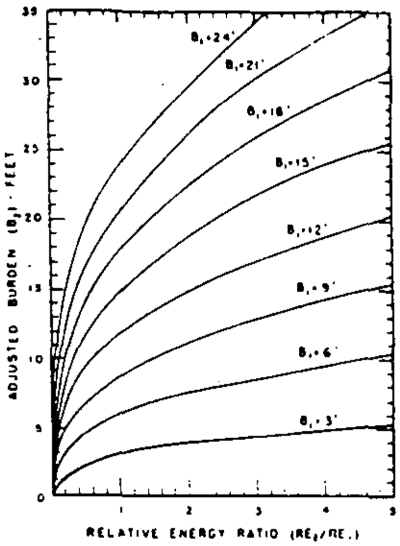

Figure 14—Relationships between burden dimensions for explosives according to their relative energy and when used under field conditions.

For Explosive No. 2, then, using Expression (5), one can approximate that $K_{B2} = 42$, or $K_{B2} = 30(2.8)^{1/3}$. The burden for the second explosive would then be 10½ ft., since $B_2 = 42D_e/12 = 3½ \times 3$. For direct calculation of the burdens for explosives used in the same diameters and under identical field conditions the following may be used:

$$B_2 = B_1(RE_2/RE_1)^{1/3}$$

The relationships given by Expressions (5) and (6) are shown on Figure 14, which permits one to determine the approximate new burden for any explosive as compared to the average explosive when used under identical field conditions.

Although the example given illustrates ideal conditions and one should recognize that many variables enter into making the final selection of a K_B ratio and its related subsequent burden dimension, the relative-energy comparison technique gives a realistic approximation. As a matter of interest, for most explosives used in blasting the maximum density variation is from 0.7 to 1.6, with a velocity variation from 8,000 to 20,000 fps, the heavier densities having the higher reaction rates. Therefore, the weakest explosives possess only 26 percent of the energy available, while the strongest have 370 percent of the energy available, as compared to that available from the average explosive. Converted to K_B values and using a $K_B = 30$ for the average explosive in average rock, the lower and upper limits for K_B values would be 19 and 46, respectively. From Table 1 it can be seen that these values satisfy results from actual field experiences.

The Mechanics of

ROCK BREAKAGE

MATERIAL PROPERTIES, POWDER FACTOR, BLASTING COST

Part IV of a Series

MATERIALS PROPERTIES AND INFLUENCE

MOST materials requiring blasting are not homogeneous nor are their properties the same throughout. Of all the physical properties, there are essentially five that predominantly influence blasting results. These include in order of their importance the following characteristics: (1) structure, (2) resilience, (3) strength, (4) density, and (5) velocity of energy propagation. Blastability, elasticity, hardness, toughness, and other terms may also be used to describe a material, but often such expressions are too indefinite and difficult for the ordinary quarry man to understand. Drillability, or ease of drilling, should in no way be confused with the manner in which a material can be blasted.

Structure The structural features of a material usually have the greatest influence on blast effects. To better understand their importance one should recognize that rock, as we think of it, is essentially an accumulation of small particles bonded together. The constituents are oriented in definite structural patterns, established during the formation and alteration processes. Of primary importance to blasting is compression jointing, existing within all rocks (igneous, sedimentary, and metamorphic) and composed of planes along which there is no resistance to separation. Igneous rock may also have tension jointing, formed during the cooling process.

Sedimentary rocks are unique in that they have stratification planes (in addition to joints), which were originally horizontal and formed by interruptions in the initial deposition of sediments. Stratification and jointing are not the same thing. For metamorphic rocks, the relationship of their jointing to schistosity is similar to that between jointing in sedimentary rocks and their stratification, both in angular position and mechanical development.

Jointing is usually easily detected, the planes being generally smooth and often short distances apart. One set of planes is parallel with the dip and strike of the rock formation, with two or more sets being nearly perpendicular to the first set. Rocks when broken will separate into blocks of a shape characteristic of their particular jointing pattern, and the new faces produced from blasting tend to follow the jointing directions. (See Figures 3, 4, 8, 10, and 15.)

For the sedimentary rocks there is one particular direction along which jointing is the most pronounced, the other planes being less dominant. The horizontal angles between the vertical jointing planes are usually near 75 and 105 degrees, which form rhombohedrons when the rock is broken. Igneous rocks, however, have jointing planes of uniform strength, the angles between planes being most often near 60 degrees. The fragments produced from blasting are generally hexagons or pyramids in shape.

Jointing directions can be found quite easily if it is recognized that most faults, cliffs, mud seams, caves, etc., produced by weathering and the other geologic actions tend to follow the jointing planes. It is particularly important that the blaster endeavor to locate the planes before laying out a drill pattern. Blastholes located in tight corners will generally overbreak, opening large cracks in the ledge. Subsequent blasts will usually do no more in those areas than give large boulders, and possibly be quite violent. It can be seen from Figure 15, which illustrates a representative quarry in a sedimentary rock formation, that there are tight (75-degree) and open (105-degree) corners. This means that normal blasts under those conditions should be directed out of the open angles in so far as possible, or toward the east or west. If blasting is done in the other directions, or to the north or south, cracking of the solid ledge will occur along the planes forming the tight angles.

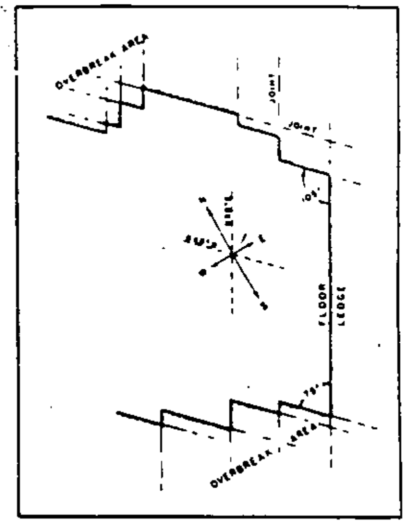

Figure 15—A representative plan sketch of a quarry in a sedimentary rock formation, showing tight (75-degree) and open (105-degree) corners.

Another structural feature that is

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very important, particularly to rock fracturing, is the type and strength of the bonding between individual grains. For example, rock may have pronounced jointing at widely separated distances, but the material between joint planes may be strongly bonded, or massive in character. Large boulders invariably result when blasting is carelessly done under this condition. On the other hand, rocks may be highly laminated or stratified, or the bond between grains may be very weak, so that fragmentation is always easily accomplished by merely moving the material from its original place.

Resilience This property, sometimes called sponginess or toughness, refers to the elasticity of a material. It is used to express the capability of a rock to resist shock and recover its original position and shape without being ruptured. If a rock on being dropped, for example, makes a dull thud and does not rebound, it would be very difficult to break by impact. Brittle rocks, however, shatter easily, particularly those types having a high silica (quartz) content. A blaster can generally determine quite easily whether or not a material will break into small sizes or large coarse fragments by conducting a simple drop test. Furthermore, the test provides a clue as to the energy absorption power of the material, which is important for estimating the amount of additional charge, or energy, that would be necessary to overcome expected energy losses.

Strength Of the characteristic strengths of materials, blasting is normally concerned only with that of tension. Most rocks are very weak in tension, more resistant to shear, and strongest in compression, having approximately only one-tenth the resistance to tensile rupture that they have to failure by compression (Table 7). However, shear is not actually a force by itself but rather the result of two forces, either two tensile or two compressive forces, or a combination of one of each, which act along different lines and directions.

To know the actual strengths of a material, samples must be tested in a laboratory. (Regular tensile-strength tests are usually difficult to conduct.) However, tests for what is known as the modulus of rupture are much easier to perform; yet they provide information that is just as useful in providing tensile-strength data of equal practical value. In fact, the laboratory test for the modulus breaks samples in tension by bending test slabs until they fracture, much in the same manner that rock is stretched and broken at an open face during blasting (Figure 3).

Quite often it is impossible or quite impracticable for quarry operators to have tests conducted. Also, test results on samples may not necessarily provide information on the over-all strength of a rock deposit, except when the material is homogeneous and very massive. Nevertheless, if tests could be made, the data would aid greatly in determining the stress levels (psi) required for fracture. It is the resistance to tensile rupture that must be exceeded by the energy pulses at the free faces, and thus, if known, could also give an approximation of the required burden dimension and the explosive pressures needed for proper breakage. In the event specific test data cannot be obtained, the operator may find the information in Table 7 quite useful. From the various moduli listed for many of the representative rock-types, a practical estimate can be made that will approximate the characteristics of his particular deposit.

Table 7—Properties of Various Selected Materials

Name and Location	Compressive Strength (psi)	Modulus of Rupture (psi)	Specific Gravity (SG)	Density (d_r) (ton/cu. ft.)	Longitudinal Velocity (v_l) (fps)
Amphibolite (fine grain, India)	61,400	7,400	3.12	0.097	19,000
Basalt (New York)	46,600	8,000	2.94	0.092	18,700
Basalt (Michigan)	33,400	3,800	2.85	0.089	15,200
Basalt glass	—	—	2.81	0.088	21,000
Diabase (fine grain, Michigan)	44,200	5,300	2.94	0.092	16,700
Dolomite (Missouri)	8,800	1,000	2.80	0.087	—
Dolomite (Tennessee)	46,700	3,800	2.84	0.089	17,900
Gabbro (altered, New York)	40,200	5,400	2.93	0.091	17,600
Granite (Georgia)	28,000	2,000	2.64	0.082	8,900
Granite (Vermont)	33,200	2,900	2.66	0.083	11,100
Granite (Nevada)	39,500	3,900	2.63	0.082	14,500
Granite (North Carolina)	30,400	1,600	2.60	0.081	8,000
Greenstone (Michigan)	45,500	3,300	3.30	0.103	16,600
Gypsum (Indiana)	3,200	1,200	2.32	0.072	—
Limestone (Ohio)	28,500	2,900	2.69	0.084	15,400
Limestone (Utah)	28,000	2,200	2.78	0.087	15,900
Limestone (fossiliferous, Indiana)	10,900	1,600	2.31	0.072	12,400
Limestone (West Virginia)	23,000	1,900	2.68	0.084	16,400
Marble (Maryland)	30,800	2,800	2.37	0.074	13,700
Marble (New York)	18,400	1,700	2.72	0.085	14,500
Obsidian	—	—	2.35	0.073	16,100
Quartzite (taconite, Minnesota)	91,200	3,400	2.75	0.086	18,200
Rock salt (Louisiana)	5,000	Negligible	2.50	0.078	—
Sandstone (Ohio)	10,400	500	2.06	0.064	5,600
Sandstone (West Virginia)	19,400	3,400	2.50	0.078	12,900
Sandstone (Utah)	11,500	620	2.17	0.068	8,400
Sandstone (Alabama)	26,800	2,200	2.76	0.086	12,500
Shale (Utah)	31,300	2,500	2.81	0.088	14,900
Shale (West Virginia)	11,600	4,200	2.40	0.075	13,600
Syenite (New York)	34,300	2,800	2.72	0.085	14,500
Alluvium, broken rock, loess	—	—	1.3-1.5	0.044	2,300
Clay	—	—	2.58	0.081	5,900
Air	—	—	0.0012	—	1,080
Water	—	—	1.00	0.031	4,750

Density Denser materials require greater amounts of work energy to be satisfactorily broken and displaced, and heavier explosives or large charges will therefore be needed. However, from Table 7 it can be concluded that for most rocks there is a very narrow range of density differences, with SG values varying from 2.3 to 3.3 in most instances. The materials generally requiring blasting have densities confined to the 2.5-2.9 SG range. This can be interpreted to mean that the influence of rock density alone has a limited effect on blasting, the extreme conditions being within 15 percent of the average 2.7 SG. One may then reasonably assume that rock density by itself is of little importance to blasting and would not appreciably affect a K_B value or burden dimension.

Its importance, however, lies in the fact that it does influence costs and the other physical properties. Although densities are most often given by specific gravity, for calculations in costing and powder factor determinations it is more convenient to use the density ratio, d_r, expressed in units of tons/cu. ft. of solid material. If the d_r value is not known, one can utilize the following expression for converting any SG that may be given:

$$d_r = SG(62.4/2000) = 0.0312(SG), \text{ tons/cu. ft.} \quad (7)$$

Velocity The velocity of energy transmission in rock, v_r, is like the reaction velocity for explosives, v_e, in that it increases as rock density becomes greater. The denser rocks are often the least porous and are generally composed of small grains, which permit easier propagation of energy through the material. For this reason most dense rocks have smaller energy losses due to dampening, and they often have a tendency to shatter rather than break into slabs. Most brittle rocks also transmit energy at very high rates, except in the unique case of certain sandstones. The characteristic low velocities of many of the sandstones are due to a peculiarity in their composition: the matrix bonding the sand grains may be clay, lime, or other energy-absorbing substances. However, if the matrix is silica, the velocity is quite high.

Velocities for materials are usually specified as longitudinal velocities, v_l, as are also those given in Table 7. But these values are normally slightly lower than the velocity of energy propagation, v_r. The two velocities are related by the following expression:

$$v_r = v_l[(1-\mu)/(1+\mu)(1-2\mu)]^{1/2} \quad (8)$$

Because μ, or Poisson's Ratio, is usually considered as 0.25 for estimations, it is more convenient to convert velocities by using $v_r = 1.095v_l$ for approximations. However, it is more practical and will not introduce any great error if the two velocities are considered equal.

The importance of velocity in rocks on blasting is that it has a strong influence on the amount and manner in which a material will be stressed. In order that the momentive forces be conserved, there should be nearly perfect coupling of the energy from an explosive's reaction with the surrounding material. The matching of the momentive energies is considered necessary theoretically for the most efficient blasting results. This condition is known as acoustical coupling. Since the energy required for stressing strong and dense rocks would be relatively large compared to that needed for lighter materials, the use of denser, fast-reacting explosives is generally advisable.

The velocity of a rock will determine the time it takes the stress energy to reach free faces and return. The velocity of an explosive, on the other hand, will determine the total time it takes for an entire charge to complete its reaction. The relationship of the two velocities, called the velocity ratio or $K_v = v_e/v_r$, has a very important influence on the manner in which an entire blast will function. This is because the K_v ratio defines the shape of the composite wave produced by all the individual stresses introduced into the rock from each point along a charge column (see Figure 6, PIT AND QUARRY, September, 1963, page 119) the primer positions thus controlling which faces are fractured first and the direction in which the composite wave will travel in the rock.

The K_v ratio, primer location, and general design features of a blast must follow certain definite relationships, if results are to be satisfactory. In particular, the influence of rock velocity is such that there will be a certain optimum of critical hole depth for each blasting situation. For example, when a charge is bottom-primed, there will be a specific *minimum* hole depth. If the depth is less than the minimum value, blast effects will begin near the collar region, which quite likely may promote violence and air blast. In some instances, toe will be left

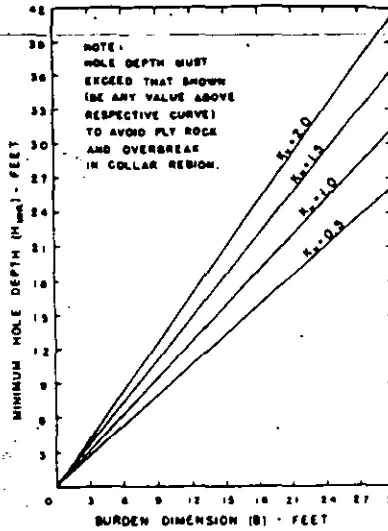


Figure 16—A graphic presentation of the relationship between minimum hole depth and burden dimension.

at the floor. However, when holes are deeper than the minimum value, stressing and rock movement will always begin at the ledge bottom before action occurs in the collar region. The particular minimum required depth of hole can be determined from the following expression:

$$H_{min} = K_v[(B^2+J^2)^{1/2} - T] + T \quad (9)$$

The relationship is illustrated graphically in Figure 16, in which $K_T = 0.7$ and $K_J = 0.3$ are considered average conditions. The values for the H_{min} represent balanced stressing at both the toe and collar regions.

If charges are collar-primed, stressing will always begin in the collar region, unless the amount of stemming used exceeds the burden dimension. Even under that condition, collar overbreak and air blast may occur, with possible toes resulting, if a particular *maximum* hole depth is exceeded. This limiting condition can be determined from the following relationship:

$$H_{max} = K_v(T - B) + T \quad (10)$$

From a practical viewpoint, the expression shows that under no circumstances should the stemming dimension be less than that for the burden in blasting massive rock. Otherwise, collar cratering and air blast can be expected. The condition becomes particularly critical when detonating fuse is used and initiation is done on the surface, since the fuse on detonating has the tendency to loosen the stemming. For deep holes, collar priming would definitely be undesirable under conditions where massive cap rock occurs in the collar region and where column loading is practiced; i.e., the charges are continuous from just below the stemming to the hole bottoms.

An unusual situation exists when the K_v is less than 1, or when the rate of travel of the compressive stress-wave in the rock exceeds the speed of the detonation wave in the charge column (Figure 6). Stress waves will reach free faces before the explosive has completed its reaction, with rock at the faces being repeatedly stressed by the pressures produced by the still reacting explosive column. The action reinforces the stresses and reduces the resistance of the rock to fracture, giving the impression that the explosive is stronger than it actually is. Under certain conditions, blasts are extremely efficient, but they are usually difficult to control, producing greater heave or throwing action.

Since there are critical hole depths for each blasting condition, the best results can often be insured by first estimating the particular K_v value for the conditions present, and then placing primers accordingly. Control for very deep holes, for example, is achieved by using primers both near the collars and in the hole bottoms; or primers may be placed at strategic intervals throughout the columns, with or without the use of deck charges. Either detonating fuse or close-interval delay blasting caps can be used for initiating the primers, those near the collar being preferably of a longer delay. The composite effect of using primers at both the collar and hole bottom is that it extends the optimum hole depth and better distributes the stresses in the ledge, notably in the toe and collar regions.

POWDER FACTOR AND ITS SIGNIFICANCE

A guideline used by many for estimating and evaluating blasting is the Powder Factor, Pf, an expression which relates the yield of material blasted to the quantity of explosives used. For quarry work and mining, the Pf is most often stated in tons/lb., or vice versa, while for most construction excavation it is customarily expressed in lb./cu. yd. or cu. yd./lb. The latter ratio is also commonly used for much of the work in overburden removal for coal and metal-ore operations. Of all the different ratios in common use, only those utilizing weights, e.g., tons/lb., take into account any of the properties of the materials being blasted.

Because of its extremely variable character Pf is not normally a sound index upon which to judge blasting efficiency or design blasts, as many believe. Different values will be obtained by merely changing the blasthole pattern or configuration, and values will also change for other reasons, such as variable hole depths and deck loading. Also, the many different standards employed tend to confuse rather than assist persons in evaluating results. The most practical value of Pf is in cost analysis, because explosives are sold by weight, and payment for materials mined or removed is generally made on a weight or volume basis.

One of the ways in which the powder factor can vary is shown by the examples given in Figure 17. These sketches illustrate four possible ways of blasting with a single charge and six different patterns utilizing a V-cut arrangement for multiple charges. All the blasts are conducted under identical conditions except for the relative positions of open faces. Pertinent data for Figure 17 are given in Table 8. The information there given is merely representative and used for comparative purposes. It may or may not fit actual blasting situations.

In determining the possible yields given in Table 8 for the various blasts shown in Figure 17, the surface blast areas, A, were approximated based on the locations of open faces, assumed rock structural features, and the particular mechanics of how each specific blast would be expected to function. The excavation volume would then be the product of the blast area and the ledge height, L, not the hole depth, H, as some might assume. Simple conversion to tonnage yield, W, is accomplished by multiplying the volume by the material density, d_r, using the following relationship:

$$W = AL(d_r), \text{ tons} \qquad (11)$$

The quantity of explosives used, E,

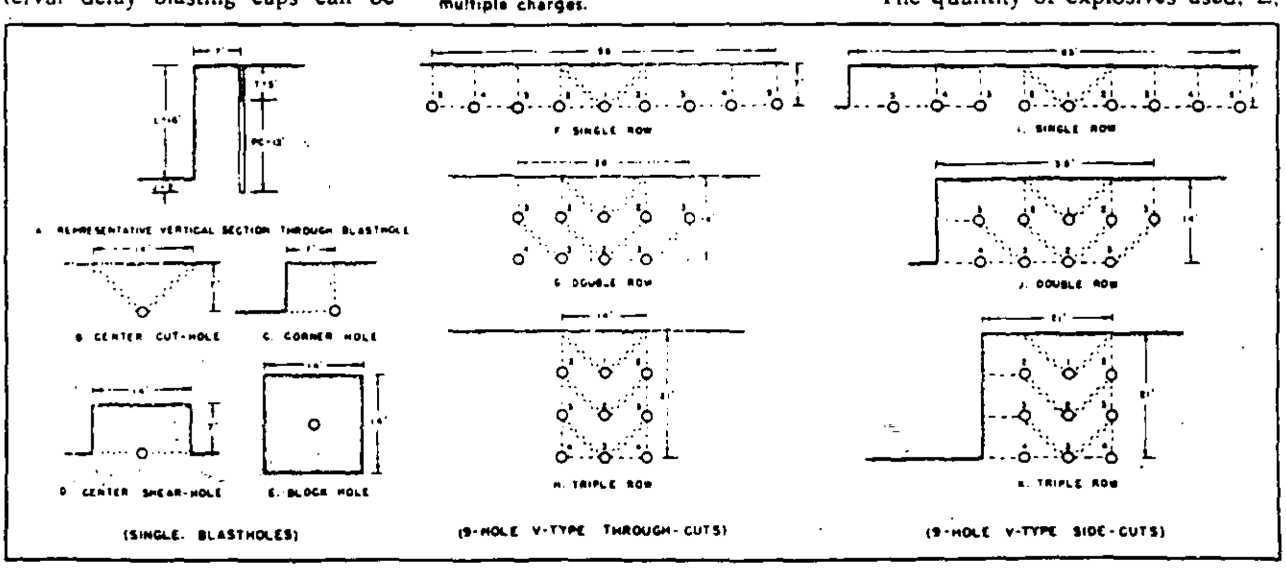


Figure 17—These sketches show four possible ways of blasting with a single charge and six patterns utilizing a V-cut arrangement for multiple charges.

Table 8—Change in Powder Factor (Pf) With Variation In Drill-Pattern Configuration [a]

(For blasting limestone with $d_r = 0.084$ ton/cu. ft. [b] by Extra 60% dynamite, D_e=2 inches[c], and blastholes located according to average K_B ratio of 30 [d].)

	Total No. Blastholes	Total Yield (tons)	Total Expl. Used (lb.)	Powder Factor (tons/lb.)
Single charges:				
(B) Center cut-hole (2 free faces)	1	66	22.7	2.91
(C) Corner hole (3 free faces)	1	66	22.7	2.91
(D) Center shear-hole (4 free faces)	1	132	22.7	5.82
(E) Block hole (5 free faces)	1	264	22.7	11.64
Multiple charges: V-type through-cut:				
(F) Single row	9	528	205	2.58
(G) Double row	9	462	205	2.25
(H) Triple row	9	396	205	1.93
Multiple charges: V-type side-cut:				
(I) Single row	9	594	205	2.90
(J) Double row	9	627	205	3.06
(K) Triple row	9	594	205	2.90

Notes: (a)—See Figure 17 for design specifications.
(b)—Rf. Table 7.
(c)—Rf. Figure 12.
(d)—Rf. Table 2.

would be the product of the explosive's loading density, d_e, the total average length of one charge, PC, and the total number of blastholes, N, calculated as follows:

$$E = (d_e)(PC)N, \text{ lb.} \quad (12)$$

The powder factor, Pf, would then be the ratio of the above two expressions, or

$$Pf = W/E, \text{ tons/lb.} \quad (13)$$

In studying Figure 17 and Table 8, it will be noticed that the number of free faces has a very pronounced influence on the value of the Pf. For multiple-hole blasts, when there is a free face added on one side, the over-all Pf's for all blasts will usually be the same as that for a single corner or cut hole. However, the Pf may be affected by the initiation-timing pattern employed, which may change the blast area outline, as shown in Figure 17J and line J of Table 8. For the particular blast in point, the additional tonnage results from overbreak in the tight corner of the second row of holes. If a later-interval initiation delay were used in the corner hole, the blast would then be expected to cut squarely without any overbreak, to give the same yield as for the other two examples (Figures 17I and 17K).

Estimating or evaluating an entire blast on a single-hole Pf basis can be very misleading, but unfortunately it is a practice often followed. For the design and evaluation of underground face-blasting, the errors produced would be even more serious and costly when based on a single-hole Pf. This is because there is an automatic elimination of potential tonnage for one complete row of holes. The row may be considered as serving merely to shear the cut out of the solid without achieving any effective production. It is also very important to recognize that in all blasting, when rows are added into the solid, with a subsequent reduction in the number of open faces, the Pf value will continue to change toward lower yields even though all other fundamental blasting relationships and the resulting rock fragmentation may remain substantially the same.

In surface or open-pit blasting the hole depths may vary within a particular cut or excavation, with no other changes being made in any of the other design dimensions. If column-loading is practiced, the Pf will change with the hole-depth variations. The trend is illustrated by data given in Table 9, in which the values represent conditions for the 9-hole blast shown in Figure 17F. The cause for the Pf variations is the result of changes in the ratio of the amount of hole used for stemming relative to the total hole depth. To counteract the lowering of yields, deck loading could be used, a practice commonly followed for deep holes particularly. This practice produces no detrimental effects on fragmentation when the decking is done properly.

Blasters should be cautioned regarding difficulties that may result from reducing the explosive loading density as a means for improving their Pf, or use of lighter grades or smaller diameter explosives. Attempts to extend drill-pattern dimensions by increasing burdens, etc., will produce similar difficulties for the same reason. Rather than sacrifice good fragmentation and displacement effects by decreasing the explosive energy, adjusting the blasthole arrangement is generally preferred. This can be done by redesign, so that more free faces are made available and charges are located more advantageously.

COST OF BLASTING

The primary concern of the quarry operator is to make a profit. To do this, costs must be kept to the minimum. Some costs, however, are interdependent, so that no single cost reduction may necessarily guarantee an over-all decrease in production expenses. It is the composite effect with which one must be

Table 9—Change in Powder Factor (Pf) With Variation of Hole Depth (H)

(9-hole single-row V-type through-cut, using Extra 60% dynamite with 2-in. D. column loaded and drill pattern dimensions* constant for blasting limestone with SG of 2.69)

Avg. H (ft.)	Avg. PC (ft.)	Avg. L (ft.)	Total Expl. Used (lb.)	Yield (tons) Total	Pf (tons/lb.)
10	5	8	79	264	3.34
12	7	10	110	330	3.00
14	9	12	142	396	2.79
16	11	14	173	462	2.67
18	13	16	205	528	2.58
20	15	18	236	594	2.52
22	17	20	268	660	2.47
24	19	22	300	726	2.42

Note: *See Figure 17 for drill pattern specifications.

concerned. In this respect many different costs and their effects on one another must be considered, some of which include the following: drilling, primary blasting, secondary breakage, loading, haulage, crushing, screening, stockpiling and reclaiming, loading and weighing for delivery to customers, supervision and engineering, maintenance, equipment and materials purchases and replacements, insurance, depletion and depreciation allowances, sales and other administrative services, royalties, stripping expenses (including ground breaking and removal), and taxes. Of all costs or expenses, the first seven (and in some instances those for stripping) generally constitute the major portion of costs for quarry production.

The percentage of total production costs attributed to drilling and blasting may be as low as 10 percent or as high as 40 percent. The relative importance of primary and secondary breakage costs to loading, haulage, crushing, etc., will depend largely on the properties of the deposit, equipment and plant operating characteristics, and results achieved from the primary blasting. Studies on quarry efficiency show that in most cases hourly production rates for well-blasted material are nearly double that achieved for poorly blasted rock. Similar results are obtained in the other types of mining and in heavy construction work. Crushing and screening costs are likewise appreciably reduced if the material is well blasted at the very beginning. Because of these effects the trend today is to spend more for primary blasting, because the savings realized from all the other production phases more than compensate for the initial added cost for blasting. This fact is evidenced by the lower powder factor yields obtained in a great many operations.

Primary blasting expense is normally considered to be composed of costs for both drilling and explosives, including all charges for labor and material used. Before the advent of the new high-speed highly mobile drills, the respective costs for drilling and blasting were about equal. But with the new types of drilling equipment, drilling costs of many operations are only half as much as the explosive with conventional high explosives.

Table 10—Blasting Cost Analysis showing Effects from Changing the Type of Explosive (V-type side-cut[a] for vertical holes in a limestone ledge with constant Pf)

A Assumed Conditions:

(1) Kept constant are $K_T = 0.7$, $K_J = 0.3$, $K_S = 1.0$, $D_e = D_H = 3$ in., $L = 20$ ft., and $d_r = 0.084$ ton/cu. ft.[b]

(2) E_1 = Extra 60% dynamite with SG = 1.28 and v_e = 12,200 fps[c]

(3) E_2 = field-mixed AN-FO, 94/6, with SG = 0.85 and v_e = 11,100 fps[d]

(4) All holes drilled with 4½-in. hammer track-mounted air-drill with 500 cfm compressor at average drilling rate of 400 ft. per 8-hour shift[e]

B Unit Costs[e]:

(1) Drilling at $0.363/ft.[e]
(2) Extra 60% at $0.22/lb.
(3) AN-FO, 94/6 at $0.05/lb.
(4) 30-ft. MS delay EBC at $0.62
(5) 6-ft. instant EBC at $0.17
(6) Regular Primacord at $0.32/ft.
(7) MS delay Primacord connector at $0.50
(8) Cast booster (½-lb. primer) at $0.50

C Blasting Data Calculations:

E_1 (Extra 60% dynamite)

$RE_1 = (1.28)(12{,}200)^2 = 191 \times 10^6$

If $K_{B1} = 30$, then $B = 7\frac{1}{2}$ ft. for equivalent drill pattern of 10 x 10 ft.[f]

$T_1 = K_T B_1 = (0.7)(7.5) = 5$ ft.

$J_1 = K_J B_1 = (0.3)(7.5) = 2\frac{1}{2}$ ft.

$H_1 = L + J_1 = 20 + 2\frac{1}{2} = 22\frac{1}{2}$ ft.

$PC_1 = H_1 - T_1 = 22\frac{1}{2} - 5 = 17\frac{1}{2}$ ft.

Since the blast consists of 3 rows of 3 holes each, or $N_1 = 9$ holes, then $W_1 = A_1 L(d_r) = 10(10)(9)(20)(0.084)$[j]

or $W_1 = 1510$ tons

If $d_{e1} = 3.9$ lb./ft.[k] and $E_1 = d_{e1}(PC_1)N_1$[l], then $E_1 = (3.9)(17\frac{1}{2})(9) = 615$ lb.

Thus, if $Pf_1 = W_1/E_1$[m], then $Pf_1 = 1510/615 = 2.46$ tons/lb.

The total required drill footage, or $H_1N_1 = (22\frac{1}{2})(9) = 203$ ft.

E_2 (Field-mixed AN-FO, 94/6)

$RE_2 = (0.85)(11{,}100)^2 = 105 \times 10^6$

$RE_2/RE_1 = 105/191 = 0.55$,[g] or $K_{B2} = 24\frac{1}{2}$.[h]

Thus, $B_2 = 6$ ft.[i] for equivalent square drill pattern of 8 x 8½ ft.[f]

$T_2 = K_T B_2 = (0.7)(6) = 4$ ft.

$J_2 = K_J B_2 = (0.3)(6) = 2$ ft.

$H_2 = L + J_2 = 20 + 2 = 22$ ft.

$PC_2 = H_2 - T_2 = 22 - 4 = 18$ ft.

To drill a complete pattern there should be 4 rows of 4 holes each, or $N_2 = 16$ holes.

Thus, $W_2 = A_2 L(d_r) = 8(8\frac{1}{2})(16)(20)(0.084)$[j]

or $W_2 = 1830$ tons

If $d_{e2} = 2.6$ lb./ft.[k] and $E_2 = d_{e2}(PC_2)N_2$[l], then $E_2 = (2.6)(18)(16) = 750$ lb.

Thus, if $Pf_2 = W_2/E_2$[m], then $Pf_2 = 1830/750 = 2.44$ tons/lb.

The total required drill footage, or $H_2N_2 = (22)(16) = 352$ ft.

D Blasting Cost Comparison: (Calculated from B and C, above):

	E_1 (Extra 60% dynamite)		E_2 (Field-mixed AN-FO, 94/6)	
Method of Initiation:	Electric	Nonelectric	Electric	Nonelectric
Drilling:	(203') $ 73.69	(203') $ 73.69	(352') $127.78	(352') $127.78
Explosives:				
Dynamite	(615#) 135.30	(615#) 135.30	—	—
AN-FO	–	—	(750#) 37.50	(750#) 37.50
Primers	—	—	(16) 8.00	(16) 8.00
Initiators:				
30' MS EBC	(9) 5.58	—	(16) 8.12	—
6' Inst. EBC	—	(2) 0.34	—	(2) 0.34
Primacord	—	(300') 9.60	—	(505') 16.16
Primacord MS connectors	—	(9) 4.50	—	(16) 8.00
Misc.:				
Connecting wire	1.25	—	1.25	—
Labor for loading and firing blast	2.00	1.80	3.50	3.00
Total blasting cost:	$217.82	$225.23	$186.15	$200.78
Cost per ton:	0.144	0.149	0.102	0.109

E Percentage Distribution of Blasting Costs:

Drilling	33.8	32.8	68.6	63.7
Explosives (Excl. primers)	62.2	60.1	20.1	18.7
Primers	—	—	4.3	4.0
Initiators	2.6	6.3	4.4	12.1
Misc.	1.4	0.8	2.6	1.5
Total	100.0	100.0	100.0	100.0

Special Notes—Table 10

(a)—See Figure 17K for general drill pattern and initation-timing system.
(b)—Rf. Table 7.
(c)—Rf. Table 1, p. 63, *Blasters' Handbook*, 14th edition. E. I. duPont de Nemours & Co.
(d)—Rf. Figure 6, p. 8, Technical Bulletin AG-2, Nov., 1960 Monsanto Chemical Co.
(e)—Rf. *A Field Man's Guide to Drilling Costs*, A. W. Foster, Atlas Chemical Industries, Inc.
(f)—Rf. Table 2
(g)—Rf. Formula (4)
(h)—Rf. Formula (5)
(i)—Rf. Formula (6) and Figure 14
(j)—Rf. Formula (11)
(k)—Rf. Formula (2) and Figure 12
(l)—Rf. Formula (12)
(m)—Rf. Formula (13)
(n)—Explosive unit costs based on schedule 1960 prices

With the introduction of inexpensive AN blasting agents, however, the drilling-explosive cost ratio has been reversed. Even though the less dense blasting agents appreciably increase the cost of drilling because of the increased number of blastholes required, the over-all drilling and blasting cost in most instances has been materially reduced. This is because of the tremendous savings in costs of explosives. Such blasting agents often cost only 20 to 30 percent as much as the conventional high explosives.

To illustrate the effects of the various components that determine primary drilling and blasting cost, Table 10 presents representative data for a typical quarry blast. Only the type of explosive has been changed, with the powder factor, drill-pattern general arrangement, and initiation-timing system kept the same. It should be noted from the data, however, that for conventional dynamite, i.e., Extra 60 percent, a typical 10- by 10-ft. pattern is used.

In order to use a regular AN-FO 94/6 blasting agent (field-mixed), the pattern dimensions are changed to an 8- by 8½-ft. arrangement. This is done according to the principles outlined earlier in the discussion on correlating the properties of explosives to the blasting standards. In this instance, the net result is that 16 blastholes are required for the AN-FO blast, compared to only nine holes for when Extra 60 percent is used. Because of the difference in the required true-burden dimension, other design dimensions necessarily must be adjusted to give a properly balanced blast. However, the basic K_T, K_J, K_S, and K_H ratios are kept closely to the same values for both blasts, only the K_B ratios being adjusted to suit the various characteristics of the explosives.

From the costs indicated in Table 10, one would logically conclude that everyone should change to AN-FO blasting agents. However, it must be kept in mind that individual circumstances may greatly change the over-all cost relationships. The factors that have the greatest influence on the final values would be the unit costs for drilling and explosives materials used and the particular properties of the explosives themselves, since the la[...] [...]de the final required drill p[...] [...]mensions, i.e., the K_B. Fur[...] some explosives simply [...] [...]ot be suitable for use under certain quarry operating condi[...] One should, therefore, recogni[...] [...]or making a cost analysis, [...] values for expenses and [...] of materials peculiar to [...] local circumstances should be [...] not general estimates, as was [...] for Table 10 data.

The influence of the p[...] of explosives on final costs cannot be overemphasized; this is t[...] [...]cularly of the velocity of the [...] since it has a very promi[...] effect on the most desirable dri[...] [...]ern. As described earlier, the [...]ufacturer's specifications may [...] [...]ly define whether the ve[...] [...]or unconfined or confined b[...] [...]or which charge diameter a[...] As one can see from Table 11 [...] specifications vary considerably [...] fact which in turn greatly aff[...] estimates for designing blast[...] [...] on energy potential (RE) of [...] [...]od-

Table 11—A Comparison of Published Explosives Specificatio[...]

(For competitive grades equivalent to 60% ammonia dynamit[...] when used with D_e=3 in. and based on published data)

	American(a) Ammonia Dynamite	Apache(b) Standard Dynamite	Atlas(c) Extra Dynamite	Du Pont(d) Red Cross Extra	Hercules(e) Extra Dynamite	Olin(f) Special Dynamite	Trojan(g) [...]nd[...]d [...]o[...]
Velocity (fps)	10,800	12,800	10,000	12,200	12,450	13,60[...]	[...]00
Open (O) or confined (C):	Not given	(O)	(O)	Not given	(O)	Not given	(O)
Charge diameter (inches):	Not given	1¼	1¼	1¼	1¼	Not given	[illegible]
Stick count:	110	110	110	110	110	1[...]	1[...]6
Specific gravity	1.28	1.28	1.28	1.28	1.28	1.[...]	[...]22
RE factor ($X10^6$)	149	210	128	191	198	2[...]	[...]93
Relative energy Ratio (RE_2/RE_1):(h)	0.78	1.10	0.67	1.00	1.04	1.[...]	[...]01
Adjusted burden (B_2 in feet):(i)	6.5	8.0	6.2	7.5	7.7	8.2	7.5
Equivalent drill Pattern (square):	8x9	10x11	8x8	10x10	10x11	11x	[...]10

References:
(a)—p. 2, Ammonia Dynamites specification sheet, A-2141-300-4/61, American Cyanamid Co.
(b)—p. 16, Apache Explosives catalog, third revision, Apache Powder C[...]
(c)—p. 21, Atlas Explosives Products, Catalog No. 13, 1957, Atlas Chemic[...] [...]stries, Inc.
(d)—p. 63, Table 1, *Blasters' Handbook*, 14th edition, 1958, E. I. du Pon[...] [...]mours & Co.
(e)—p. 4, Hercules Explosives, Blasting Agents and Blasting Supplies, [...] 1959, Hercules Powder Co.
(f)—p. 9 Olin Explosives Products catalog, fourth edition, 1955, Olin Mathie[...]n Chemical Co.
(g)—p. 4, Trojan Explosives and Blasting Supplies, Catalog No. 101, Troj[...] [...]der Co.
(h)—Relative energy ratios calculated on basis of Du Pont Red Cross [...] 60% as unity.
(i)—Figures 7 and 14, with K_B=30 for Du Pont Red Cross Extra 60%

uct. The suggested drill-pattern arrangements will not give the same powder factor yields but should produce comparable blast results, if the published specifications are not in error.

The expenses for primers and initiators may have a greater influence on final costs than one might expect, from the data indicated in Table 10. For blastholes with deck charges and those having extremely short depths, the costs for primers and initiators may constitute a considerable share of the over-all cost. Nevertheless, under such conditions the inherent savings resulting from higher powder factor yields usually compensate for the added costs. As experience has clearly shown, it is simply good practice always to use the best primers available. As a rule, the total required quantity of powerful high-energy primers is much smaller than that needed when cheaper low-energy explosives are used for priming. Initiator costs are also normally relatively low; so if improved blasting results can be insured by using additional initiators, the added expense could be considered insignificant, as compared to the benefits received.

As powder factor yields are reduced, costs will be increased proportionately. But irrespective of the actual powder factor value, blasts should always be designed to give the yield most suitable for maximum production at the least expense. In this respect, the percentage of usable material from a blast must also be given consideration. Well-blasted rock does not mean it must necessarily be pulverized. On the contrary, the required particle sizing and its uniformity must be such that maximum recovery is achieved. If, for example, 10 percent of the production is lost due to spoiling or waste, which in quarrying is quite common, the loss must be included in the final cost analysis. If recovery is reduced in order to increase rates of production, the value of the wasted material should logically be less than the savings accomplished from the lower operating costs for the material salvaged.

CONCLUSIONS

Effective blasting depends largely on a knowledge of how materials fracture, the particular characteristics of those materials, qualities of the various explosives that may be used, and recognition that the secret of efficient, economical, and safe results lies essentially in the suitable placement of charges where they will do the most good. Since explosives are merely very powerful tools for performing work, they should always be used accordingly.

As has been shown by these discussions, there are no easy, simple methods for solving blasting problems. The mechanisms and factors involved are too complex and numerous to permit clear-cut solutions. Each situation must be handled according to its own requirements, with the prudent use of one's best judgment. However, with a reasonable amount of study and understanding of operating conditions, blasters can evaluate results and make adjustments toward improvements by using certain basic standards. It has been the purpose of this article, therefore, to outline those standards and explain how they can be adjusted to apply to on-the-job conditions. But it must be realized that there can be no substitute for initial tests to ascertain what may be expected.

The burden dimension is the m[illegible] critical of the important fact[illegible] blasting. Its value must su[illegible] characteristics of the material being blasted and the properties of the explosives, and it must produce the desired degree of fragmentation and displacement. All other blasting standards are controlled by the burden value, and they should be designed on that basis. It should be, therefore, of primary concern to all blasters first to establish the best burden for their particular needs.

It has been shown that the powder factor as such has little meaning except as a relative basis for cost comparisons. For many years it has been used all too frequently, and unfortunately, as a means of judging blast efficiency. But under no circumstances can it be used as a reliable index for judging what one can expect in rock breakage or control of throw. Its value in costing is even questionable under many conditions.

FACULTAD DE INGENIERIA U.N.A.M.
DIVISION DE EDUCACION CONTINUA

CURSOS ABIERTOS

VI CURSO INTERNACIONAL DE INGENIERIA GEOLOGICA APLICADA A OBRAS SUPERFICIALES Y SUBTERRANEAS.

CUARTO MODULO:

TECNOLOGIA SOBRE EL USO DE EXPLOSIVOS.

THE LAWS OF ENERGY DIVERGENCE.

ING. RAUL CUELLAR BORJA.

THE LAWS OF ENERGY DIVERGENCE

By Richard L. Ash, P.E.

Energy transmitted through a homogeneous isotropic medium from an explosion-induced confined source is diverged outward equally in all directions. The effect is to reduce the unit energy at any one position within the medium as the distances increase away from the explosion center. This phenomenum as applied to mass, volumetric, or weight considerations is described by what is known as the cube-root law: whereas, the decrease of the effective stress or pressure with distance in any single direction is expressed by the square-root law. Both of the two laws serve as the principle bases for all explosive-energy design criteria. It is most important, therefore, that the laws be clearly distinguished from one another as to their specific applications.

The first assumption made in the use of the laws is that the explosion source consists of a single concentric or point charge, i.e., one that is initiated in its center and whose length-to-diameter ratio (L_e/D_e) is not more than 6. However, the laws can be applied equally as well to the long-length cylindrical charge normally used for the bulk of industrial blasting, providing one recognizes that only in the planes of the charge length will the effects differ from those produced by the concentric charge.

To comprehend the unique characteristics of the long-length charge it is convenient to consider that it is essentially no more than a continuous series or succession of individual point charges. Thus, stress energy from each segment normally will be released in a definite sequence or progressive order, from the point of initiation and then preceding to each adjacent portion of the charge remaining in the column. The resultant composite stress-form transmitted into the surrounding medium will vary in shape from that of a sphere to that of a cylinder with hemispherical ends, depending on the method of initiation used and the properties of both the explosives and containing material. For the cylindrical wave to form, for example, all points along the column must be initiated simultaneously. This is highly improbable for most types of blasting except for very short columns that are initiated in the center or moderate length charges that contain closely-spaced primers which are all initiated by means of instantaneous electrical blasting caps. Instead, the composite wave-form usually is either some form of conical shape or spherical. The conical wave results when the reaction velocity through the explosive column exceeds the compressive wave velocity of the surrounding medium. In this instance stresses from the charge ends in each of the planes intersecting the axis of the explosive column will be diverged like those from the spherical charge with changing angles of incidence at free-face planes, while stresses in the planes of the axis from the central portion of the column will be directed with a constant angle of incidence. Stresses in the planes of the charge diameter and perpendicular to the column axis, however, will be diverged much in the same circular manner as those produced from the spherical point charge.

/

conditions, it is possible to make reasonable approximations of the energy capabilities of various kinds of explosives and the estimated level of the transmitted stresses at any distance d from the explosion center. With respect to the relative energies of explosives with different densities but having a constant charge diameter and velocity of reaction, the following expression can be used:

$$RE_2 = RE_1 (SG_{e2}/SG_{e1})$$

In the case where explosives' densities, charge diameters, and reaction velocities all differ, the general relationship for making the comparison would be

$$RE_2 = RE_1 \; (d_{e2}v_{e2}{}^2/d_{e1}v_{e1}{}^2)$$

For all practical considerations, the comparisons would be reasonably valid for all center-initiated 1-ft long charges with diameters that would vary from 2 to 72 inches. This is because the range of values for L_e/D_e and its reciprocal, or D_e/L_e, would not exceed 6, which was defined earlier as the limitation for a point charge.

The significance of the foregoing relationships becomes apparent when one considers their application for cratering in materials. The problem, in gist, is one involving the accomplishment of mechanical work, whereby the energy supplied by the explosive (Q_e) is used for fracturing the materials by overcoming their strength properties and then displacing the broken particles. In general, the required diverged value of δQ_e, or δQ_d at distance d from the explosion's center, will be unique for any given type of material.

The specific depth of charge burial, which would correspond with the maximum limit for distance d, at which optimum crater results will be achieved is called the burden, B. The volume of the developed crater (V_c), in turn, will always be a function of B, as well as the explosive's Q_e. For example, V_c for a simple cone-type crater with one free surface is $\pi r^2 B/3$, but the value for the crater radius r is dependent on the material's properties and is related to B. Thus, as a general rule one can assume that $V_c = B^3 = Q_e$ for approximation purposes. From the previous discussions it was shown that $Q_e = RE_e = SG_e = D_e{}^3$. Therefore, for any confined explosive charge it can be concluded that

$$B = V_c{}^{1/3} = RE_e{}^{1/3} = SG_e{}^{1/3} = D_e,$$

and $$B_2 = B_1(RE_2/RE_1)^{1/3} \text{ or } B_2 = B_1(D_{e2}/D_{e1}).$$

In summation, the cube-root law describes the effect of three-dimensional divergence in reducing the stress energy produced by a confined explosive charge as the energy propagates in all directions

TABLE I. Strength Data For Some Competent Rocks (13).

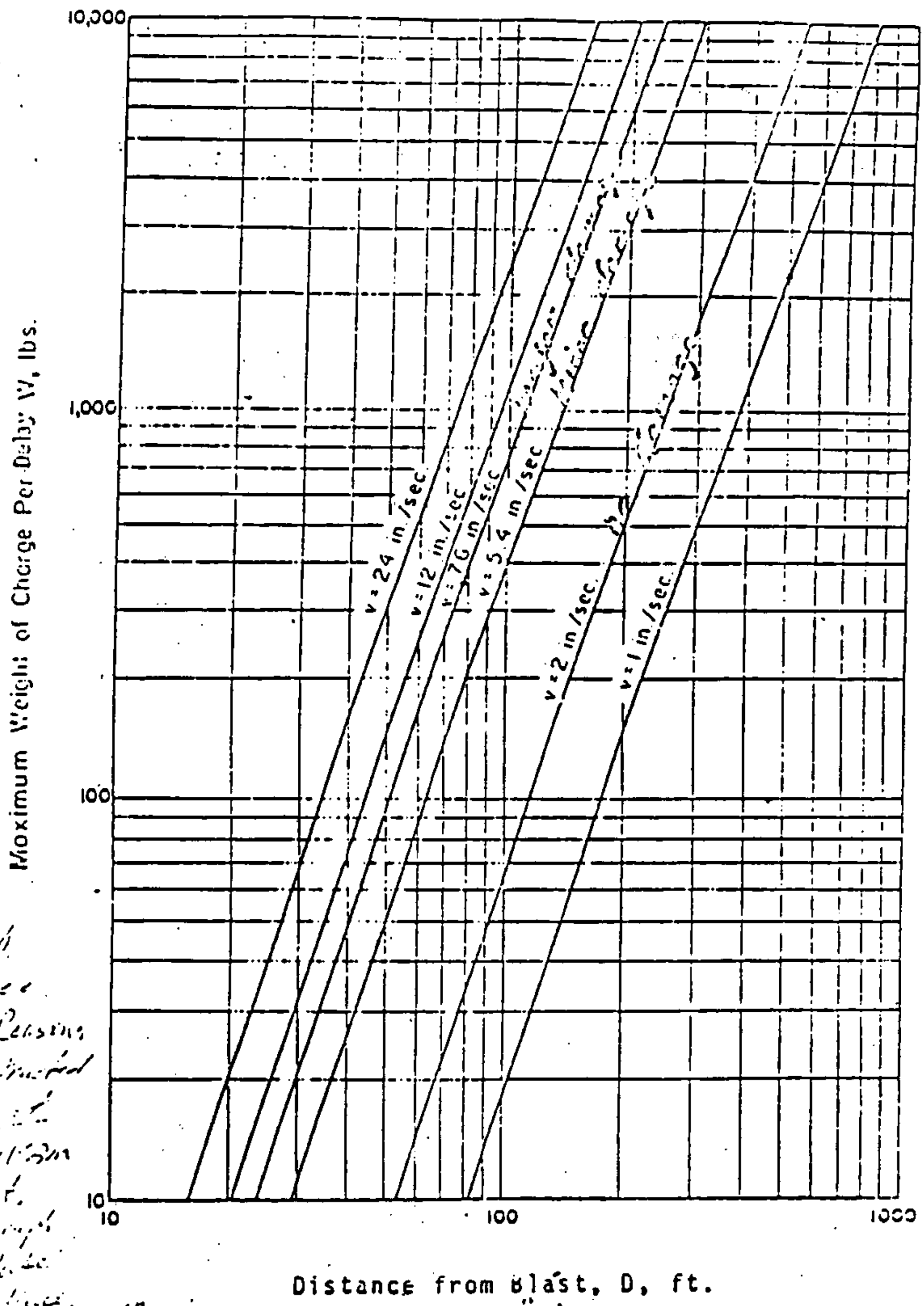
Rock Type	Compressive Strength psi x 10^3	Elasticity Modulus (Compression) psi x 10^6	Tensile Strength psi	$\frac{\text{Tens.}}{\text{Comp.}}$	C psi	ϕ deg.	Equation of Mohr's Envelope (ref. Fig. 3)	C/Compressive Strength
Chert	29.3	8.15	820	0.0280	2550	71.5	$Y = 2550 + 3.0\,x$	0.0870
Coal	6.2	–	–	—	1600	38.5	$Y = 1600 + 0.8\,x$	0.258
Granite	28.0	3.17	410	0.0146	1720	76.5	$Y = 1720 + 4.2\,x$	0.0614
Green Stone	29.1	8.82	380	0.0130	1700	77.5	$Y = 1700 + 4.5\,x$	0.0584
Greywacke	7.9	1.80	700	0.0886	1200	59.5	$Y = 1200 + 1.7\,x$	0.1519
Limestone	21.3	9.50	350	0.0164	1320	75.5	$Y = 1320 + 3.9\,x$	0.0620
Marble	30.8	7.15	863	0.0279	2650	71.0	$Y = 2650 + 2.9\,x$	0.0860
Salt Rock	2.2	1.35	85	0.0386	210	72.5	$Y = 210 + 3.2\,x$	0.0954
Sand Stone	14.8	2.00	230	0.0155	900	76.0	$Y = 900 + 4.0\,x$	0.0608
Shale	5.2	1.09	1538	0.2957	1420	31.0	$Y = 1420 + 0.6\,x$	0.2731
Silt-Stone	5.0	12.60	440	0.0880	750	59.5	$Y = 750 + 1.7\,x$	0.1500

C – Cohesion

ϕ – Angle of Internal Friction

Y – Shear Stress, τ_s

x – Normal Stress, σ_n



minor damage: fine plaster cracks, opening of old cracks

major damage: serious new cracking, plaster falls

SOLUTION (CONT.)

C.(1) From Eq. 36, $K_r = \frac{V_c}{V_p}$. Thus, from part E and determining K_r for the respective V_c values for each D_c from 1 to 6 inches, inclusive, the following summary table can be prepared:

D_c, in	B, ft	V_c, fps	K_r
1	0	0	0
2	4	9450	0.56
3	7	12,500	0.74
4	10.3	14,000	0.82
5	13.5	14,900	0.88
6	16.2	15,000	0.88

(a) Floor Priming

From Eq. 38(a), $B' = \frac{3L}{9K_r + 2}$

1. At $D_c = 5$ in,

$$B' = 3(30)/(9 \times 0.88 + 2) = 90/9.9 = 9.1 \text{ ft.}$$

From above Table, $B = 13.5$ ft.

Thus, $B > B'$ or $13.5 > 9.1$. Diameter can be reduced.

2. At $D_c = 4$ in,

$$B' = 3(30)/(9 \times 0.82 + 2) = 90/9.4 = 9.6 \text{ ft.}$$

From Table, $B = 10.3$ ft.

Thus, $B > B'$ or $10.3 > 9.6$. Diameter can be reduced.

3. At $D_c = 3$ in,

$$B' = 3(30)/(9 \times 0.74 + 2) = 90/8.66 = 10.4 \text{ ft.}$$

From Table, $B = 7$ ft.

Thus, $B < B'$ or $7 < 10.4$. Diameter too small.

Note that at $D_c = 4$ in., optimum burden B and the minimum burden B' at which misfire might occur are approximately equal. Therefore, use $D_o = 4$ in. ⟸

SOLUTION (CONT.)

B (4) (CONT.)

In general form, therefore, $B = 2.33\,D_c \left(\frac{V_c}{12{,}000}\right)^{\frac{2}{3}}$, ft.

(a) $D_c = 2$ in.

$$B = 2.33(2)\left(\frac{9450}{12{,}000}\right)^{\frac{2}{3}} = 4.66\,(0.86) = 4.0 \text{ ft} \Leftarrow$$

(b) $D_c = 4$ in.

$$B = 2.33\,(4)\left(\frac{14{,}000}{12{,}000}\right)^{\frac{2}{3}} = 9.32\,(1.11) = 10.3 \text{ ft} \Leftarrow$$

Also, for $D_c = 3$ in.

$$B = 2.33\,(3)\left(\frac{12{,}500}{12{,}000}\right)^{\frac{2}{3}} = 7\,(1.02) \simeq 7 \text{ ft}$$

And for. $D_c = 5$ in. and larger,

$$B = 2.33\,D_c\left(\frac{15{,}000}{12{,}000}\right)^{\frac{2}{3}} = 2.33\,D_c\,(1.16) = 2.7\,D_c$$

Thus, at $D_c = 5$ in, $B = 2.7\,(5) = 13.5$ ft.

At $D_c = 6$ in, $B = 2.7\,(6) = 16.2$ ft

B (5) From Eq. 33, $T \simeq 2B/3$

Thus, (a) $D_c = 2$ in., $T \simeq 2(4)/3 = 2.7$ ft $\Leftarrow$

(b) $D_c = 4$ in, $T \simeq 2(10.3)/3 = 6.9$ ft $\Leftarrow$

B (6) From Eq. 32, $J \simeq B/3$

Thus, (a) $D_c = 2$ in, $J \simeq 4/3 = 1.33$ ft $\Leftarrow$

(b) $D_c = 4$ in, $J \simeq 10.3/3 = 3.43$ ft $\Leftarrow$

SOLUTION (CONT.)

B (1) (CONT.)

Therefore, basic velocity equation for the explosive is

$$V_e = \frac{5000\,(D_c - 1)}{0.26 + 0.27\,(D_c - 1)}$$

with the D_c range of values from 1 to 5 inches.

Check: @ D_c = 3 in.

$$V_e = \frac{5000(3-1)}{0.26+0.27(3-1)} = \frac{10,000}{0.26+0.54}$$

$$= 12,500 \text{ fps} \qquad \underline{OK}$$

@ D_c = 5 in.

$$V_e = \frac{5000(5-1)}{0.26+0.27(5-1)} = \frac{20,000}{0.26+1.08}$$

$$= 14,900 \text{ fps} \qquad \underline{OK}$$

(a) D_c = 2 in.

$$V_e = \frac{5000(2-1)}{0.26+0.27(2-1)} = \frac{5000}{0.26+0.27} = 9450 \text{ fps} \Leftarrow$$

(b) D_c = 4 in.

$$V_e = \frac{5000(4-1)}{0.26+0.27(4-1)} = \frac{15,000}{0.26+0.81} = 14,000 \text{ fps} \Leftarrow$$

B (2) From Eq. 4 (a)

$$P_d = \frac{6.06\times10^{-3}\,V_e^2\,(SG_e)}{1 + 0.80\,(SG_e)}$$

From Eq. 1

$$SG_e = \frac{141}{SC} = \frac{141}{117} = 1.2$$

Then

$$P_{d_{max}} = \frac{6.06\times10^{-3}\times1.5^2\times10^8\times1.2}{1+0.80(1.2)} = \frac{6.06\times2.25\times10^5\times1.2}{1.96}$$

or $\qquad P_{d_{max}} = 835,000$ psi

SOLUTION TO SINGLE BLASTHOLE DESIGN PROBLEM

A(1) From Eq. 21,

$$\sigma_t = \sigma_c \left(\frac{1-\sin\phi}{1+\sin\phi}\right)$$

Then

$$\frac{1750}{25{,}000} = \frac{1-\sin\phi}{1+\sin\phi}$$

or

$$0.07(1+\sin\phi) = 1-\sin\phi$$

$$0.07 + 0.07\sin\phi = 1-\sin\phi$$

$$1.07\sin\phi = 0.93$$

or

$$\sin\phi = 0.87$$

Thus,

$$\phi = 60 \text{ deg}$$

From Eq. 22(b),

$$\tau_s = \frac{\sigma_c}{2}(\cos\phi)$$

$$= \frac{25{,}000}{2}(0.5)$$

or

$$\tau_s = 6250 \text{ psi}$$

A(2) From Eq. 14(b)

$$V_p = \left[\frac{E_r(1-\mu)}{\rho_r(1+\mu)(1-2\mu)}\right]^{1/2}$$

From Eq. 8(b)

$$\rho_r = 1.941\, SG_r$$

Thus, substituting given values of V_p, μ, and SG_r and squaring both sides of Eq. 14(b),

$$(17{,}000)^2 = \frac{E_r(1-0.25)}{1.941(2.9)(1+0.25)(1-2\cdot 0.25)}$$

Rearranging

$$E_r = \frac{1.7^2\times 10^8(1.941)(2.9)(1.25)(0.5)}{0.75}$$

or

$$E_r = 13.5\times 10^8 \text{ psf} = 9.4\times 10^6 \text{ psi}$$

SOLUTION (CONT.)

C(1) (CONT.)

(b) Primer at Center of Charge Column.

$$\text{From Eq. 38(b)}, \quad B' = \frac{3L}{18K_v+1}$$

1. At D_e = 5 in.,

$$B' = 3(30)/(18 \times 0.88+1) = 90/16.8 = 5.4 \text{ ft.}$$

From Table B = 13.5 ft.

B is much greater than B' indicating diameter can be much smaller, i.e., $B > B'$ or $13.5 > 5.4$.

2. At D_e = 3 in.,

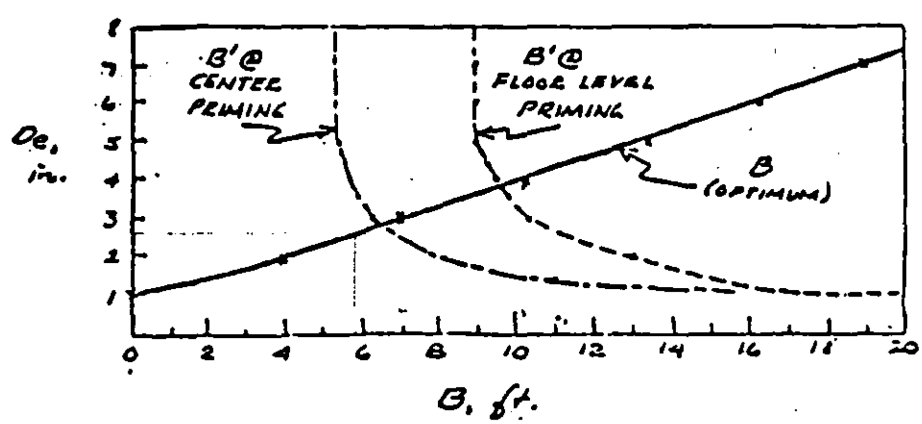
$$B' = 3(30)/(18 \times 0.74+1) = 90/14.3 = 6.3 \text{ ft.}$$

From Table B = 7 ft.

Values of B and B' are approximately equal with $B > B'$ a small amount, which is desirable.

Therefore, use D_e = 3 in. ⟸

NOTE The previous solutions can be solved quite simply by plotting values as shown below:



C(2)

From Eq. 39, $B'' = 0.62L = 0.62(30) = 18.6$ ft ⟸

This burden would be for $D_e = \frac{B''}{2.7} = \frac{18.6}{2.7} = 6.9$ in.

VI *CURSO INTERNACIONAL DE INGENIERIA GEOLOGICA APLICADA A OBRAS SUPERFICIALES Y SUBTERRANEAS*
CUARTO MODULO
TECNOLOGIA SOBRE EL USO DE EXPLOSIVOS

A N E X O 2

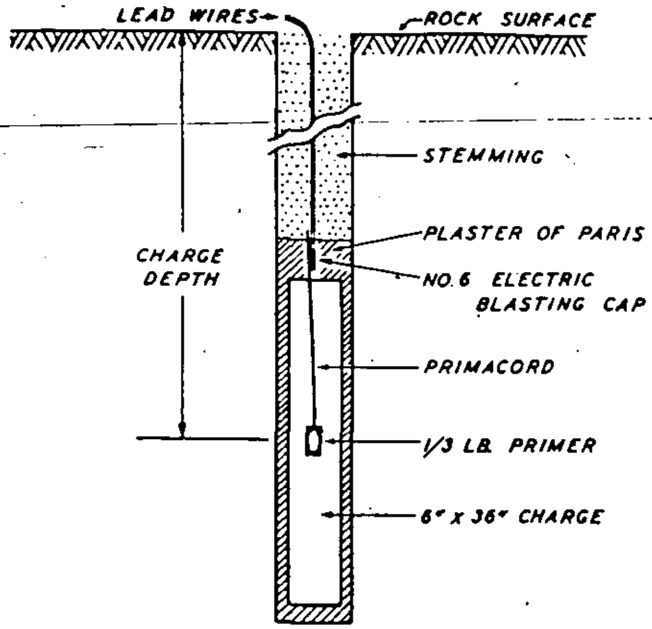


FIG. 1. Typical Dow volume crater charge showing position of explosives charge, detonators and stemming.

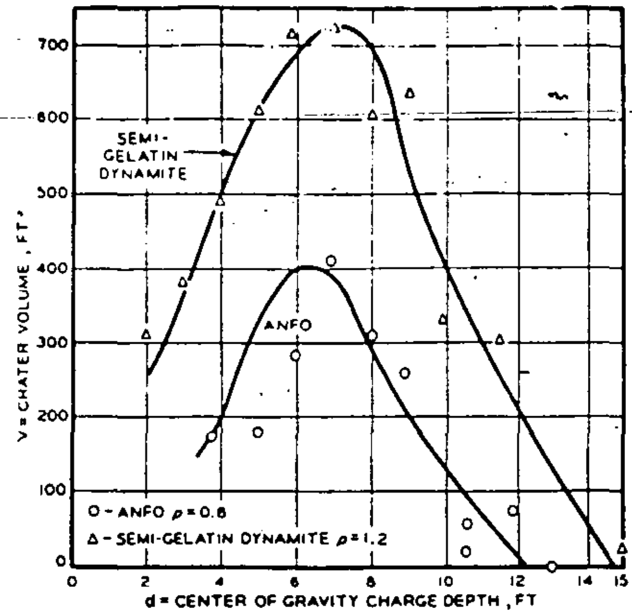


FIG. 2. Crater depth relationships for semi-gelatin dynamite are compared with those for ANFO charges.

Simplified Explanation of Crater Method

CHARLES H. GRANT

Editor's Note: Widespread interest in the "explosives algebra" article published in E&MJ, August 1964, prompted us to ask Mr. Grant for a more complete explanation of the cratering methods employed in introducing Dow's new metallized blasting agents. Mr. Grant prepared this article to assist readers in understanding the crater method.

AT THE RISK OF RIDICULE from some of my fellow engineers, I am going to try to reduce the principles and practices of rock cratering with explosives to the simplest possible terms for two reasons:

1) Because, too often, crater methods are ignored or misunderstood at field level which is where they should be best known; and, as a result, new explosives are usually tested in full-scale production blasting where poor performance can result in heavy losses.

2) Because only if those at field level have an understanding of cratering, can we at Dow (or anyone else) bring out the true characteristics of any new blasting agent or explosive as proved in the only proper laboratory for a rock-breaking material—that of rock itself.

Let's start out by giving full credit to the untiring enthusiasm of consultant C. W. Livingston, whose efforts to put his original theories into practical applications were so significant in gaining recognition for all crater methods. Credit must also be given to Messrs. Duvall, Rinehart, Nichols and Atchison of the U.S. Bureau of Mines, M. A. Cook of the University of Utah, Allen Bauer of C.I.L., K. Hino of Japan, and many others who have helped to increase this knowledge.

All of us who have gotten involved in cratering, however, have often forgotten that the work which seems so close and clear to us is almost a foreign language to those whose daily job is planning and executing the primary blasting which our investigations have endeavored to improve.

So this is an effort at communication, in the hope that proper testing methods will replace production blasting as a proving ground for new explosives and blasting agents.

For many years "Curly" Livingston's methods for determining explosives performance in rock were simply branded as "theories" and not given the practical application they deserved. It took much patient work before he established their validity and gained justly deserved recognition. Before stating the Livingston formula let's simply say that it measures how far down in a blast hole you can detonate a given weight of explosive and still pull rock at the top of the hole. The depth at which the explosive no longer breaks rock is called the critical depth. Its algebraic symbol is N.

The trick is to take this critical depth, N, and find some reproducible relationship between it and the explosive being used, and the size of the crater formed. Livingston determined theoretically and experimentally that there was a constant factor between critical depth and the cube root of the weight of the explosive. He expressed it this way:

$$N = EW^{1/3}$$

That E is called the strain energy factor. You could also call it a weight crater constant. It is simply there. You have the critical depth and the cube root of the weight of your explosive, and between them there is this factor. You can transpose the equation and solve for this factor as follows:

$$E = \frac{N}{W^{1/3}}$$

A practical application of this might work out this way. Say you have an 8-lb charge of explosive and in a series of crater tests you establish that the critical depth for this explosive in this rock is 6 ft. You solve for the strain energy factor (or weight crater constant) as follows:

$$E = \frac{N}{W^{1/3}} = \frac{6}{8^{1/3}} = \frac{6}{2} = 3$$

Now what do you have? Nothing yet, but since you have measured volumes of various craters you have blasted in searching for the critical depth of the explosive, you have found a certain depth at which the explosive produced the greatest volume of crater, and this gives you

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another relationship. The depth at which the explosive produced the largest crater you can call the optimum depth, and there is a ratio you can form between this depth and the critical depth. This can be stated as follows:

$$\text{Optimum Depth Ratio} = \frac{\text{Optimum Depth}}{\text{Critical Depth}}$$

Say you found that a given explosive in a certain rock had a critical depth of 10 ft and an optimum depth of 5 ft. Then you have this equation:

$$\text{Optimum Depth Ratio} = \frac{5\text{ ft}}{10\text{ ft}} = .5$$

With this ratio, you can go to the practical application of the weight crater method which is expressed in this formula:

$$W = \left(\frac{\text{Distance}}{\text{Optimum Depth Ratio} \times E}\right)^3$$

Here Distance equals the number of feet to the center of gravity of the explosive charge. Now you have something you can take right into the pit to determine the charge you want to put into a specific blast hole or series of blast holes.

According to H. E. Farnam, manager of operations of the Iron Ore Co. of Canada, "From this point on, bench geometry becomes a problem of mathematically turning the crank, substituting numbers for burdens, spacing, and depth." In calculating bench geometry, a number of burden distances are arbitrarily chosen and substituted for Distance in the above equation. Then Iron Ore Co. engineers solve for the number of pounds of explosive required for each chosen burden. With the burden and the explosive weight known, the depth of hole and the spacing are calculated, and thus they solve a number of bench configurations for a specific rock and a specific explosive.

Using a typical example, the above equation can be easily applied. In an iron formation with a C.I.L. slurry, the strain energy factor has been determined as being 4.26 and the optimum depth ratio as being .52. Substituting an 18-ft burden for distance, we have:

$$W = \left(\frac{18}{.53 \times 4.26}\right)^3$$

$$\left(\frac{18}{2.25}\right)^3 \quad \text{or} \quad (8)^3$$

W = 512 lb of slurry per hole (when burden equals depth of center of gravity of the charge)

In practice, Iron Ore Co. furnishes pit foremen with precalculated cards designating the type of material and explosive and listing information on bench height, burden, hole spacing, depth of hole, weight of explosive per hole, height of explosive column in the borehole, and the height from the top of the explosive column to the collar. With the cars (Table I) the foremen are able to lay out blast patterns for any bench in any of Iron Ore Co.'s rock types for any of their available explosives. The cards allow a margin of safety in the burden calculations to guard against underloading.

According to Farnam, this system of calculating bench and blast configurations is the first systematic method Iron Ore Co. has tried that has produced satisfactory results, increasing broken rock yield 30% per foot of drilled borehole and reducing blasting cost 40% with numerous other benefits.

"The system is not perfect, however, and considerable work must be done to perfect the theoretical aspects of the procedure," Farnam has reported.

This was the situation we at Dow faced when we first tried to gain accurate measurements of the effectiveness of our line of metallized blasting agents. We found that the existing crater methods which compared explosive using charges of constant weight made it necessary to vary the charge size in accordance with the density of each explosive being tested. If we kept the boreholes at a uniform diameter, this uniform weight changed the shape of each explosive charge, and we found that this shape affected our results. This got us into an intensive investigation of charge geometry. Results from underwater testing showed that with some explosive compositions there was a significant variation in performance with changes in the length-to-diameter (L/D) ratio of the charge, up to an L/D radio of 4.

A series of cratering experiments comparing performance of charges at L/D ratios of 2 and 6 with both ANFO and semi-gelatin dynamite, showed that a difference in performance also manifested itself in rock. At an L/D ratio of 2, more rock was broken per unit of explosive. At an L/D ratio of 6, the depth at which any explosive would affect the surface of the rock (critical depth) was greater. But of most significance for the purposes of the study, data obtained at an L/D of 6 was more reproducible and acted more like the typical column charge used in actual production blasting. Therefore, we determined that this charge geometry was essential for valid testing, and an L/D ratio of 6 was selected for all comparative work.

But this was not the only reason we began comparing explosives and blasting agents on the basis of a constant explosive volume and shape rather than a constant weight basis. Many other factors indicated that a volume method would give us more meaningful comparative data on explosive performance.

Using a constant volume, we avoid the problem of having to scale charges by weight and we can correlate the data with a single function. More important, the use of a constant volume of explosive means that each sample faces an identical rock resistance, and this also helps reproducibility.

To ensure that all experiments would be conducted with a charge diameter greater than critical diameter for the least sensitive composition expected to be evaluated, a constant diameter of 6 in. was selected.

These considerations committed us to a charge geometry for all experiments of 6x36 in.—a massive charge for cratering but possessing many advantages. The charge is of sufficient size so that the effect of the priming system does not distort our results. The effect of minor variations in rock structure is overcome. And, from the point of view of cost, having all holes of the same diameter simplifies our drilling problem. Fig. 1 shows a typical Dow volume crater charge as loaded.

In making a crater test with the volume method, the weight of the explosive varies with its density, and this is recorded. The charge is primed with a ½-lb high density pentolite primer placed at the center of the 6x36 in. cartridge. The primer is initiated with 100-grain detonating fuse. The fuse extends only to the top of the charge where the electric blasting cap is attached. The charge is cemented into the borehole with plaster of paris to ensure good con-

TABLE FOR PIT FOREMEN

MATERIAL—IRON FORMATION EXPLOSIVE—HYDROMEX

E = 4.26 Δ_r = 0.53

BH	BURDEN	SPACING	DEPTH	WEIGHT	COLUMN	COLLAR
21	17	24	25	506	10	15
24	18	25	28	603	12	16
26	19	26	30	710	14	16
28	20	28	32	825	17	16
31	21	29	35	958	19	16
33	22	31	37	1102	22	16
36	23	32	40	1258	25	15
39	24	34	43	1426	29	14

TABLE I. This is the card which the Iron Ore Co. of Canada gives to pit foremen to slove blasting problems.

tact between the explosive and the rock. The hole is stemmed with finely crushed rock or tailings. After the charge has been shot, the crater, if one is obtained, is carefully excavated and cleaned, and its volume is calculated from measurements taken in a meticulous sectioning procedure.

As in the weight crater method, the critical depth is determined as the charge depth where the rock just starts to fail at the surface by cracking or spalling. At all depths less than critical, of course, craters will result.

With the volume crater method we use a somewhat different set of symbols than those in the weight crater method. The symbol N for critical depth stays the same, but instead of E (the strain energy factor or *weight* crater constant) we use the symbol Σ or sigma; to stand for the *volume* crater constant. And instead of taking the cube root of the *weight* of the explosive, we work in terms of the cube root of the volume, for which we use the symbol v—the explosives volume in cubic inches. Thus the Dow crater formula appears as follows:

$$N = \Sigma v^{1/3}$$

or in solving for Σ it can be written

$$\Sigma = \frac{N}{v^{1/3}}$$

Since the volume of the explosive in cubic inches is expressed as v, the volume of the crater in cubic feet is expressed as V. This can be plotted against the center of gravity depth of the explosive charge, d, which is expressed in feet. This is shown in Fig. 2, which compares a semi-gelatin dynamite with ANFO.

Three features of the curves may be noted:

1) Crater values at relatively shallow depths are not meaningful, because of the lack of confinement in this region, which is referred to as the airblast range.

2) The peaks of the curves represent the optimum depth, as described under the weight crater method but here determined volumetrically.

3) The point at which no crater is produced and the confinement of explosives energy is complete in the critical depth and will ordinarily be different for different explosives.

All this, as you can see, is relatively close to the weight crater method, but at this point the symbols and reasoning of the volume method begin to differ somewhat from conventional methods.

One difference is in the use of the symbol Δ (delta) to express the ratio of the center of gravity of the charge depth to the critical depth, or

$$\Delta = \frac{d}{N}$$

Table II Symbols

d	=	Center of gravity charge depth, ft
v	=	Explosive volume, cu in.
W	=	Explosive weight, lb
ρ	=	Explosive density, lb per cu in.
V	=	Crater volume, cu ft
N	=	Critical depth, ft (to center of gravity of charge)
E	=	Weight crater constant
Σ	=	Volume crater constant
Σ	=	$E\rho^{1/3}$ when $W = \rho v$
Λ	=	Reduced charge depth = d/N
$K(\Lambda)$	=	Reduced crater volume = V/N^3
$K(\Lambda)'$	=	A variation of $K(\Lambda) = \frac{V}{d^3} = \frac{K(\Lambda)}{\Lambda^3}$

In conventional weight crater calculations, the delta sign represents optimum depth only. We at Dow, and a growing number of other crater testers, find there is a greater flexibility in our calculations when we use delta to express the ratio of d/N wherever the charge may be placed in the hole—at depths either more or less than critical. If the center of gravity of the charge depth is halfway to the collar from critical depth, we have a delta of .5. If it is below critical depth we have a delta of over 1.

Thus, it is convenient to call delta the reduced crater depth and think of it simply as the ratio of center of gravity charge depth, d, to the critical depth, N. It represents points of constant interrelationship of rock and explosive, or points of equal relative confinement.

Another ratio which comes in handy in finer crater calculations is the relationship of the actual crater volume, V, to the cube of the critical depth. We call this the K (Δ), and the formula for finding it is expressed as follows:

$$K(\Delta) = \frac{V}{N^3}$$

Being nothing but ratios, both Δ and K (Δ) are dimensionless. K (Δ) has the same value for any explosive at points of equal relative confinement and so may be considered a function of the reduced charge depth, Δ.

Our object in working out these ratios was to reduce the large number of experiments required to define a complete cratering curve.

In Fig. 2, you can see that the general shape of the two cratering curves is the same. You might say that only the dimensions are different. Therefore, it is mathematically possible to drop the dimensions and keep only the relationships. Since these relationships are constants, the curves can become the same. Fig. 3 shows what happens when we plot the Δ and K (Δ) ratios of the tests in Fig. 2.

In Fig. 3, within experimental error, a common curve is obtained over the major portion of the range. Values for ANFO drop off in the range of poor confinement, apparently because ANFO does not detonate well in this range. Normally crater experiments are performed in the range between optimum and critical depth to avoid the uncertainties of the airblast region.

The objective of all this is to compare explosive performance by comparing the crater volumes produced at points of equal relative confinement. In Fig. 3, this condition is satisfied for craters obtained at equal values of Δ or, because of the dependency of K (Δ) on Δ, at equal values of K (Δ). To compare two craters formed by equal volumes of explosive 1 and explosive 2,

$$K(\Delta)_1 \text{ must equal } K(\Delta)_2 \text{ or } \frac{V_1}{N_1^3} = \frac{V_2}{N_2^3} \text{ or } \frac{V_1}{V_2} = \frac{N_1^3}{N_2^3}$$

The ratio of the cubes of the critical depths has the same value as the ratio of crater volumes and is an equivalent measure of relative explosive performance.

Although in practice it is usually more convenient to determine the critical depth and make the comparison on this basis, it is now theoretically possible to shoot one crater with a new explosive and from the values of crater volume, V, and depth of charge, d, to determine the critical depth, N, for the explosive. The data point must be moved by trial and error on Fig. 3 until its location on the reduced crater curve gives a value of N that satisfies the values of both Δ and K (Δ) at that point.

For instance, if you had an explosive (say ANFO) which gave you a crater volume (V) of 200 cu ft at a depth (d) of 9 ft, you could assume various critical depths to find the one which would satisfy the values in Fig. 3. The only critical depth which would do this would be 12 ft:

$$\Delta = \frac{d}{N} = \frac{9}{12} = .75$$

and

$$K(\Delta) = \frac{V}{N^3} = \frac{200}{1728} = .108$$

— or with a semi-gelatin you would find that only an assumed critical depth of 15 ft, based on a charge depth of 10 ft and a crater volume of 400 cu ft would give you a correlation of Δ and K (Δ) as follows:

$$\Delta = \frac{10}{15} = .67$$

$$K(\Delta) = \frac{400}{3475} = .108$$

To avoid trial and error solutions, another parameter or ratio, K (Δ)′, can be defined in terms of the experimentally determined crater volumes and charge depth

$$K(\Delta)' = \frac{V}{d^3} = \frac{K(\Delta)}{\Delta^3}$$

K (Δ)′ is a mathematical variation of K (Δ); it has the advantage that it can be determined directly from field data.

Fig. 4 is a plot of K (Δ)′ against K (Δ) using data for the calibrating explosives ANFO and semi-gelatin dynamite. Having calculated K (Δ)′ from the experimental data for a single shot, a value of K (Δ) can be found that will enable us to determine Δ from the reduced crater curve (Fig. 3) hence the critical depth.

As an example, if a given explosive produces 1000 cu ft of crater at a depth of 10 ft,

$$K(\Delta)' = \frac{V}{d^3} = \frac{1000}{10^3} = \frac{1000}{1000} = 1$$

But K (Δ)′ also equals $\frac{K(\Delta)}{\Delta^3}$, so we can determine that

$$1 = \frac{K(\Delta)}{\Delta^3}$$

and by employing the rock function curve to determine that K (Δ) is .185 [dropping down from K (Δ)′ = 1 on a finer calibration than shown here] we find

$$\Delta^3 = .185$$
$$\text{or } \Delta = .57$$
$$\text{hence } N = 17.5$$

This can be checked against the reduced crater curve, Fig. 3, if you don't trust your algebra.

Those unfamiliar with "explosives alegebra" may find the latter part of this analysis somewhat confusing. Actually K (Δ)′ can be plotted against Δ to obtain the critical depth from a single curve, but this presentation attempted to trace the manner in which the method developed.

In practice, because of the experimental error inherent in crater work, data from more than one shot with a test explosive are used in comparing explosive performance.

To make comparison more meaningful, results of experiments comparing various explosives are reported in terms of the volumes of standard explosive equivalent in crater performance to one volume of the test explosive. This is equal to the inverse ratio of the cubed critical depths.

$$\frac{V_1}{V_2} = \frac{N^3_2}{N^3_1}$$

Examples of such comparisons are given in Table III.

That about wraps up the basic algebra involved in volume crater testing. Now all you have to do is translate all this into action in your rock with the explosives or blasting agents available at your operation. With these techniques, you should be able to determine the best explosive for your job without gambling a single production blast.

Along with certain other companies, Dow offers the services of experienced crater technicians on a per diem—excess cost basis to conduct crater tests in the user's rock formation or to assist in the loading and firing of trial production blasts.

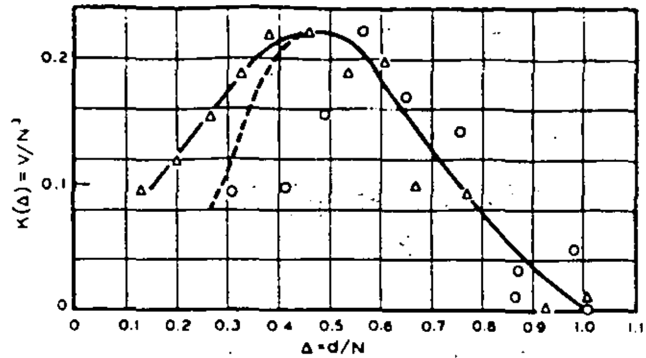

FIG. 3. Reduced crater curve. Data are plotted for both ANFO and semi-gelatin dynamite.

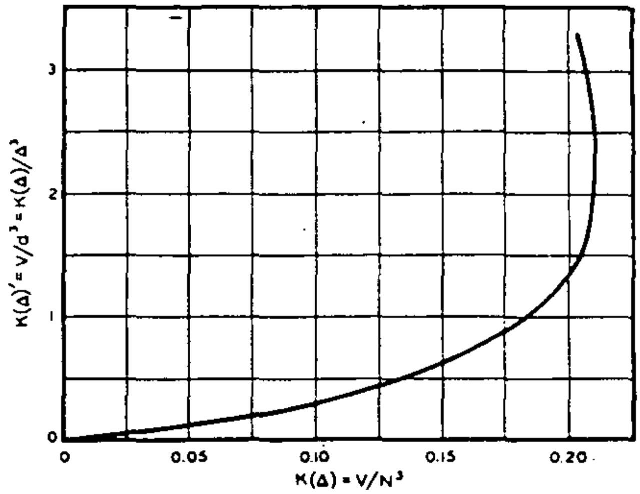

FIG. 4. Rock function curve derived from ANFO and semi-gelatin dynamite data.

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Results of Minnesota and Georgia Crater Shooting in Granite

Explosive Type	Density, g per cc	V AN-FO per V Explosive	Wt AN-FO per Wt Explosive
Ammonium nitrate and fuel oil	.80	1.00	1.00
Hi-Explosive slurries nonmetallized	1.5	1.0 - 1.65	.53 - .88
Hi-Explosive slurries iron-metallized	1.7	1.1 - 2.2	.52 - 1.04
Hi-Explosive slurries metallized with aluminum	1.5	2.6	1.48
Slurried explosive metallized with aluminum (containing no Hi-Explosives)	1.20	5.0 ± .2	3.33

TABLE III. Comparison of various blasting agents on the basis of weight and volume, using the inverse ratio of their cubed critical depths as derived by the Dow volume-crater method described in this article.

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DIVISION DE EDUCACION CONTINUA

CURSOS ABIERTOS

VI CURSO INTERNACIONAL DE INGENIERIA GEOLOGICA APLICADA A OBRAS SUPERFICIALES Y SUBTERRANEAS

MODULO IV: TECNOLOGIA PARA EL USO DE EXPLOSIVOS

LAS TROJES, COL.
PROCEDIMIENTO DE EXCAVACION DEL VERTEDOR

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LAS TROJES, COL.

PROCEDIMIENTO DE EXCAVACION DEL VERTEDOR

860831

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1. TIPO DE ROCA

Origen: Ignea, piroclástica

Estructura: Pseudo estratificada, formada por estratos cuyo espesor varía entre 2 m y 10 m en actitud sensiblemente horizontal.

Clasificación: Brecha volcánica con fragmentos angulosos de andesitas de color gris y rosa cuyos tamaños varían desde 3 cm hasta 1 m, empacados en matriz vítrea andesítica de color gris, de bajo grado de cementación.

De esta manera se tiene una secuencia rítmica de estratos compuestos por brechas con matriz tobácea y tobas brechoides dependiendo del porcentaje relativo de matriz, apreciándose variaciones desde 50% matriz 50% fragmentos hasta 80 a 90% matriz y 10 a 20% fragmentos.

Resistencia: Los fragmentos o clastos andesíticos deben tener más o menos los siguientes valores:

Compresión simple: 300 a 700 kg/cm^2

Dureza Mohs: 6.5

Indice de abrasión: 0.30

Indice de perforabilidad: 1.0

La matriz tobácea es blanda con grado de cementación variables desde deleznable a compacta.

Debe tener más o menos los siguientes valores de resistencia:

Compresión simple: 15 a 300 kg/cm^2

Dureza Mohs: 6

Indice de abrasión: 0.6

Indice de perforabilidad: 2.0

2. USO DE EXPLOSIVOS EN EL CANAL VERTEDOR

DATOS:

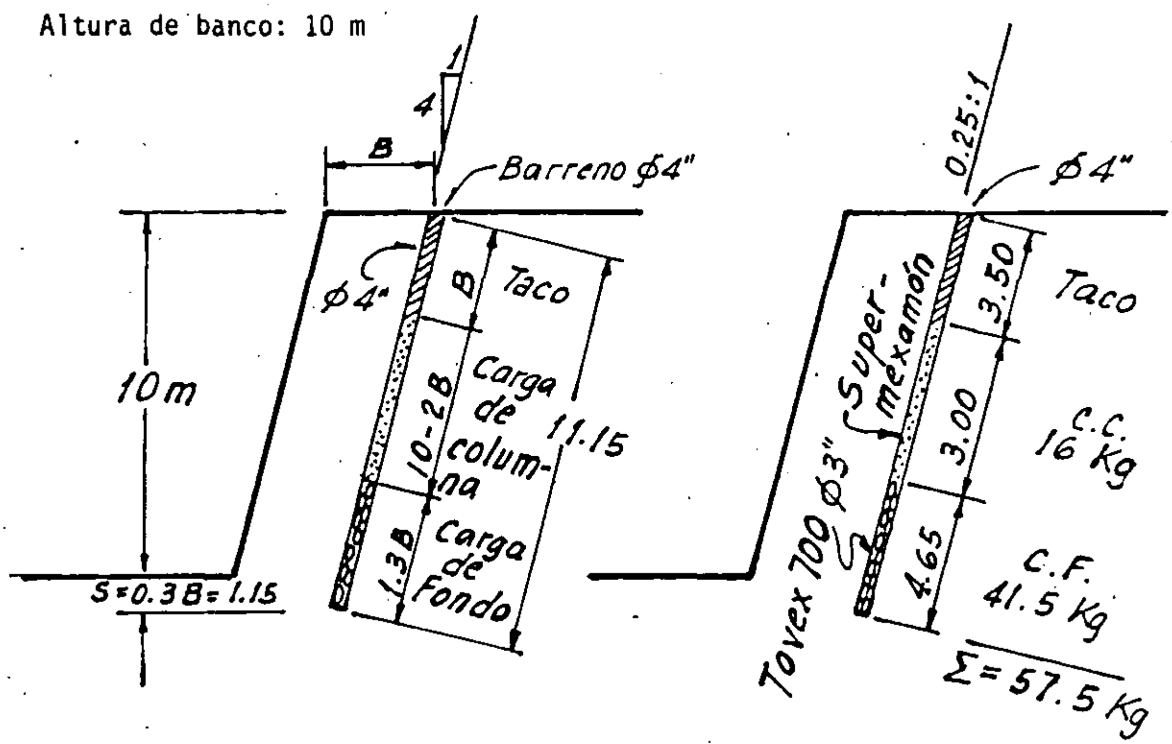
Constante de roca: 0.280 kg/m^3

Explosivo: Tovex 700; Densidad 1.2 g/cm^3 (teórica)

Densidad 1.1 g/cm^3 (práctica)

Anfomex: Densidad: 0.75 g/cm^3, en saco; = 0.65 g/cm^3, práct.

Altura de banco: 10 m



El bordo máximo en función de la potencia del Tovex 700 es 40 ϕ

$Bordo_{máx}$ = 40 ϕ ; Utilizando ϕ = 4"

Bordo práctico = B_1 = $B_{máx}$ - Falla de barrenación

Falla de barrenación = F = (error en emboquillado + % desviac.)

$\therefore$ $B_{máx}$ = 40 x 10.16 = $\underline{406.4 \text{ cm}}$

$\therefore$ F = (0.10 + 0.05 x 10) = 0.6 m

$\therefore$ B_1 = 406.4 - 60 = $\underline{\underline{346.4 \text{ cm}}}$

Consideraciones sobre el bordo máximo

1) El bordo máximo teórico para el Tovex 700 es:

$$B = d \times 30\sqrt{\frac{q \times S}{\bar{c} \times f\ (\frac{E}{F})}}$$

<u>Fórmula actual</u>

en donde:

d = diámetro del barreno

q = densidad del explosivo, práctica

S = Potencia del explosivo en relación a la de un explosivo con NG = 40% y densidad $\rho = 1.4\ g/cm^3$: Para Tovex 700 S = 0.9

f = Factor de confinamiento = 1.02

E/B = 1.25

c = Constante de roca + 0.05 kg/m^3

(Factor de seguridad)

$$\therefore\ B_{máx}\ \text{Tovex 700} = 10.16 \times 30\sqrt{\frac{1.1 \times 0.9}{0.33 \times 1.02 \times 1.25}}$$

$$\therefore\ B_{máx} = 10.16 \times 30 \times 1.5339 = \underline{467.5\ cm}$$

e) Considerando el bordo máximo $B_{máx} = 45\phi$

$$B_{máx}\ \text{Tovex 700} = B_{máx}\sqrt{\frac{\text{Pot. Tovex 700 x Densidad}}{\text{Pot. NG 40\% x Densidad}}}$$

$$B_{máx}\ \text{Tovex 700} = B'_{máx}\sqrt{\frac{\text{Factor de roca 0.4}}{\text{Factor de roca 0.28}}}$$

$$B_{máx}\ \text{Tovex 700} = 45 \times 10.16\sqrt{\frac{0.75 \times 1.1}{1.00 \times 1.4}} = 457.2 \times 0.77$$

$$= \underline{350.96}$$

$$\therefore\ B_{máx} = 350.96\sqrt{\frac{0.4}{0.28}} = \underline{\underline{419.47\ cm}}$$

Utilizando el valor menor del bordo máximo se tiene:

$$B_{máx} = 406.4 \text{ cm}$$

Fallas en la barrenación $F = (0.10 + 0.05 \times 10) = 0.6$ m

en donde 0.10 = falla de emboquillado y 0.05 es el % de desviación de la barrenación.

$$\therefore B_{práctico} = B_1 = B_{máx} - \text{Fallas}$$

$$B_1 = 406.4 - 60 = \underline{346.4 \text{ cm}}$$

Para un espaciamiento $E_1 = 1.25\ B_1$

resulta: $E_1 = 1.25 \times 346.4 = \underline{433 \text{ cm}}$

$$\therefore E_1 B_1 = 3.464 \times 4.33 = 15 \text{ m}^2$$

Utilizando $\underline{B_1 = 3.5 \text{ m}}$

Resulta $\underline{E_1 = 4.5 \text{ m}}$

Altura de carga de fondo = 1.3 B

Sub-barrenación = $\frac{1}{3}$ B = $\frac{350}{3} \doteq 115$ cm

Altura de carga de fondo = 1.3 B

$\therefore$ Altura de carga de fondo = 1.3 x 3.5 = 4.55 m

Carga de fondo = $4.55 \times 8.107 \frac{\ell}{m} \times 1.1 \frac{kg}{\ell} = 40.58$ kg

Altura carga de columna = Altura banco - 2B

Altura carga columna = 10 - 2 x 3.5 = 3 m

Carga de columna = 3 m x 8.107 ℓ/m x 0.65 kg/ℓ = 16 kg

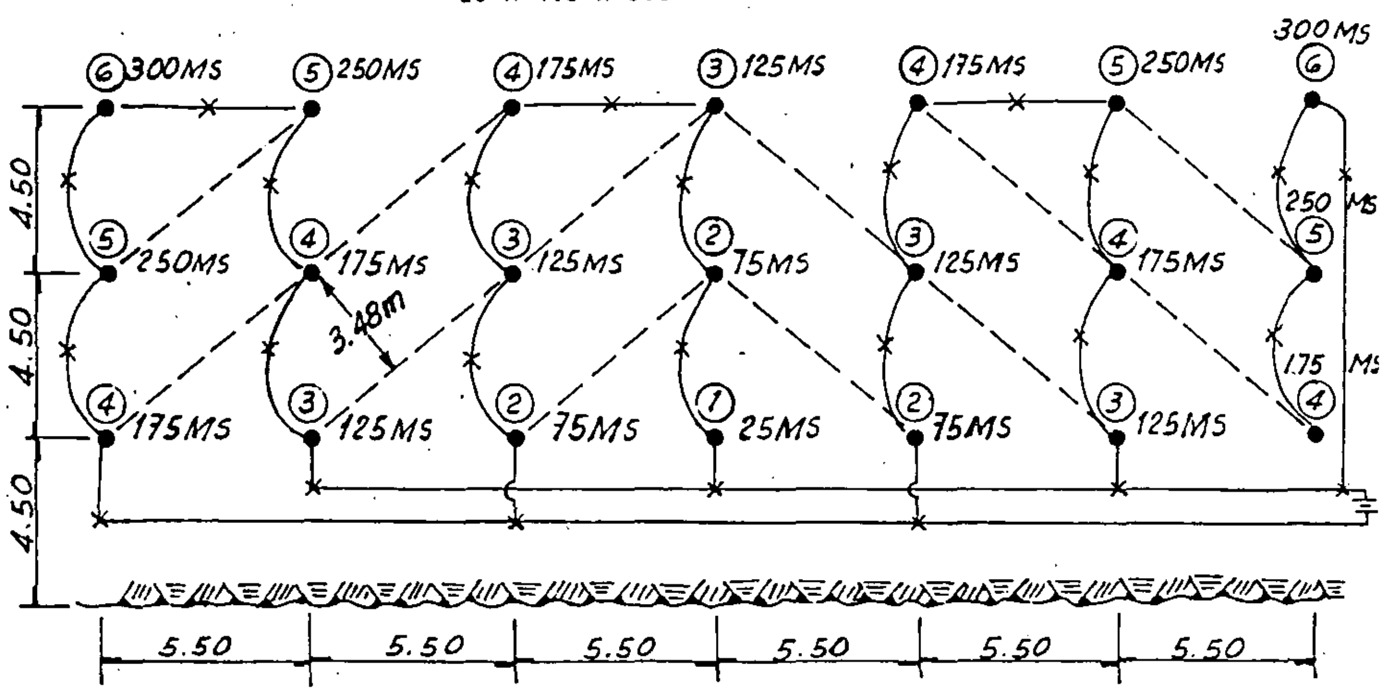
Carga total = 40.6 kg + 16 kg = 56.6 kg

$$\therefore \text{ Factor de carga F.C.} = \frac{56.6}{10 \times 3 \times 3.5 \times 4.5} = 0.360 \text{ kg/m}^3$$

Realizando voladuras con sistema de ignición en V, se tiene:

B = 3.5 x 1.414 = 4.95 m —>4.5

$$\text{Factor de barrenación} = \frac{11.15}{10 \times 4.5 \times 5.5} = 0.0451 \text{ m/m}^3$$



PLANTA

$$\text{Factor de carga F.C.} = \frac{56.6}{10 \times 4.5 \times 5.5} = 0.229 \text{ kg/m}^3$$

Factor de perforabilidad:

Velocidad de perforación:

De los datos Ingersoll-Rand

Para una roca: Granito Barre

con Trackdrill CM 350 y perforadora VL-140

con Compressor DXL-750; v = 44 pies/hora

con Trackdrill ECM 350 y perforadora VL-140

con Compresor DXL-750; v = 48 pies/hora

Factor de perforabilidad de la brecha

Para matriz 50% y fragmentos 50% (2 x 0.5) = 1

Para matriz 80% y fragmentos 20% (2 x 0.8) = 1.6

Factor de perforabilidad promedio = 1.3

∴ Velocidad de perforación = 44 x 1.3 = 57.2 pies/h

∴ v = 17 m/h

Duración de brocas

Indice de abrasión = 0.6

Para el granito Barre la duración de brocas ϕ 3" varía entre 400 a 900 ft; promedio = 650

$$\therefore \frac{650}{0.6} = 1083 \text{ pies} = 330 \text{ m} \doteq 350 \text{ m}$$

Duración de brocas = 350 m

3. PROPIEDADES DE LA ROCA

Resistencia en compresión simple; R_c = 40 a 80 kg/cm^2

Módulo elástico: E = 20 000 kg/cm^2; Toba

E = 112,000 kg/cm^2; Andesita

Relación de Poisson: $\nu = 0.3$ supuesta; $\rho = 2.2$ ton/m^3

Velocidad de transmisión de ondas de compresión V_L

$$V_L^2 = \frac{E\,(1-\nu)}{(1+\nu)(1-2\nu)} \times g$$

<u>Para la Toba</u>:

$$V_L^2 = \frac{200\ 000\ \frac{ton}{m^2}\,(1-0.3)}{2.2\ \frac{ton}{m^3}\,(1+0.3)(1-0.6)} \times 9.81\ \frac{m}{seg^2} = \frac{200\ 000\ \frac{ton}{m^2}}{2.2\ \frac{ton}{m^3}} \times 1.3462 \times 9.81\ \frac{m}{seg^2}$$

$$V_L^2 = 1\ 200\ 565\ \frac{m}{seg^2} \quad \therefore \quad \underline{V_L = 1100\ m/seg} = \underline{3600\ pies/seg}$$

Para E = 112 000 kg/cm^2 = 1 120 000 $\frac{ton}{m^2}$

resulta: $V_L^2 = \frac{1\ 120\ 000}{2.2} \times 1.3642 \times 9.81 = 6\ 723\ 167\ m^2/seg^2$

$$\therefore\ V_L = 2600\ m/seg = 8500\ pies/seg$$

DISEÑO DE UN SOLO BARRENO

DATOS:

Roca masiva

Altura de banco = 10 m = 32.8 pies

Densidad de roca SG_r = 2.2

Velocidad ondas P: V_p = 3600 pies/seg; Rel. Poisson $\nu = 0.3$

Compresión simple = 80 kg/cm² = 1140 lb/pulg²

D_e = Diámetro del explosivo

D_n = Diámetro del barreno

Densidad encartuchada del explosivo SC = 117

Diámetro crítico D_c = 1"

Velocidad confinada del explosivo:

$$V_e = 12\,500 \text{ pies/seg para } D_e = 3"$$

$$V_e = 15\,000 \text{ pies/seg para } D_e = 5"$$

SOLUCION

La relación entre V_e y D_e en el intervalo 1" a 5" puede determinarse por la expresión:

$$y = \frac{C_x}{a + bx} \quad \text{en donde } y = V_e;\ x = D_e - D_c$$

De donde:

$$V_e = \frac{C(D_e - D_c)}{a + b(D_e - D_c)}$$

Sabemos que D_c = 1" y que: V_e = 15000 pies/seg para D_e = 5"

V_e = 12 500 pies/ seg para D_e = 3"

Para $D_e = 3"$; $12\ 500 = \frac{C(3 - 1)}{a + b(3 - 1)} = \frac{2C}{a + 2b}$

Suponiendo C = 5000 como valor de constante

Se tiene: $a + 2b = \frac{2 \times 5000}{12\ 500} = \frac{4}{5} = 0.8$ (1)

y para $D_e = 5"$ $15\ 000 = \frac{C(5 - 1)}{a + b(5 - 1)} = \frac{4C}{a + 4b}$

$$a + 4b = \frac{4 \times 5000}{15\ 000} = \frac{4}{3} = 1.33 \quad (2)$$

Agrupando: $a + 2b = 0.8$ (1)

$\underline{a + 4b = 1.33}$ (2)

Restando (1) de (2) $2b = 0.53$ $\therefore b \cong 0.27$

Sustituyendo en I $a + 2(0.27) = 0.8$

$$\therefore a = 0.26$$

Por lo tanto: a = 0.26, b = 0.27 y C = 5000

Empleando la expresión: $V_e = \frac{5000(D_c - 1)}{0.26 + 0.27(D_c - 1)}$

con D_e variando desde 1" a 5"

Comprobación:

Para $D_e = 3"$: $V_e = \frac{5000(3 - 1)}{0.26 + 0.27(3 - 1)} = \frac{10\ 000}{0.26 + 0.54}$

$\therefore V_e = \underline{12\ 500 \text{ pies/seg}}$ - O.K.

y para $D_e = 5"$; $V_e = \frac{5000(5 - 1)}{0.26 + 0.27(5 - 1)} = \frac{20\ 000}{0.26 + 1.08}$

$V_e = \underline{14\ 900 \text{ pies/seg}}$ - O.K.

Para D_e = 2"; $V_e = \frac{5000(2 - 1)}{0.26 + 0.27(2 - 1)} = \frac{5000}{0.26 + 0.27} = 9450$ pies/seg

Para D_e = 4"; $V_e = \frac{5000(4 - 1)}{0.26 + 0.27(4 - 1)} = \frac{15\ 000}{0.26 + 0.81} = 14\ 000$ pies/seg

<u>Presión de detonación</u>:

$$P_d = \frac{6.06 \times 10^{-3}\ V_e^2(SG_e)}{1 + 0.8(SG_e)}$$

Densidad del explosivo: $SG_e = \frac{141}{SC} = \frac{141}{117} = \underline{1.2\ g/cm^3}$

La densidad práctica del Tovex 700 es $SG_e = 1.1\ g/cm^3$

De donde: $P_d = \frac{6.06 \times 10^{-3} \times 15\ 000^2 \times 1.1}{1 + 0.8 \times 1.1} = \frac{6.06 \times 2.25 \times 10^5 \times 1.1}{1.88}$

$\therefore\ P_{d\ máx} = 796\ 790$ lb/pulg $= 56\ 182\ kg/cm^2$

Para D_e = 2"; $P_d = P_{d\ máx}(\frac{9450}{15000})^2 = 797\ 790\ (\underline{0.397})$

$\therefore\ P_d = 316\ 723\ lb/pulg^2 = 22\ 304\ kg/cm^2$

Para D_e = 4"; $P_d = P_{d\ máx}(\frac{14000}{15000})^2 = 797\ 790\ (\underline{0.87})$

$\therefore\ P_d = 694\ 077\ lb/pulg^2 = 48\ 878\ kg/cm^2$

Para D_e = 3"; $P_d = P_{d\ máx}(\frac{12500}{15000})^2 = 797\ 790\ (\underline{0.69})$

$\therefore\ P_d = 554\ 021\ lb/pulg^2 = 39\ 016\ kg/cm^2$

Determinación del bordo óptimo

Utilizando la expresión: $K_B = 30(\frac{160}{d_r})^{1/3}(\frac{SG_e}{1.3})^{1/3}(\frac{V_e}{12000})^{2/3}$

en donde:

$$d_r = 62.4\ (SG_r) = 62.4\ (2.2) = 137\ \text{lb/pie}^3$$

siendo:

d_r = peso volumétrico de la roca

SG_e = Densidad práctica del Tovex 700 = 1.1 g/cm^3

V_e = Velocidad del explosivo Tovex 700 ± 15 000 pies/seg

12 000 = Velocidad de un explosivo base

30 = Relación de bordo promedio = 30

1.3 = Densidad del explosivo base

$$\therefore\quad K_B = 30(\frac{160}{137.3})^{1/3}\ (\frac{1.1}{1.3})^{1/3}(\frac{15000}{12000})^{2/3}$$

$$= 30(1.05)(0.95)(\frac{V_e}{12000})^{2/3} = \underline{29.8}(\frac{V_e}{12000})^{2/3}$$

Para tener el bordo en pies:

$$B = \frac{K_B D_e}{12} = \frac{29.8}{12}(\frac{V_e}{12000})^{2\ 3}\ D_e$$

$$\therefore\quad \underline{B = 2.48\ D_e(\frac{V_e}{12000})^{2/3}}$$

Cálculo del bordo:

En forma general tenemos $B = 2.48\ D_e(\frac{V_e}{12000})^{2/3}$, pies

Para D_e = 2" $B = 2.48\ (2)\ (\frac{9450}{12000})^{2/3} = 4.96\ (0.85) = \underline{4.23\ pies}$

$\therefore B = \underline{25.4\ \phi}$

Para D_e = 4" $B = 2.48\ (4)\ (\frac{14000}{12000})^{2/3} = 0.92\ (1.11) = \underline{11\ pies}$

$\therefore B = \underline{33\ \phi}$

Para D_e = 5" $B = 2.48\ (5)\ (\frac{15000}{12000})^{2/3} = 12.4\ (1.16) = \underline{14.39\ pies}$

$\therefore B = \underline{34.5\ \phi}$

Para D_e = 6" $B = 2.48\ (6)\ (\frac{15000}{12000})^{2/3} = 14.88\ (1.16) = \underline{17.27\ pies}$

$\therefore B = \underline{34.5\ \phi}$

Para D_e = 3" $B = 2.48\ (3)\ (\frac{12500}{12000})^{2/3} = 7.44\ (1.03) = \underline{7.65\ pies}$

$\therefore B = \underline{30.6\ \phi}$

Velocidad de propagación de fracturas:

$V_f = \frac{V_p}{3}$; $V_f = \frac{3600}{3} = 1200$ pies/seg

Tiempo de arribo de fracturas al frente libre:

Si $t = \frac{B}{V_f}$; Para D_e = 2"; $t_f = \frac{4.23}{1200} = 3.5$ ms

Para D_e = 4"; $t_f = \frac{11}{1200} = 9.2$ ms

Para D_e = 3"; $t_f = \frac{7.65}{1200} = 6.4$ ms

Para D_e = 5"; $t_f = \frac{14.39}{1200} = 12$ ms

Tiempo de arranque de la roca:

La velocidad de desprendimiento de la roca es $\doteq \frac{1}{6}$ de la velocidad de propagación de las fracturas.

$$t = \frac{B}{V_d} \qquad V_d = \frac{V_f}{6} = \frac{1200}{6} = \underline{200 \text{ pies/seg}}$$

Para: $D_e = 2"$; $t = \dfrac{4.23 \text{ pies}}{200 \ \frac{\text{pies}}{\text{seg}}} = 0.212 \text{ seg} \times 1000 = \underline{21.2 \text{ ms}}$

$D_e = 4"$; $t = \dfrac{11 \text{ pies}}{200} = 0.055 \text{ seg} \times 1000 = \underline{55 \text{ ms}}$

$D_e = 3"$; $t = \dfrac{7.65}{200} = 0.383 \text{ seg} \times 1000 = \underline{38.3 \text{ ms}}$

$D_e = 5"$; $t = \dfrac{14.39}{200} = 0.072 \text{ seg} \times 1000 = \underline{72 \text{ ms}}$

$D_e = 6"$; $t = \dfrac{17.27}{200} = 0.0864 \text{ seg} \times 1000 = \underline{86 \text{ ms}}$

Bordo Mínimo

Utilizando la relación de bordo en función de las velocidades de la roca y del explosivo se tiene:

$$K_v = \frac{V_e}{V_p}$$

donde: V_e = Velocidad explosivo

V_p = Velocidad roca

V_p = 3600 pies/seg

Tabulando valores:

D_e"	B, pies	V_e, pies/seg	K_v
1	0	0	0
2	4.23	9 450	2.63
3	7.65	12 500	3.47
4	11	14 000	3.89
5	14.39	14 900	4.14
6	17.27	15 000	4.17

Bordo mínimo para el primer o cebo a nivel del piso

$$B_1 = \frac{3L}{9K_v + 2} \; ; \qquad L = 32.8 \text{ pies (altura banco)}$$

Para D_e = 5"; $B' = \frac{3 \times 32.8}{9 \times 4.14 + 2} = \frac{98.4}{39.26} = \underline{2.51 \text{ pies}}$

De la tabla B = 14.39 > 2.51 Se puede reducir el diámetro

Para D_e = 6"; $B' = \frac{3 \times 32.8}{9 \times 4.17 + 2} = \frac{98.4}{39.53} = \underline{2.49 \text{ pies}}$. Se puede reducir el diámetro

Para D_e = 4"; $B' = \frac{3 \times 32.8}{9 \times 3.89 + 2} = \frac{98.4}{37.01} = 2.66$ pies

De la tabla B = 11 pies > 2.96 Se puede reducir el diámetro

Para D_e = 3"; $B' = \frac{3 \times 32.8}{9 \times 3.47 + 2} = \frac{98.4}{32.23} = 2.96$ pies

De la tabla B = 7.65 pies > 2.96 Se puede reducir el diámetro

Para D_e = 2"; $B' = \frac{3 \times 32.8}{9 \times 2.63 + 2} = \frac{98.4}{25.67} = 3.83$ pies

De la tabla B = 4.23 > 3.83 pies

El valor de Bordo óptimo B de la tabla y el bordo mínimo son aproximadamente iguales para D_e = 2". Por tanto, deberíamos utilizar D_e = 2".

Bordo mínimo para el primer o cebo al centro de la carga de columna.

$$B' = \frac{3L}{18K_v + 1}$$

Para D_e = 5"; $B' = \frac{3 \times 32.8}{18 \times 4.14 + 1} = \frac{98.4}{75.52} = 1.30$ pies

De la tabla B = 14.39 >> 1.30 El diámetro puede ser mucho más pequeño

Para D_e = 3"; $B' = \frac{3 \times 32.8}{18 \times 3.47 + 1} = \frac{98.4}{63.46} = 1.55$ pies

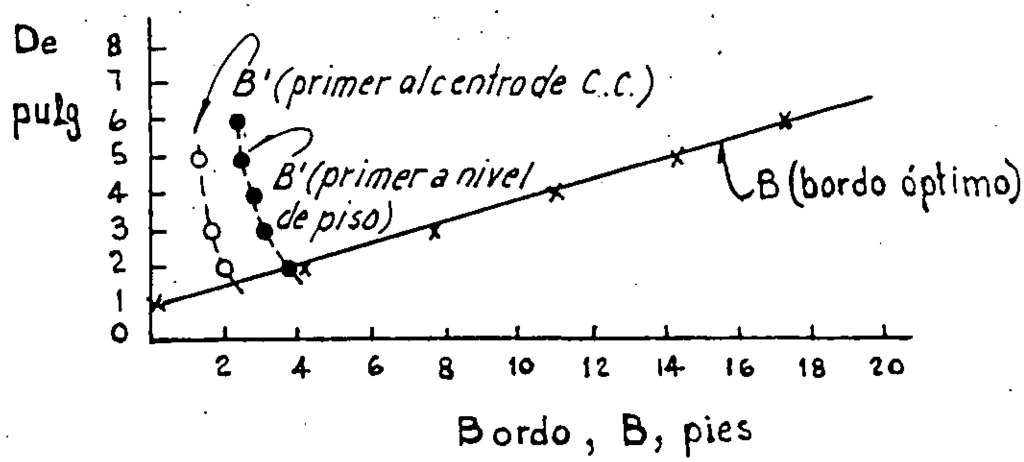
De la tabla B = 7.65 >> 1.55 El diámetro puede ser mucho menor

Para D_e = 2"; $B' = \frac{3 \times 32.8}{18 \times 2.63 + 1} = \frac{9.84}{48.34} = 2.04$ pies

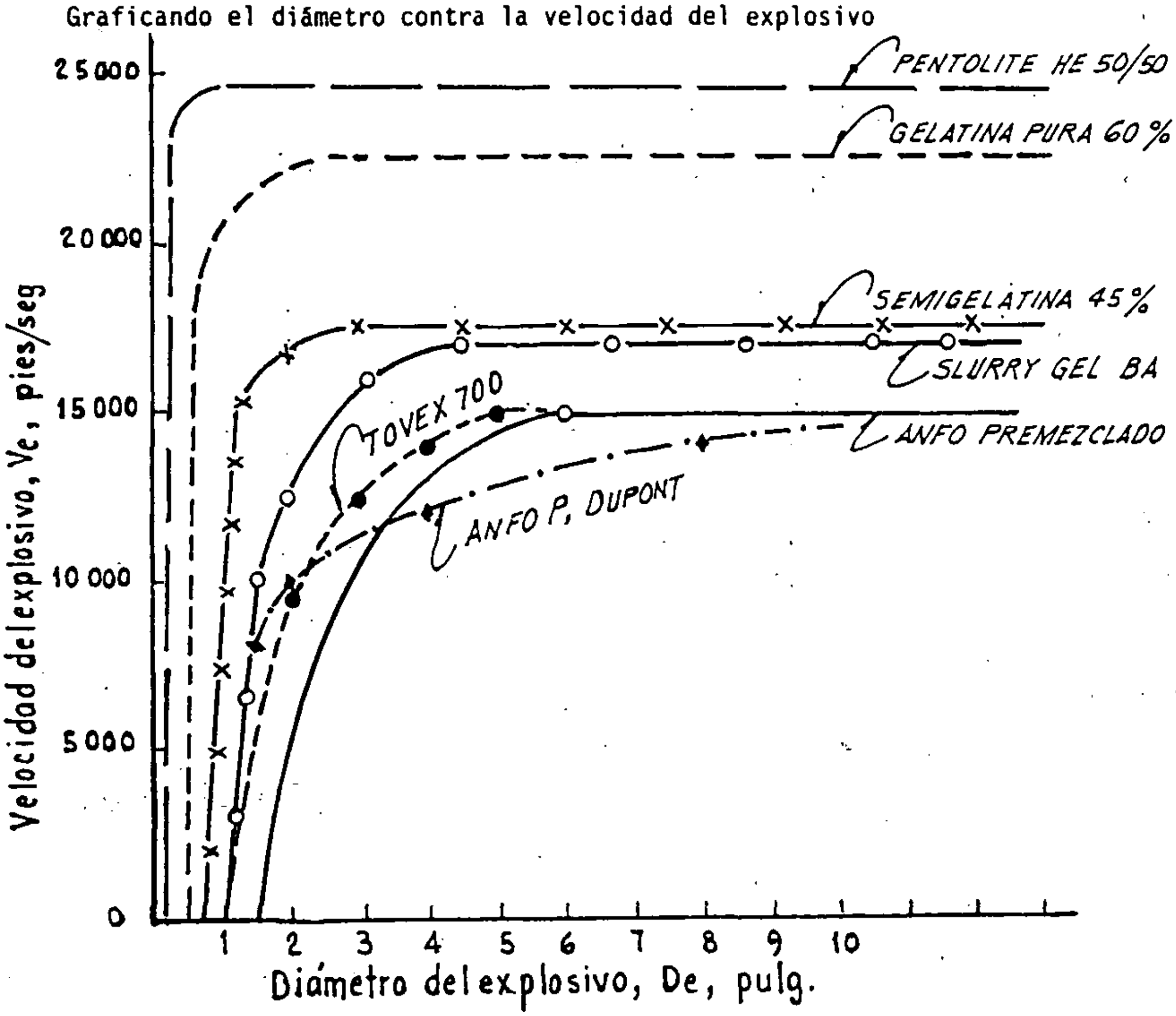
De la tabla B = 4.23 > 2.04 pies

Se observa que el diámetro que más se aproxima es D_e = 2; Deberíamos usar D_e = 2"

Graficando la relación entre los bordos y los diámetros se tiene:



Graficando el diámetro contra la velocidad del explosivo



RESULTADOS:

METODO SUECO

Diámetro de barreno ϕ = 4" (10.16 cm)

Bordo máximo = 346 cm = 34 ϕ

Bordo práctico = 350 cm

Espaciamiento = 350 x 1.25 = 437.5 ≐ 450

Patrón de Barrenación

B = 3.5 m

E = 4.5 m

METODO AMERICANO

Diámetro barreno ϕ 4" (10.16 cm)

Bordo óptimo = 335 cm = 33 ϕ

Bordo práctico = 350

Espaciamiento = 335 x 1.25 = 418.75

Area = E x B = 3.35 x 4.1875 = 14.028 m^2

Espaciamiento = 14.028/3.5 = 4.0 m

Patrón de Barrenación

B = 3.5 m

E = 4.0 m

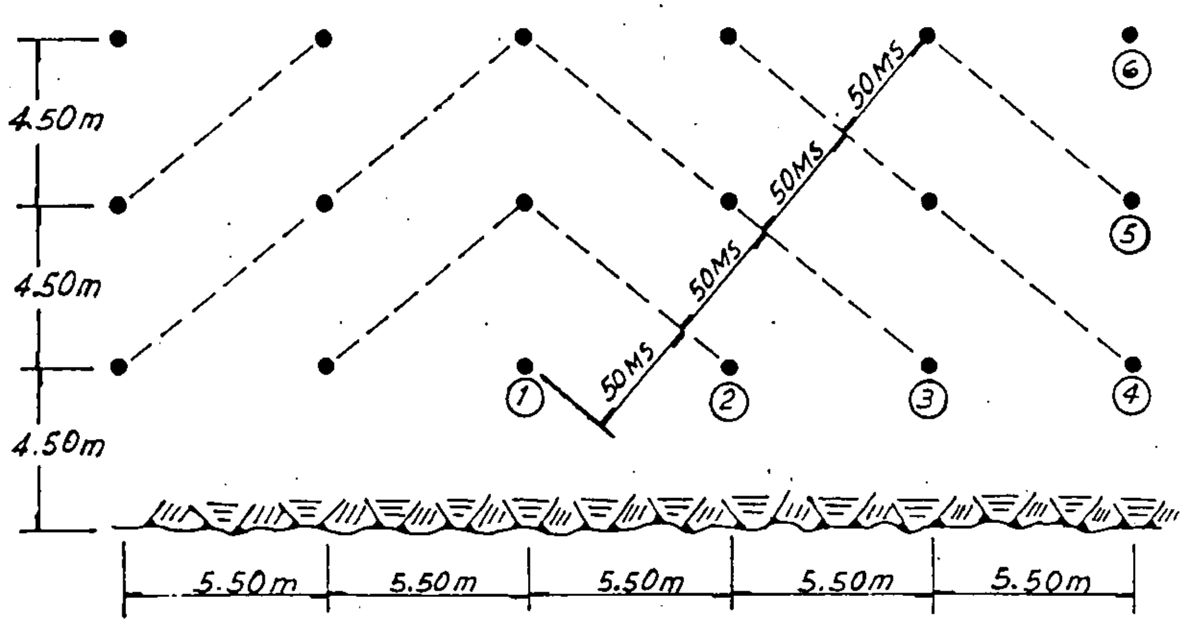
RECOMENDACION

Utilizar el patrón resultante del Método Sueco realizando la voladura con secuencia de ignición en V, de manera que el bordo máximo se presente en forma diagonal resultando entonces un patrón rectangular de 4.5 x 5.5 que tiene un bordo diagonal de 3.48 m.

Resultando:

B = 4.5 m   Diámetro barreno ϕ = 4"

E = 5.5 m

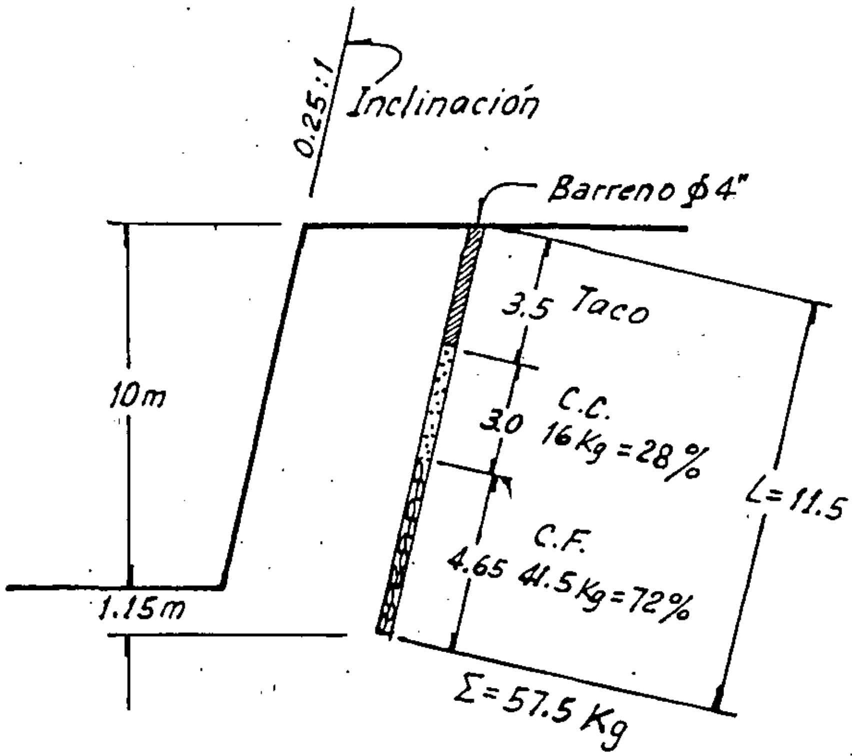


Explosivo: Tovex 700, 3" + Supermexamón

Carga de Fondo = 41.5 kg = 72%

Carga de columna = 16 kg = 28%

Total 57.5 kg



Factor de carga

$$F.C. = \frac{57.5 \text{ kg}}{10 \times 4.5 \times 5.5 \text{ m}^3} = 0.232 \text{ kg/m}^3$$

Factor de barrenación

$$F.B. = \frac{11.15}{10 \times 4.5 \times 5.5} = 0.045 \text{ m/m}^3 = 4.5 \text{ cm/m}^3$$

Velocidad de barrenación en ϕ 4" → 17 m/h

Duración de brocas: 350 m

NOTA: De la pág. 13 se observa que el tiempo de arranque de la roca para ϕ = 4" es de 55 ms por lo que se recomienda que la separación entre líneas sea de 50 ms.

COMENTARIOS

Las voladuras de Peñitas, Chis. tuvieron las siguientes características:

Diámetro de barreno: ϕ 2 1/2"

Patrón de barrenación:

2.5 x 3.0 m

2.75 x 2.75 m

3.0 x 3.0 m

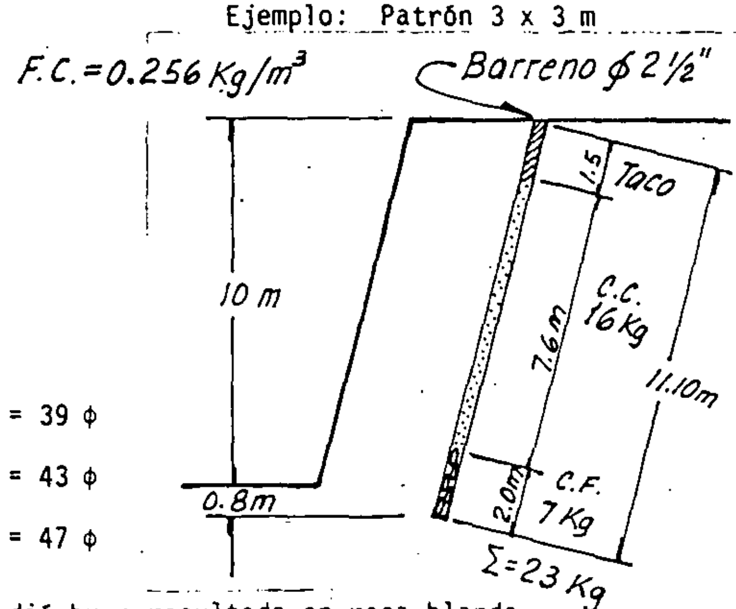
Factor de carga: 0.180 a 0.36 kg/m³

Factor de barrenación: 0.12 a 0.14 m/m³

$$\text{Suponiendo un banco de 10 m} = \frac{10.8}{10 \times 2.5 \times 3} = 0.14 \text{ m/m}^3$$

$$= \frac{10.8}{10 \times 3 \times 3} = 0.12 \text{ m/m}^3$$

Ejemplo: Patrón 3 x 3 m



Relación de cargas:

C.F. = 29%

C.C. = 71%

Relación diámetro a bordo:

Para B = 2.5 m; ϕ 6.35 cm; K_V = 39 ϕ

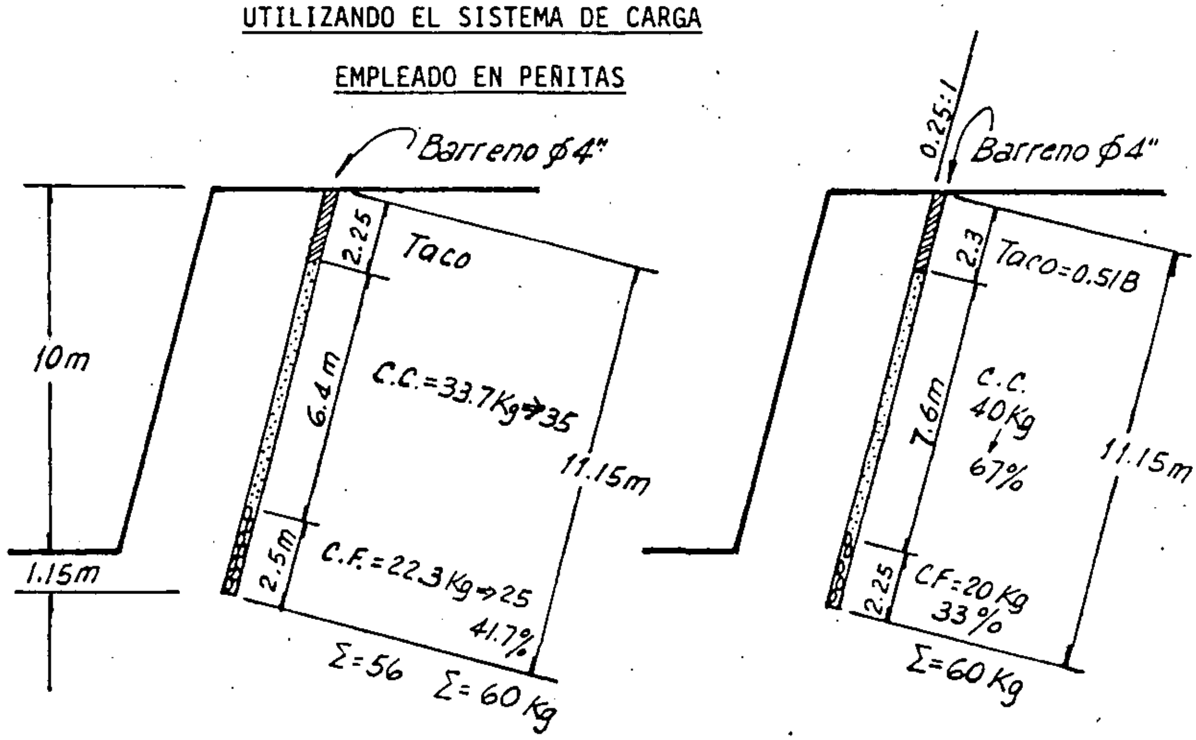
B = 2.75 m; ϕ 6.35 cm; K_V = 43 ϕ

B = 3.00 m; ϕ 6.35 cm; K_V = 47 ϕ

NOTA: Se tiene la experiencia que dió buen resultado en roca blanda.

UTILIZANDO EL SISTEMA DE CARGA

EMPLEADO EN PEÑITAS



ALTERNATIVA 1

ALTERNATIVA 2

$$\text{Factor de carga} = \frac{60}{10 \times 4.5 \times 5.5} = 0.242 \text{ kg/m}^3$$

$$= \underline{242 \text{ g/m}^3}$$

$$\text{Factor de barrenación} = \frac{11.15}{10 \times 4.5 \times 5.5} = 0.045 \text{ m/m}^3$$

$$= \underline{4.5 \text{ cm/m}^3}$$

NOTA: Esta carga es más económica que la indicada en la pág. 20 y debe dar buen resultado ya que no se requiere explosivo muy potente, pues la roca es blanda y por tanto se debe usar la mayor cantidad posible de ANFO, recordando que conviene utilizar velocidad de explosivo igual a velocidad de roca. Es mejor la Alternativa 2.

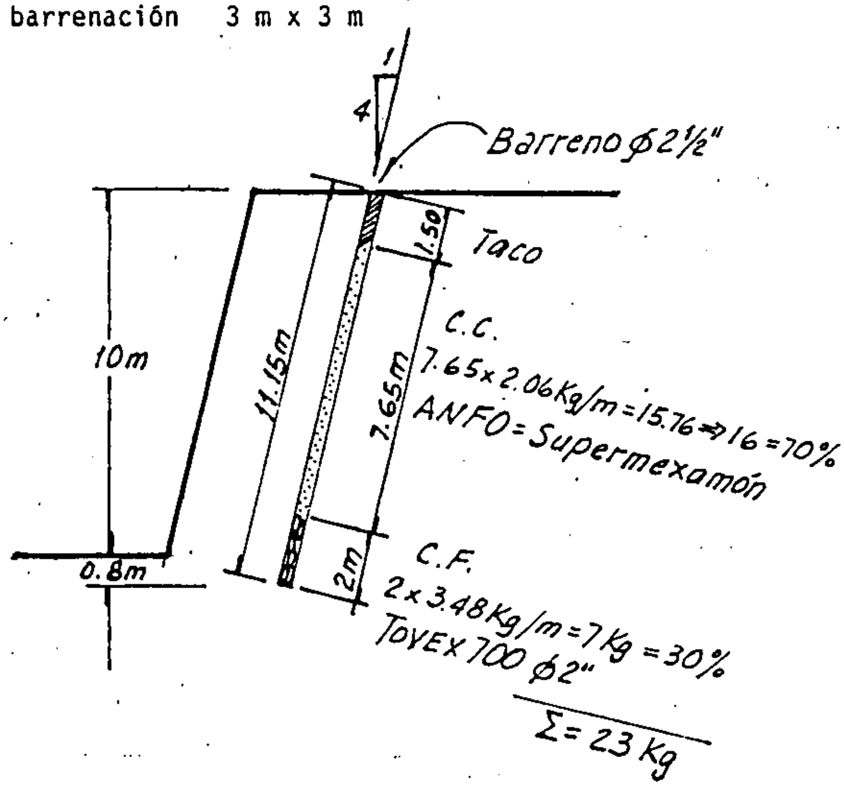
PROPUESTA:

VERTEDOR TROJES

Sept. 2, 1986

Diámetro de barreno ϕ 2 1/2" (6.35 cm) A = 31.67 cm^2

Plantilla de barrenación 3 m x 3 m



$$\text{Factor de carga} = \frac{23\ \text{kg}}{10 \times 3 \times 3} = 0.256\ \text{kg/m}^3$$

$$\text{Coeficiente de barrenación} = \frac{11.15}{10 \times 3 \times 3} = 0.0124\ \text{m/m}^3 = 12.4\ \text{cm/m}^3$$

Rendimiento de barrenación = 17 m/h

Utilizar 6 tiempos: 25 ms, 50, 75, 100, 125 y 150 ms.

CANTERA TROJES

(CORTINA)

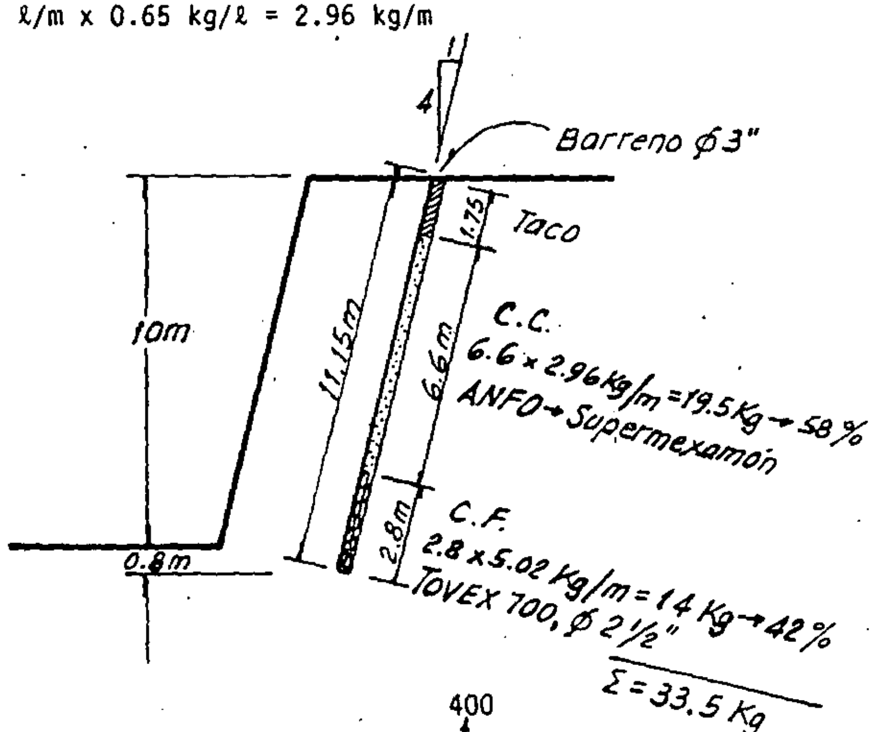
Sep. 2, 1986

Diámetro barreno ϕ = 3" (7.6 cm) A = 45.6 cm^2

Plantilla de barrenación 2.5 m x 2.5 m $\xrightarrow{\text{Cambia a}}$ 2.75 m x 3.0 m

Tovex 4.56 ℓ/m x 1.1 kg/ℓ = 5.02 kg/m

ANFO 4.56 ℓ/m x 0.65 kg/ℓ = 2.96 kg/m



$$\text{Factor de carga} = \frac{33.5}{10 \times \underbrace{2.5 \times 2.5}_{2.75 \times 3}} = \overset{400}{0.536}\ kg/m^3$$

$$\text{Factor de barrenación} = \frac{11.15}{10 \times \underbrace{2.5 \times 2.5}_{2.75 \times 3}} = 0.18\ m/m^3$$

$= 18\ cm/m^3$

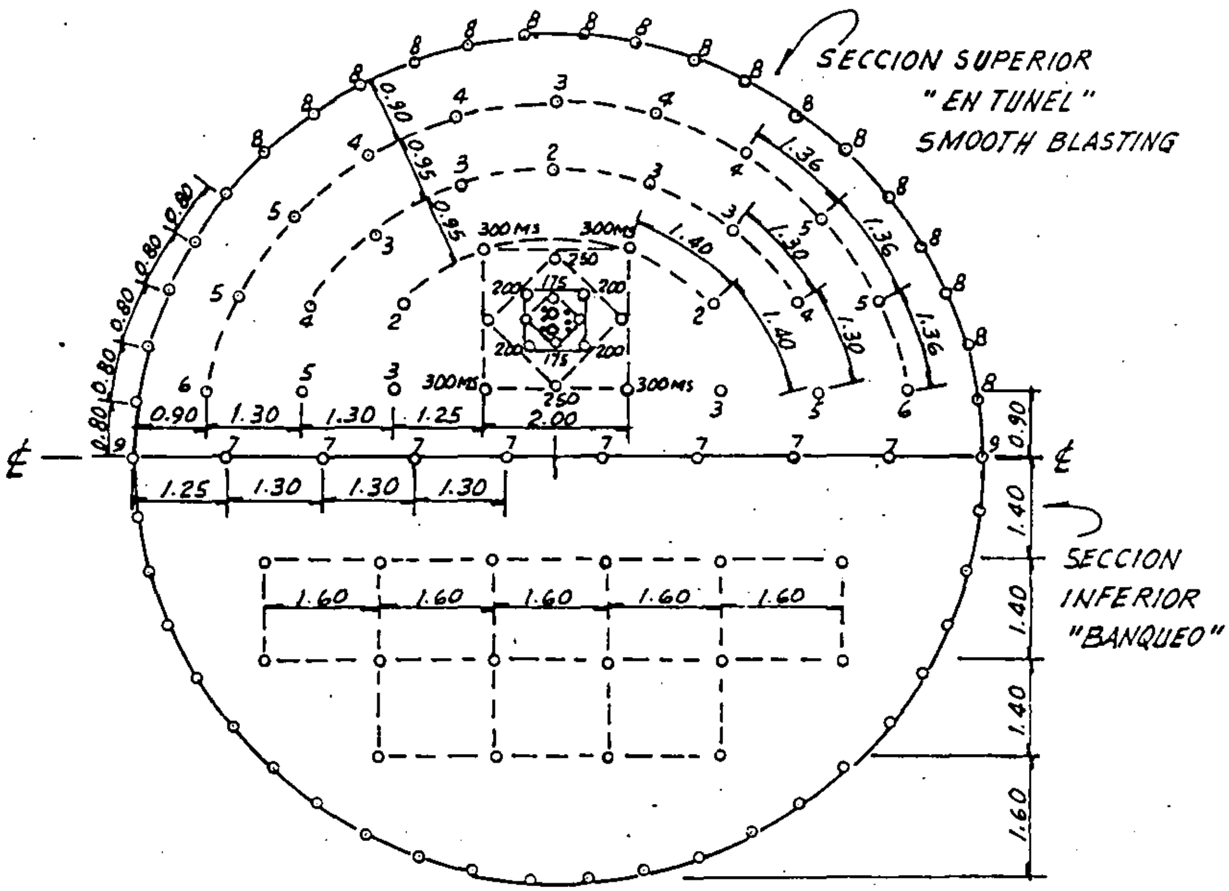
$= 13.5\ cm/m^3$

Rendimiento de barrenación: 12 m/h

Utilizar 6 tiempos: 25, 50, 75, 100, 125, 150 ms.

TUNEL DE DESVIO. P. TROJES, COLIMA

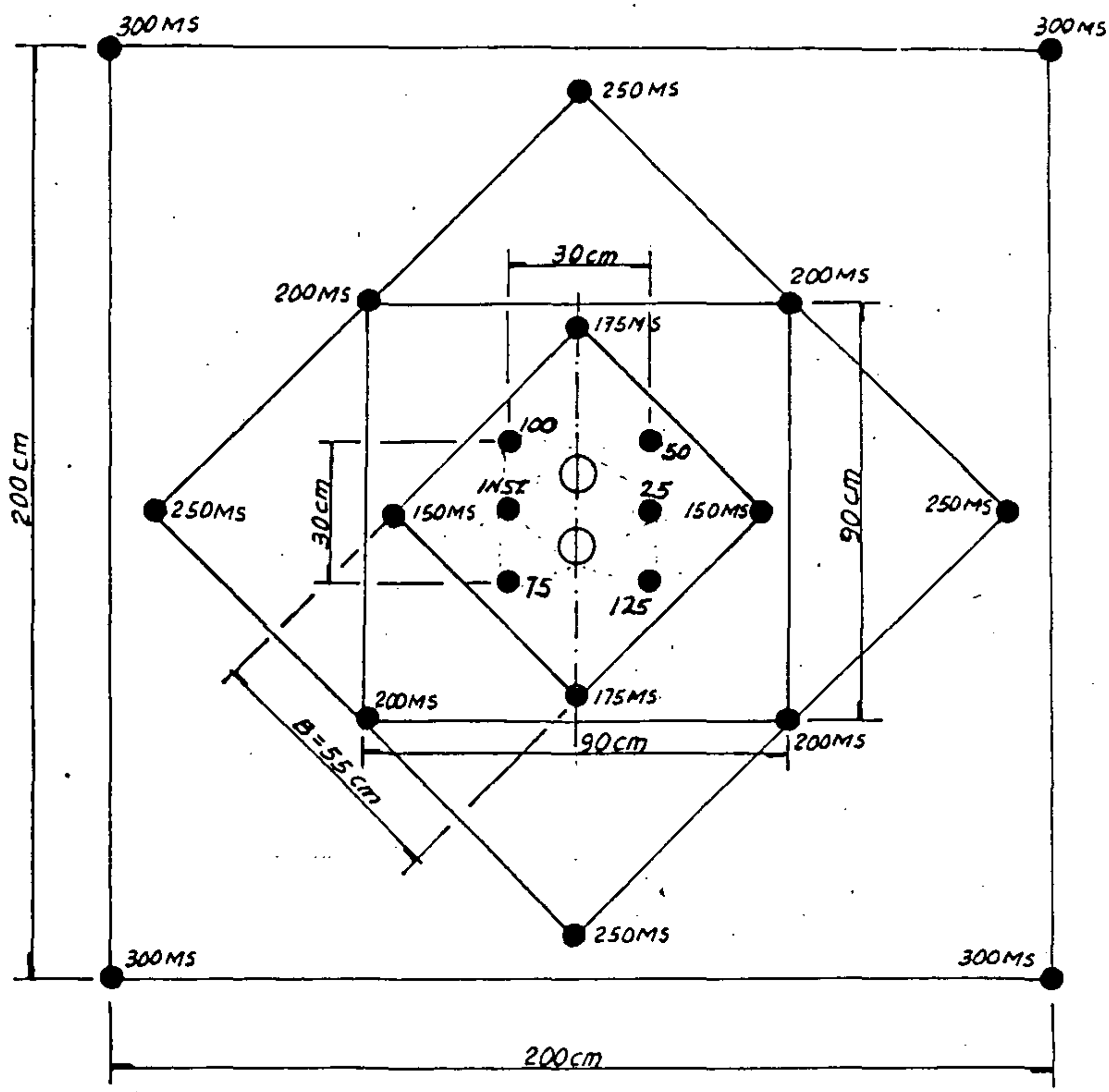
ϕ 11.60 m



Longitud de Barrenación = 4 m				Avance real = 3.90 m
Sección Superior. Area = 52.84 m²				ϕ Barreno 1 7/8"
Precorte:	24 x 1.2 kg	=	28.80 kg	TOVEX 100 ϕ 1 7/8" x 8"
Piso:	8 x 7.35 kg	=	58.80 kg	TOVEX 700 ϕ 1 3/4" x 16"
Abierta:	24 x 2.85 kg	=	68.40 kg	
	24 x 2.4 kg	=	58.5 kg	ANFO (SUPERMEXAMON)
Contracuña:	12 x 3.40 kg	=	40.70 kg	
	4 x 2.70 kg	=	11.00 kg	TOVEX 700 ϕ 1 3/4" x 16"
Cuña:	6 x 2 kg	=	12.00 kg	

CUÑA DE EXPANSION DE BARRENOS PARALELOS

CON 2 BARRENOS HUECOS DE $\phi 3"$



● Barreno ϕ 1 7/8" (22)

○ Barreno ϕ 3" (2)

CANTIDAD DE EXPLOSIVO EN LA SECCION SUPERIOR

TOVEX 100	=	28.8 kg	→	10.4%
TOVEX 700	=	190.9 kg	→	68.6%
SUPERMEXAMON	=	58.5 kg	→	21.0%
Σ	=	278.2 kg		

No. DE BARRENOS

Barrenos con explosivo	=	78 φ 1 7/8"
Barrenos huecos	=	2 φ 3"
TOTAL	=	80 Barrenos

COEFICIENTE DE BARRENACION

$$C.B. = \frac{80 \times 4 \text{ m}}{\frac{\pi}{2} \times \overline{5.8}^2 \times 3.9} = \underline{\underline{1.55 \text{ m/m}^3}}$$

COEFICIENTE DE CARGA

$$C.C. = \frac{278.2}{\frac{\pi}{2} \times \overline{5.8}^2 \times 3.9} = \underline{\underline{1.35 \text{ kg/m}^3}}$$

CALCULO DE CARGAS DE EXPLOSIVO

ϕ barreno 1 7/8" = 4.8 cm   Area = 18.1 cm^2; Volumen = 1.81 ℓ/m

ϕ explosivo, TOVEX 700;   Densidad = 1.18 g/cm^3

ϕ 1 3/4"; Long. = 16" = 40.6 cm; Area = 15.5 cm^2

Peso de 1 cartucho = 630.02 x 1.18 g/cm^3 = 743.4 g

TOVEX 100   Densidad = 1.10 g/cm^3

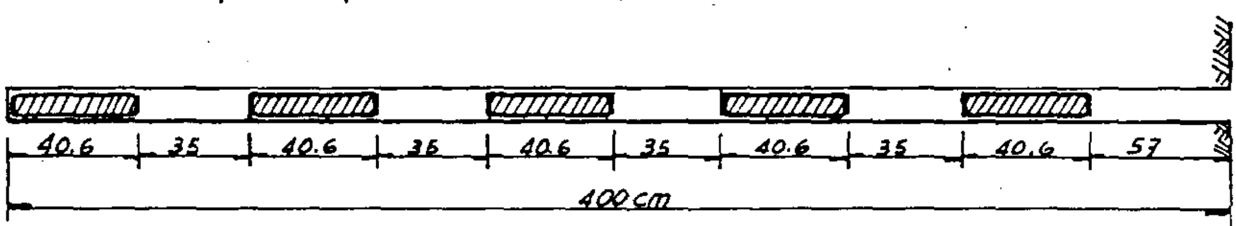
Peso de 1 cartucho = 120 g ϕ 1" x 8"

Peso de explosivo por metro de barreno

TOVEX 700 = 1.81 ℓ/m x 1.18 kg/ℓ = 2.14 kg/m

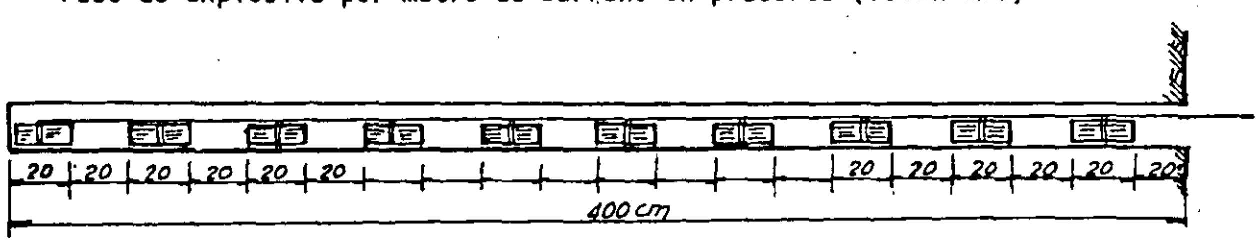
SUPERMEXAMON = 1.81 ℓ/m x 0.65 kg/ℓ = 1.18 kg/m

Peso de explosivo por metro de barreno en contra cuña (TOVEX 700)



Peso = 5 cartuchos x 0.743 kg = 3.72 kg; 3.72/4 = 0.93 kg/m

Peso de explosivo por metro de barreno en precorte (TOVEX 100)



Peso = 10 cartuchos x 120 = 1200 g → 1.2 kg/4 m = 0.30 kg/m

CARGA DE EXPLOSIVO A COLOCAR

- BARRENOS DE PISO Long. = 4 m

  TACO DE 57 cm, BARRENOS LLENOS DE TOVEX 700

  CARGA DE FONDO (TOVEX 700)

$$C.F. = 2.14 \frac{kg}{m} \times \frac{1}{3} 4 \ m = 2.85 \ kg$$

  CARGA DE COLUMNA (TOVEX 700)

$$C.C. = 2.14 \ kg/m \ (2/3 \ 4.00 - 0.57) = 4.5 \ kg$$

$$TOTAL = 2.85 + 4.5 = 7.35 \ kg$$

- BARRENACION ABIERTA Long. = 4 m

  CARGA DE FONDO (TOVEX 700)

$$C.F. = 2.14 \ kg/m \times 1/3 \times 4 = 2.85 \ kg$$

  CARGA DE COLUMNA (ANFO)

$$C.C. = 1.18 \ kg/m \times (4 - 0.6 - 1/4 \times 4) = 2.44 \ kg$$

- CONTRACUÑA

CONCENTRACION PARA B = 0.70, 1.15 kg/m

(TABLA I-22 CFE) B = 0.25, 0.75 kg/m

12 barrenos con 0.93 kg/m y 4 barrenos con 0.75 kg/m

$$12 \times 0.93 \times (4 - \underset{B/2}{0.7/2}) = 40.7 \ kg$$

$$4 \times 0.75 \times (4 - 0.35) = 11.0 \ kg$$

- CUÑA

CONCENTRACION = 0.65 kg/m (TABLA I-21 CFE)

NO. BARRENOS = 6 de ϕ 1 7/8"

6 x 0.55 kg/m (4 - 0.35) = 12 kg

- BARRENOS DE PRECORTE

TOVEX 100 ϕ 1" x 8" con 120 g c/u, de L = 20 cm

Densidad 1.10 g/cm^3

10 cartuchos x 120 g = 1.2 kg

12 kg/4 m = 0.30 kg/m (TABLA I-24 CFE)

No. de barrenos = 24

24 x 0.3 kg/m x 4 m = 28.8 kg

BANQUEO SECCION INFERIOR

PRECORTE:

0.30 kg/m x 4 m = 1.2 kg/barreno

22 Barrenos x 1.2 = 26.4 kg   TOVEX 100

BANQUEO (IGUAL QUE LA ABIERTA)

Concentración 21.4 kg/m   CARGA DE FONDO

C.F. = 214 kg/m x 1/3 x 4 = 2.83 kg/barreno

16 barrenos x 2.85 = 45.60 kg   TOVEX 700

CARGA DE COLUMNA. Concentración = 1.18 kg/m

C.C. = 1.18 kg/m x (2/3 x 4 - 0.6) = 2.4 kg/barreno

16 barrenos x 2.4 = 38.4   ANFO

SUMA

TOVEX 100 ϕ 1" x 8"	=	26.4 kg	23.9%
TOVEX 700 ϕ 1 3/4" x 16"	=	45.6 kg	41.3%
SUPERMEXAMON	=	38.4 kg	34.8%
		110.4 kg	

$$\text{FACTOR DE BARRENACION} = \frac{38 \text{ barr x } 4 \text{ m}}{52.84 \times 3.90} = \underline{\underline{0.74 \text{ m/m}^3}}$$

$$\text{FACTOR DE CARGA} = \frac{110.4 \text{ kg}}{52.84 \times 3.90} = \underline{\underline{0.54 \text{ kg/m}^3}}$$

SECCION COMPLETA

VOLUMEN DE ROCA POR DE TUNEL

$$= \frac{\pi}{4} \times \overline{11.6}^2 \times 1 = 105.7 \ m^3/m$$

No. TOTAL DE BARRENOS = 80 + 38 = 118 barrenos

TOTAL DE EXPLOSIVO = 278.2 + 110.4 = 388.6 kg

$$\text{FACTOR DE BARRENACION} = \frac{118 \times 4}{105.7 \times 3.9 \ m^3} = \underline{1.15 \ m/m^3}$$

$$\text{FACTOR DE CARGA} = \frac{388.6 \ kg}{105.7 \times 3.9 \ m^3} = \underline{\underline{0.943 \ kg/m^3}}$$

CONSUMO DE EXPLOSIVO

		(kg)	%	(kg/mℓ Tun.)*
TOVEX 100, ϕ 1" x 8"	28.8 + 26.4	= 55.2	14.2	14.15
TOVEX 700, ϕ 1 3/4" x 16"	190.9 + 45.6	= 236.5	60.9	60.64
SUPERMEXAMON	58.5 + 38.4	= 96.9	24.9	24.85
		388.6		99.64

* kg/mℓ de TUNEL = EXPLOSIVO/AVANCE (3.90 m)

BARRENACION POR mℓ DE TUNEL

$$1.15 \ m/m^3 \times 105.7 \ m^3/m = 121.6$$

$\rightarrow$ 121.6 m de barrenación/mℓ de túnel

FACULTAD DE INGENIERIA U.N.A.M.
DIVISION DE EDUCACION CONTINUA

CURSOS ABIERTOS
VI *CURSO INTERNACIONAL DE INGENIERIA GEOLOGICA*
APLICADA A OBRAS SUPERFICIALES Y SUBTERRANEAS
CUARTO MODULO:
TECNOLOGIA SOBRE EL USO DE EXPLOSIVOS.

PROPIEDADES GEOMETRICAS Y MECANICAS DE LAS ROCAS.

ING. RAUL CUELLAR BORJA.

PROPIEDADES GEOMETRICAS Y MECANICAS DE LAS ROCAS

Raúl Cuéllar Borja

1.- PROPIEDADES GEOMETRICAS DE LOS MACIZOS ROCOSOS

1.1.- INTRODUCCION

En este capítulo se describen algunos de los sistemas usuales que se utilizan para describir, ya sea en forma gráfica o escrita las características geométricas de la estructura de los macizos rocosos.

1.2.- Mapas geológicos

Estos mapas contienen la delimitación geográfica de las formaciones de roca existentes en el lugar y se aprovecha para señalar mediante una simbología las características de la estructura de los macizos rocosos como son: plegamientos, cabalgaduras, hundimientos, etc así como los razgos más significativos de las discontinuidades como son: fracturas o juntas, fallas, planos de estratificación, planos de foliación, oquedades, etc.

Es importante anotar la orientación (rumbo y echado) y espaciamiento de las discontinuidades, así como una descripción de las características de las caras o planos de discontinuidad. Por ejemplo: Los términos cerradas o abiertas se aplican para describir el grado de estanqueidad de las discontinuidades, esta información debe acompañarse de una descripción del material de relleno. También deben anotarse las características de los planos en los que están contenidas las caras de las juntas, por ejemplo: si el plano es recto o alabeado y si las caras son lisas, rugosas o escalonadas. A estas características de los planos de las juntas se les puede asignar un número en función de su resistencia al corte.

Existen dos formas básicas para representar las estructuras y los razgos estructurales.

1) Mapas con la localización geográfica de estructuras y razgos estructurales, anotando sus características físicas de orientación y posición y 2) Gráficas en las que se muestra la frecuencia relativa de las discontinuidades, mostrando el intervalo de variación de las orientaciones que ocurren en el sitio.

Ambos sistemas tienen sus ventajas y limitaciones.

El primer sistema es preferido para usos generales. Permite la construcción de secciones transversales en cualquier área crítica particular del proyecto; ya sea una ladera o la pared de una excavación subterránea. De esta manera se puede identificar la presencia de alguna discontinuidad cuya orientación sea adversa a alguna excavación.

En este sistema se incluyen tambien los mapas de socavones y lumbreras de exploración geológica.
La simbología de banderas de Mueller es muy adecuada en este sistema de representación. En las Figs 1 y 2 se presentan ejemplos de estas simbologías.
Las gráficas estadísticas más usuales son: a) La roseta de fallas y fracturas y b) Las proyecciones estereográficas.
Las ventajas de estos sistemas es que permiten ubicar un grán número de observaciones y pueden sacarse valores "promedio". Hay que tener cuidado con esto pues el peligro puede venir de una sola discontinuidad mayor y que no necesariamente se encuentre en el promedio.
En la Fig 3 se presenta una roseta de fracturas y fallas y en las Figs. 4 y 5 un sistema de representación estereográfica con base en el hemisferio inferior.

1.3.- Clasificación de la roca "in situ" para propósitos ingenieriles.

Además de los mapas y las graficas estadísticas de las discontinuidades geológicas, es deseable tener alguna forma de clasificación que permita la caracterización de los macizos rocosos. El propósito de esta clasificación es para facilitar la comunicación entre ingenieros

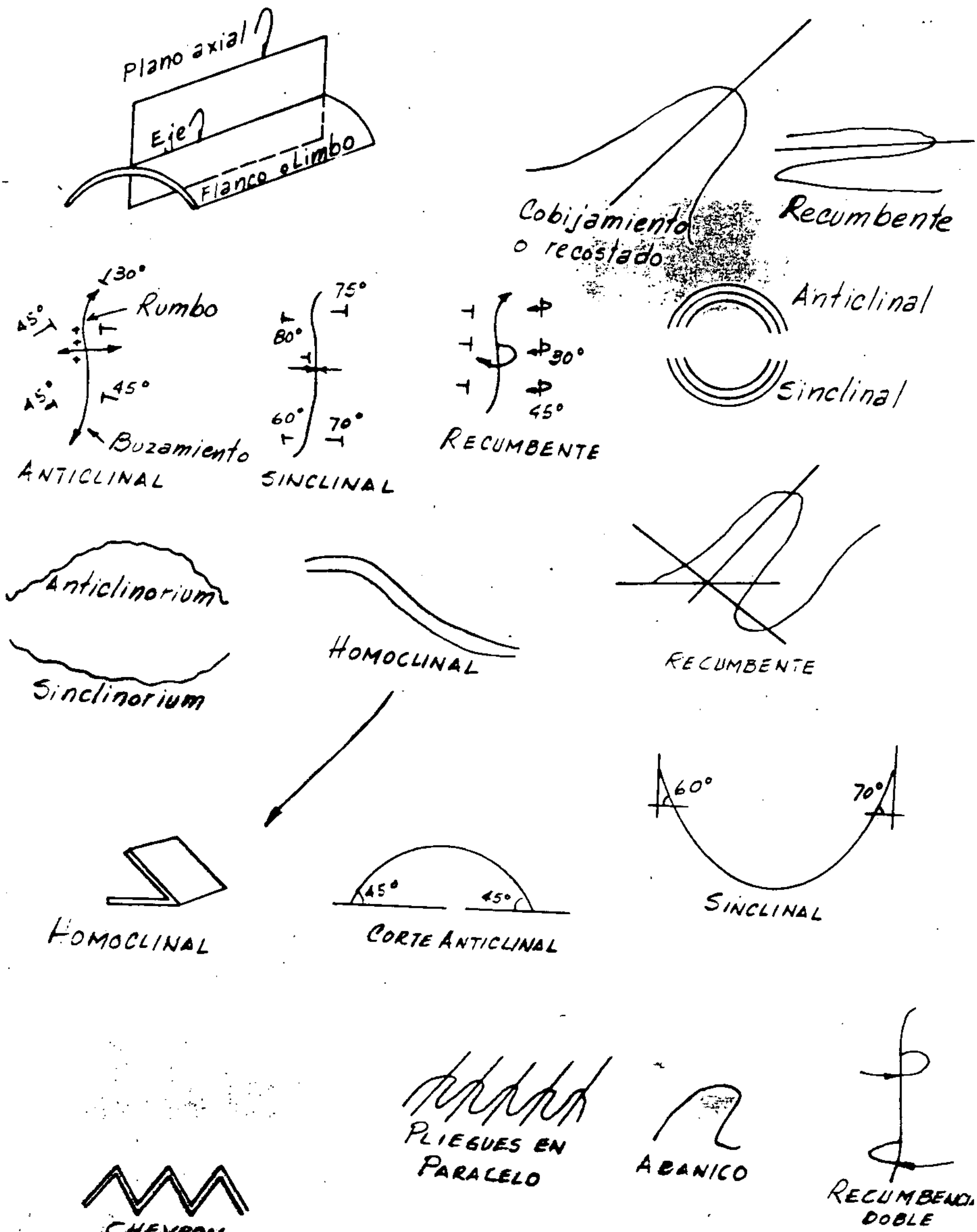
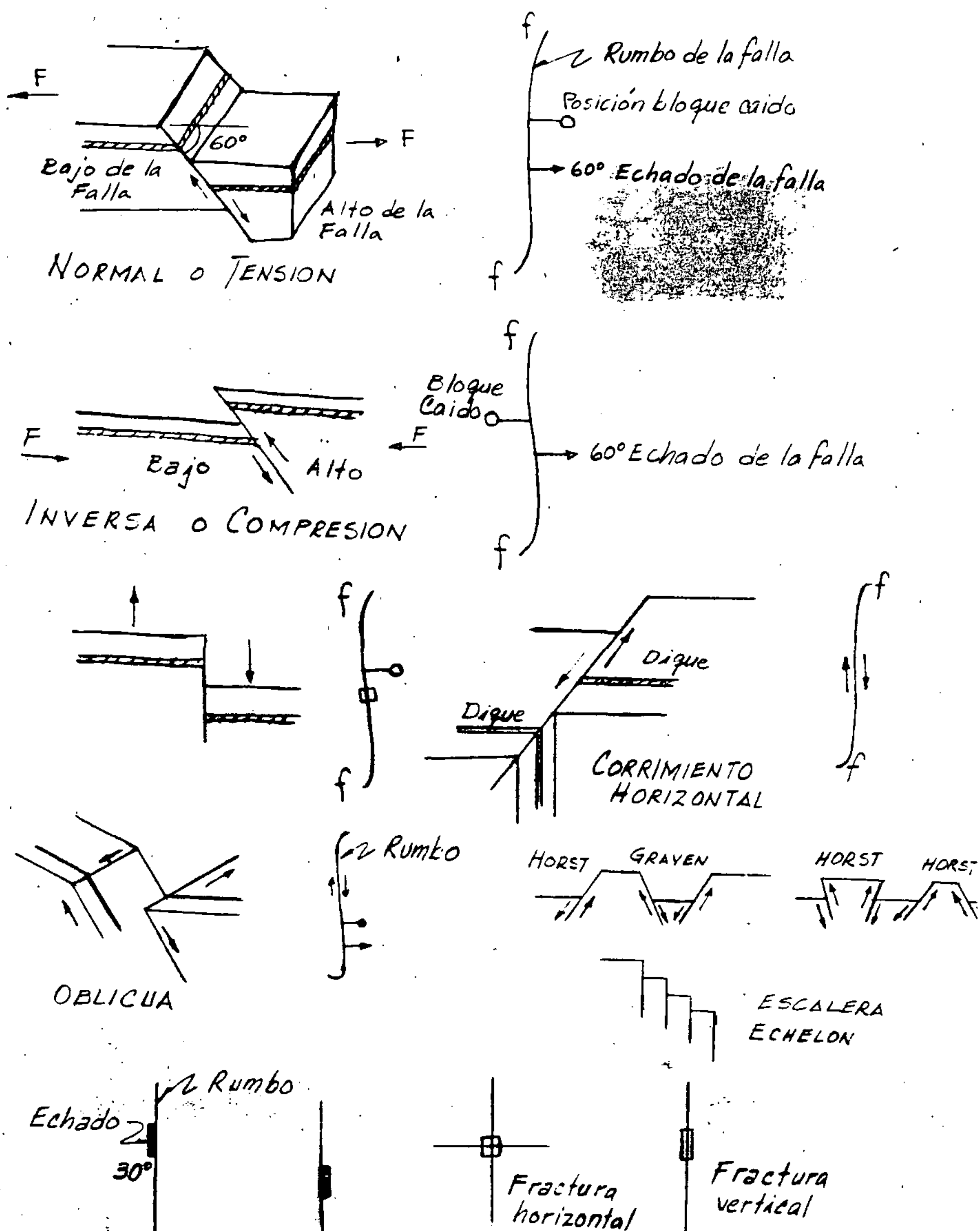

Plano axial
Eje
Flanco o Limbo
Cobijamiento o recostado
Recumbente
30°
Rumbo
45°
45°
45°
Buzamiento
ANTICLINAL
75°
80°
60°
70°
SINCLINAL
30°
45°
RECUMBENTE
Anticlinal
Sinclinal
Anticlinorium
Sinclinorium
HOMOCLINAL
RECUMBENTE
60°
70°
SINCLINAL
HOMOCLINAL
45°
45°
CORTE ANTICLINAL
PLIEGUES EN PARALELO
ABANICO
RECUMBENCIA DOBLE
CHEVRON

FIG 1

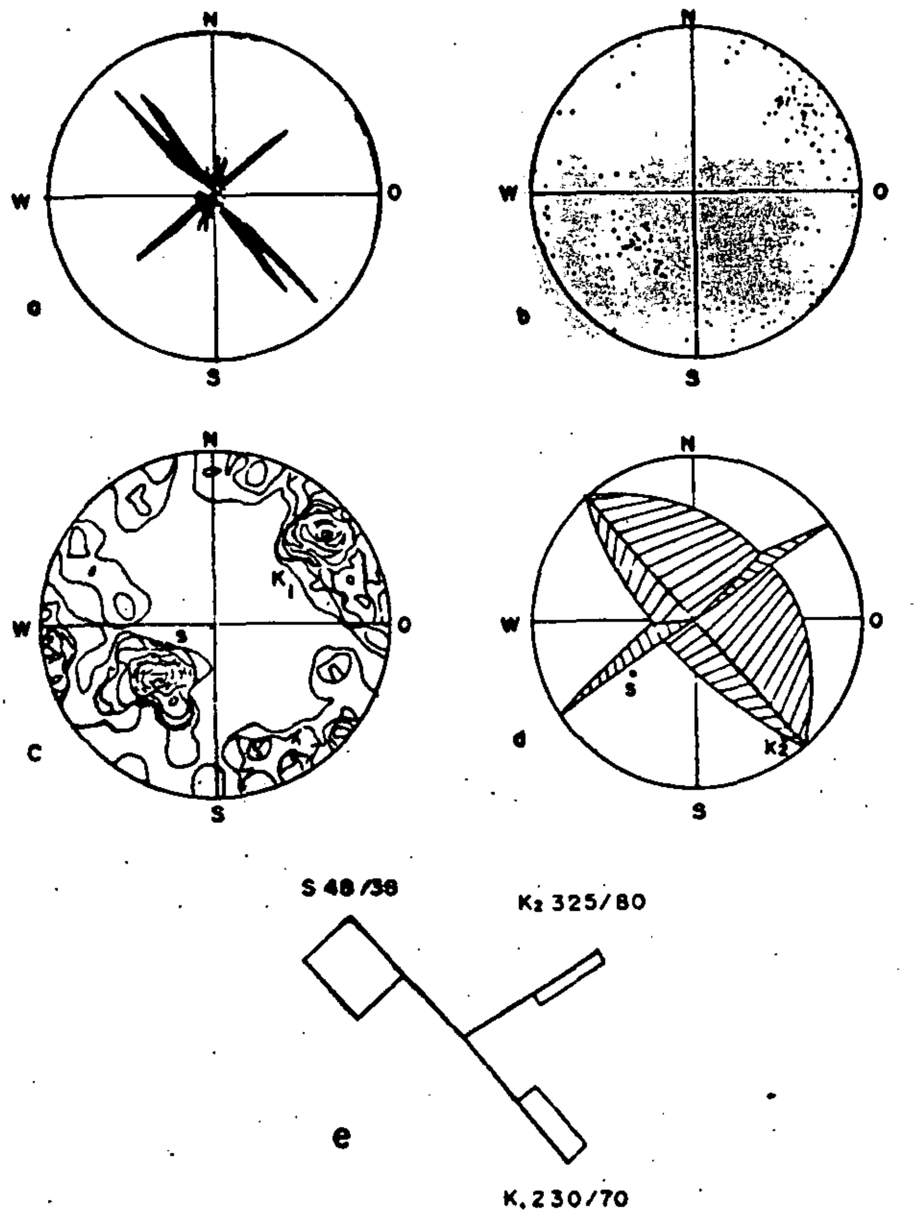
FALLAS



F
F
60°
Bajo de la Falla
Alto de la Falla
NORMAL O TENSION
f
Rumbo de la falla
Posición bloque caido
60° Echado de la falla
f
F
F
Bajo
Alto
INVERSA O COMPRESION
f
Bloque Caido
60° Echado de la falla
f
f
f
Dique
Dique
CORRIMIENTO HORIZONTAL
f
f
Rumbo
OBLICUA
HORST
GRAVEN
HORST
HORST
ESCALERA
ECHELON
Rumbo
Echado
30°
Fractura horizontal
Fractura vertical

FIG. 2

Representación de planos estructurales —



a) Rosa de vientos b) Diagrama de puntos (polos)

c) Diagrama de frecuencias d) Circulos grandes y polos

e) Cuadrados unitarios segun MULLER

PREPARACION DE DIAGRAMAS

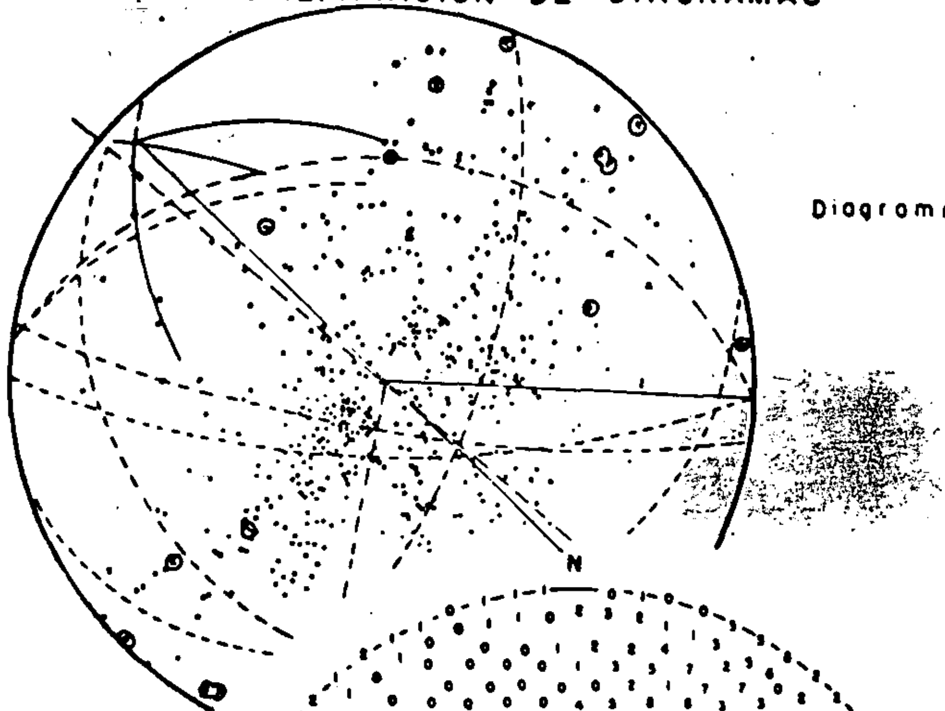
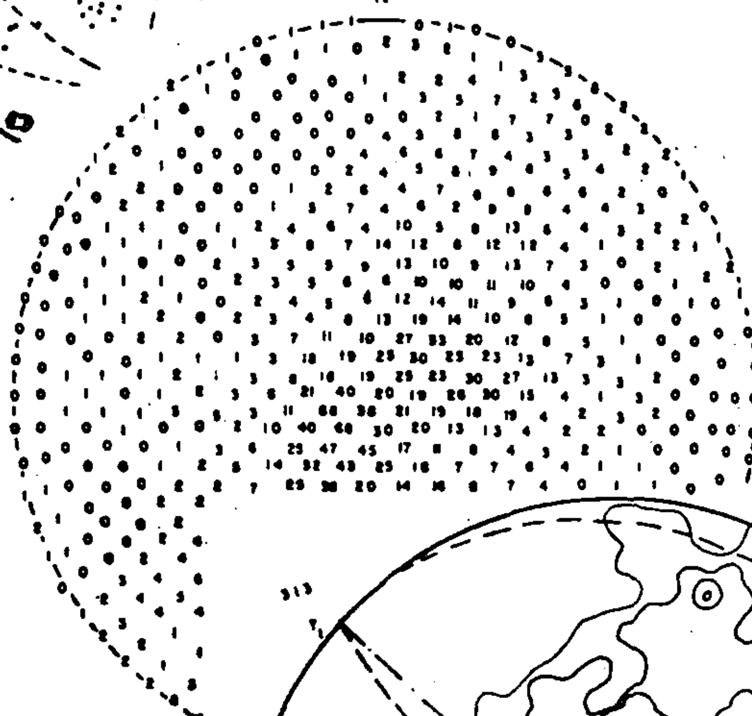


Diagrama de puntos



Conteo

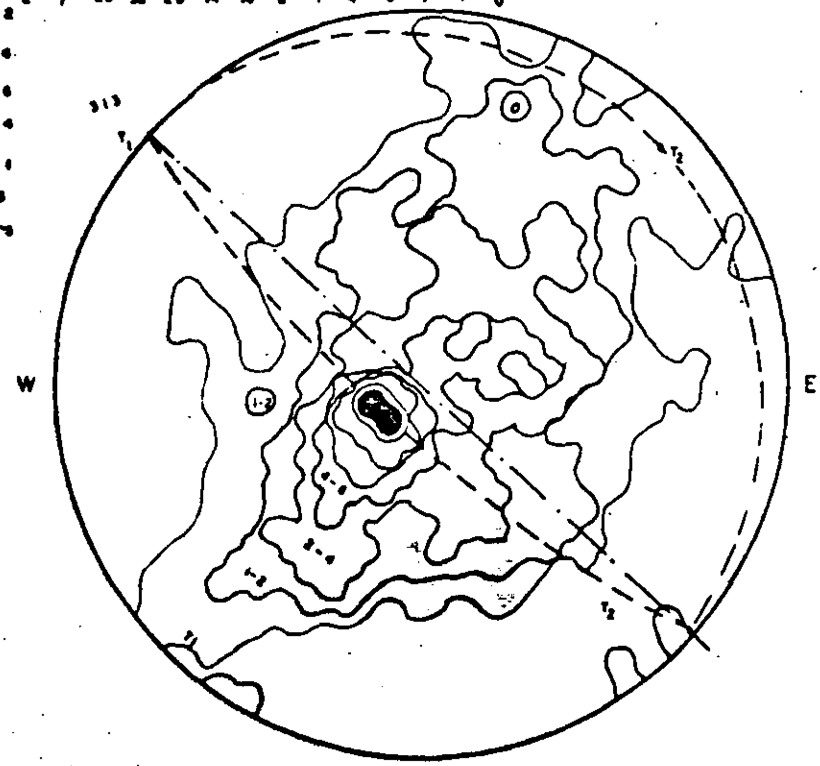
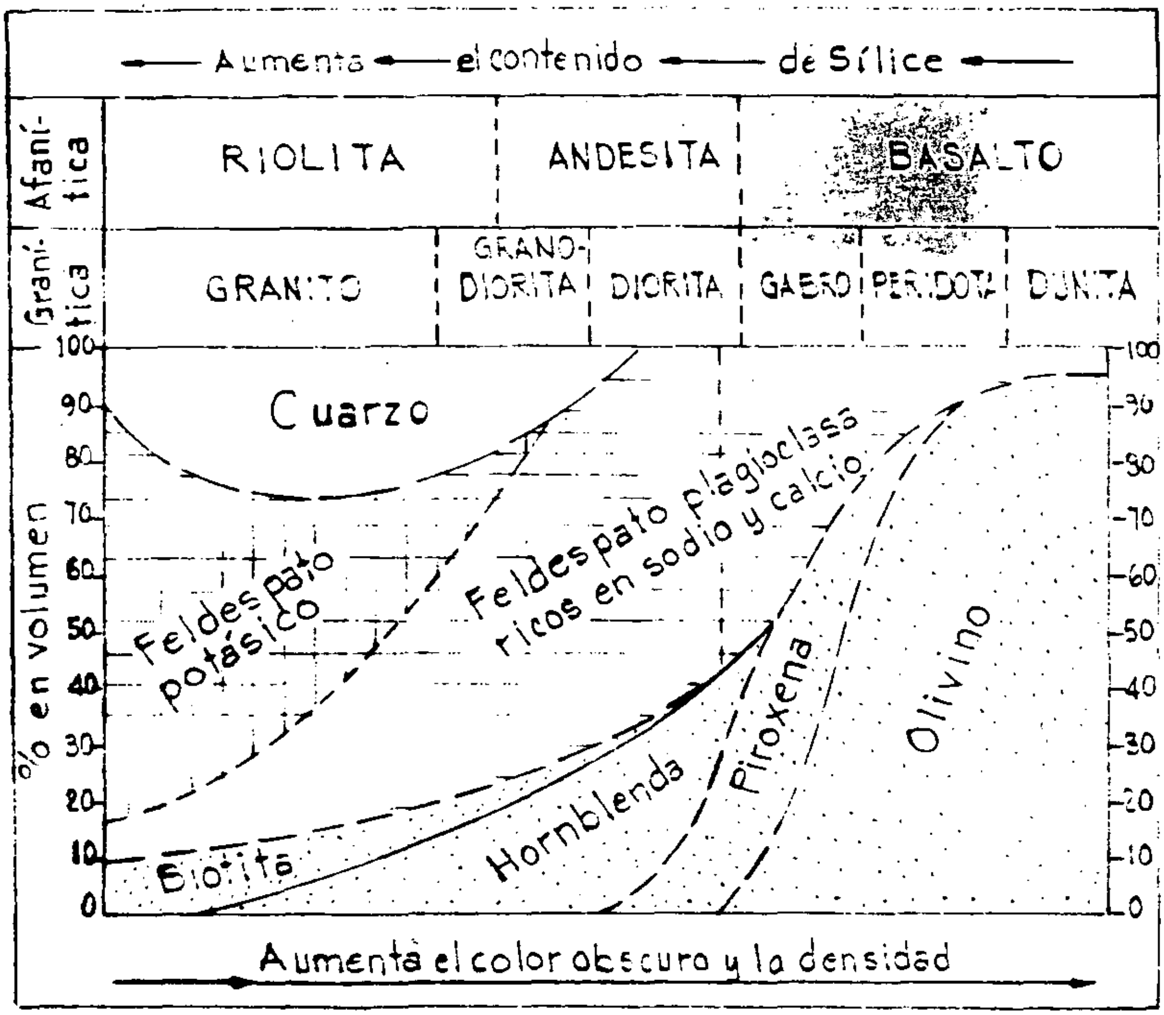


Diagrama de frecuencia



COMPOSICION MINERALOGICA DE LAS ROCAS

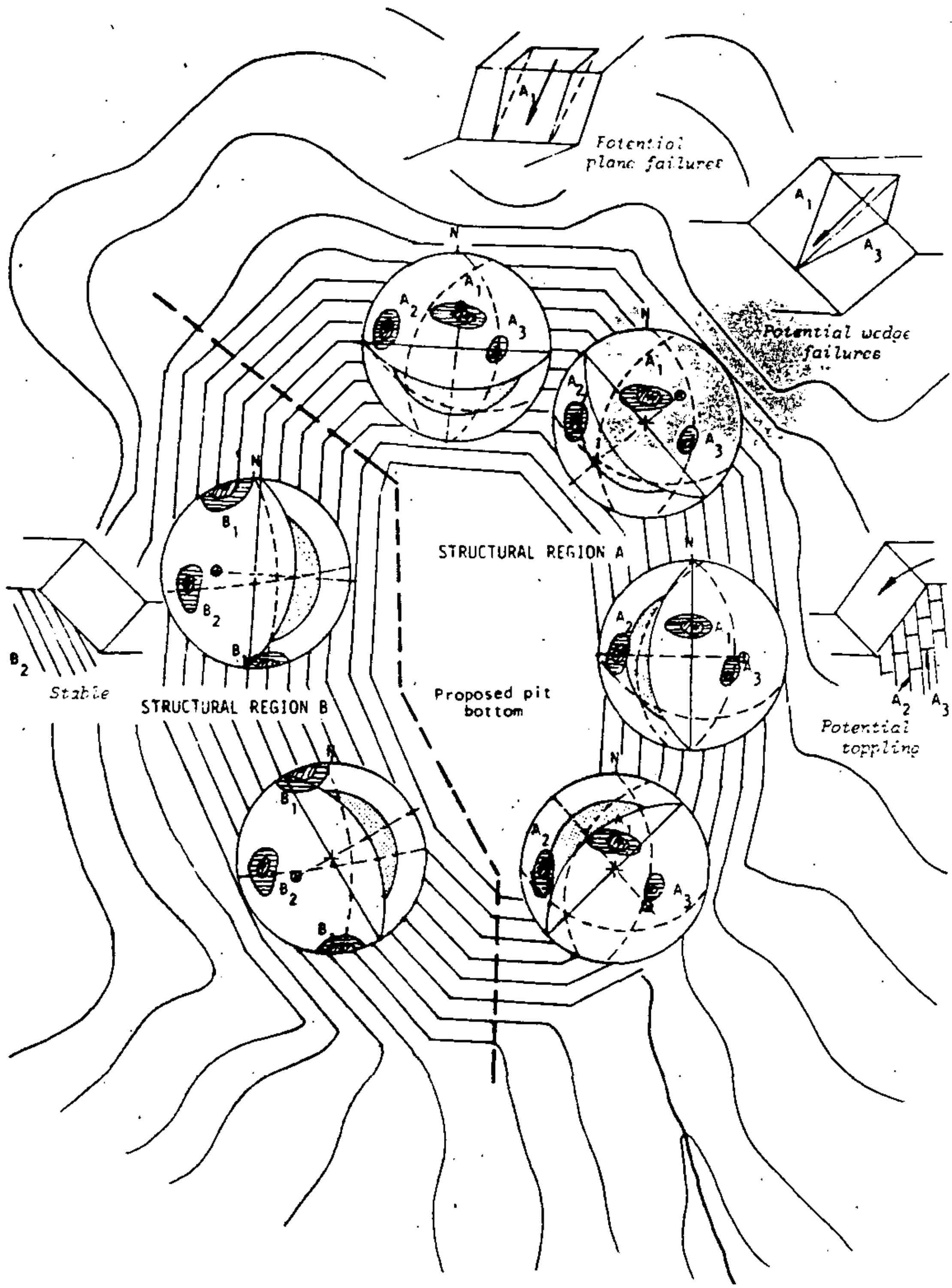
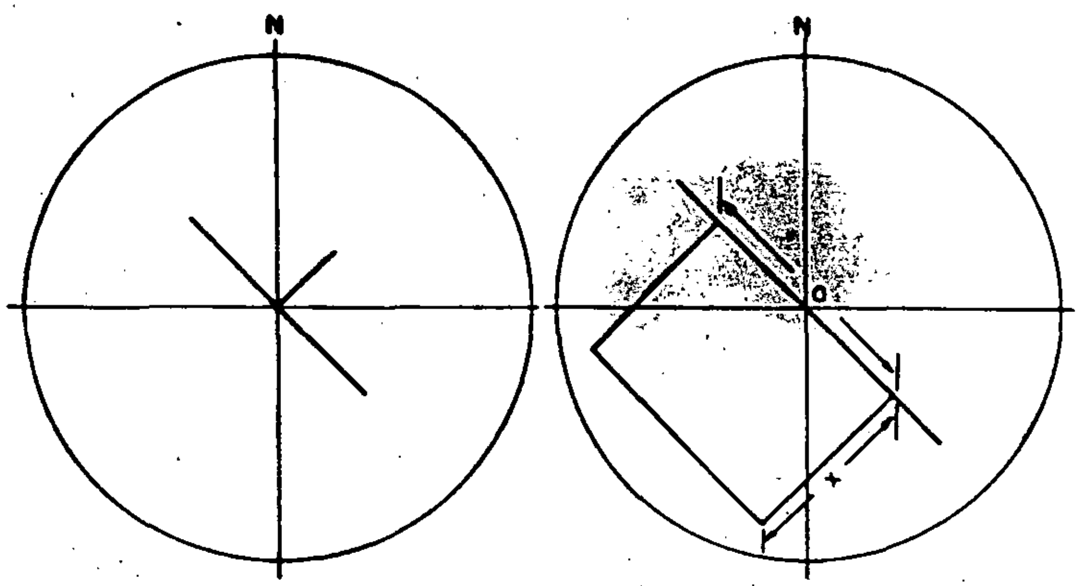


Figure 21 : Presentation of structural geology information and preliminary evaluation of slope stability of a proposed open pit mine.

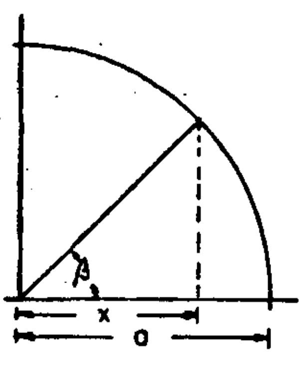
La banderola de MÜLLER

La representación de un plano geológico por medio del escorzo de un cuadrado unitario



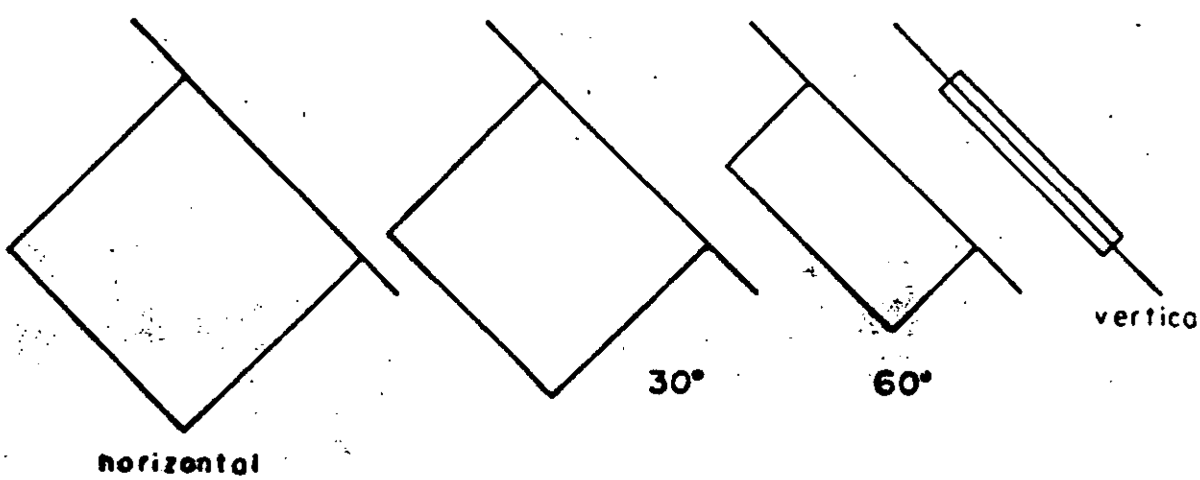
Símbolo convencional

"Banderola de MÜLLER"



Determinación gráfica del escorzo x

$x = a \cdot \cos\beta$



geotecnistas, proyectistas y contratistas.
Una propuesta de Don U. Deere que proporciona una terminología uniforme para la descripción de juntas es la siguiente:

DESCRIPCION	Espaciamiento de juntas	
	Sist. Inglés	Sist. Métrico
Muy cercanas	Menos de 2"	Menos de 5 cm
Cercanas	2" - 1'	5cm - 30cm
Medianamente cercanas	1' - 3'	30 cm - 1m
Espaciadas	3' - 10'	1m - 3m
Muy espaciadas	Más de 10'	Más de 3 m

TERMINOLOGIA DESCRIPTIVA PARA EL ESPACIAMIENTO DE JUNTAS

En los siguientes apartados se describirán dos métodos generales para calificar la calidad de la roca "in situ" basadas sobre la cantidad relativa de fracturamiento y alteración.
En uno de los métodos se utiliza una recuperación de núcleos modificada como base de la clasificación y en el otro, se utiliza la velocidad sísmica.

1.3.1.- Indice de Calidad de la Roca, R.Q.D.
(Rock quality designation)

El índice de calidad de la roca (RQD) está basado sobre un procedimiento de recuperación de núcleos modificado, el cual esta apoyado indirectamente sobre el número de fracturas y la cantidad de alteración o suavidad de la roca como se observa en los núcleos recuperados en un barreno. En lugar de contar las fracturas, se obtiene una medida indirecta mediante la suma de todos los fragmentos duros y sanos con longitud ≥ 10 cm.

Ejemplo:

Recuperación (pulg)	Recuperación modificada (pulg)
10	10
2	
2	
3	
4	4
5	5
3	
4	4
6	6
4	
2	
5	5
50"	34"

L=60"

$$\%\ \text{Recuperación} = \frac{50}{60} = 83\%$$

$$RQD = \frac{34}{60} = 57\%$$

En este caso la recuperación total es de 83% mientras que el índice de calidad de la roca es 57%. Se ha visto que el RQD es un indicador más sensible de la calidad general de la roca que el porcentaje de recuperación total.

Si el núcleo se rompe por manejo o por efectos de la perforación p.ej. si la superficie de la fractura se observa fresc deberán unirse los dos fragmentos y considerarse como una sola pieza.

Este criterio puede cambiarse cuando se trate de rocas sedimentarias con estratificación delgada y rocas metamórficas foliadas. Sin embargo este sistema ha sido aplicado exitosamente aún para lutitas, siendo necesario realizar inmediatamente el registro después de sacar los núcleos del muestreador y antes del efecto de fracturamiento por secado al aire.

Este método penaliza la roca con pobre recuperación. Esto es apropiado porque una recuperación pobre generalmente coincide con un calidad pobre. Esto no siempre es cierto pues el equipo de perforación y la técnica empleada pueden ocasionar pobre recuperación. Por esta razón se recomienda el uso del doble barril giratorio con diámetro mínimo NX (2⅛")

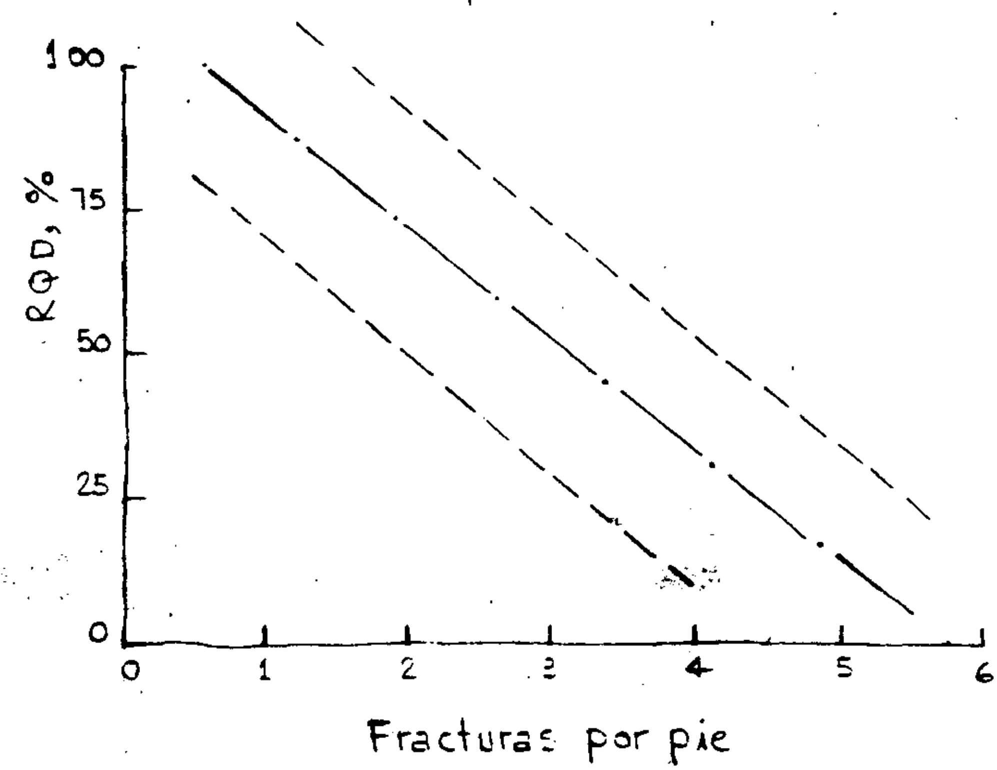
Tan simple como parece este método, se ha encontrado que existe una buena correlación entre los valores numéricos del RQD y la calidad ingenieril de las rocas.

El RQD usualmente ha sido utilizado por algunas Compañías Americanas, Consultores en Geotecnia y Contratistas para la evaluación de la calidad de la roca.

Indice de Calidad de roca RQD, %	Descripción de la Calidad
0 - 25	Muy pobre
25 - 50	Pobre
50 - 75	Regular
75 - 90	Buena
90 - 100	Excelente

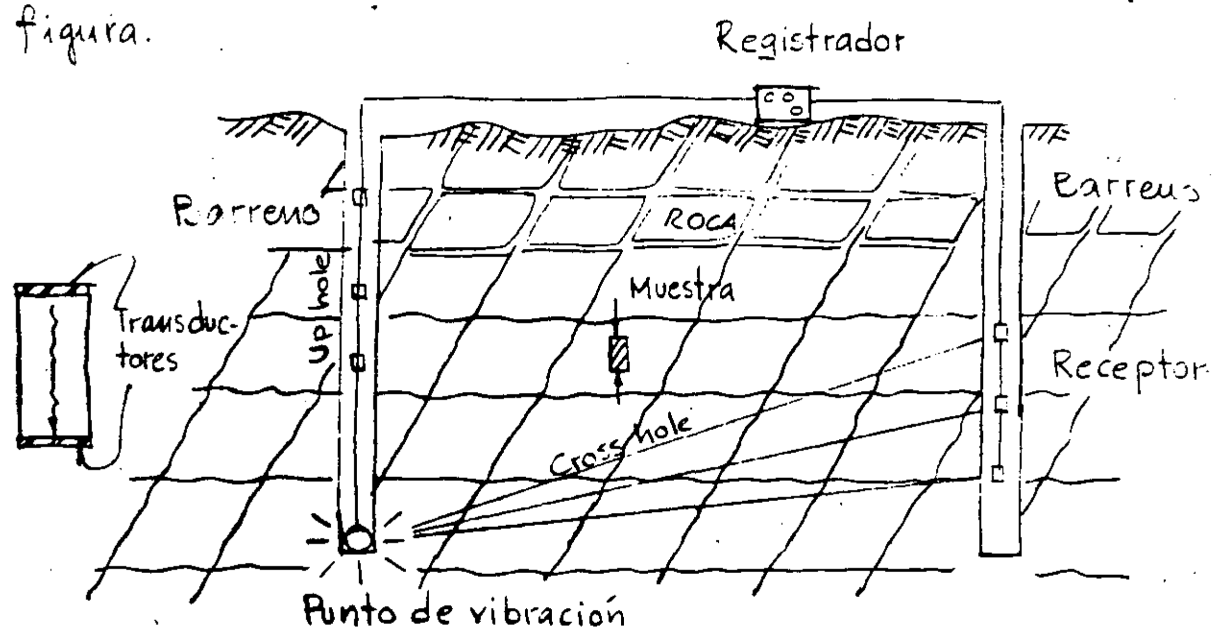
RELACION ENTRE EL RQD Y LA CALIDAD DE LA ROCA.

A continuación se presenta la correlación entre la frecuencia de fracturas y la calidad RQD, observándose una correlación lineal con límites aceptables:



1.3.2. Relación de velocidades sísmicas

El efecto de las discontinuidades en la masa de roca puede ser estimado por comparación de la velocidad de la onda compresional "in situ" con la velocidad sónica de laboratorio obtenida en núcleos intactos de la misma roca como se observa en la siguiente figura.



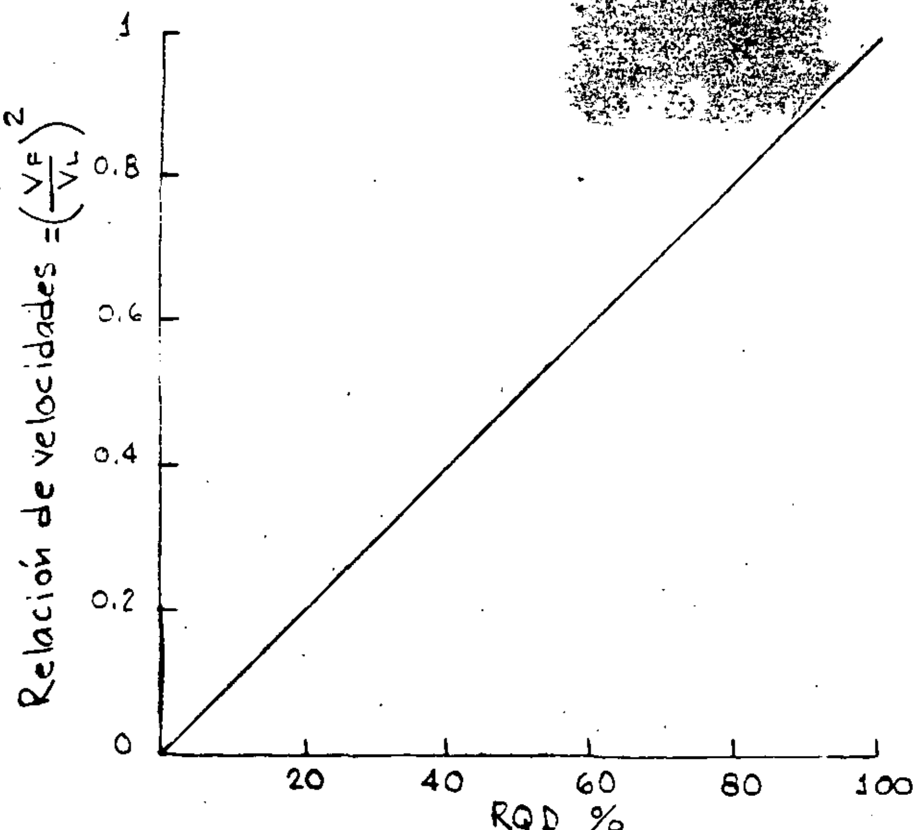
RELACION DE VELOCIDADES COMO INDICE DE CALIDAD DE ROCA ;

$$\text{Relación de velocidad} = \frac{V_F}{V_L} \; ; \; \begin{array}{l}\text{Velocidad de campo} \\ \text{Velocidad de lab.}\end{array}$$

La diferencia entre estas dos velocidades dilatantes es ocasionada por las discontinuidades estructurales existentes en el campo.

La relación entre estas ondas compresionales V_F/V_L, donde V_F y V_L son las velocidades de ondas compresionales de la masa de roca "in situ" y de un espécimen intacto fue propuest por Onodera, 1963.

Para una roca masiva de alta calidad con pocas juntas, la relación de velocidades se aproxima a la unidad.



CORRELACION ENTRE EL RQD Y LA RELACION DE VELOCIDADES $\left(\frac{V_F}{V_L}\right)^2$

La velocidad sónica debe determinarse en núcleos sujetos a un esfuerzo axial igual al que produce la cobertura de roca a la misma profundidad a la que fué tomada la muestra y con un contenido de agua equivalente al de la roca "in situ". La velocidad sísmica es mejor obtenerla entre barrenos con el sistema "Up-hole" o "Cross-hole" que mediante refracción superficial.

Parece que el cuadrado de la relación de velocidades es intercambiable con el RQD, pero todavía hay poca información

2.- PROPIEDADES MECANICAS DE LA ROCA

2.1.- Introducción

En muchos problemas de la mecánica de rocas las propiedades ingenieriles de la roca intacta son de importancia primordial. En otros casos resulta más importante el comportamiento de la roca "in situ" con sus inherentes discontinuidades geológicas

2.2.- Propiedades ingenieriles de la roca intacta.

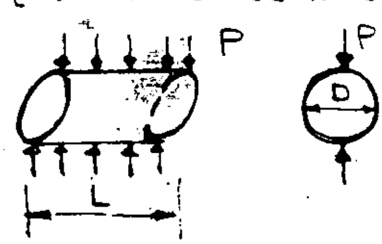
2.2.1.- Resistencia en tensión

La determinación de la resistencia a la tensión por extensión directa de un espécimen cilíndrico ha sido difícil de realizar, pues los dispositivos de sujeción introducen flexiones.

A causa de la dificultad arriba mencionada es más rápido obtener la resistencia en tensión en forma indirecta mediante la prueba "Brasileña". En esta prueba la probeta cilíndrica se ensaya acostada aplicando una carga lineal diametralmente opuesta.

La resistencia a la tensión σ_t se calcula mediante la expresión:

$$\sigma_t = \frac{2P}{\pi DL}$$



en donde:

P= carga a la falla ; D=diámetro
L= longitud del espécimen

Otra prueba indirecta para determinar la resistencia en tensión de la roca es la "Carga puntual" que se realiza aplicando una carga puntual de compresión sobre la superficie curva de un especimen cilíndrico con su eje horizontal. Esta carga produce esfuerzos de tensión perpendiculares al eje de carga. La resistencia a la tensión σ_t esta dada por una expresión empírica:

$$\sigma_t = \frac{0.96\, P}{D^2}$$

en donde:

P = carga de falla en lb y D = diámetro en pulg.

Miller relacionó esta resistencia en tensión de punta con la resistencia uniaxial de compresión mediante la siguiente expresión:

$$\sigma_{a(ult)} = 21\,\sigma_t + 4000\ \text{lb/pulg}^2$$

en donde $\sigma_{a(ult)}$ = resistencia en compresión y

σ_t = resistencia en tensión bajo carga puntual.

Para propósitos ingenieriles se tiene suficiente aproximación suponer una resistencia a la tensión comprendida entre 5 y 10 % de la resistencia en compresión. No se requiere mayor aproximación en la determinación en vista del amplio intervalo de variación en la resistencia, sobre todo en rocas metamórficas y sedimentarias con estratificación delgada

2.2.2.- Resistencia en compresión simple

El comportamiento de la roca intacta bajo compresión uniaxial esta influenciada por las características intrínsicas de la prueba como son la relación de esbeltez, la velocidad de carga y las condiciones de fricción de los apoyos.

En especímenes con relación de esbeltez pequeña no pueden desarrollarse los planos de cortante por el efecto de fricción de los apoyos, resultando un valor mayor de la resistencia en compresión. Obert y Duvall han encontrado una relación empírica entre la resistencia a la compresión y la relación de esbeltez como sigue:

$$\sigma_{a(u1t)} = \sigma_{al} \left(0.778 + \frac{0.222}{L/D} \right)$$

en donde: $\sigma_{a(u1t)}$ = resistencia en compresión para $L/D \neq 1$

y σ_{al} es la resistencia en compresión para $L/D = 1$

Se recomienda una relación de esbeltez entre 2 y 2.5 para asegurar una distribución de esfuerzos más o menos uniforme en la muestra alejándose también del efecto de fricción de los cabezales. En la Fig. se presenta en forma gráfica la relación entre la resistencia en compresión simple y la esbeltez del espécimen.

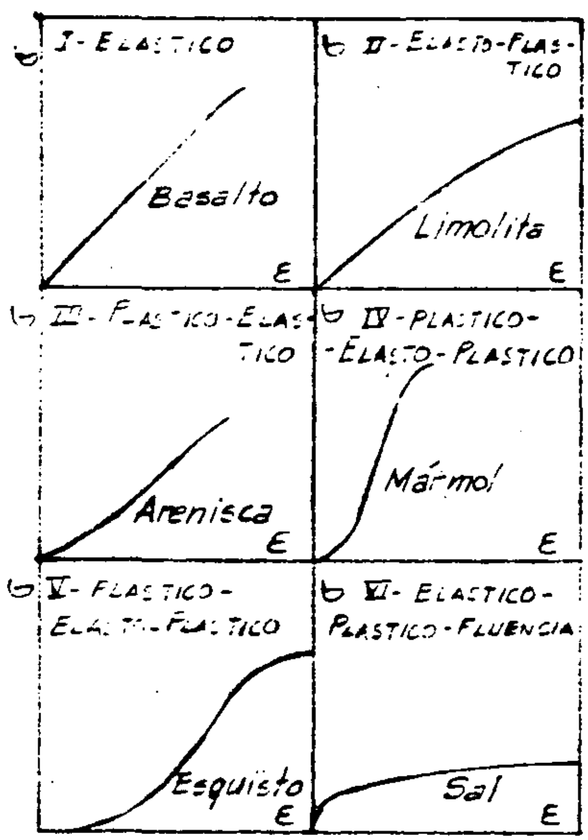
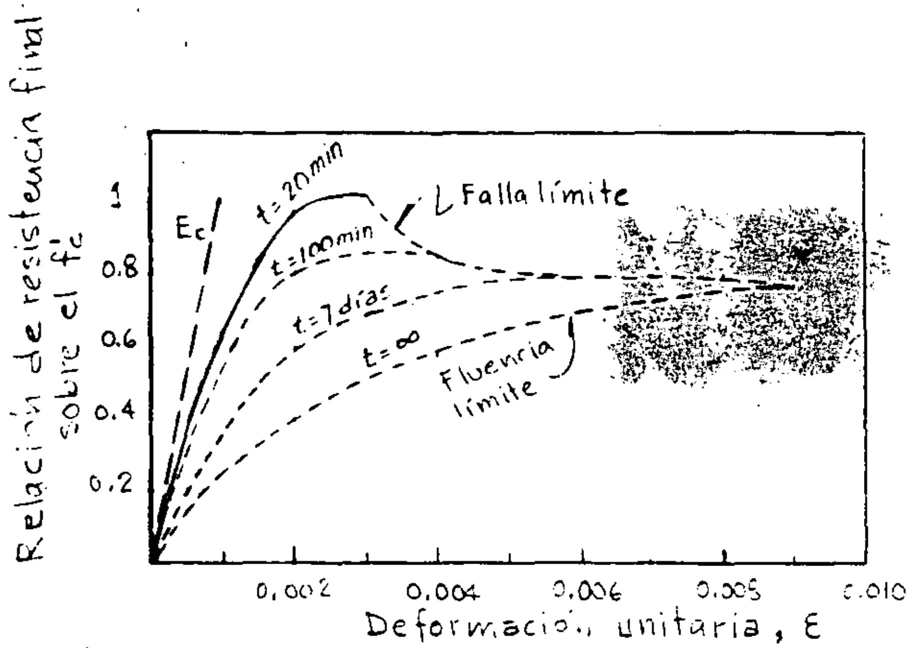


FIG. CURVAS ESFUERZO - DEFORMACION TIPICAS PARA ROCAS A LA FALLA EN COMPRESION SIMPLE



INFLUENCIA DE LA VELOCIDAD DE CARGA EN LA RESISTENCIA Y MODULO EN ESPECIMENES DE CONCRETO

En la Fig. .. se presenta varias gráficas esfuerzo-deformación en ensayes de compresión simple para varios tipos de roca.

A continuación se presentan algunos valores de resistencia obtenidos en varios tipos de roca

TIPO DE ROCA		σ_a (UH) (lb/pulg^2 x 10^3)	c, Cohesión lb/pulg^2 x 10^3	ϕ (grados)	$N_\phi = K$
Granito	Intervalo	10-40	1.4-5.8	51-58	8-17
	Promedio	25	3.6	55	11
Calizas	Intervalo	3-30	0.5-5	37-58	4-13
	Promedio	15-20	2.5-3.3	50	8
Areniscas	Intervalo	3-30	0.6-6	48-50	6-7
	Promedio	8-20	1.6-4.1	48	6

$N_\phi = \frac{1+\operatorname{sen}\phi}{\ldots}$

La velocidad de carga afecta tanto la resistencia en compresión como el módulo elástico. En la siguiente tabla se presentan los resultados obtenidos para dos velocidades de carga con la arenisca Berea y un Gabro.

Roca	Resistencia en compresión simple (lb/pulg²)		
	Tiempo a la falla = 30 seg	Tiempo a la falla = 0.02 seg	Incremento en resistencia (%)
Arenisca Berea	8 000	12 000	50
Gabbro	31 000	40 000	30

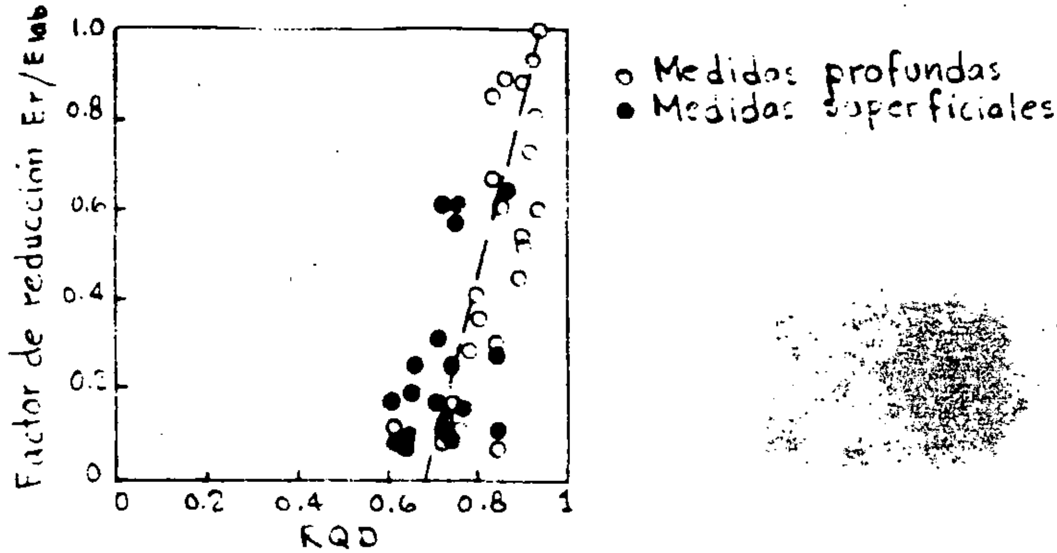
En el concreto simple sucede lo mismo, al incrementar la velocidad de carga, la resistencia y el módulo elástico aumentan y la deformación unitaria disminuye. Resultados similares se han obtenido en rocas carbonatadas de manera que pudiera esperarse este mismo efecto en otros tipos de roca. Werker y Watstein encontraron diferencias despreciables en resistencia variando la velocidad de carga entre 10 y 100 lb/pulg²/seg, por lo que se ha considerado que este tipo de prueba se realice con velocidades de carga comprendidas entre esos límites.

2.3 Propiedades ingenieriles de la roca "in situ"

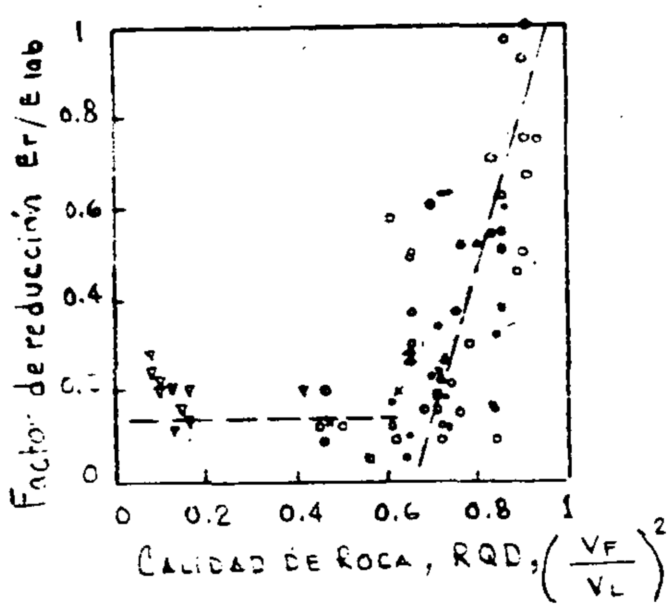
En la Fig se presenta la variación del factor de reducción contra calidad de la roca (RQD) a partir de pruebas de placa flexible de 34" de diámetro en granitos de la Presa Dworshak, considerando al factor de reducción como la relación entre módulos elásticos de campo y laboratorio.

Se observa que los módulos de deformación son consistentemente más altos con la profundidad que los módulos superficiales, y que a mayor calidad de roca el factor de reducción vá aproximándose a la unidad.

En la Fig se presenta la variación entre la calidad de roca, RQD, o, $(V_F/V_L)^2$ contra el factor de reducción observándose que para valores de RQD menores de 65% el factor de reducción varía más o menos entre 0.1 y 0.2 y que para valores de RQD mayores de 65% se tiene una relación lineal con el factor de reducción. Para valores de RQD entre 90 y 100% el factor de reducción varía entre 0.8 y 1.



VARIACION DEL FACTOR DE REDUCCION CONTRA CALIDAD DE LA ROCA (R.Q.D.) EN PRUEBAS DE PLACA Ø 24" EN GRANITOS DE LA PRESA DWORSHAK

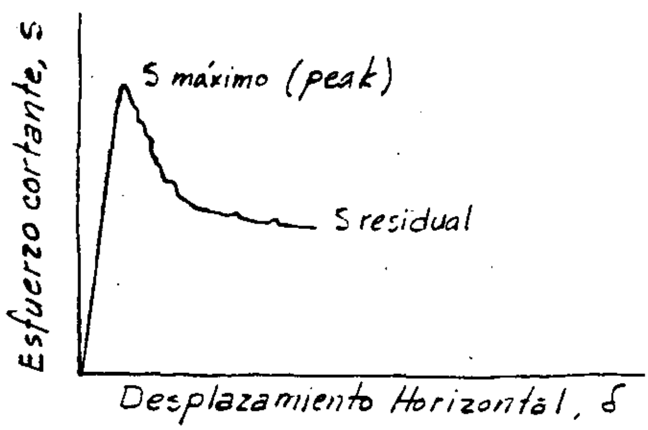


REF. ROCK MECHANICS
Stagg y Zienkiewicz
Ed. John Wiley & Sons 1968

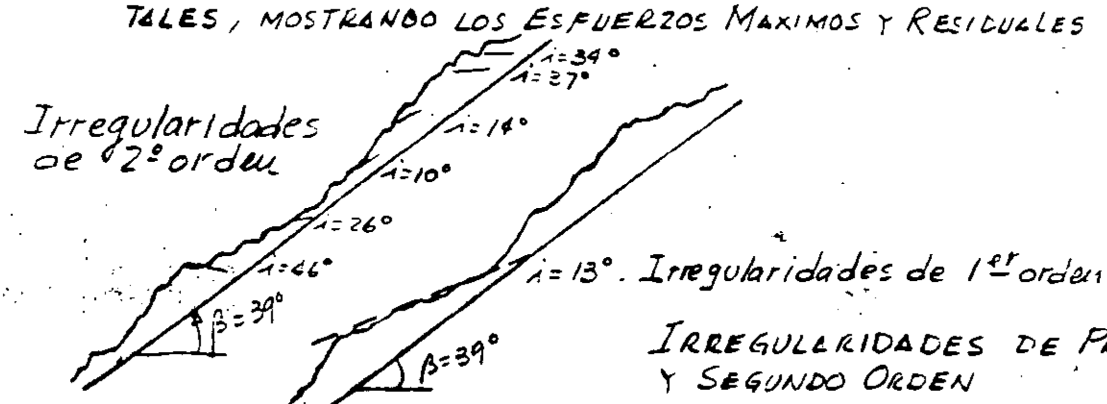
2.3.1. Resistencia al esfuerzo cortante

La resistencia al esfuerzo cortante "in situ" depende de los siguientes factores: a) del material de relleno b) del espesor de la fractura c) de la irregularidad del plano de falla y d) de las irregularidades secundarias del plano de falla.

A continuación se presenta una gráfica típica de esfuerzo cortante - desplazamiento obtenida en ensayes de campo



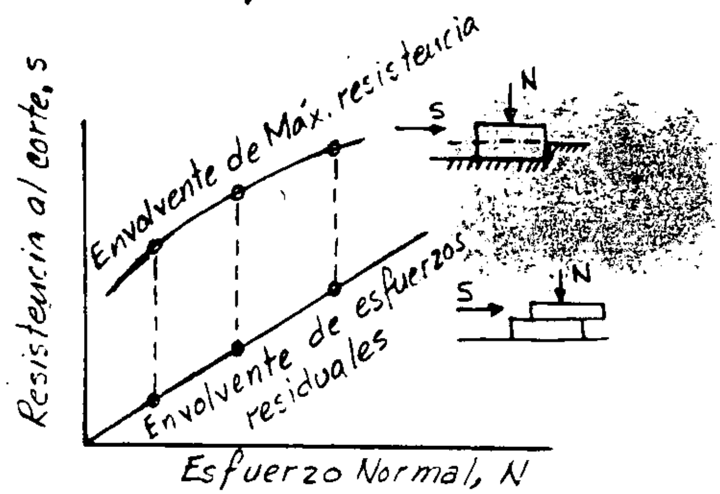
RESISTENCIA AL ESFUERZO CORTANTE CONTRA DESPLAZAMIENTOS HORIZONTALES, MOSTRANDO LOS ESFUERZOS MAXIMOS Y RESIDUALES



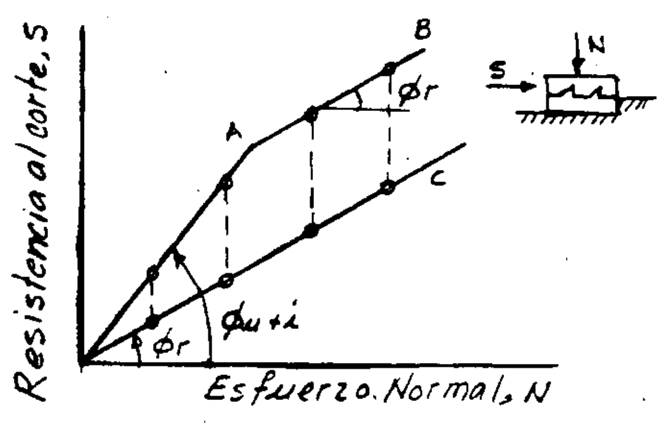
IRREGULARIDADES DE PRIMERO Y SEGUNDO ORDEN

Se ha obtenido por F. D. Patton que $i = \pm 10 \text{ ó } 15°$ como componente de resistencia adicional por las irregularidades "in situ"

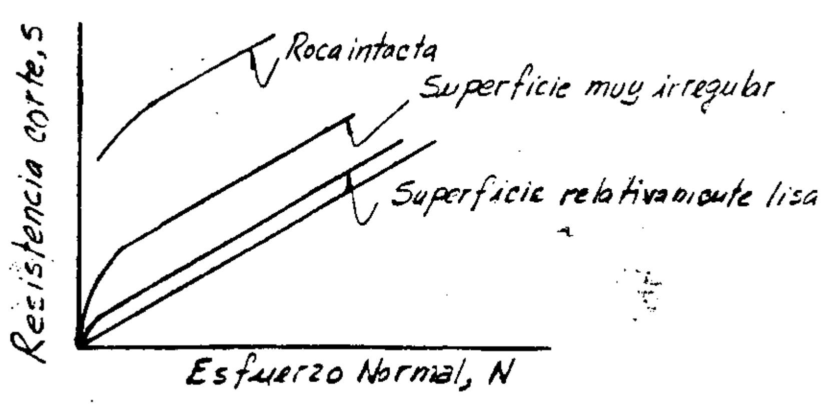
Efecto de las irregularidades del plano de falla.



ESFUERZOS CORTANTES MAXIMOS Y RESIDUALES PARA ROCA INTACTA



ENVOLVENTES DE FALLA PARA SUPERFICIES IRREGULARES



ENVOLVENTES DE FALLA COMUNES PARA MACIZOS ROCOSOS

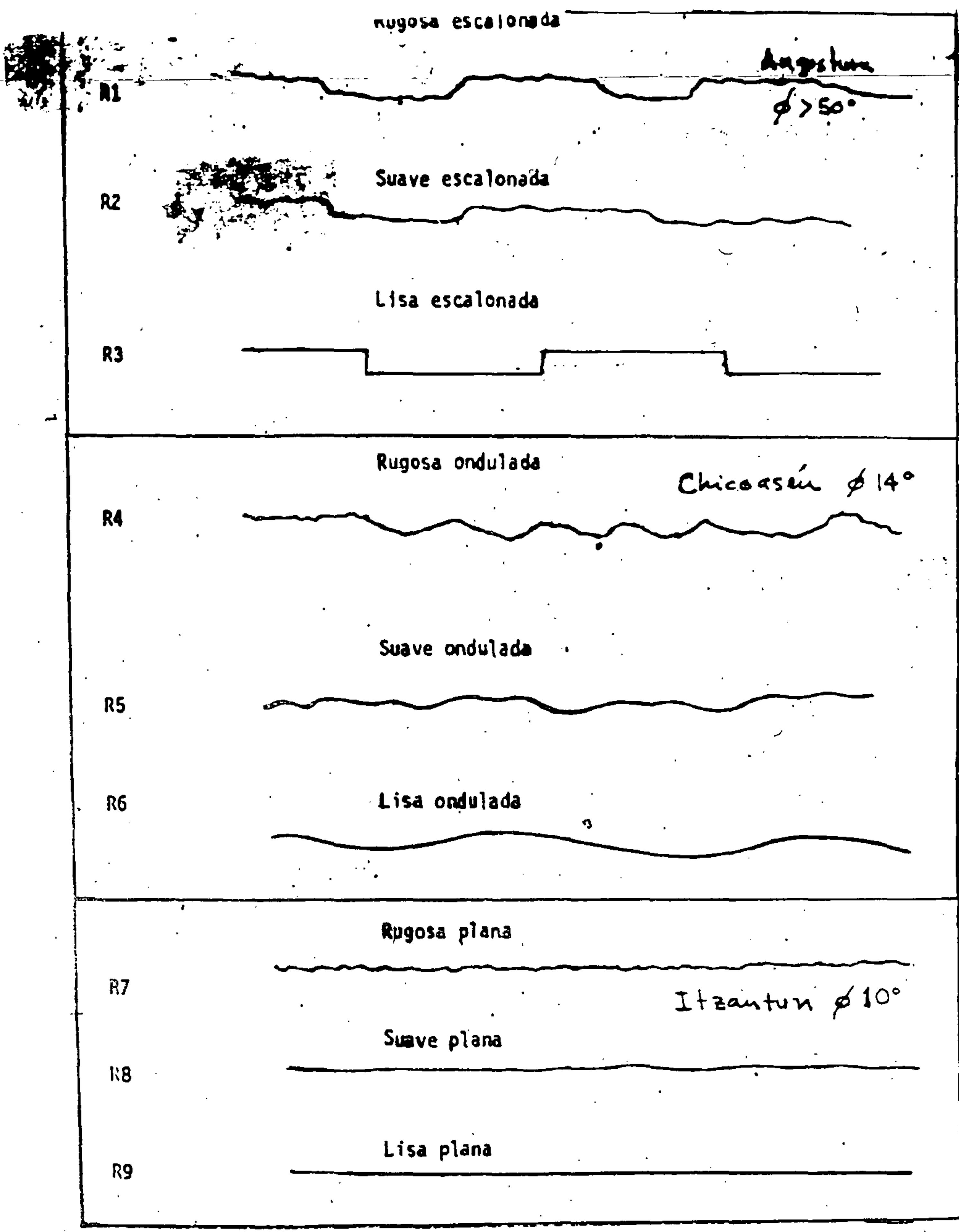


FIGURA Nº .- JUNTAS RUGOSAS CERRADAS.

FACULTAD DE INGENIERIA U.N.A.M.
DIVISION DE EDUCACION CONTINUA

CURSOS ABIERTOS

VI CURSO INTERNACIONAL DE INGENIERIA GEOLOGICA APLICADA A OBRAS SUPERFICIALES Y SUBTERRANEAS

MODULO IV: TECNOLOGIA SOBRE EL USO DE EXPLOSIVOS

SURFACE BLAST DESIGN

EXPONENTE: ING. RAUL CUELLAR BORJA

SURFACE BLAST DESIGN

This article is an excerpt from Bureau of Miles circular IC 8925, "Surface Blast Design."

BLASTHOLE DIAMETER

The size of blasthole is the first consideration of any blast design. The blasthole diameter, along with the type of explosive being used and the type of rock being blasted, will determine the burden. All other blast dimensions are a function of the burden. This discussion assumes that the blaster has the freedom to select the borehole size. In many operations one is limited to a specific size borehole based on available drilling equipment.

Practical blasthole diameters for surface mining range from 2 to 17 in. As a general rule, large blasthole diameters yield low drilling and blasting costs because large holes are cheaper to drill per unit volume and less sensitive, cheaper blasting agents can be used in larger diameters. However, larger diameter blastholes also result in large burdens and spacings and collar distances and hence, they tend to give coarser fragmentation. Figure 65 illustrates this comparison using 2- and 20-in-diameter blastholes as an example. Pattern A contains four 20-in blastholes and pattern B contains 400 2-in blastholes. In all bench blasting operations some compromise between these two extremes is chosen. Each pattern represents the same area of excavation, 15,000 sq ft, each involves approximately the same volume of blastholes, and each can be loaded with about the same weight of explosive.

In a given rock formation, the four-hole pattern will give relatively low drilling and blasting costs. Drilling costs for the large blastholes will be low, a low-cost blasting agent will be used, and the cost of detonators will be minimal. However, in a difficult blasting situation, the broken material will be blocky and nonuniform in size, resulting in higher loading, hauling, and crushing costs as well as requiring more secondary breakage. Insufficient breakage at the toe may also result.

On the other hand, the 400-hole pattern will yield high drilling and blasting costs. Small holes cost more to drill per unit volume, powder for small-diameter blastholes is usually more expensive, and the cost of detonators will be higher. However, the fragmentation will be finer and more uniform, resulting in lower loading, hauling, and crushing costs. Secondary blasting and toe problems will be minimized. Size of equipment, subsequent processing required for the blasted material, and economics will dictate the type of fragmentation needed, and hence the size of blasthole to be used.

Geologic structure is a major factor in determining blasthole diameter. Planes of weakness such as joints and beds, or zones of soft, incompetent rock tend to isolate large blocks of rock in the burden. The larger the blast pattern, the more likely these blocks are to be thrown unbroken into the muckpile.

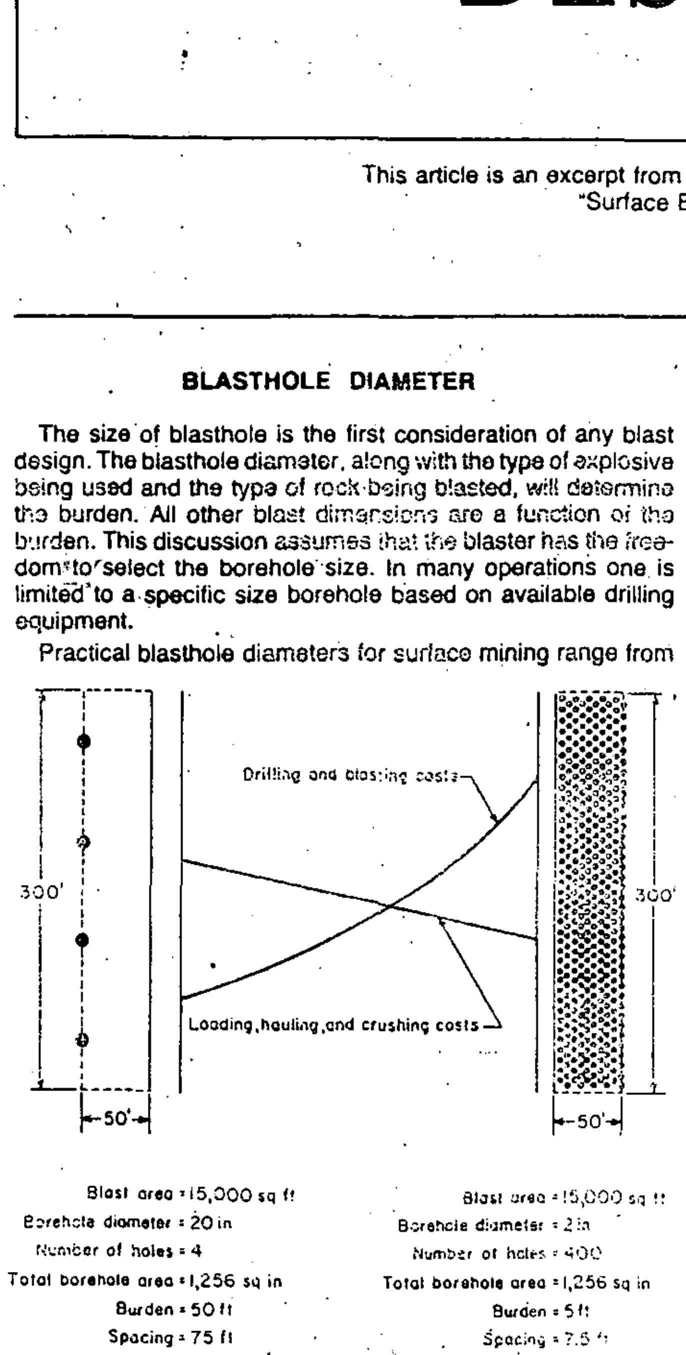


Figure 65.—Effect of large and small blastholes on unit costs.

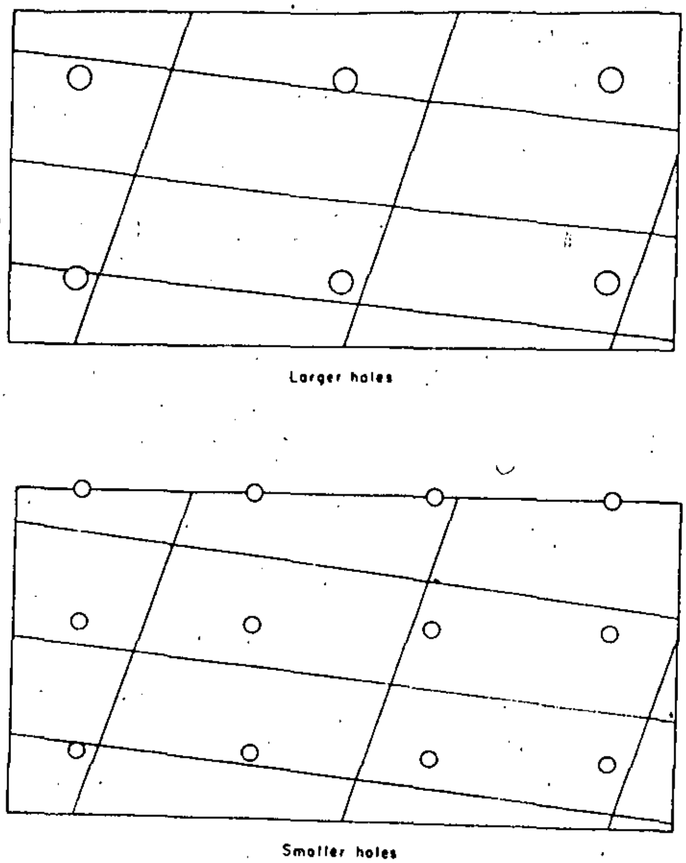


Figure 66.—Effect of jointing on selection of blasthole size.

Note that in the top pattern in figure 66 some of the blocks are not penetrated by a blasthole, whereas in the smaller bottom pattern all of the blocks contain at least one blasthole. Owing to the better explosives distribution, the bottom pattern will give better fragmentation.

As more blasting operations are carried out near populated areas, environmental problems such as airblast and flyrock often occur because of an insufficient collar distance above the explosive charge. As the blasthole diameter increases, the collar distance required to prevent violence increases. The ratio of collar distance to blasthole diameter required to prevent violence varies from 14:1 to 28:1, depending on the relative densities and velocities of the explosive and rock, the physical condition of the rock, the type of stemming used, and the point of initiation. A larger collar distance is required where the sonic velocity of the rock exceeds the detonation velocity of the explosive or where the rock is heavily fractured or low in density. A top-initiated charge requires a larger collar distance than a bottom-initiated charge. As the collar distance increases, the powder distribution becomes poorer resulting in poorer fragmentation of the rock in the upper part of the bench.

Ground vibrations are controlled by reducing the weight of explosive fired per delay interval. This is more easily done with small blastholes than with large blastholes. In many situations where an operator uses large-diameter blastholes near populated areas, several delayed decks must be used within each hole to control vibrations.

Large holes with large blast patterns are ideally suited to an operation with the following characteristics: A large volume of material to be moved; large loading, hauling, and crushing equipment; no requirement for fine, uniform fragmentation; an easily broken toe; few ground vibration or airblast problems (few nearby neighbors); and a relatively homogeneous, easily fragmented rock without excessive, widely spaced planes of weakness or voids. Many blasting jobs, however, present constraints that require smaller blastholes.

In the final analysis, the selection of blasthole size is based on economics. It is important to consider the economics of the overall excavation or mining system. Savings realized through indiscriminate cost cutting in the drilling and blasting program may well be lost through increased loading, hauling, and crushing costs and increased litigation costs owing to disgruntled neighbors.

TYPES OF BLAST PATTERNS

There are three commonly used drill patterns: square, rectangular, and staggered. The square drill pattern (fig. 67) has equal burdens and spacings, while the rectangular pattern has a larger spacing than burden. In both the square and rectangular patterns, the holes of each row are lined up directly behind the holes in the preceding row. In the staggered pattern (fig. 67), the holes in each row are positioned in the middle of the spacings of the holes in the preceding row. In the staggered pattern, the spacing should be larger than the burden.

The staggered drilling pattern is used for row-on-row firing; that is, where the holes of one row are fired before the holes in the row immediately behind them as shown in figure 68. The square or rectangular drilling patterns are used for firing V-cut (fig. 69) or echelon rounds. Either side of the blast round in figure 69 by itself would be called an echelon blast round. In V-cut or echelon blast rounds the burdens and subsequent rock displacement are at an angle to the original free face. Looking at figure 69, with the burdens developed at a 45° angle with the original free face, you can see that the originally square drilling pattern has been transformed to a staggered blasting pattern with a spacing twice the burden. The simple patterns discussed here account for the vast majority of the surface blasts fired.

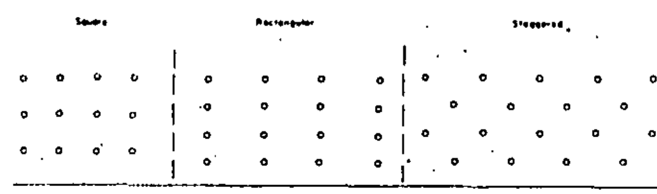
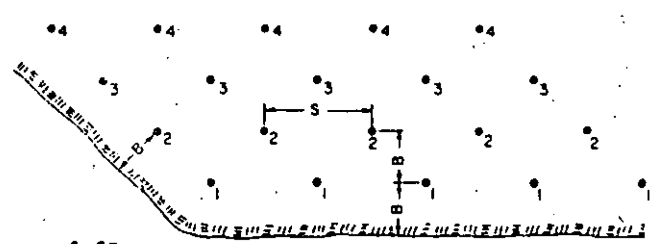


Figure 67.—Three basic types of drill pattern.



S = 2B

Figure 68.—Corner cut staggered blast pattern—Simultaneous initiation within rows (blasthole spacing, S, is twice the burden, B).

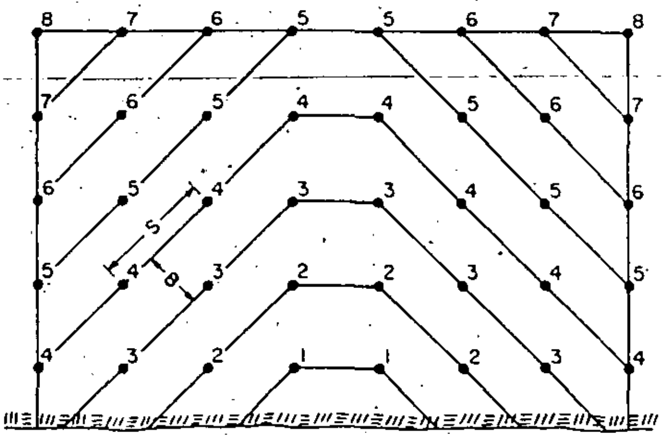


Figure 69.—V-echelon blast round (true spacing, S, is twice the true burden, B).

BURDEN

Figure 70 is an isometric view showing the relationship of the various dimensions of a bench blast. The burden is defined as the distance from a blasthole to the nearest free face at the instant of detonation. In multiple row blasts, the burden for a blasthole is not necessarily measured in the direction of the original free face. One must take into account the free faces developed by blastholes fired on lower delay periods. As an example, in figure 68, where one entire row is blasted before the next row begins, the burden is measured in a perpendicular direction between rows. However, in figure 69 the blast progresses in a V-shape. In this situation, the true burden on most of the holes is measured at an angle of 45° from the original free face, as shown in the figure.

It is very important that the proper burden be calculated, taking into account the blasthole diameter, the relative density of the rock and the explosive, and to some degree, the length of the blasthole. An insufficient burden will cause excessive airblast and flyrock. Too large a burden will give inadequate fragmentation, toe problems, and excessive ground vibrations. Where it will be necessary to drill a round before the previous round has been excavated, it is important to stake out the first row of the second round before the first round is fired. This will assure a proper burden on the first row of blastholes in the second blast round.

The burden dimension is a function of the charge diameter. For bulk-loaded charges, the charge diameter is equal to the blasthole diameter. For tamped cartridges, the charge diameter will be between the cartridge diameter and the blasthole diameter, depending on the degree of tamping. For untamped cartridges the charge diameter is equal to the cartridge diameter. When blasting with AN-FO or other low density blasting agents with densities near 0.85 g/cu cm, in typical rock with a density near 2.7 g/cu cm, the normal burden is approximately 25 times the charge diameter. When using denser products such as slurries or dynamites, with densities near 1.2 g/cu cm, the normal burden is approximately 30 times the charge diameter. It should be stressed again that these are first approximations, and field testing often results in minor adjustments to these values. The burden-to-charge-diameter ratio is seldom less than 20 or seldom more than 40, even in extreme cases. For instance, when blasting with a low density blasting agent, such as AN-FO, in a dense formation such as iron ore, the desired burden may be about 20 times the charge diameter. When blasting with denser slurries or dynamites in low density formations such as some sandstones or marbles, the burden may approach 40 times the charge diameter. Table 4 summarizes these approximations.

Table 4.-Approximate B/D ratios for bench blasting

	Ratio
AN-FO (density—0.85 g/cu cm):	
Light rock (density-2.2 g/cu cm)	28
Average rock (density—2.7 g/cu cm)	25
Dense rock (density—3.2 g/cu cm)	23
Slurry, dynamite (density—1.2 g/cu cm):	
Light rock (density—2.2 g/cu cm)	33
Average rock (density—2.7 g/cu cm)	30
Dense rock (density—3.2 g/cu cm)	27

B Burden D Charge diameter

High-speed photographs of blasts have shown that flexing of the burden plays an important role in rock fragmentation. A relatively long, slender burden flexes, and thus breaks more easily than a short, stiffer burden. Figure 71 shows the difference between using a 6-in blasthole and a 12¼-in blasthole in a 40-ft bench, with a burden-to-charge-diameter ration of 30 and appropriate subdrilling and stemming dimensions. Note the inherent stiffness of the burden with the 12¼-in blasthole as compared with the 6-in blasthole. Based on this consideration, lower burden-to-charge-diameter ratios should be used as a first approximation when the blasthole diameter is large in comparison to the bench height. Care must be taken that the burden ratio is not so small as to create violence. Once the burden has been determined, it becomes the basis for calculating subdrilling, collar distance (stemming), and spacing.

SUBDRILLING

Subdrilling is the distance drilled below the floor level to assure that the full face of rock is removed. Where there is a pronounced parting at floor level, to which the explosive charge can conveniently break, subdrilling may not be required. In coal stripping, it is common practice to drill down to the coal and then backfill a foot or two before loading explosives, resulting in a negative subdrill. In most surface blasting jobs, however, it is necessary to do some subdrilling to make sure the shot pulls to grade. A good first approximation for subdrilling under average conditions is 30 pct of the burden. Where the toe breaks very easily, the subdrill can sometimes be reduced to 10 to 20 pct of the burden. Even under the most difficult conditions, the subdrill should not exceed 50 pct of the burden. If the toe cannot be pulled with a subdrill-to-burden ratio of 0.5, the fault probably lies in too large a burden.

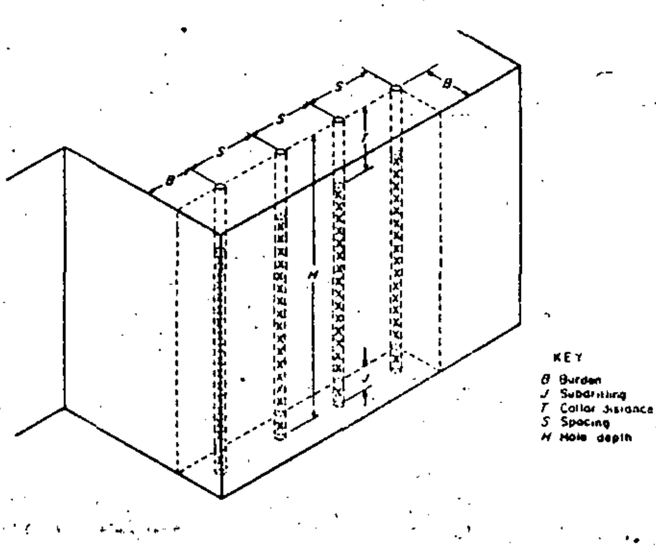


Figure 70.—Isometric view of a bench blast.

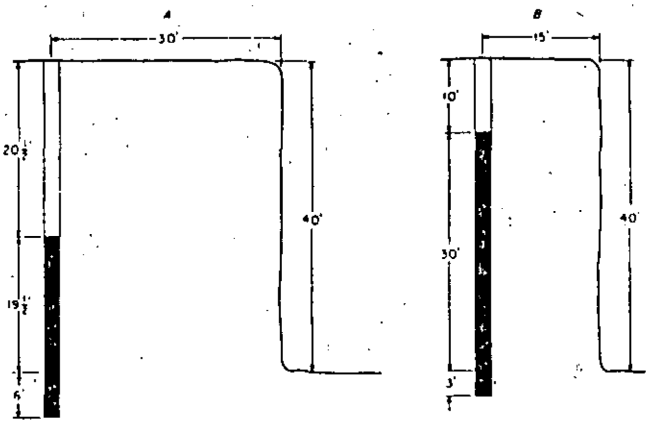


Figure 71.—Comparison of a 12¼-in-diameter (A) blasthole (stiff burden) with a 6-in-diameter (B) blasthole (flexible burden) in a 40-ft bench.

Priming the explosive column at the toe level gives maximum confinement and normally gives the best breakage. Other factors being equal, toe priming usually requires less subdrilling than collar priming.

Too much subdrilling is a waste of drilling and blasting expense and may also cause excessive ground vibrations owing to the high degree of confinement of the explosive in the bottom of blasthole, particularly when the primer is placed in the bottom of the hole. In multiple-bench operations, excessive subdrilling may cause undue fracturing in the upper portion of the bench below, creating difficulties in collaring holes in the lower bench. Insufficient subdrilling will cause high bottom, resulting in increased wear and tear on equipment and expensive secondary blasting. Table 5 summarizes the recommended subdrilling approximations.

Table 5.-Approximate J/B ratios for bench blasting

	Ratio
Open bedding plane at toe	0
Easy toe	0.1-0.2
Normal toe	.3
Difficult toe	.4-.5

B Burden   J Subdrilling

COLLAR DISTANCE (STEMMING)

Collar distance is the distance from the top of the explosive charge to the collar of the blasthole. This zone is usually filled with an inert material called stemming to give some confinement to the explosive gases and to reduce airblast. Research has shown that crushed, sized rock works best as stemming but it is common practice to use drill cuttings because of economics. Too small a collar distance results in excessive violence in the form of airblast and flyrock and may cause backbreak. Too large a collar distance creates boulders in the upper part of the bench. The selection of a collar distance is often a tradeoff between fragmentation and the amount of airblast and flyrock that can be tolerated. This is especially true where the upper part of the bench contains rock that is difficult to break. In this situation the difference between a violent shot and one that fails to fragment the upper zone properly may be a matter of only a few feet of stemming. Collar priming of blastholes normally causes more violence than center or toe priming, and requires the use of a longer collar distance.

Field experience has shown that a collar distance equal to 70 pct of the burden is a good first approximation except where collar priming is used. Careful observation of airblast, flyrock, and fragmentation will enable the blaster to further refine this dimension. Where adequate fragmentation in the collar zone cannot be attained while still controlling airblast and flyrock, deck charges or satellite holes may be required.

A deck charge is an explosive charge near the top of the blasthole, separated from the main charge by inert stemming. If boulders are being created in the collar zone but the operator fears that less stemming would cause violence, the main charge should be reduced slightly and a deck charge added. The deck charge is usually shot on the same delay as the main charge or one delay later. Care must be exercised not to place the deck charge too near the top of the blasthole, or excessive flyrock may result. As an alternative, short satellite holes between the main blastholes can be used. These satellite holes are usually smaller in diameter than the main blastholes and are loaded with a light charge of explosives.

From the standpoint of public relations, collar distance is a very important blast design variable. One violent blast can permanently alienate neighbors. In a delicate situation, it may be best to start with a collar distance equal to the burden and gradually reduce this if conditions permit. Collar distances greater than the burden are seldom necessary.

SPACING

Spacing is defined as the distance between adjacent blastholes, measured perpendicular to the burden. Where the rows are blasted one after the other as in figure 68, the spacing is measured between holes in a row. However, in figure 69, where the blast progresses on an angle to the original free face, the spacing is measured at an angle from the original free face.

Spacing is calculated as a function of the burden and also depends on the timing between holes. Too close a spacing causes crushing and cratering between holes, boulders in the burden, and toe problems. Too wide a spacing causes inadequate fracturing between holes, accompanied by humps on the face and toe problems between holes (fig. 72).

When the holes in a row are initiated on the same delay period, a spacing equal to twice the burden will usually pull the round satisfactorily. Actually, the V-cut round in figure 69 also illustrates simultaneous initiation within a row, with the rows being the angled lines of holes fired on the same delay. The true spacing is twice the true burden even though the holes were originally drilled on a square pattern.

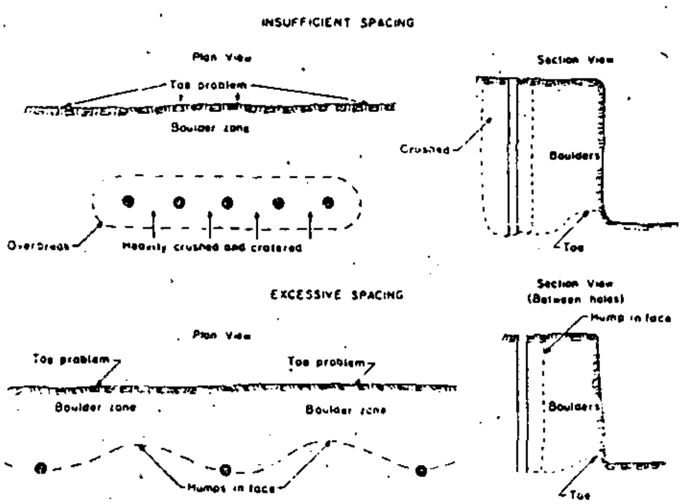


Figure 72.—Effects of insufficient and excessive spacing.

Field experience has shown that the use of millisecond delays between holes in a row results in better fragmentation and also reduces the ground vibrations produced by the blast. When millisecond delays are used between holes in a row, the spacing-to-burden ration must be reduced to somewhere between 1.2 and 1.8, with 1.5 being a good first approximation. Various delay patterns may be used within the rows, including alternate delays (fig. 73) and progressive delays (fig. 74). Generally, large-diameter blastholes require lower spacing-to-burden ratios (usually 1.2 to 1.5 with millisecond delays) than small-diameter blastholes (usually 1.5 to 1.8). Because of the complexities of geology, the interaction of delays, differences in explosive and rock strengths, and other variables, the proper

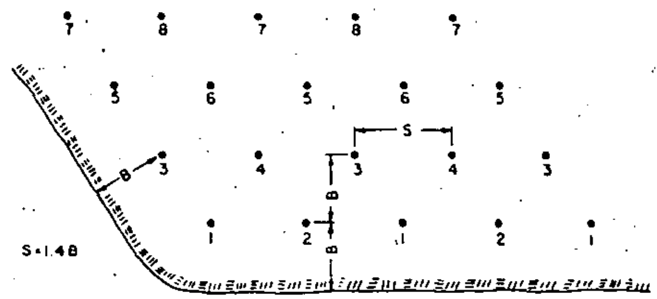


Figure 73.—Staggered blast pattern with alternate delays (spacing, S, is 1.4 times the burden, B).

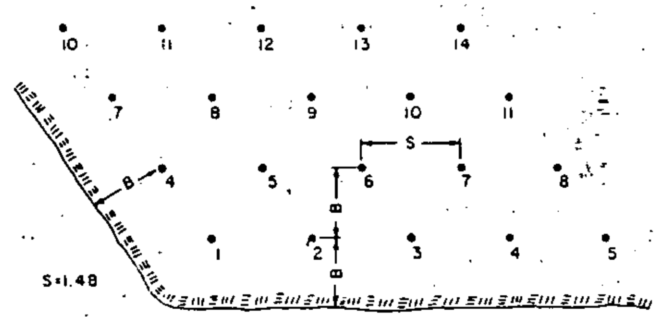


Figure 74.—Staggered blast pattern with progressive delays (spacing, S, is 1.4 times the burden, B).

spacing-to-burden ratio must be determined through onsite experimentation, using the preceding values as first approximations.

Except when using controlled blasting techniques such as smooth blasting and cushion blasting, the spacing should never be less than the burden.

HOLE DEPTH

In any blast design it is important that the burden and the blasthole depth (or bench height) be reasonably compatible. As a rule of thumb for bench blasting, the hole depth-to-burden ratio should be between 1.5 and 4.0. Hole depths less than 1.5 times the burden cause excessive airblast and flyrock and, because of the short, thick shape of the burden, give coarse, uneven fragmentation. Where operational conditions require a ratio of less than 1.5, the primer should be placed at the toe of the bench to assure maximum confinement. Keep in mind that placing the primer in the subdrill can cause increased ground vibrations. If an operator continually finds use of a hole depth-to-burden ratio of less than 1.5 necessary, consideration should be given to increasing the bench height or using a smaller drill.

Hole depths greater than four times the burden are also undesirable. The longer a hole is in respect to its diameter the more error there will be in its location at toe level, which is the most critical portion of the blast. A poorly controlled blast will result. Extremely long, slender holes have even been known to intersect.

High benches with short burdens also create hazards, such as a small drill having to put in the front row of holes near the edge of a high ledge or a small shovel having to dig at the toe of a precariously high face. The obvious solution to this problem is to use a lower bench height. There is no real advantage to a high bench height. Lower benches give more efficient blasting results, lower drilling cost and chances for cutoffs, and are safer from an equipment operation standpoint. If it is impractical to reduce the bench height, larger drilling and rock handling equipment should be used, which will effectively reduce the blasthole depth-to-burden ratio.

A major problem with long slender charges is the greater potential for cutoffs in the explosive column. Where it is necessary to use blast designs with large hole depth-to-burden ratios, multiple priming should be used as insurance against cutoffs.

DELAYS

Millisecond delays are used between charges in a blast round for three reasons:

1. To assure that a proper free face is developed to enable the explosive charge to efficiently fragment and displace its burden.

2. To enhance fragmentation between adjacent holes.

3. To reduce the ground vibrations created by the blast.

There are numerous possible delay patterns, several of which were covered in figures 68, 69, 73, and 74.

Andrews, of du Pont, conducted numerous field investigations to determine optimum delay intervals for bench blasting and reached the following conclusions.

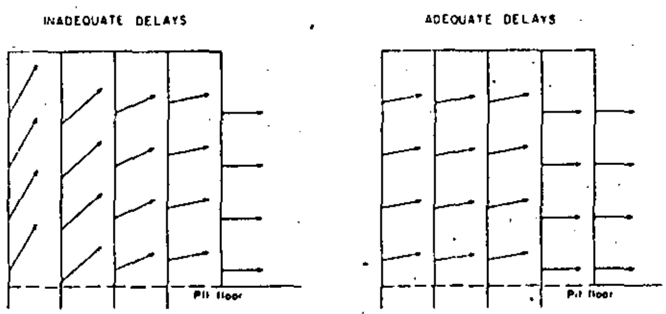


Figure 75.—The effect of inadequate delays between rows.

1. The delay time between holes in a row should be between 1 and 5 ms per foot of burden. Delay times less than 1 ms per foot of burden cause premature shearing between holes, resulting in coarse fragmentation. If an excessive delay time is used between holes, rock movement from the first hole prevents the adjacent hole from creating additional fractures between the two holes. A delay of 3 ms per foot of burden gives good results in many kinds of rock.

2. The delay time between rows should be two to three times the delay time between holes in a row. This is longer than most previous recommendations. However, in order to obtain good fragmentation and control flyrock, a sufficient delay is needed so that the burden from previously fired holes has enough time to move forward to accommodate broken rock from subsequent rows. If the delay between rows is too short, movement in the back rows will be upward rather than outward (fig. 75).

3. Where airblast is a problem, the delay between holes in a row should be at least 2 ms per foot of spacing. This will prevent airblast from one charge from adding to that of subsequent charges as the blast proceeds down the row.

4. For the purpose of controlling ground vibrations, most regulatory authorities consider two charges to be separate events if they are separated by a delay of 9 ms or more.

Following these recommendations should yield good blasting results. However, when using surface delay systems such as detonating cord connectors and sequential timing blasting machines, the chances for cutoffs will be increased. To solve this problem, in-hole delays should be used in addition to the surface delays. For instance, when using surface detonating cord connectors, one might use a 100-ms delay in each hole. This causes ignition of the in-hole delays well in advance of rock movement, thus minimizing cutoffs. With a sequential timer, the same effect can be accomplished by avoiding the use of electric caps with delays shorter than 75 to 100 ms.

From the standpoint of simplicity in blast design it is best if all the explosive in a blasthole is fired as a single column charge. However, it is sometimes necessary, where firing large blastholes in populated areas, to use two or more delayed decks within a blasthole to reduce ground vibrations. Blast rounds of this type can become quite complex, and should be designed under the guidance of a competent person.

All currently used delay detonators employ pyrotechnic delay elements. That is, they depend on a burning powder train for their delay. Although these delays are reasonably accurate, overlaps have been known to occur. Therefore, when it is essential that one charge fires before an adjacent charge, such as in a tight corner of a blast, it is a good idea to skip a delay period. Development of blasting caps with electronic delays is a good future possibility.

POWDER FACTOR

Powder factor, in the opinion of the authors, is not the best tool for designing blasts.

Blast designs should be based on the dimensions discussed earlier in this chapter. However, powder factor is a necessary calculation for cost accounting purposes. In blasting operations such as coal stripping or construction work where the excavated material has little or no inherent value, powder factor is usually expressed in terms of pounds of explosive per cubic yard of material broken. Powder factors for surface blasting can vary from 0.25 to 2.5 lb/cu yd, with 0.5 to 1.0 lb/cu yd being most typical.

Powder factor for a single blasthole is calculated by the following formula:

$$P.F. = \frac{L(0.3405d)(D^2)}{(B)(S)(H)/(27)}$$

where P.F. = powder factor, pounds of explosive per cubic yard of rock,
L = length of the explosive charge, feet,
d = density of the explosive, grams per cubic centimeter,
D = charge diameter, inches,
B = burden dimension, feet,
S = spacing dimension, feet,
and H = bench height, feet.

Many explosives companies publish tables that give loading densities in pounds per foot of blasthole for different combinations of d and D. Powder factor is a function of type of explosive, rock density, and geology. Table 6 gives typical powder factors for surface blasting.

Higher energy explosives, such as those containing large amounts of aluminum, can break more rock per pound than lower energy explosives. However, most of the commonly used explosive products have fairly similar energy values and thus have similar rock breaking capabilities. Soft, light rock requires less explosive per yard than hard, dense rock. Large-hole patterns require less explosive per yard of rock blasted because a larger proportion of stemming is used. Of course, larger blastholes frequently result in coarser fragmentation because of poorer powder distribution. Massive rock with few existing cracks or planes of weakness requires a higher powder factor than a formation that has numerous, closely spaced geologic flaws. Finally, the more free faces a blast has to break to, the lower will be the powder factor. For instance a corner cut, with two vertical free faces, will require less powder than a box cut with only one vertical free face; and a box cut will require less powder than a sinking cut, which has only the ground surface as a free face. In a sinking cut it is desirable, where possible, to open a second free face by using a V-cut somewhere near the center of the round.

Table 6.-Typical powder factors for surface blasting

Degree of difficulty in rock breakage	Powder factor, lb/cu yd
Low	0.25-0.40
Medium	.40- .75
High	.75-1.25
Very high	1.25-2.50

When blasting materials that have an inherent value per ton, such as limestone or metallic ores, powder factors are sometimes expressed as pounds of explosive per ton of rock or tons of rock per pound of explosive.

SECONDARY BLASTING

Some primary blasts, no matter how well designed, will leave boulders that are too large to be handled efficiently by the loading equipment or large enough to cause plugups in crushers or preparation plants. Secondary fragmentation techniques must be used to break these boulders.

In the case of boulders too large to be handled, the loader operator will set the boulders aside for treatment. Identifying material large enough to cause plugups is not always quite so apparent. The operator must be instructed to watch for material that is small enough for convenient loading but which is large enough to cause a bottleneck later in the processing cycle.

Secondary fragmentation can be accomplished in four ways:

1. A heavy ball suspended from a crane may be dropped repeatedly on the boulder until the boulder breaks. This is a relatively inefficient method, and breaking a large or tough (nonbrittle) rock may take a considerable period of time. This method is adequate where the number of boulders produced is not excessive.

2. A hole may be drilled into the boulder and a wedging device inserted to split the boulder. This is also a slow method but may be satisfactory where only a limited amount of secondary fragmentation is necessary. An advantage of this method is that it does not create the flyrock associated with explosive techniques or, to some degree with drop balls.

3. Loose explosive may be packed into a crack or depression in the boulder, covered with damp earthen material, and fired. This type of charge is called a mudcap, plaster, or adobe charge. This method is inefficient because of a lack of explosive confinement, and relatively large amounts of explosives are required. The result is considerable noise and flyrock, and often, an inadequately broken boulder. The system is hazardous because the primed charge, lying on the surface, is prone to accidental initiation by external impacts from falling rocks or equipment. External charges should be used to break boulders only where drilling a hole is impractical, and when used, extreme caution concerning noise, flyrock, and accidental initiation through impact must be exercised. If it is found necessary to shoot a multiple mudcap blast, long delays or cap and fuse are not recommended.

4. The most efficient method of secondary fragmentation is through the use of small (1- to 3-in) boreholes loaded with explosives. The borehole is normally collared at the most convenient location such as a crack or a depression in the rock, and is directed toward the center of mass of the rock. The hole is drilled two-thirds to three-fourths of the way through the rock. Because the powder charge is surrounded by free faces, less explosive is required to break a given amount of rock than in primary blasting. One-quarter pound per cubic yard will usually do the job. Careful location of the charge is more important than its precise size. When in doubt it is best to estimate on the low side and underload the boulder. With larger boulders it is best to drill several holes to distribute the explosive charge, rather than placing the entire charge in a single hole. All secondary blastholes should be stemmed. As a cautionary note, secondary blasts are usually more violent than primary blasts.

Any type of initiation system may be used to initiate a secondary blast. For connecting large numbers of boulders, where noise is not a problem, detonating cord is often used. Electric blasting is also frequently used.

Although secondary blasting employs relatively small charges, its potential hazards must not be underestimated. Flyrock is often more severe and more difficult to predict than with primary blasting. Secondary blasts require at least as much care in guarding as do primary blasts. Secondary blasting can truly be called an art, with experience being an important key to success.

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TECNOLOGIA SOBRE EL USO DE EXPLOSIVOS.

CHARGE CALCULATION METHODS FOR TUNNEL BLASTING

ING. RAUL CUELLAR BORJA.

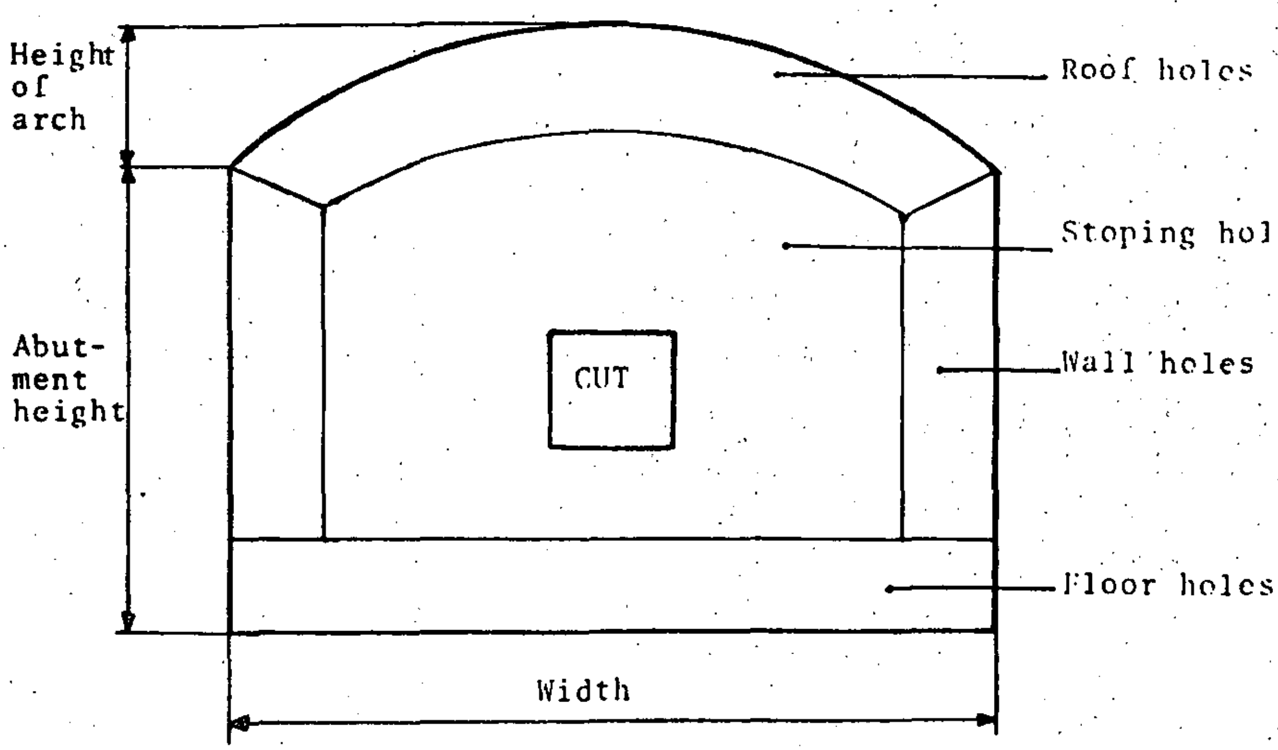
CHARGE CALCULATION METHODS FOR TUNNEL BLASTING

1 INTRODUCTION

Calculation of drilling patterns and charges is often carried out in a less systematic way in tunnelling than in bench blasting. As the demands, however, are about the same, i.e. to try to aim at an optimal result of the blasting operation, there are methods available also for this type of calculations. In tunnel blasting there is only one free surface, the face. This means that further free surfaces have to be obtained in some way, an opening is blasted by a cut. When stoping towards this opening, a higher specific charge is required than in bench blasting, depending on deviations in drilling, lack of hole inclination, need for swelling, lack of co-operation between adjacent holes, etc. The smaller the cross section of the tunnel is the higher specific charge is required.

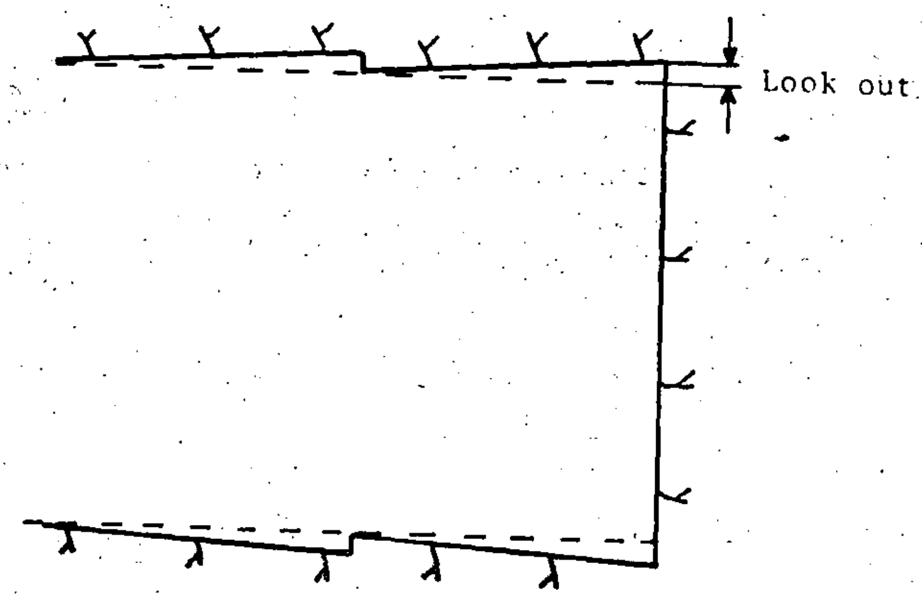
2. NOMENCLATURE

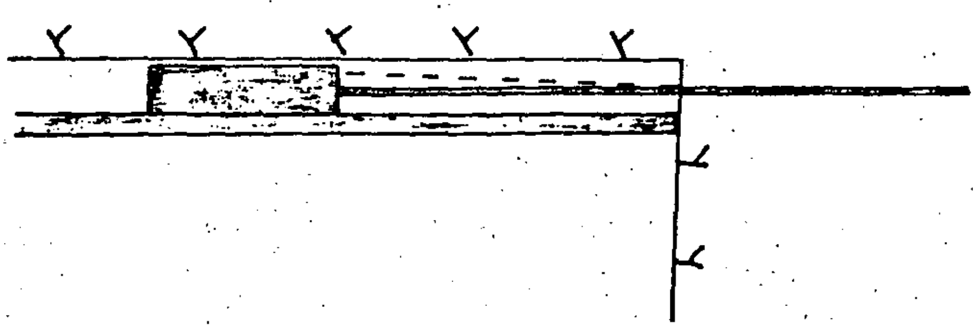
Tunnel face



Look out

To maintain the theoretical cross section from one round to another, the contour holes have to be angled outside the theoretical cross section. This is to make sure that, when drilling the next round, there is required space for the rock drill. The look out is depending on the equipment used, but normally amounts to not less than 0.1-0.2 metres.





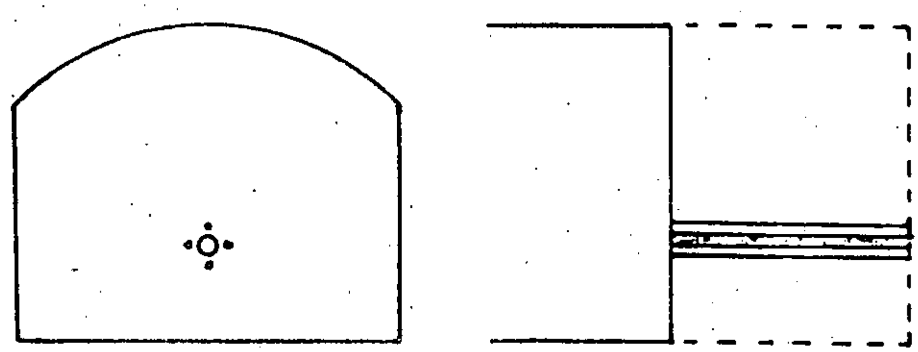
3 TYPES OF CUTS

To obtain an opening in the tunnel face, there are various types of cuts available. Some examples are:

- Parallel hole cuts
- V-cut
- Fan cut
- Norwegian cut

The types of cuts most used in Scandinavia today are a type of parallel hole cut, with an empty large hole, and the V-cut. Which type to select depends on cross section, hole depth, drilling equipment, etc. The skill of the drillers may also be a factor to take into consideration. Drilling and charging of a cut have to be carried out extremely carefully, as the advance per round to a great extent is depending on the success in the blasting of the cut.

Parallel hole cut with an empty large hole



The parallel hole cut is mainly used in smaller cross sections, but can also be used in larger ones. The cuts consist of one or several empty large holes (normally 57-127 mm) surrounded by weakly charged parallel holes with a small burden. The principle is that the empty large hole is acting as an opening to these cut holes. When the cut holes have detonated, a larger opening is obtained, why next group of holes can be drilled with a larger burden. In this way the blasting proceeds until the size of the opening permits a free breakage of the stoping holes. The area of the opening normally then is 2-4 sq.metres, depending on bit diameter used.

Some advantages with parallel hole cuts of this type are:

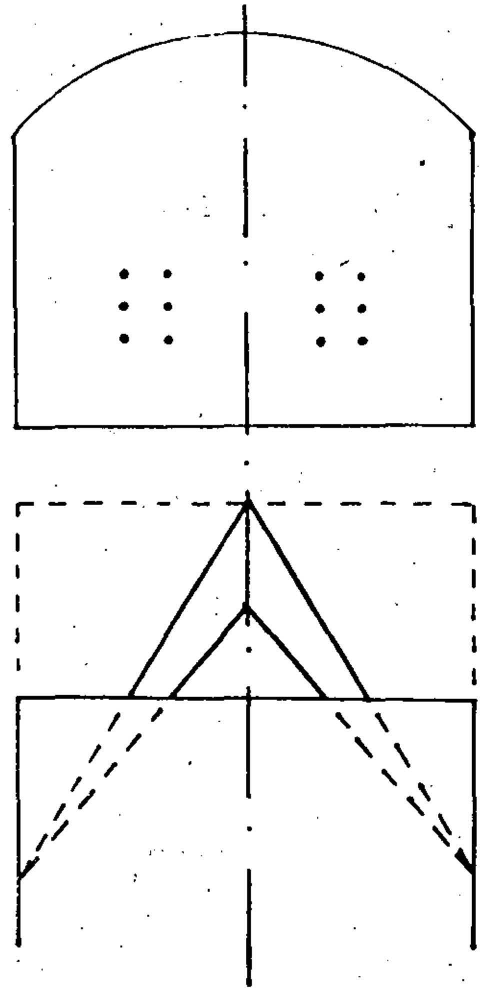
- Relatively simple design
- As all holes are parallel the drill steel length is utilized for all holes
- Can be used also in the smallest cross sections.

Among the disadvantages we have:

- Special drill steel equipment required when drilling the empty large hole
- One rock drill is occupied during a relatively long time when drilling the empty large hole
- Requires great precision when drilling and charging the cut holes.

Other types of parallel hole cuts are the Coromant cut, the Fagersta cut, burn cuts, etc.

V-cut



The V-cut is made up by angled holes and to attain a good blasting result the angle of the inner plough must not be too acute. Due to space requirements of the drilling equipment this implies that a minimum width of the tunnel is necessary when drilling a V-cut.

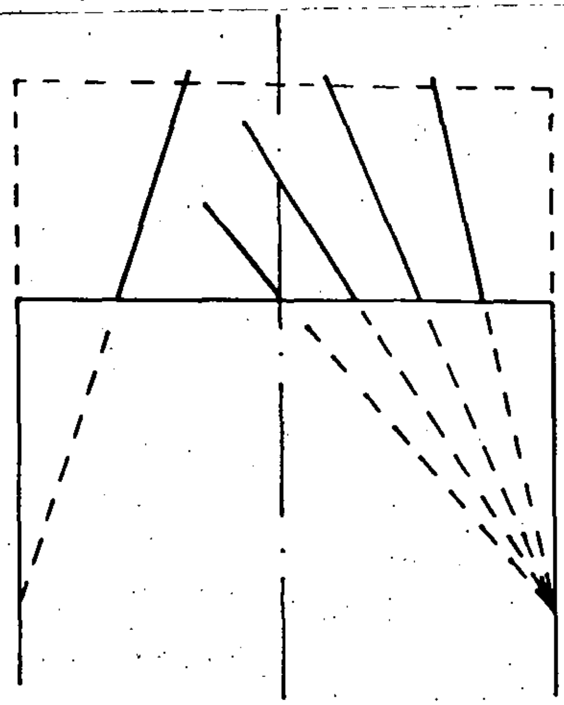
Some advantages with the V-cut are:

- The same type of drill steel equipment can be used for all the holes
- Symmetric, which makes the drilling easier, particulary when using mechanized equipment.

Disadvantages:

- Due to angled holes, the drill steel length is not completely utilized for all the holes
- A minimum width of the tunnel is required.

Fan cut



The fan cut is, like the V-cut, made up by angled holes, implying that a certain width of the tunnel is necessary to carry out the drilling.

As the cut is not symmetric and with an unequal distribution of the holes, it is not particulary well suited for mechanized drilling. The fan cut is today not used to the same extent as earlier.

4 CALCULATION METHODS

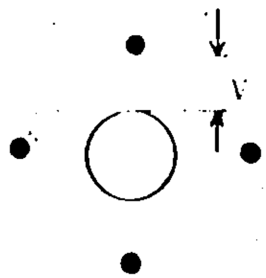
When calculating drilling pattern and charges of a tunnel cross section, it is suitable to calculate in two steps, the cut and the remaining part of the round.

The calculations provide that DYNAMEX®B explosive is used (weight strength s = 1.0 or 78 % of Blasting Gelatine).

4.1 Calculation of the cut

Parallel hole cut

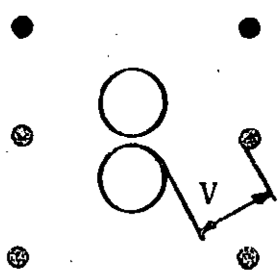
For the holes closest to the empty large hole following relationship can normally be applied when calculating the burden.



$$V = 0.7 \cdot d$$

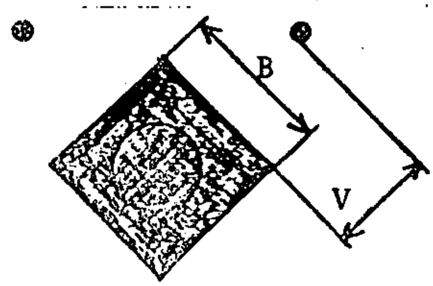
where d is the diameter of the empty large hole.

In case of two empty large holes, the formula is changed to



$$V = 0.7 \cdot 2\ d$$

After breakage of these holes a square opening is obtained and the breakage continues by stoping towards this opening. If the width of the square is designated B, the burden will be



$$V = 0.7\ B$$

This procedure is repeated until the burden obtained by the formula equals the burden obtained by the diagramme (item 4.2) for stoping holes.

When charging the holes in the cut it is important to select a charge concentration which is well balanced. This is particularly important for the holes in the first square, i.e. the holes closest to the large hole. In case of a too weak charge there is a risk of a complete failure of the round since the cut will remain unblasted. If on the other hand the charge is too strong, a similar result is obtained due to burning of the cut holes. As a guide line for charging of the holes in a parallel hole cut following table may be used, where the concentration of the bottom charge is l_b (kg/m).

Square no	h_b (x H)	l_p (x l_b)	h_o (x V)
1	0.05	0.5	-
2	0.05	0.5	0.5
3	0.20	0.5	0.5
4 -	0.33	0.5	0.5

where

h_b = Height of bottom charge

H = Hole depth

l_p = Concentration of column charge

V = Burden

h_o = Uncharged part

for all the holes the height of the bottom charge is equal to

$$h_b = \frac{1}{3} H$$

where H is the hole depth.

The column charge concentrations is equal to

$$l_p = 40\text{-}50\ \%\ \text{of}\ l_b$$

Uncharged part for the cut holes

$$h_o = 0.3\ V_1$$

and for subsequent holes

$$h_o = 0.5\ V_2$$

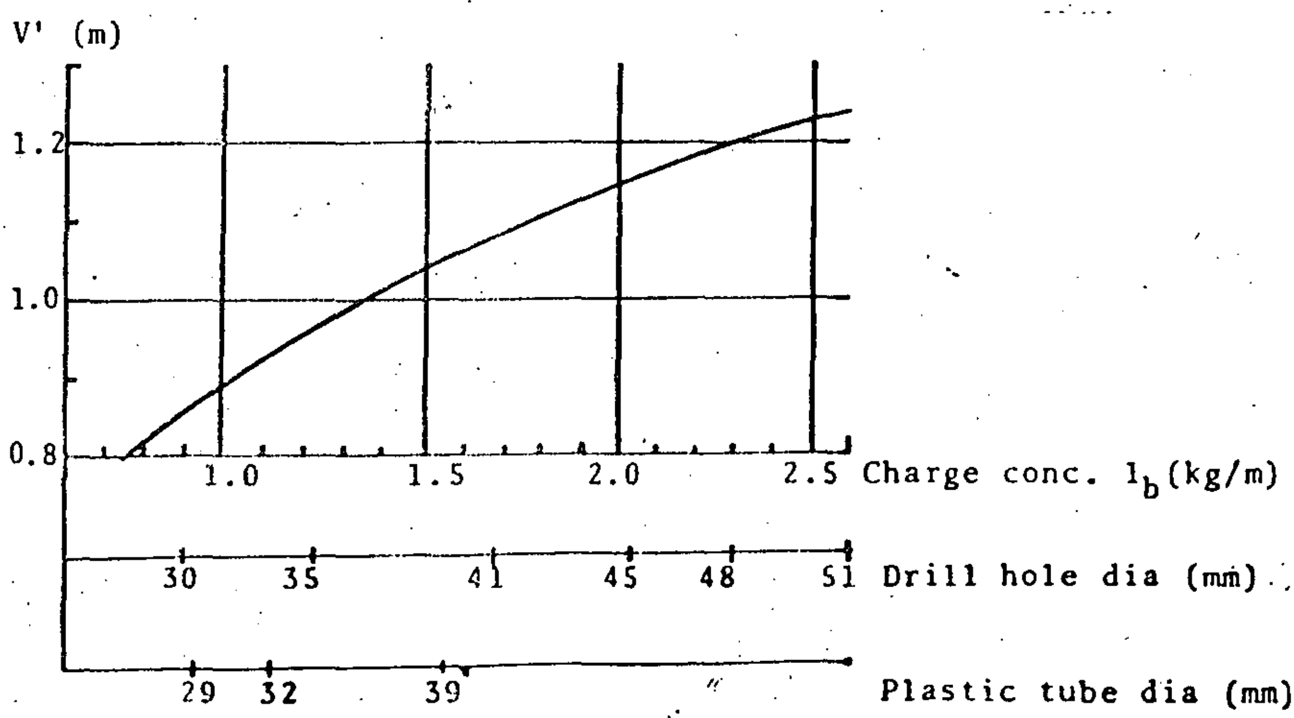
4.2 Calculation of remaining holes in the round

The remaining holes of the round are divided into

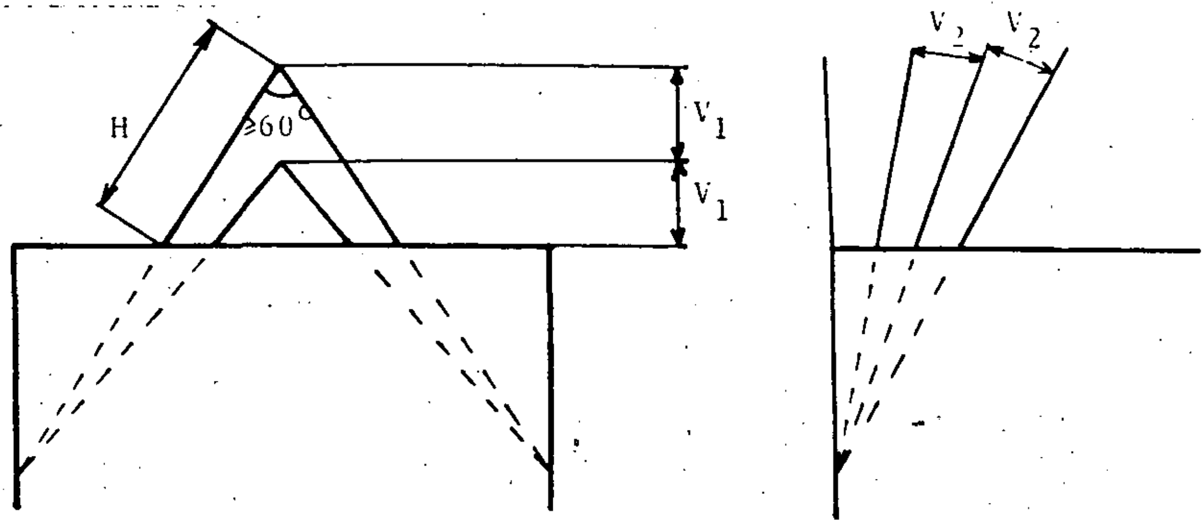
floor holes
wall holes
roof holes
stoping holes upwards and horizontally
stoping holes downwards.

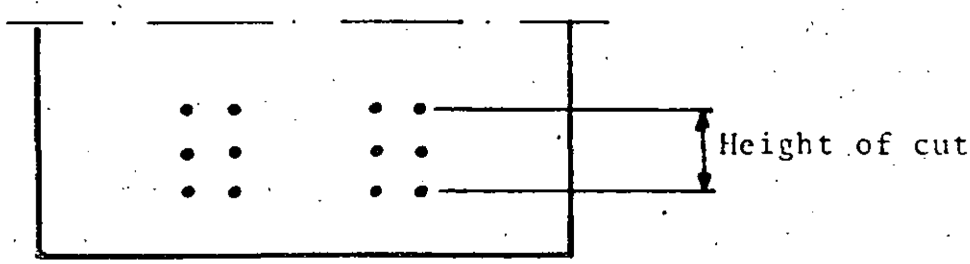
To calculate burdens, spacings and charges for these different types of holes following diagramme and table may be used.

As indata in the diagramme the concentration of the bottom charge, l_b, is used. If the bottom charge consists of DYNAMEX® B packed to 1.25 kg/dm^3, corresponding hole diameter can be used. In case of DYNAMEX® B in plastic tubes corresponding tube diameter can be used as indata. From the diagramme V' is obtained which then is used in the table.

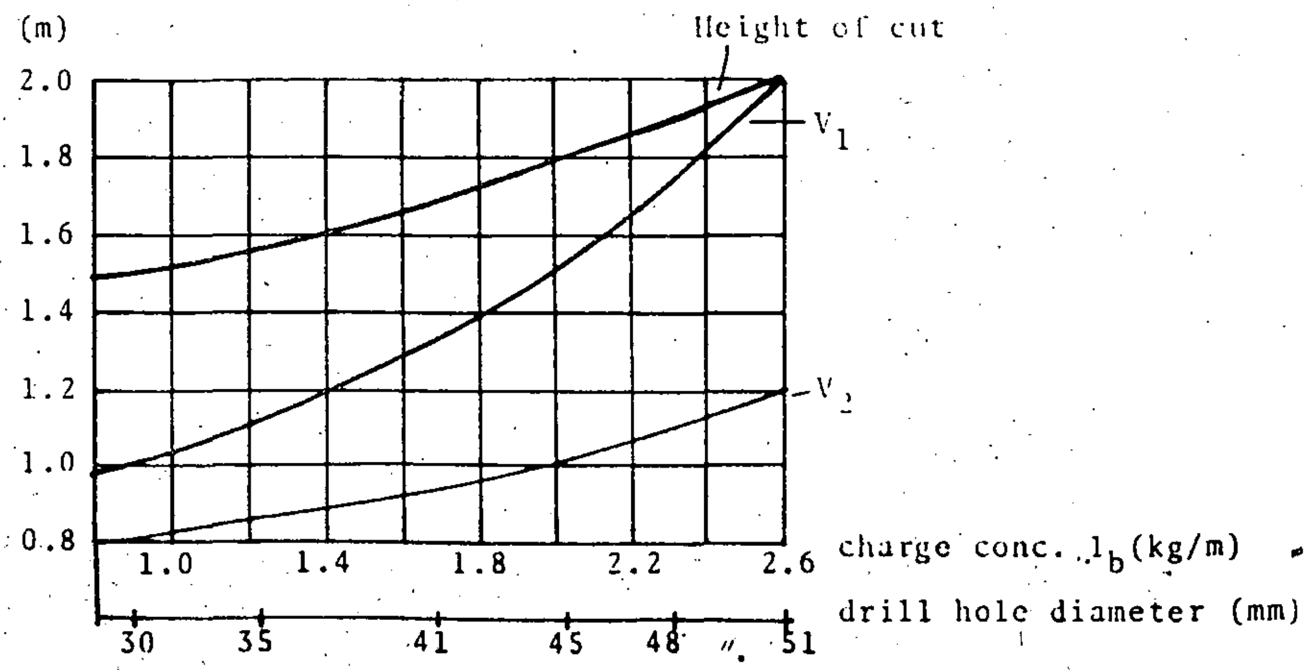


V-cut





When calculating a V-cut, the height of the cut and the burdens V_1, and V_2 are found in the diagramme below. The height of the cut provides three ploughs vertically according to figure above.



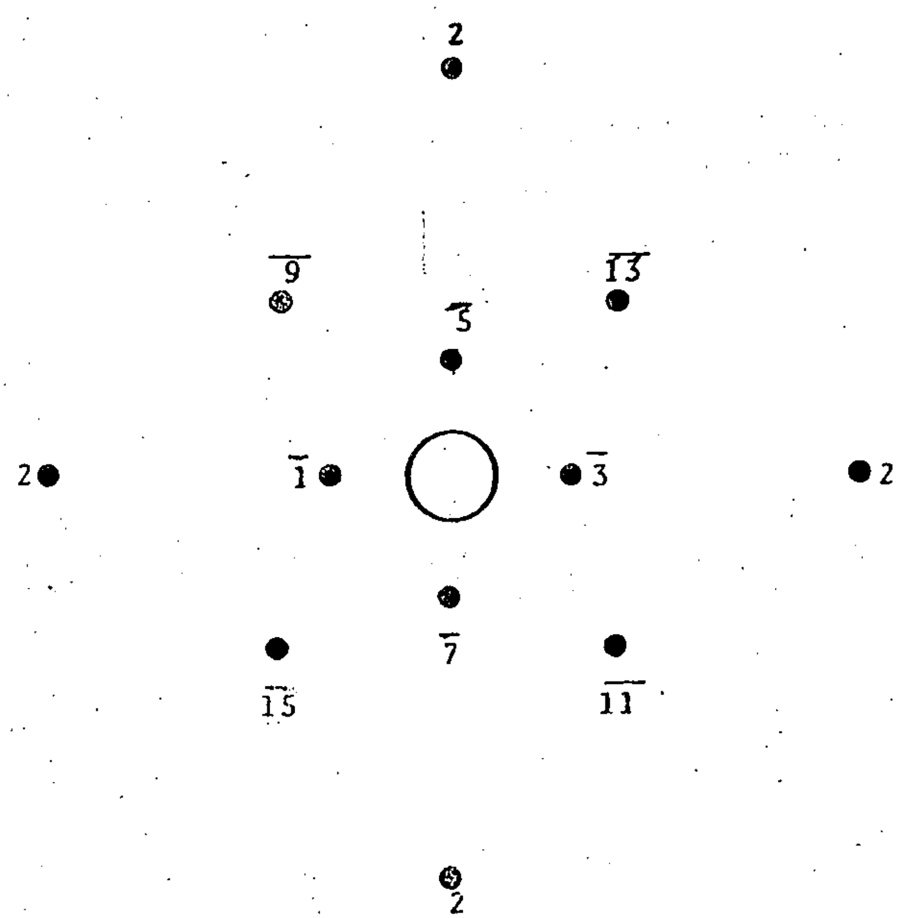
Knowing V', hole depth H and the concentration of the bottom charge l_b, the table below provides burden V, spacing E, height of bottom charge h_b, concentration of column charge l_p and uncharged part h_o.

HOLE TYPE	V (m) xV'	E (m) xV	h_b (m) xH	l_p (kg/m) xl_b	h_o (m) xV
Floor	1	1.1	1/3	1.0	0.2
Wall	0.9	1.2	1/6	0.4	0.5
Roof	0.9	1.2	1/6	0.3	0.5
Stoping ↑ →	1	1.1	1/3	0.5	0.5
Stoping ↓	1	1.2	1/3	0.5	0.5

The design of the drilling pattern can now be carried out and the cut is fitted into the cross section in the most suitable way.

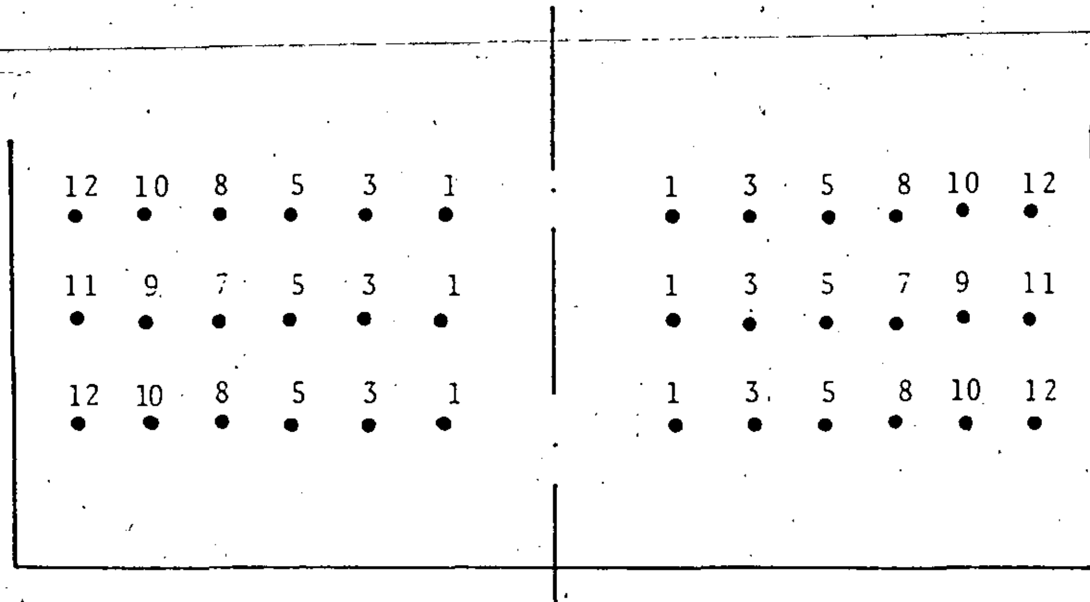
5 EXAMPLES OF IGNITION PATTERNS

Parallel hole cut



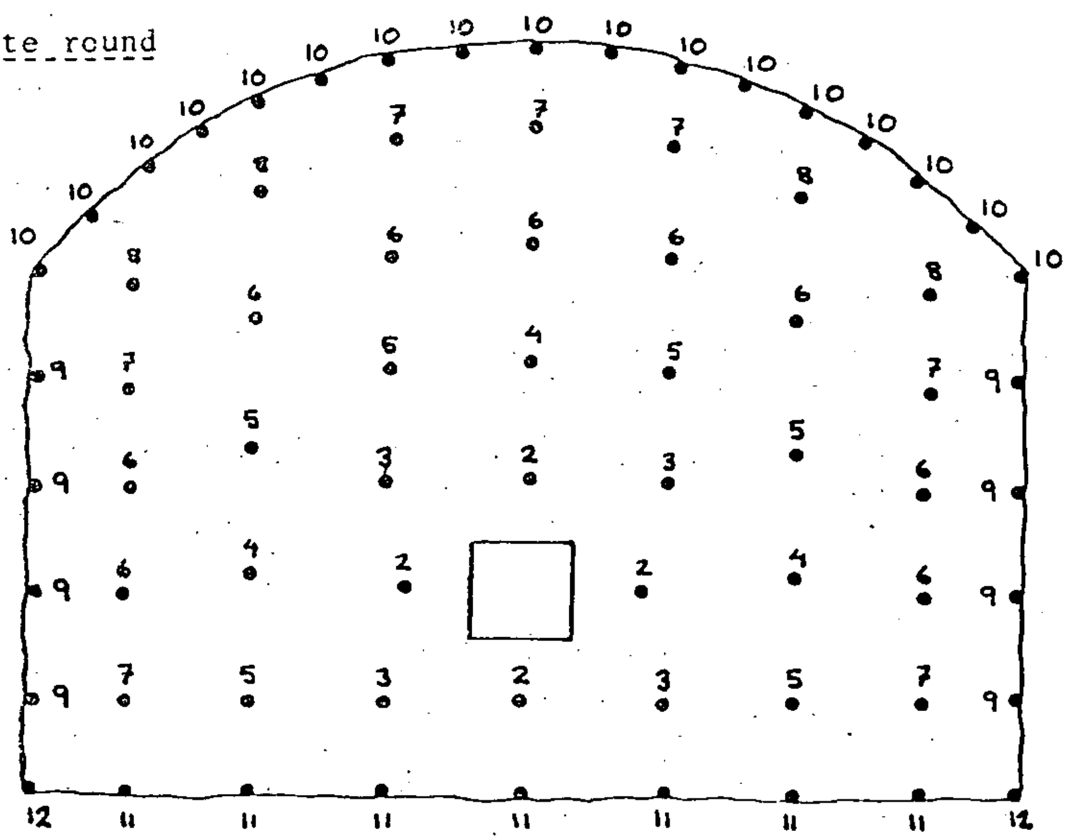
$\bar{1}$-$\bar{15}$ MS
2- HS

V-cut



MS-detonators

Complete round



HS-detonators

A method used more and more when blasting tunnels is the smooth blasting technique. This method is referring to a technique where the charges along the contour are more distributed, i.e. more contour holes are drilled, each of them having a weaker concentration. This means that the damage of the remaining contour is less than when using normal charges. The advantages of this is obvious as it implies

- less overbreak
- smoother surfaces
- higher rock strength
- less reinforcement and scaling, etc.

The success of the method is to a great extent depending on the accuracy in drilling, which means that the look out has to be kept as small as possible and equal for all holes along the contour. The charging has to be adapted to the local conditions, but to start with following values may be used:

DRILL HOLE DIAMETER (mm)	CHARGE CONCENTRATION (kg DxB/m)	CHARGE UNIT	BURDEN V (m)	HOLE SPACING E (m)
25-32	0.07	11 mm GURIT®	0.30-0.45	0.25-0.35
25-48	0.16	17 mm GURIT®	0.70-0.90	0.50-0.[illegible]
51-64	0.37	22 mm GURIT®	1.00-1.10	0.80-[illegible]

As bottom charge in the roof normally one cartridge is sufficient while two cartridges are recommended in the wall holes.

To obtain an optimum result of the smooth blasting it is also important to reduce the cracks from the holes closest to the contour. This means that these holes have to be charged weaker than other stoping holes.

March 1981

The Nitro Nobel Group

BLASTING TECHNIQUES DEPT

USE OF PARALLEL HOLE CUTS IN DRIFTING AND TUNNELLING

 May 1980

USE OF PARALLEL HOLE CUTS IN DRIFTING AND TUNNELLING

INTRODUCTION

During the end of the 50's and the beginning of the 60's the attention was drawn to the cuts in tunnel blasting. Many valuable and empirical results were published and presented, a.o. in "Bergsprängningskommittén". During the last 20 years the drilling equipment used in tunnelling has changed fundamentally. Many of the people, who worked with these matters in the beginning of the 60's, have now retired and the knowledge has not been followed up and utilized to a desired extent.

Because of the progress of the drilling equipment the hole depths in tunnelling have increased considerable. In 1968 the main part of the number of rounds was shorter than 3.0 m, whereas the round depth 1978 is quite a bit over 4.0 m.

The hole depth increases partly because of the fact that longer feeds have been designed and partly because bigger drilling rigs make the drilling time shorter compared to the set up times. This also affects the costs in a favourable way at the same percentage of advance per round (fig. 1).

As the hole depth has increased the types of cuts used have also changed. Earlier V-cut, fan cut and Norwegian cut were used but today almost all tunnelling is carried out by using parallel hole cuts.

When blasting a tunnel round the result is very much depending on how you manage with the cut. If the cut does not work satisfactorily, maximum advance per round can not be achieved. Therefore, it is worth paying much attention to how the cuts are made, both when it comes to calculation and execution.

By studying empirical resultats from practical cases it has been possible to describe the relation between advance per round and hole depth at a certain large hole diameter (in this example 102 mm, figure 3). When the hole depth increases, for example from 3.0 to 4.0 m, the advance per round is reduced from 95 to 87%, i.e. 8%.

Figure 4 shows the change in costs for drilling/blasting at a changed percentage advance per round. If previous example is utilized, that is a reduction in advance per round from 95 to 87%, the costs increase by 8.5%. Normally this is not acceptable, which means that something has to be done to improve the

the advance per round.

CALCULATIONS FOR OPTIMAL ADVANCE PER ROUND

Today the most common type of cut, in Sweden, is a parallel hole cut consisting of one or more uncharged large holes, figure 5, surrounded by small holes with small burdens placed in a number of squares. The number of quadrants, in the cut, is limited by the fact that the burden of the last quadrant may not exceed the burden for stoping holes for a certain charge concentration.

When a cut is designed it is, besides the choice of explosives and detonators, principally the following parameters that must be settled in such a way that an optimal result is obtained.

- Large hole diameter
- Burden
- Charge concentration

Besides it is important that the execution is made with most possible precision, that means a.o. that the precision in drilling is of great importance. This is especially the case for the inner cut holes, for which a faulty drilling beyond the calculated, of just a couple of cm/m may result in a crossing of the small hole and the large hole, or a doubled burden. The consequence will normally be a loss in advance per round to a greater or smaller extent. For natural reasons it can be assumed that the carefulness is greater for drilling of holes with small burdens than it is with larger burdens. This means that the faulty drilling is a function of hole depth as well as of burden and can be described through a relation of the type

$$F = V\ (0.1 + 0.3H)$$

F is faulty drilling
V is greatest burden
H is the hole depth

Large holes

As mentioned before there is a close connection between hole depth; large hole diameter and the advance per round expected. A common reason for too low advance per round is probably that a too small size of the large hole in relation to the hole depth in question has been chosen. It appears from the diagram (figure 6) that if there is for instance a hole depth of 3.9 m and a large hole diameter of 102 mm an advance per round of barely 90% can be expected. If this common combination was changed to a large hole of 127 mm, or two large holes of 89 mm, an increase of the advance per round up to about 95% could theoretically be expected.

The first quadrant

The next important step is the connection between large hole diameter and the burden together with the charge concentration for the cut holes in the first quadrant close to the large hole. A bigger large hole makes it possible to have larger burdens, which demand larger charge concentrations, diagram (figure 7). The burden must here be chosen in a way which does not make the angle of breakage too small, but it must not become so large that risk occurs that drilling will be done into the large hole. As basic value is normally used a largest burden (c/c-distance) corresponding to approximately 1.5 Ø where Ø stands for the large hole diameter.

A general opinion has been that the inner cut holes shall be lightly charged and also that the risk of failure is bigger at a too heavy charging than at too light charging. However, the opinion today is that the cut holes can be charged stronger than blasters earlier believed. The stronger charging is often an advantage. (Please notice the headline "Selection of explosives and firing devices").

Other quadrants

In general the same rules are applicable to the cut holes of the following quadrants as those for the first quadrant, including, however, the difference that the breakage is done towards a plane surface, diagram (figure 8). Also here the angle of breakage must not be too acute, as small angles only to a certain extent can be compensated with increased charge concentrations. In the diagram we have marked out the line V = 1.5 B, which is the largest burden for a decided width of opening.

If, on the other hand, the width of the opening is large in relation to the burden the same condition is obtained as for stoping holes. This limit value in the diagram is V = 0.5 B, i e an angle of breakage equivalent to 90°. This involves that a width of the opening, which has more than the double size of the burden, does not lead to a decrease of the charge concentration. Normally, a drilling pattern for cut holes is done for width of openings approximately of the same size as the burden, i e V = B.

SELECTION OF EXPLOSIVES AND FIRING DEVICES

Explosives

To obtain a rational charging of a tunnel round today, pipe charges are mostly used. Except for a fast charging a controlled charge concentration is obtained, which means that the risk for too much or too less charging becomes smaller in comparison with the use of cartridges. Exact knowledge about the charge concentration is obtained, and the burdens can be adapted hereto. This is very important regarding the cut holes. One of the complications that might occur in a parallel hole cut is flash-over between drill holes. As the distance is very small between the holes in a parallel hole cut, there is risk that the detonation from one drill hole will be transmitted by fractures and joints to the nearest hole, which will detonate at a time not suitable. This may result in a loss of advance per round. The risk of flash-over, except for the condition of the rock and the hole distance, is dependent on what kind of explosive that is used.

At the time when the basic investigations regarding parallel hole cuts were done, much more sensitive explosives existed compared to our present explosives. This is one reason why the risk of over charging and flash-over was extremely pointed out at that time. The explosives of today are, however, much more insensitive and involve consequently less risks for flash-over. When comparison is made between DYNAMEX B and NABIT , which today are the most common explosives, NABIT is considered the most advantageous regarding the flash-over point of view. This is one reason why NABIT is used when charging cuts. In many cases even DYNAMEX would be successfully used, which would be an advantage considering the limit of assortment. One example is a big tunnelling project in Sweden, where in holes of 43 mm DYNAMEX B of 32 mm was used in the cut holes with a very good result.

EXPLOSIVES	WEIGHT STRENGTH rel DxB %	DENSITY kg/m^3	BULK STRENGTH rel DxB %
DYNAMEX B	100	1 450	100
NABIT A	93	1 000	64
PRILLIT A (ANFO)	91	850	53

Table 1 Weight strength, density and bulk strength for explosives suitable for the cut.

Firing Devices

When choosing detonators the first thing is to make use of the interval numbers in such a way that a functional firing pattern is obtained. On one hand one want to have a sufficient delay between the cut holes and on the other hand the number of intervals must be enough for the round. At big tunnel areas this may mean that the delay between the cut holes must be reduced to a minimum. A too narrow division of intervals in the cut results, however, undoubtedly in a cut that does not function satisfactorily and this means an impaired advance per round.

To find the shortest delay time that can be used without a negative effect on the function of the cut, we have tested a cut consisting of two large holes Ø 76 mm, according to fig. 9. We found, as expected, that the delay times between the two first holes were the most critical. Under the actual circumstances it was proved that the delay time must be at least 50 ms. As for the other holes this time could be reduced to 25 ms according to fig. 9. It must be pointed out that these delay times is an absolute minimum and ought to be somewhat higher to be sure on a good result.

RECOMMENDATION

At drifting and tunnelling one try to obtain the best total economical result. This is achieved as a rule at an advance per round of 95% or more of drilled depth.

Drilling and charging calculations shall be made so, that the conditions for a big advance per round increase, fig. 10. This is achieved by the following steps.

- Choose a large hole diameter in relation to the hole depth, which gives at least 95% advance per round and gladly more. If the drilling equipment is not suitable for drilling large holes, make more holes with less diameter according to the formula

  $D = d \cdot \sqrt{n}$

  D = fictitious large hole diameter

  d = used large hole diameter

  n = number of large holes

- Calculation of the conditions in relation to large hole diameter, alternatively opening width, according to the following

  First quadrant V = 1,5 Ø

  V is the largest burden = c/c-distance large hole/ small hole (m)

  Ø is the diameter (m) of the large hole

  Other quadrants V = B

  V is the largest burden (m)

  B is opening width (m)

- Always calculate with faulty drilling. A useful formula is

  F = V (0,1 + 0,03 H)

  F = faulty drilling

  V = the largest burden

  H = hole depth (m)

- Always drill the hole to a contemplated plan. A too deep hole damages the rock and if the hole is too shallow rock parties remain undestroyed. The result is poorer conditions for next round and a reduced advance per round.

- Always calculate the charges after the largest burden and always calculate with a certain safety margin.

- Choose interval number in a way that enough time is obtained for the rock to move. The two first holes are the most important.

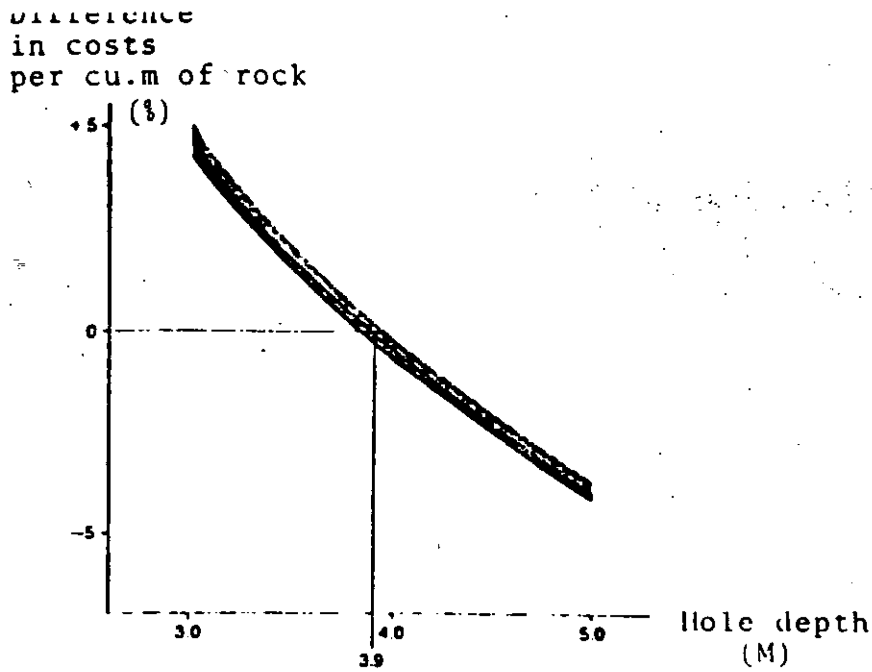


Fig 1. Change in costs per cu.m of rock at different hole depths and at the same percentage of advance per round.

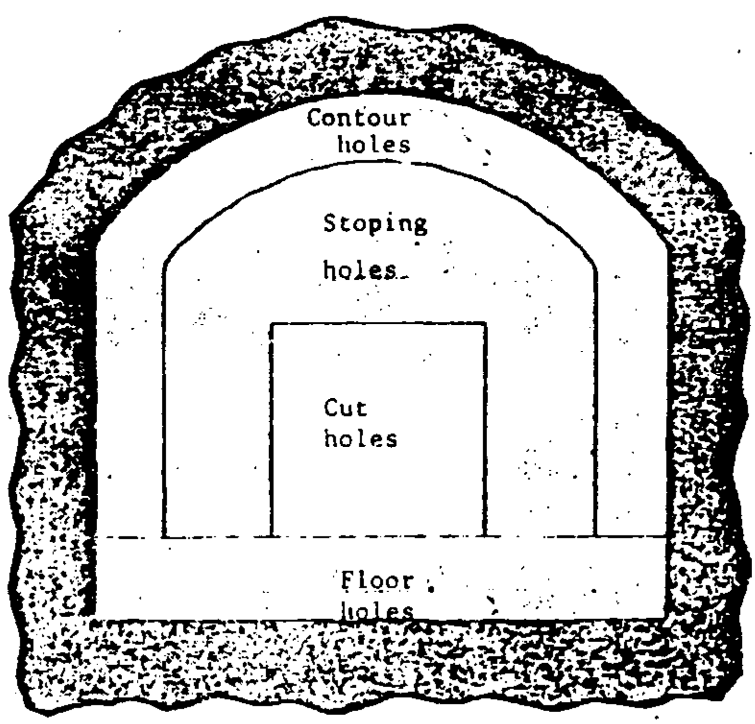


Fig 2. When blasting a tunnel round the result is very much depending on how you manage with the cut.

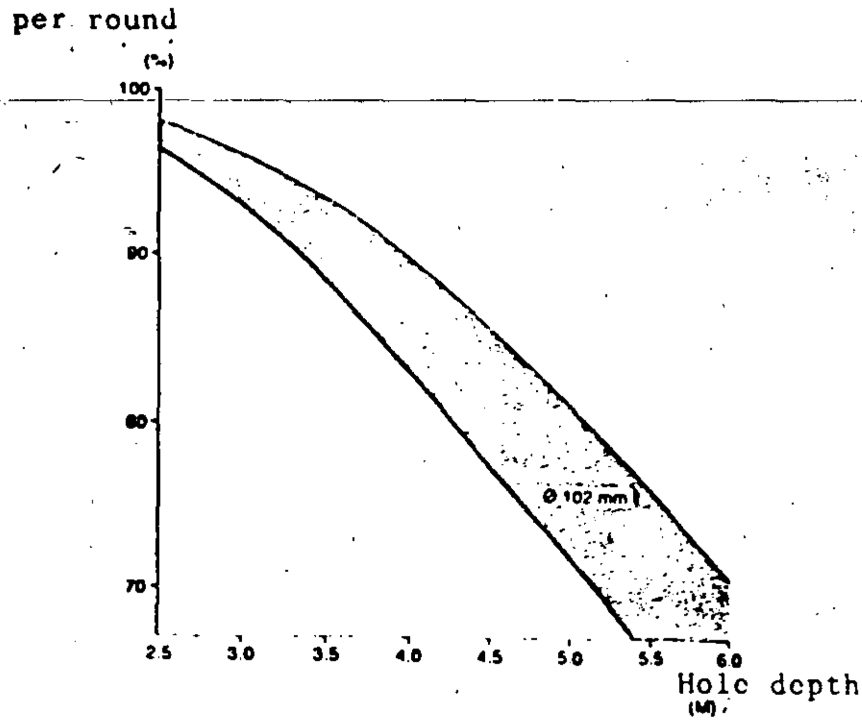


Fig 3. The relation between percentage of advance per round and hole depth at 102 mm large hole diameter.

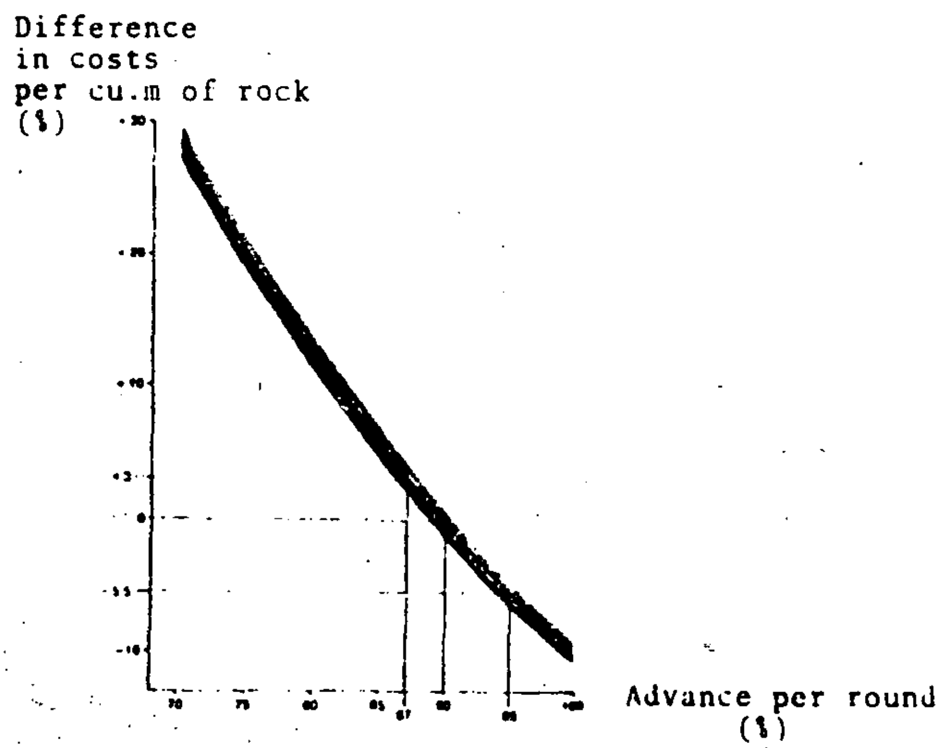


Fig 4. The change in costs for drilling/blasting per cu.m of rock at a changed percentage of advance per round but with the same hole depth.

Four section cut

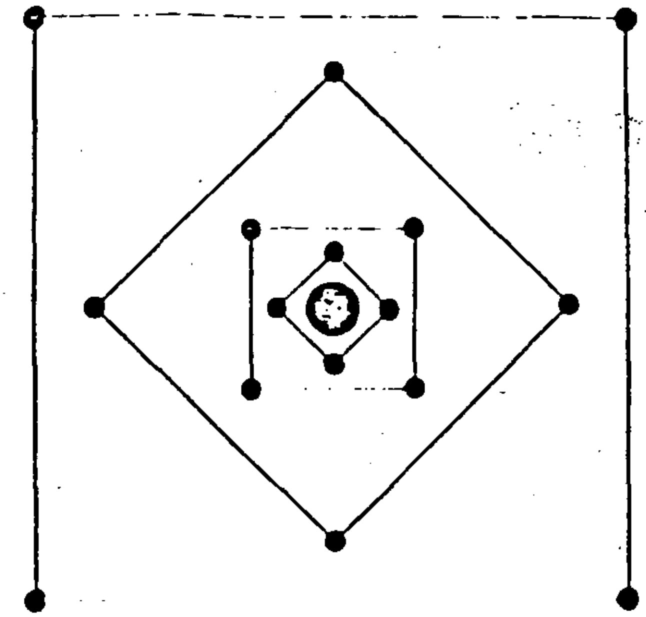


Fig 5. Parallel hole cut with a large hole surrounded by small hole with small burdens placed in a number of squares.

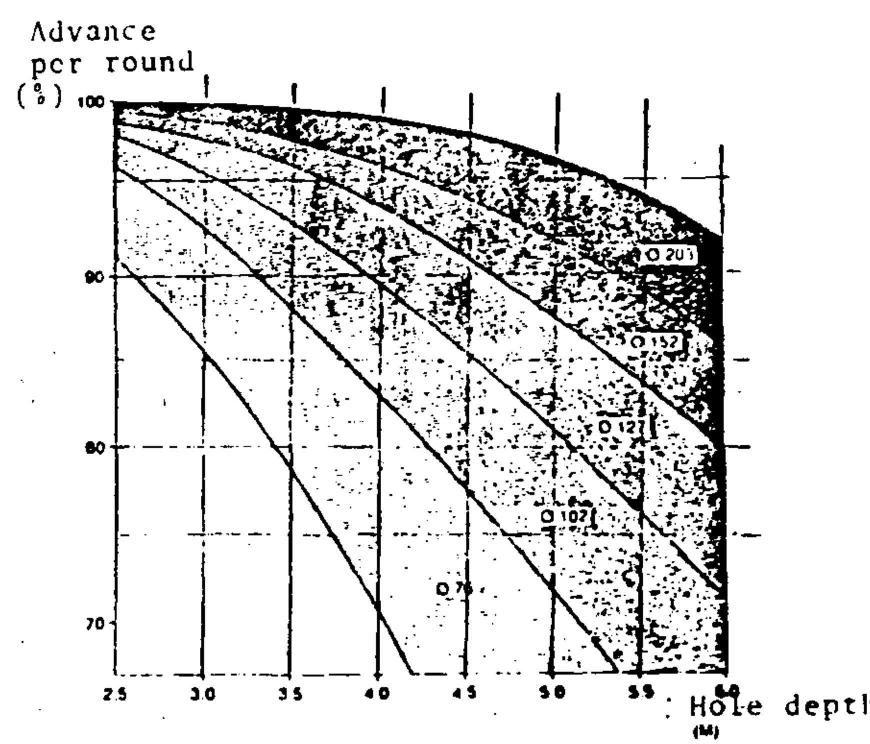


Fig 6. The relation between percentage of advance per round and the hole depth at different **large hole diameters.**

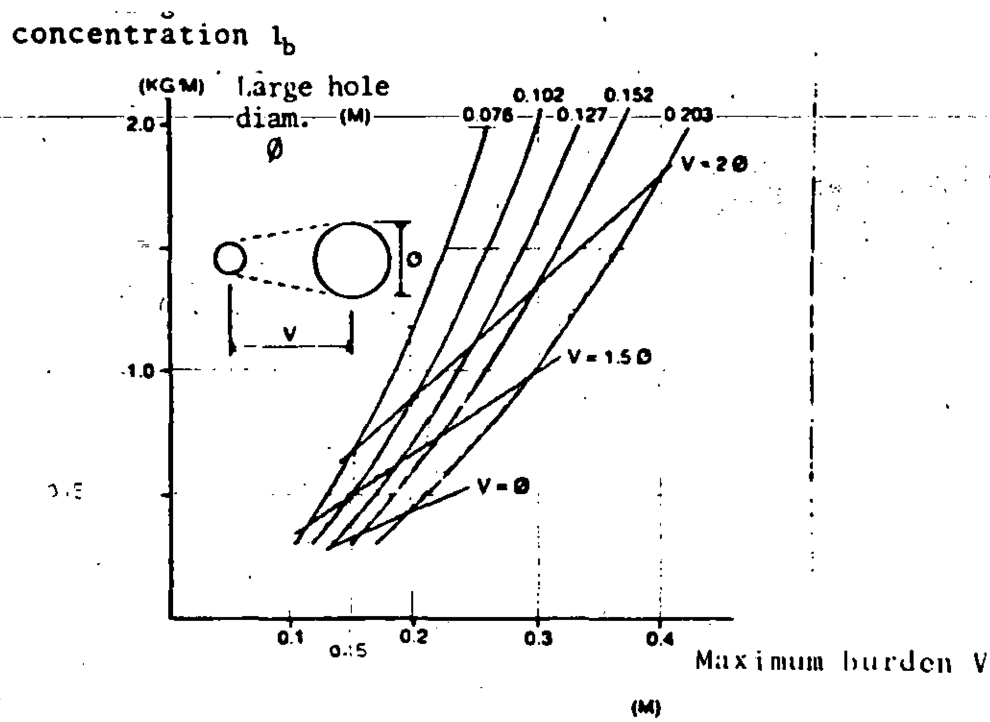


Fig 7. The relation between charge concentration (l_b) and largest burden (V) at different hole diameters (Ø).
As basic value is normally V=1.5 used.

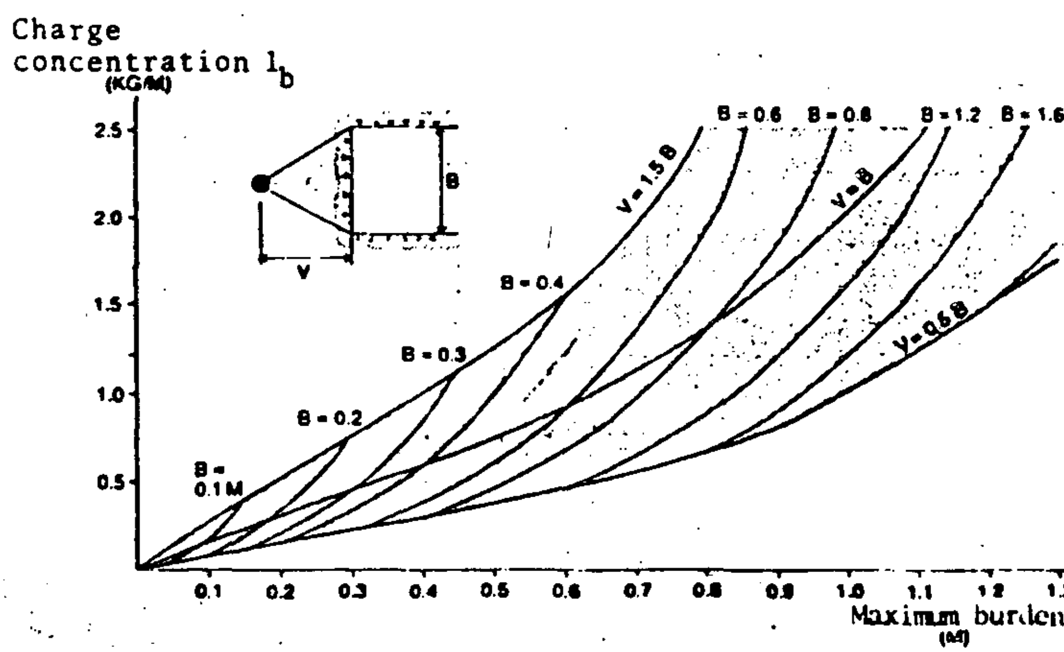
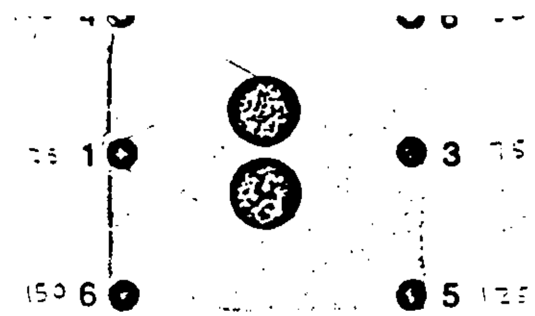


Fig 8. The relation between charge concentration (l_b) and largest burden (V) at different width of opening.
As basic value is normally V = B used.



MILLISECOND DETONATORS

DELAY NUMBERS	DELAY TIMES
1	25 MS
3	75 MS
4	100 MS
5	125 MS
6	150 MS

Fig 9. Shortest delay time for a large hole cut with 2 large holes of 76 mm.

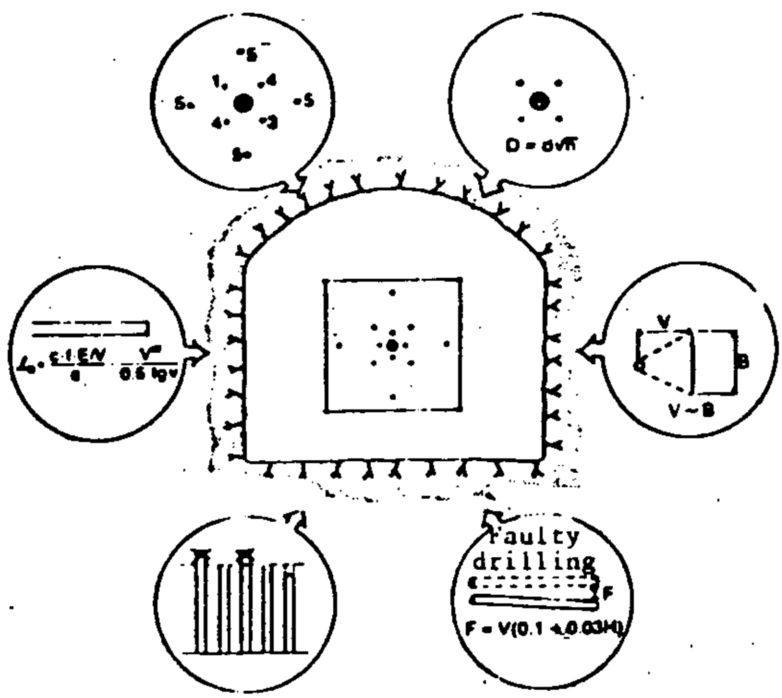


Fig 10. **Recommendations to achieve a maximum advance per round.**

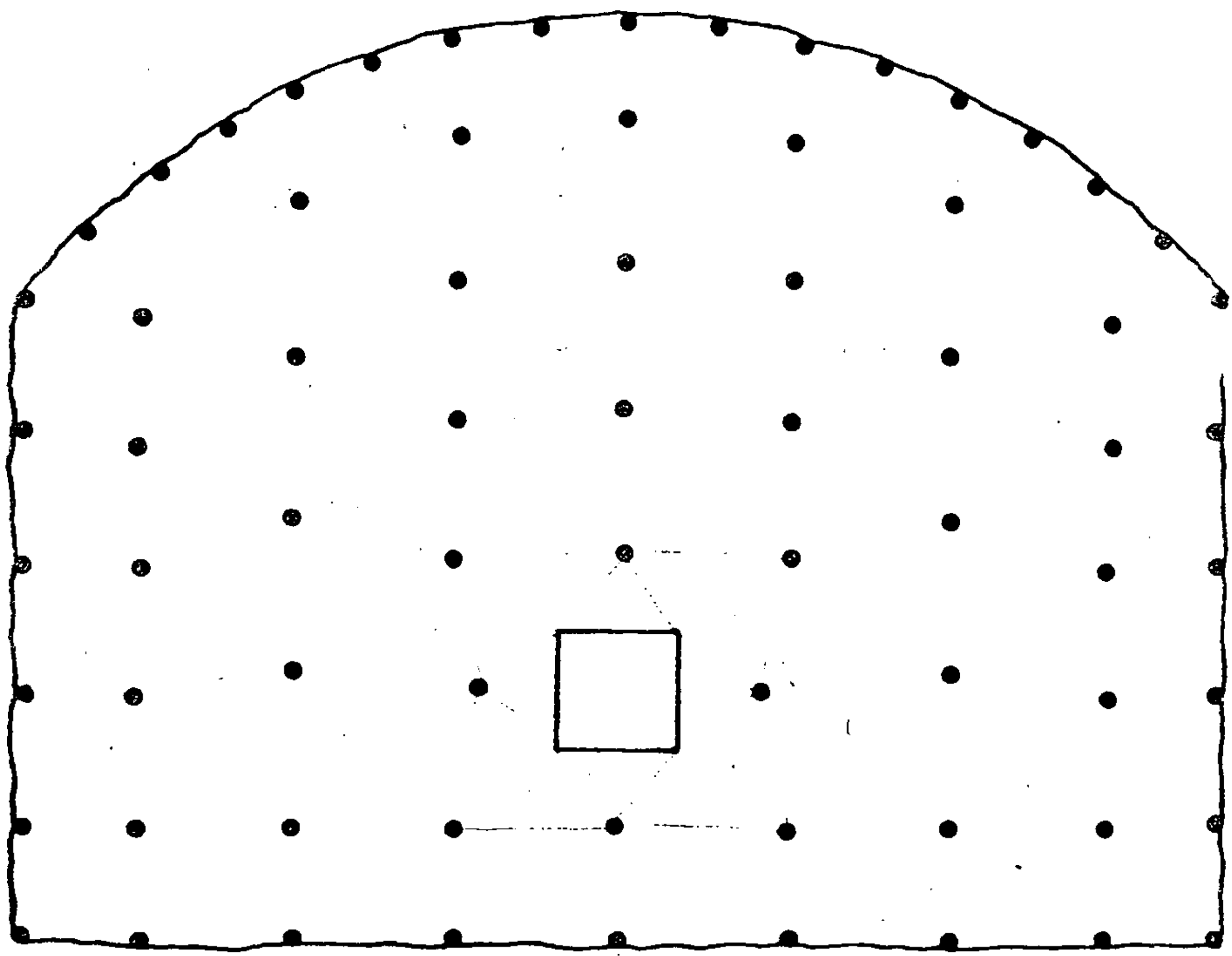
TUNNEL BLASTING

Calculation example

<u>Conditions:</u>

Cross section	53	m^2
Width	8.55	m
Abutment height	4.65	m
Height of arch	2.00	m
Hole diameter	45/102	mm
Hole depth	3.9	m

Smooth blasting to be carried out of the roof, using 17 mm GURIT.



FACULTAD DE INGENIERIA U.N.A.M.
DIVISION DE EDUCACION CONTINUA
C U R S O S A B I E R T O S
VI CURSO INTERNACIONAL DE INGENIERIA GEOLOGICA APLICADA A OBRAS SUPERFICIALES Y SUBTERRANEAS
MODULO IV: TECNOLOGIA SOBRE EL USO DE EXPLOSIVOS

TUNEL PILOTO

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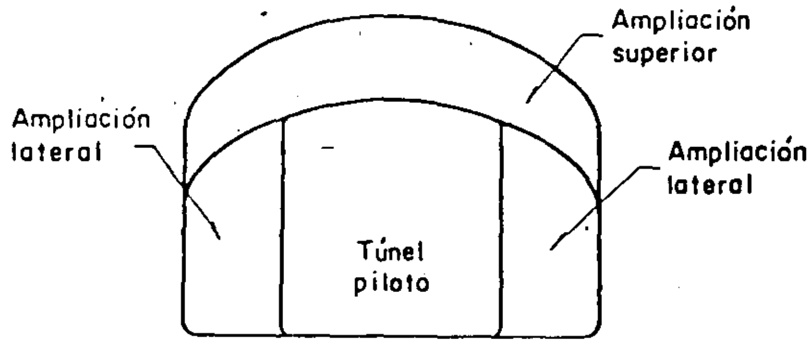


FIG I.27 Túnel piloto y ampliaciones laterales y superior

7.2.1.3 Cuñas iniciales

En los socavones y túneles el frente de avance es la única cara de liberación de la voladura. Es por este confinamiento de los barrenos cargados, que la carga específica es mayor en los túneles de pequeña sección y túneles pilotos que en los banqueos o en las excavaciones a cielo abierto. A fin de dar mayor eficiencia a la voladura se produce una abertura a todo lo largo del avance previsto, creando así un espacio vacío que permite la expansión y fragmentación de la roca removida por las sucesivas etapas de la voladura. Es obvio que este espacio inicial no es suficiente para acomodar la expansión y movimiento de toda la roca de la voladura completa. Por tanto, la mayor parte de la roca se proyecta hacia la zona previamente excavada. El espacio producido inicialmente se ha denominado "cuña".

Los principales tipos de cuña son dos: la cuña de barrenos paralelos y la cuña en "V". Cada tipo de cuña tiene una variedad de diseños para ajustarse a cada formación particular.

La cuña inicial es la parte más crítica en el diseño de voladuras en túneles.

Es muy difícil determinar un tipo de cuña inicial que resulte el adecuado para el terreno por excavar sin haber efectuado algunas voladuras previas.

a) Cuña de barrenos paralelos o cuña quemada

La cuña de barrenos paralelos consiste de uno o más barrenos vacíos y uno o más barrenos cargados, paralelos unos a otros, que son perforados en el centro del frente, y con la profundidad del tramo de avance fijado. Las perforaciones que rodean la cuña están dispuestas en tal forma que se disparan después de abierta la cuña. Es muy importante para lograr una fragmentación eficiente, que se mantenga el paralelismo de los barrenos de la cuña. Una barrenación inapropiada puede dar lugar a la propagación entre los barrenos cercanos, destruyendo así la secuencia de detonación prevista y provocando zonas de fragmentación deficiente por exceso de confinamiento.

La cuña quemada es empleada casi exclusivamente en túneles de sección transversal menor de 10 m^2 y permite voladuras más profundas. En túneles reducidos el espacio resulta pequeño para acomodar las máquinas para perforar con cualquier ángulo, lo cual limita la longitud del tramo excavado empleando cuñas en "V".

La cuña quemada queda emplazada en la zona central del frente, pero no exactamente al centro sino que se va cambiando su posición en voladuras sucesivas para evitar que la perforación de la cuña se ejecute en la parte más fracturada del frente. Además, la rotación del sitio de la cuña resulta una medida de seguridad, ya que, la zona de la cuña es el sitio donde con más alta probabilidad pueden quedar explosivos sin disparar. El diseño de la cuña quemada depende de las características de la roca, del tipo de los explosivos empleados y del diámetro de los barrenos. Toda roca tiene un determinado porcentaje de expansión que varía con el tamaño de los fragmentos producidos por la voladura. Por tanto, el diseño de la cuña quemada debe tomar en cuenta un espacio vacío para permitir esta expansión. Un 15 por ciento del área de influencia de los barrenos que disparan en primer término es el espacio mínimo que ha resultado adecuado para una fragmentación y desalojo apropiados. Este porcentaje varía de acuerdo con la formación rocosa. Sin embargo, a medida que el espacio vacío proporcionado es mayor, mayor es también la probabilidad de que la voladura actúe eficientemente en la longitud total de las perforaciones.

En una cuña con barrenos de 41.3 mm ($1\,^5/_8$ pulg) de diámetro con un área de influencia de 225 cm^2 (fig I.28) si el espacio vacío está constituido por un barreno central (fig I.28a) proporcionará únicamente el 5.9 por ciento para la expansión. Si con la misma área de influencia se dejan vacíos tres barrenos (fig I.28b) el porcentaje para expansión será, entonces, de 17.8 por ciento. La mayor longitud de los tramos de avance que se logran cuando se deja un espacio de expansión suficiente compensa con amplitud el tiempo invertido en la perforación de los barrenos adicionales.

Para lograr la remoción de la cuña en toda la longitud de la pertoración se recomienda cargar el tercio interior del barreno con la mitad de la carga total del barreno. Además para una adecuada expulsión del material fragmentado, la columna de explosivos debe alcanzar casi hasta la boca del barreno, con menor densidad en la carga.

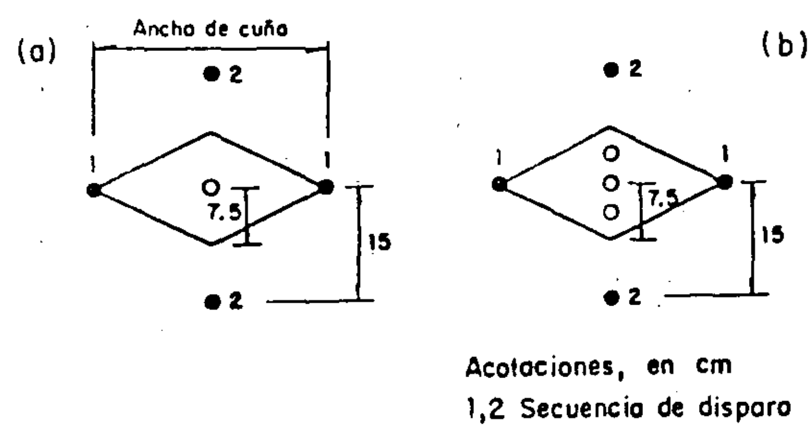


FIG I.28 Cuña quemada cuadrada: a) con un barreno vacío central; b) con tres barrenos vacíos

Si no se reduce la densidad de la carga en la mitad exterior del barreno se corre el riesgo de impedir la acción eficiente de la carga del interior para expulsar el material fragmentado. Cuando este error se comete el avance sólo se logra hasta donde la cuña es fragmentada y desalojada.

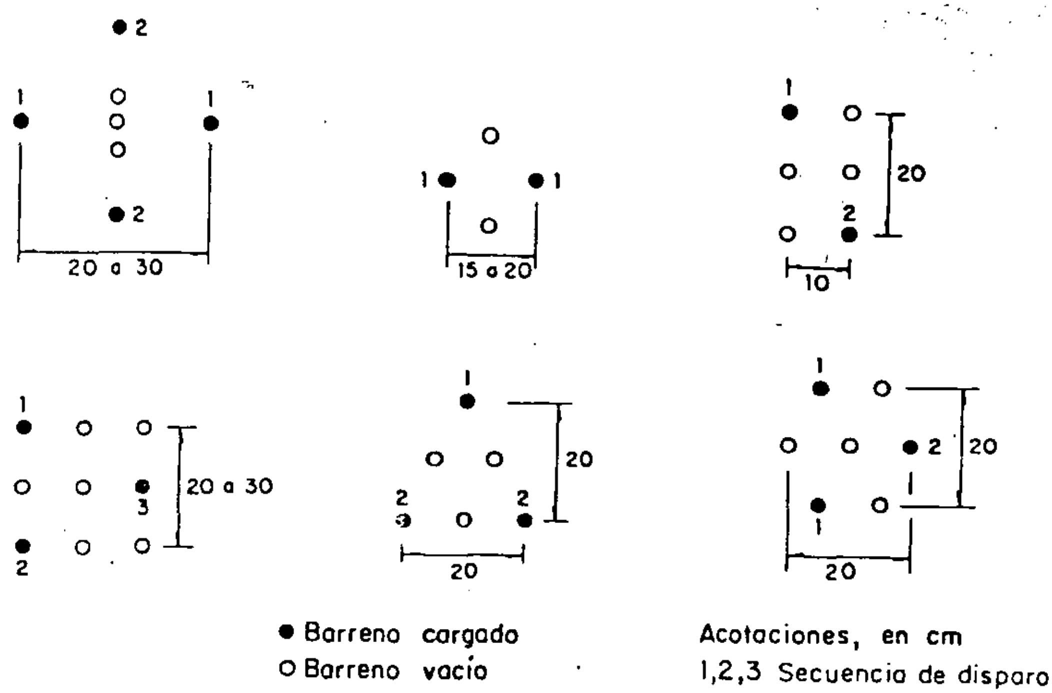


FIG I.29 Algunos diagramas típicos de cuñas quemadas

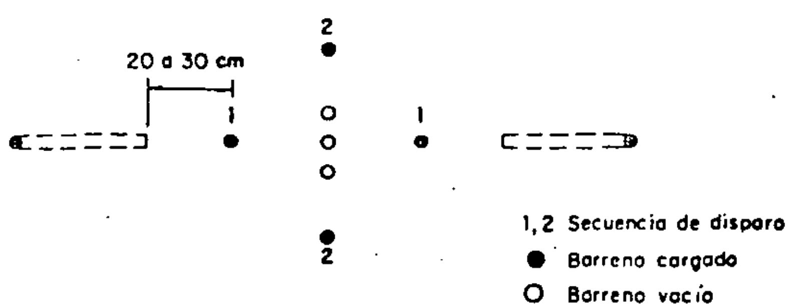


FIG I.30 Barrenos de alivio inclinados o diagonales

Los barrenos de la cuña y los de alivio se cargan dejando, en general, 30 cm para el retacado. Los barrenos restantes se retacan en un tramo de longitud igual al espaciamiento entre los mismos.

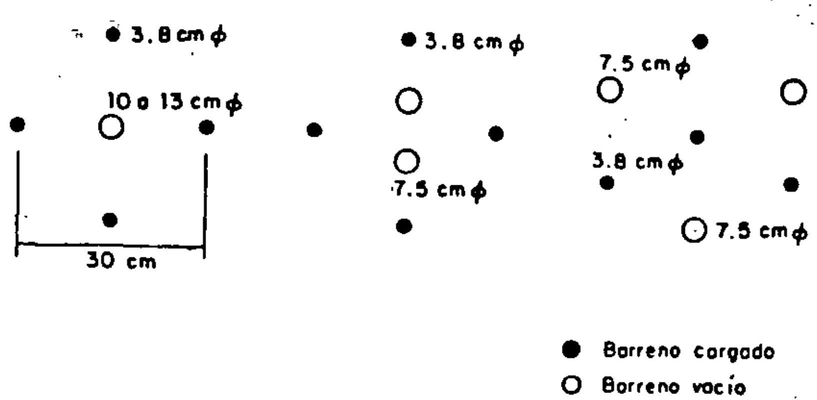


FIG I.31 Cuñas quemadas con barrenos vacíos de mayor diámetro

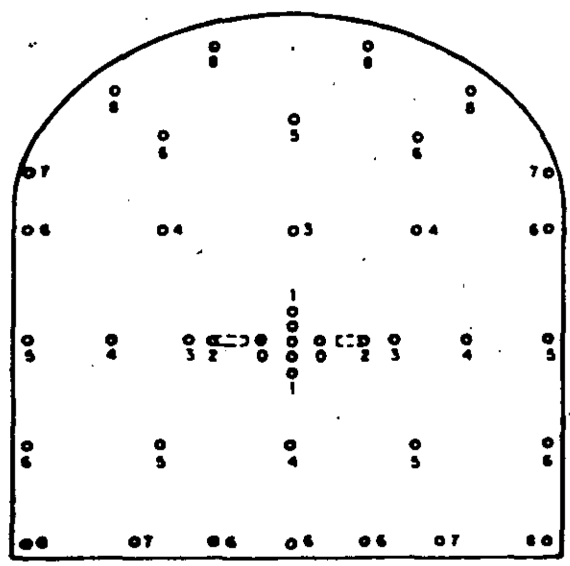


FIG I 32 **Distribución** típica de retardos en un frente de 3 por 3 m

b) Cuña en V

Este tipo de cuña es el más utilizado en túneles mayores de 20 m², aunque recientemente ha podido notarse una tendencia hacia la cuña paralela.

La cuña en V es simétrica. Esto permite una mejor organización del trabajo en el frente respecto a los tipos de cuñas no simétricas. La cuña en V, por otra parte, no exige una barrenación tan perfecta como la cuña paralela para lograr un avance razonable. El ángulo mínimo recomendable para la cuña es de 60°. Este requisito limita el avance por tronada a la mitad del ancho del túnel (fig I.33).

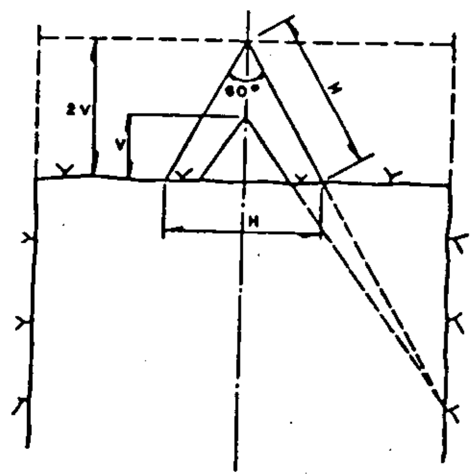


FIG I.33 Barrenos inclinados de la cuña en V

La cuña puede estar formada por uno o más pares de barrenos en V perforados en planos paralelos. El número de estos pares de barrenos depende de la estructura o estratificación de la roca. Cuando el avance por tronada es muy grande o en roca muy resistente cada V de barrenos se integra con uno o dos pares de barrenos de menor longitud.

Todos los barrenos de la cuña en V deben dispararse simultáneamente para obtener mejores resultados, particularmente en roca muy resistente.

En frentes muy grandes deben emplearse retardos mayores para lograr el desplazamiento y la fragmentación adecuados.

7.2.1.4 Cálculo de la carga

El cálculo de cargas en túneles es menos sistemático que el de las voladuras de bancos a cielo abierto. Se emplea la información teórica y experimental de las voladuras de banco a cielo abierto, aplicando factores de aumento de carga para ajustarla a la voladura en túneles. Este aumento se debe al mayor confinamiento de las voladuras en túneles, de tal manera que, a medida que el frente de ataque es menor, mayor es el confinamiento. Por tanto, a menores dimensiones del túnel corresponde una mayor carga específica.

A continuación se darán reglas para la estimación de los espaciamientos y de las cargas en las cuñas de barrenos paralelos, en las cuñas en V y en los barrenos que no pertenecen a la cuña.

a) Barrenos que no pertenecen a la cuña

En esta sección se supone que ya está abierta y expulsada la zona de la cuña y se tiene una cavidad de 1.40 por 1.40 m. Este es el espacio generalmente requerido en barrenos para el fracturamiento y expulsión de la roca hacia esa abertura. Si los barrenos son de diámetro mayor de 3 cm puede ser necesario aumentar las dimensiones de la cavidad a 2 por 2 m.

En la fig I.34 se presentan gráficas que permiten calcular la distancia máxima que debe fijarse entre la cavidad y los barrenos según su diámetro.

Todos los barrenos de la periferia, ya sean del piso, del techo y de los hastiales, deben orientarse de manera que lleguen más allá del contorno (fig I.35) y proporcionen espacio para la perforación de la voladura siguiente.

Los principios de cálculo descritos en esta sección están basados en experiencia obtenida de casos particulares.

La fig I.36 muestra el valor de las cargas específicas que se utilizan normalmente en túneles en función del área de la sección trasversal de los mismos. Los valores indicados en las figs I.36 y I.37 son valores promedio; existen ejemplos de valores que se desvían debido a la forma del túnel, condiciones de la roca, etc.

A continuación se dan recomendaciones para el diseño de las cargas y espa-

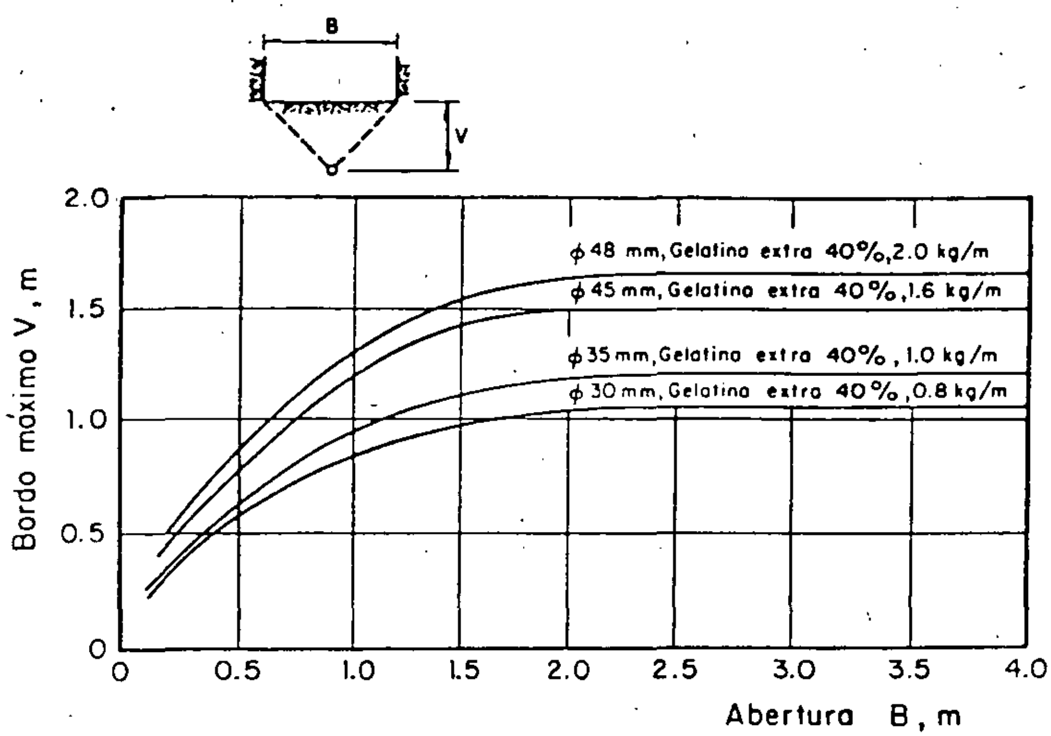


FIG I.34 Relación entre abertura, B, concentración de carga y bordo máximo, V

ciamientos de los barrenos de cada una de las zonas del túnel que se señalan en la fig I.38.

-Barrenos ayudantes con proyección horizontal o hacia arriba

El bordo o distancia entre los barrenos y la cavidad central no debe ser mayor que la mitad de la profundidad del barreno menos veinte centímetros. No deberá tomarse esta condición como base para el cálculo.

El espaciamiento de los barrenos debe ser igual a 1.1 veces el bordo.

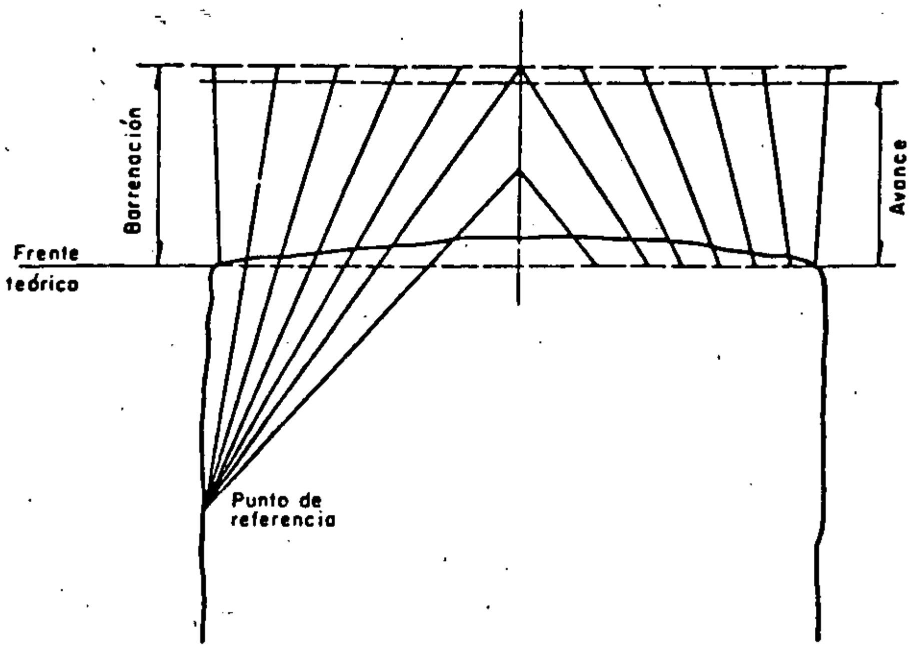


FIG I.35 Distribución en planta de los barrenos de la cuña y los de fuera de la cuña

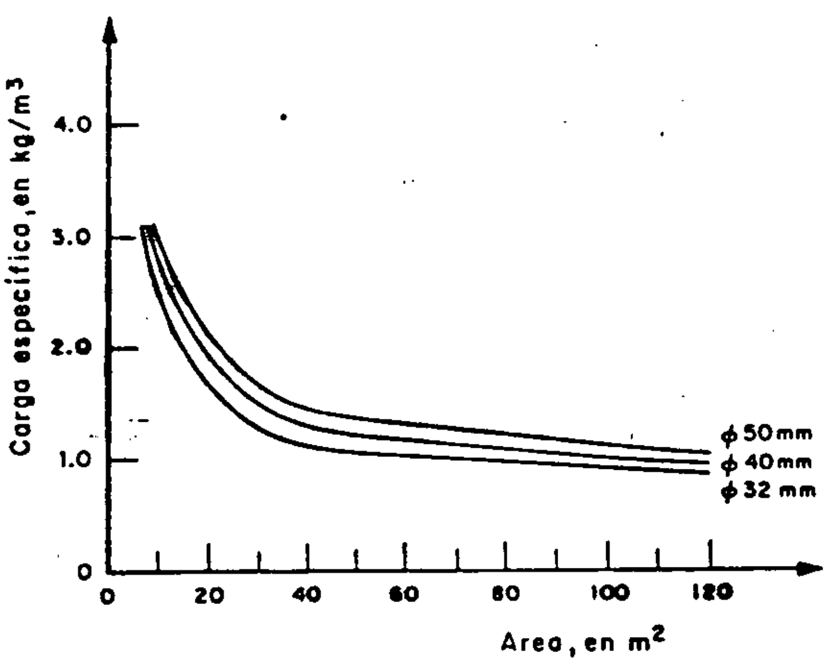


FIG I.36 Cargas específicas utilizadas normalmente en túneles

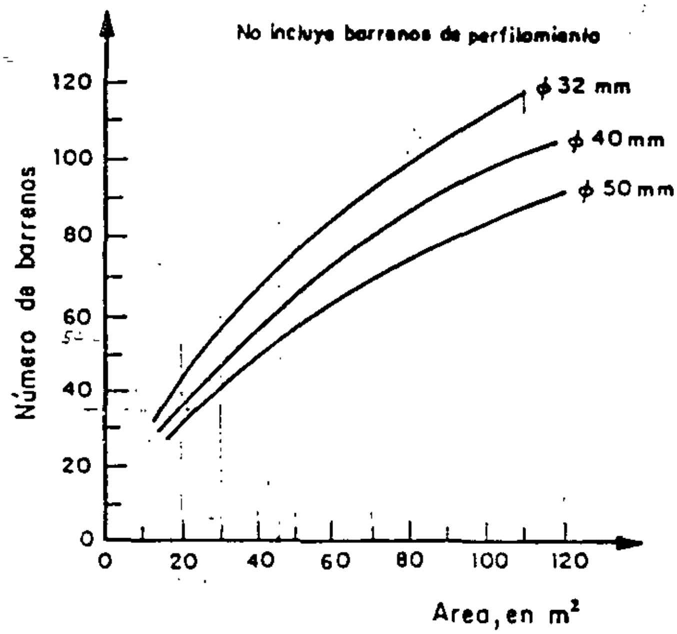


FIG I.37 Número de barrenos en función del área del frente

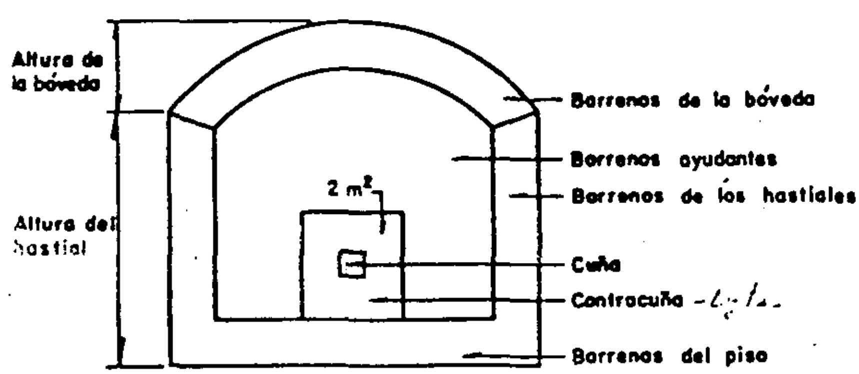


FIG I.38 Zonas de distribución de los barrenos

La carga de fondo ocupa el tercio inferior del barreno con la carga específica de la tabla I.12.

La concentración de la carga de columna en kg/m puede tomarse igual a la mi tad de la concentración de la carga de fondo. La zona de retaque debe ser igual a la mitad del bordo.

TABLA I.12 Carga específica de fondo

Aytes Horiz o arriba (ABIERTA)

Diámetro de los barrenos, en mm	Carga específica, en kg/m³
30	1.1
40	1.3
50	1.5

En la tabla I.13 se muestran los espaciamientos calculados de acuerdo con las cargas específicas de fondo necesarias, considerando explosivos de peso volumétrico de 1.3 g/cm³ y el diámetro de barrenos de la tabla I.12.

TABLA I.13 Espaciamientos y bordos en función de los diámetros de los barrenos

Aytes Horiz o arriba (ABIERTO)

Diámetro de barreno, en mm	Area por barreno, en m²	Bordo, en m	Espaciamiento, en m
32	0.91	0.90	1.00
35	1.00	0.95	1.05
38	1.15	1.00	1.15
45	1.44	1.15	1.25
48	1.57	1.20	1.30*
51	1.71	1.25	1.35*

* Estos espaciamientos son sólo para túneles de gran diámetro; en el caso de áreas menores su magnitud es menor como se muestra en las gráficas de la fig I.34.

Las concentraciones y cargas de fondo y de columna de la tabla I.14 han sido calculadas a partir de las recomendaciones anteriores, en función del diámetro de los barrenos. Estos datos han sido obtenidos de la práctica e incluyen los errores normales de perforación.

→ dicto

TABLA I.14 Cargas, espaciamientos y bordos en barrenos ayudantes con proyección horizontal o hacia arriba

Taco

Diámetro barreno. mm	Profundidad barreno, m	Bordo m	Espaciamiento m	Carga de fondo		Carga de columna		Zona de retaque m
				kg	kg/m	kg	kg/m	
33	1.6	0.60	0.70	0.60	1.10	0.30	0.40	0.30
32	2.4	0.90	1.00	0.80	1.00	0.55	0.50	0.45
31	3.2	0.90	0.95	1.00	0.95	0.85	0.50	0.45
38	2.4	1.00	1.10	1.15	1.44	0.80	0.70	0.50
37	3.2	1.00	1.10	1.50	1.36	1.15	0.70	0.50
45	3.2	1.15	1.25	2.25	2.03	1.50	1.00	0.55
48	3.2	1.20	1.30	2.50	2.30	1.70	1.15	0.60
48	4.0	1.20	1.30	3.00	2.30	2.45	1.15	0.60
51	3.2	1.25	1.35	2.50	2.60	1.95	1.30	0.60
51	4.0	1.25	1.35	3.40	2.60	2.70	1.30	0.60

-Barrenos de piso

El bordo y el espaciamiento de estos barrenos debe calcularse del mismo modo que los barrenos ayudantes. Sin embargo, debe considerarse en el bordo una corrección debido al emboquille de preparación para la voladura siguiente. Por ejemplo, con un bordo de 1.00 m y un margen para emboquille de 0.20 m, la segunda fila de barrenos del piso debe estar 0.80 m arriba de la entrada de los barrenos de la primera fila. La zona de retaque debe ser de 0.20 veces el bordo, es decir, mucho menor que en los barrenos ayudantes y la concentración de la carga de columna se fija hasta de un 70 por ciento de la concentración de la carga de fondo.

En la tabla I.15 se presentan las concentraciones de carga de fondo y de columna, el espaciamiento, el bordo y la zona de retaque para distintos diámetros de barrenos.

-Barrenos ayudantes con proyección hacia abajo

Debido a la ayuda de la gravedad, estos barrenos requieren una menor carga pecífica que los anteriores. La carga específica de fondo puede ser la de la tabla I.16.

TABLA I.15 Cargas, espaciamientos y bordos en barrenos de piso.

Diámetro barreno mm	Profundidad barreno, m	Bordo m	Espaciamiento m	Carga de fondo		Carga de columna		Zona de retaque m
				kg	kg/m	kg	kg/m	
33	1.6	0.60	0.70	0.60	1.10	0.70	0.75	0.10
32	2.4	0.90	1.00	0.80	1.00	1.00	0.70	0.20
31	3.2	0.90	0.95	1.00	0.95	1.30	0.65	0.20
38	2.4	1.00	1.10	1.15	1.44	1.40	1.00	0.20
37	3.2	1.00	1.10	1.50	1.36	1.80	0.95	0.20
45	3.2	1.15	1.25	2.25	2.03	2.60	1.40	0.25
48	3.2	1.20	1.30	2.50	2.30	3.00	1.60	0.25
48	4.0	1.20	1.30	3.00	2.30	4.25	1.60	0.25
51	3.2	1.25	1.35	2.70	2.60	3.20	1.80	0.25
51	4.0	1.25	1.35	3.40	2.60	4.75	1.80	0.25

TABLA I.16 Carga específica de fondo

Diámetro de los barrenos, en mm	Carga específica, en kg/m³
30	1.0
40	1.2
50	1.4

El espaciamiento de estos barrenos puede ser de 1.2 veces el bordo. Las demás características son las señaladas para los otros barrenos ayudantes.

En túneles de sección trasversal pequeña las cargas deberán aumentarse y el bordo y el espaciamiento disminuirse de acuerdo con las funciones de las gráficas que se presentan en las figs I.34, I.36 y I.37.

En la tabla I.17 se presentan las cargas, bordos y espaciamientos de estos barrenos. Los espaciamientos indicados son aplicables siempre que la concentración de carga en el fondo alcance, asimismo, el valor señalado. Si la concentración de carga resulta menor, el espaciamiento deberá reducirse para obtener la carga específica requerida.

Los valores de espaciamientos y bordos indicados en la tabla I.17 pueden aumentarse, particularmente cuando la roca es fácil de excavar y cuando los túneles tienen un área de más de 70 m^2. También es frecuente en estos casos utilizar los espaciamientos señalados pero con menores concentraciones de carga.

TABLA I.17 Cargas, espaciamientos y bordos en barrenos ayudantes con proyección hacia abajo

Diámetro barreno, mm	Profundidad barreno, m	Bordo, m	Espaciamiento, m	Carga de fondo		Carga de columna		Zona de retaque, m
				kg	kg/m	kg	kg/m	
33	1.6	0.60	0.70	0.60	1.10	0.30	0.40	0.30
32	2.4	0.90	1.10	0.80	1.00	0.55	0.50	0.45
31	3.2	0.85	1.10	1.00	0.95	0.85	0.50	0.45
38	2.4	1.00	1.20	1.15	1.44	0.80	0.70	0.50
37	3.2	1.00	1.20	1.50	1.36	1.15	0.70	0.50
45	3.2	1.15	1.40	2.25	2.03	1.50	1.25	0.55
48	3.2	1.20	1.45	2.50	2.30	1.70	1.15	0.60
48	4.0	1.20	1.45	3.00	2.30	2.45	1.15	0.60
51	3.2	1.25	1.50	2.70	2.60	1.95	1.30	0.60
51	4.0	1.25	1.50	3.40	2.60	2.70	1.30	0.60

-Barrenos de los hastiales

Las voladuras de los hastiales y de la bóveda corresponden por lo común al tipo de voladuras denominado recorte o poscorte perimetral (inciso 7.2.1.5). En esta sección se tratan los casos que no son voladuras de recorte.

El bordo, considerando el emboquille de preparación para la voladura siguiente, se toma igual a 0.90 veces el bordo de los barrenos ayudantes.

H profundidad del barreno, en m

q carga específica, en kg/m^3

d diámetro del barreno, en mm

Q_{bk} concentración de la carga de fondo, en kg/m

Q_{pk} concentración de la carga de columna, en kg/m

h_b altura de la carga de fondo, en m

h_o longitud del retaque, en m

E Distancia entre barrenos, en m

TABLA I.19 Cargas, espaciamientos y bordos en barrenos de la bóveda

Diámetro barreno mm	Profundidad barreno, m	Bordo m	Espaciamiento m	Carga de fondo kg	Carga de fondo kg/m	Carga de columna kg	Carga de columna kg/m	Zona de retaque m
33	1.6	0.55	0.65	0.30	1.10	0.35	0.35	0.30
32	2.4	0.80	0.95	0.40	1.00	0.50	0.30	0.40
31	3.2	0.80	0.95	0.50	0.95	0.70	0.30	0.40
38	2.4	0.90	1.10	0.60	1.44	0.70	0.45	0.45
37	3.2	0.90	1.10	0.75	1.36	0.90	0.40	0.45
45	3.2	1.00	1.20	1.10	2.03	1.30	0.60	0.50
48	3.2	1.10	1.30	1.20	2.30	1.45	0.80	0.55
48	4.0	1.10	1.30	1.50	2.30	1.95	0.90	0.55
51	3.2	1.15	1.40	1.40	2.60	1.70	0.80	0.60
51	4.0	1.15	1.40	1.70	2.60	2.25	0.80	0.60

-Barrenos ayudantes con proyección horizontal o hacia arriba

d(mm)	$q(kg/m^3)$
30	1.1
40	1.3
50	1.5
h_b	H/3

$$V_1 \leq \frac{H - 0.40\,m}{2} \quad \text{(ésta es una condición y no es una base de cálculo)} \qquad (I.4)$$

$$E = 1.1\ V \tag{I.5}$$

$$Q_{pk} = 0.50\ Q_{bk} \tag{I.6}$$

$$h_o = 0.5\ V \tag{I.7}$$

-Barrenos de piso

Las mismas características de los anteriores, excepto

$$h_o = 0.2\ V \tag{I.8}$$

$$Q_{pk} = 0.70\ Q_{bk} \tag{I.9}$$

-Barrenos ayudantes con proyección hacia abajo

Las mismas características de los ayudantes con proyección horizontal o hacia arriba, excepto

$$E = 1.2\ V \tag{I.10}$$

-Barrenos de los hastiales

Las mismas características de los anteriores, excepto

$$V = 0.90 \times (\text{bordo de los barrenos anteriores}) \tag{I.11}$$

$$Q_{pk} = 0.40\ Q_{bk} \tag{I.12}$$

$$h_b = H/6 \tag{I.13}$$

-Barrenos de la bóveda

Las mismas características de los anteriores, excepto

$$Q_{pk} = 0.30\ Q_{bk} \tag{I.14}$$

c) Cuñas de barrenos paralelos

Debe calcularse la separación entre el barreno vacío central y los barrenos cargados de la cuña de manera que el área del barreno vacío sea de cuando menos un 15 por ciento del área de influencia de los barrenos de la cuña que disparan en primer término (inciso 7.2.1.3a, fig I.31). La separación así calculada no debe rebasar la que se muestra en la tabla I.20.

TABLA I.20 Separación entre los barrenos vacíos y cargados de la cuña de barrenos paralelos

Diámetro del barreno central, mm	Diámetro de los barrenos cargados, mm	Bordo o separación entre barrenos, mm	Distancia entre centros, mm
57	32	40	85
76	32	53	107
76	45	53	113
2 x 57*	32	80	125
2 x 57*	45	80	131
2 x 76*	32	106	160
2 x 76*	45	106	167
100	45	70	143
100	51	70	146
125	51	88	176

* Dos barrenos centrales.

Las cargas que se presentan en la tabla I.21 son, en general, adecuadas para los barrenos más próximos al barreno central.

Los barrenos denominados de contracuña, situados fuera de ésta, son adaptados al área de la sección trasversal del túnel.

La carga de los barrenos de la contracuña es muy elevada debido a su gran confinamiento. La fig I.39 muestra la disposición de la contracuña para una cuña de dos barrenos centrales.

TABLA I.21 Cargas asignadas a los barrenos más próximos al central

Diámetro de los barrenos cargados, mm	Carga asignada (kg/m)	Diámetro del barreno central, mm
32	0.25	de 57 a 2 x 76
35	0.30	de 76 a 2 x 76
38	0.36	de 76 a 2 x 76
45	0.45	de 2 x 76 a 125
48	0.55	de 2 x 76 a 125
51	0.55	de 2 x 76 a 125

En la tabla I.22 se presentan valores de cargas que han dado buenos resultados en barrenos de contracuña.

TABLA I.22 Valores empíricos de carga en barrenos de contracuña (Ayudantes)

Bordo o separación entre barrenos m	Carga de fondo kg	Carga de columna en kg/m para diámetros de los barrenos cargados de:			
		32 mm	38 mm	45 mm	48 mm
0.20	0.25	0.30	0.45	0.60	0.75
0.30	0.40	0.30	0.45	0.60	0.75
0.40	0.50	0.35	0.50	0.70	0.80
0.50	0.65	0.50	0.70	1.00	1.15
0.60	0.80	0.50	0.70	1.00	1.15
0.70	0.90	0.50	0.70	1.00	1.15

* Longitud sin carga (taco) = 0.5 V.

d) Cuña en V

En esta sección se proporcionan reglas generales para el cálculo de cargas considerando una cuña de vértice interior de 60°. Si este ángulo es menor la carga debe incrementarse.

La dimensión V de la cuña (fig I.40) es función de la cantidad de explosivos que pueden cargarse en los barrenos con arreglo a su diámetro. En la

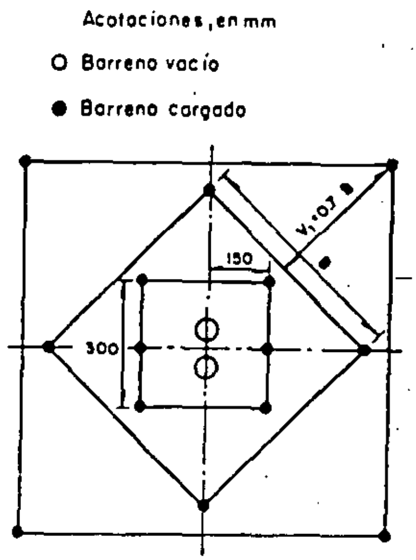


FIG I.39 Cuña de dos barrenos centrales y contracuña

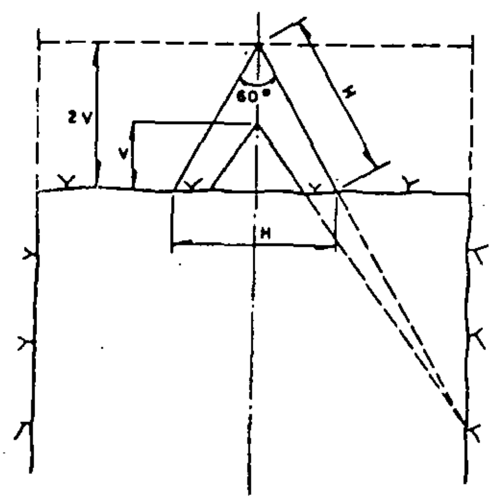


FIG I.40 Cuña en V

tabla I.23 se proporcionan valores que pueden servir de orientación en la determinación de la dimensión y carga de la cuña en V.

En cuñas en V la longitud de la carga de fondo debe ser de cuando menos un tercio de la profundidad del barreno. La carga de columna debe ser igual a la mitad de la carga de fondo. La zona de retaque debe ser un tercio de la dimensión V de la cuña, pero debe ser adaptada al espaciamiento de los barrenos de manera que no haya exceso de carga en la parte de la columna.

TABLA I.23 Dimensiones y cargas de la cuña en V

Diámetro de los barrenos mm	Altura de la cuña m	Bordo V (fig I.34) m	Concentración de la carga de fondo kg/m	Número de filas horizontales
30	1.5	1.0	0.9	3
38	1.6	1.2	1.4	3
45	1.8	1.5	2.0	3
51	2.8	2.0	2.6	3

La concentración de la carga de columna es igual al 40% de la concentración de la carga de fondo.

El bordo o separación de barrenos no debe ser superior a (Prof.barreno-0.40m)/2, lo que implica que en voladuras de poca profundidad la separación de barrenos es menor.

Los barrenos de la contracuña se perforan inclinados (fig I.35) para facilitar la remoción total hasta la profundidad de barrenación.

Los barrenos de la cuña y de la contracuña deben iniciarse con estopines de milisegundos a fin de mejorar la interacción entre los barrenos.

7.2.1.5 Poscorte perimetral

El poscorte perimetral también llamado recorte convencional tiene por objeto proteger la superficie de roca alrededor de la voladura.

Este método consiste en la aplicación de concentraciones de carga reducidas y una mayor densidad de perforación para producir un agrietamiento menor en la superficie perimetral del túnel. Al disparar instantáneamente o con un retardo mínimo entre barrenos se obtiene una acción cortante perimetral que desprende el bordo final con un daño reducido de las paredes (fig I.41).

Estos barrenos se disparan después de los barrenos de piso para asegurar

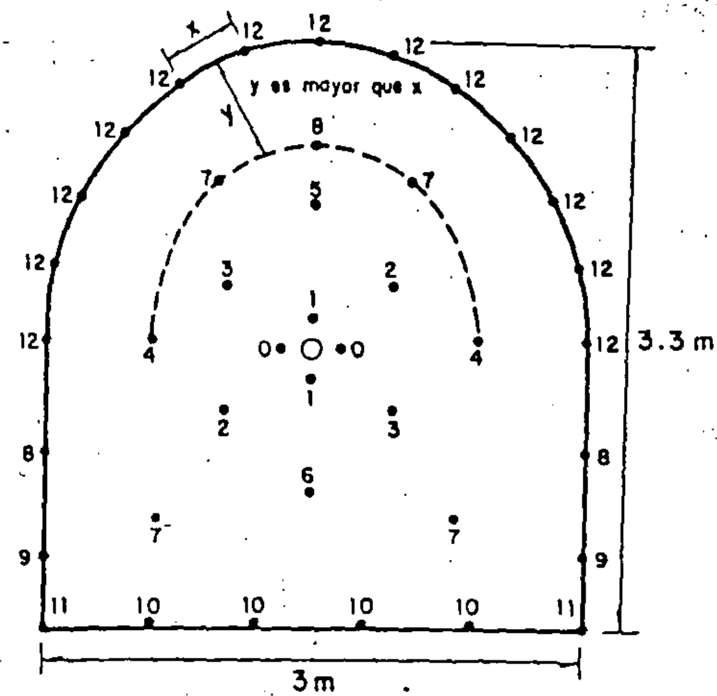


FIG I.41 Distribución típica de retardos en un túnel

que la roca fragmentada ya ha sido desplazada, ofreciéndoles un espacio de alivio suficiente. Este alivio permite una voladura del bordo final con un sacudimiento mínimo.

En la tabla I.24 se proporcionan valores prácticos recomendados de espaciamientos, bordos y concentraciones de carga promedio para dos diámetros de barreno, utilizando explosivos de 1.2 a 1.3 g/cm³ de peso volumétrico.

TABLA I.24 Poscorte perimetral

Diámetro barreno mm	Espaciamiento m	Bordo m	Concentración total de carga en el barreno kg/m
38 - 45	0.60	0.90	0.18 - 0.38
51	0.75	1.05	0.18 - 0.38

Los cartuchos largos de diámetro pequeño de explosivos de baja densidad, permiten una distribución adecuada de la carga a lo largo del barreno. Los cartuchos de 20 cm de longitud se han empleado con éxito en voladuras de poscorte perimetral utilizando espaciadores entre cartuchos para reducir la carga total en kg/m; sin embargo, este procedimiento da como resultado concentraciones de carga relativamente altas en distintos puntos.

7.2.1.6 Precorte

En el precorte los barrenos de contorno se disparan antes de efectuar la voladura propiamente dicha. El precorte produce una grieta entre los barrenos de contorno. Esta grieta evita que las ondas de choque de la voladura principal se trasmitan en toda su intensidad hacia la pared terminada y minimiza la profundidad de la fragmentación en la roca. Como los barrenos están muy próximos entre sí, las grietas se forman siguiendo las líneas de barrenos, y los mismos barrenos constituyen el inicio del agrietamiento. Esto significa que la inclusión de barrenos vacíos entre los cargados, puede mejorar los resultados.

En la tabla I.25 se indican algunas cargas y espaciamientos en función del diámetro de los barrenos.

Si no existen limitaciones en las vibraciones del terreno se utiliza el encendido instantáneo; por lo contrario, si es necesario limitar la magnitud de las vibraciones del terreno se utilizan microretardos. La formación de grietas resulta menos eficiente que con la iniciación instantánea, a menos que se reduzca el espacio entre barrenos. Si el tiempo de retardo es muy grande no se logra el precorte.

TABLA I.25 Precorte

Diámetro del barreno mm	Espaciamiento m	Concentración de carga kg/m
25 - 32	0.20 - 0.30	0.08
25 - 32	0.35 - 0.60	0.18
40	0.35 - 0.50	0.18
51	0.40 - 0.50	0.36
64	0.60 - 0.80	0.38

FACULTAD DE INGENIERIA U.N.A.M.
DIVISION DE EDUCACION CONTINUA

CURSOS ABIERTOS

VI CURSO INTERNACIONAL DE INGENIERIA GEOLOGICA APLICADA A OBRAS SUPERFICIALES Y SUBTERRANEAS

MODULO IV: TECNOLOGIA SOBRE EL USO DE EXPLOSIVOS

PLANTILLAS DE BARRENACION

Palacio de Minería Calle de Tacuba 5 Primer piso Deleg. Cuauhtémoc 06000 México, D.F. APDO. Postal M-2285
Teléfonos: 512-8955 512-5121 521-7335 521-1987 Fax 510-0573 521-4020 AL 26

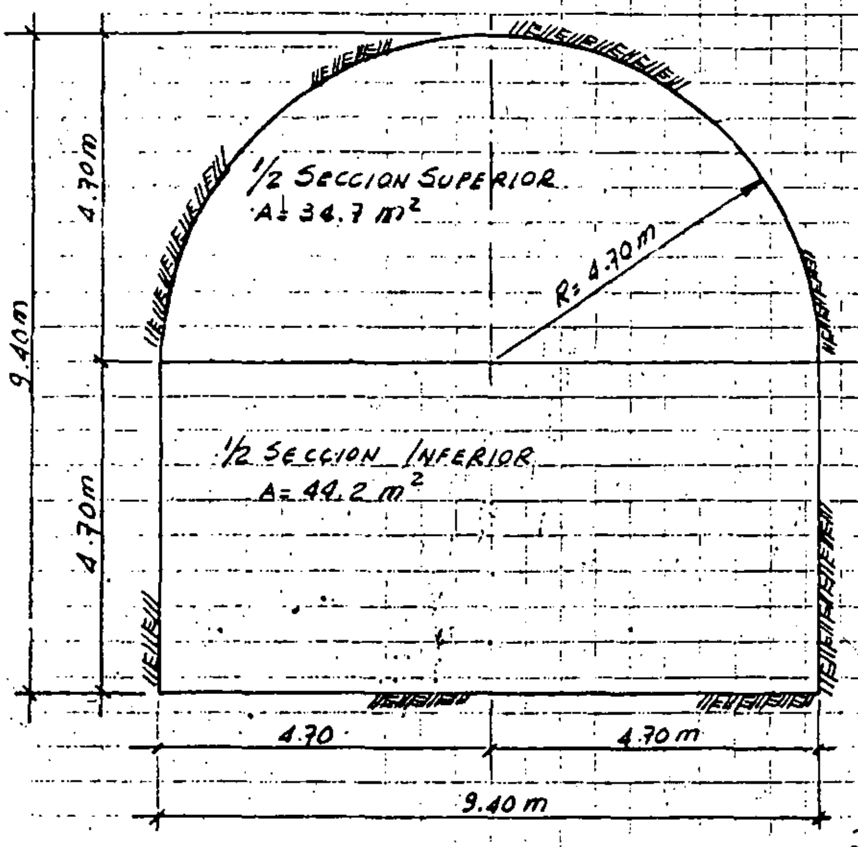
MCCSA

P. H. ZIMAPAN, HGO.

TUNEL DE DESVIO : PLANTILLAS DE BARRENACION

LONG = 522 m

J. R. T.
DIC. 12 - 1989



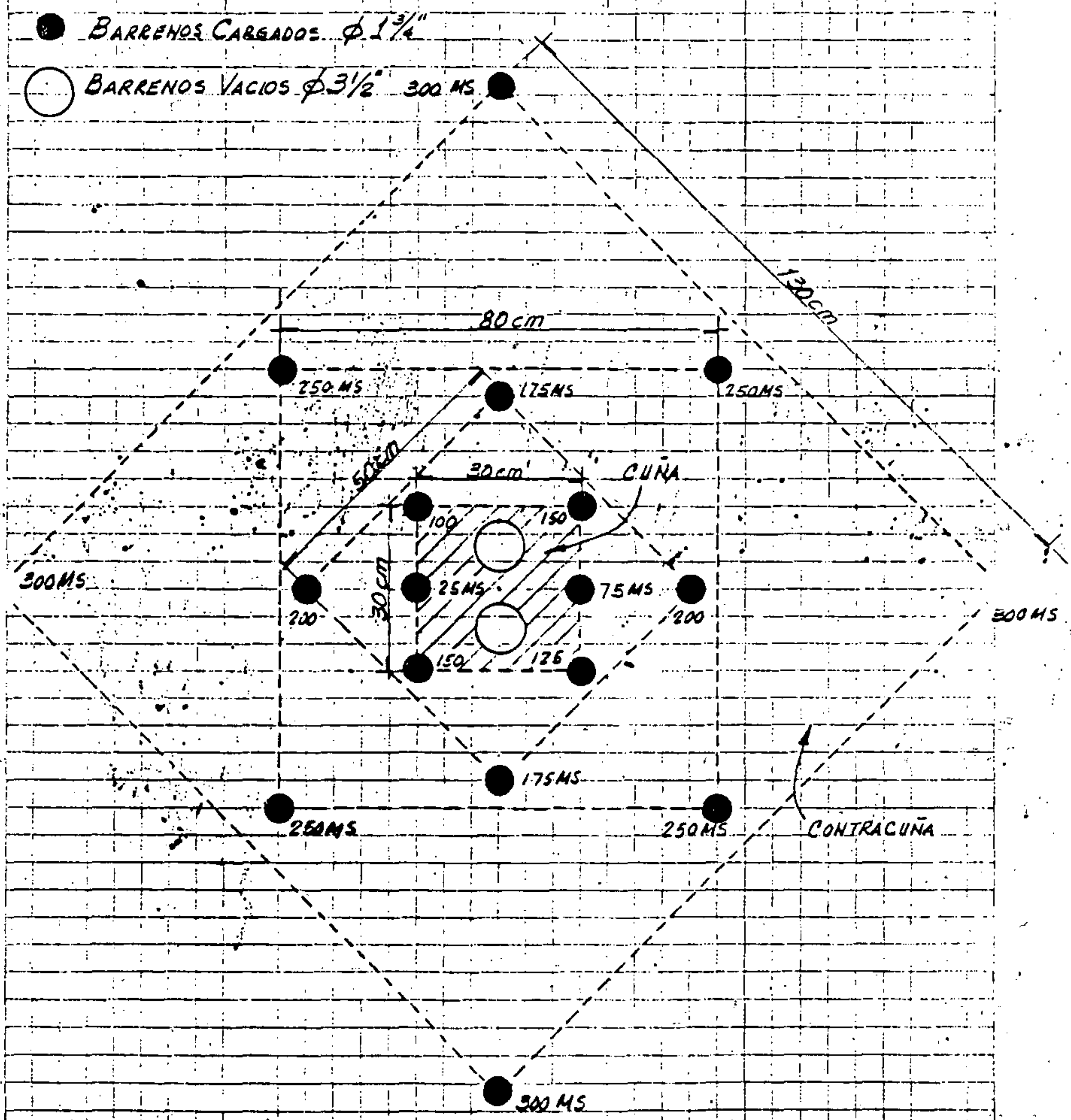
SECCION TRANSVERSAL

MCCSA.

P.H. ZIMAPAN, HGO.

TUNEL DE DESVIO : CUÑA DE BARRENOS PARALELOS J.R.T.

12 DIC. 1989.



VISTA FRONTAL

ICCSA

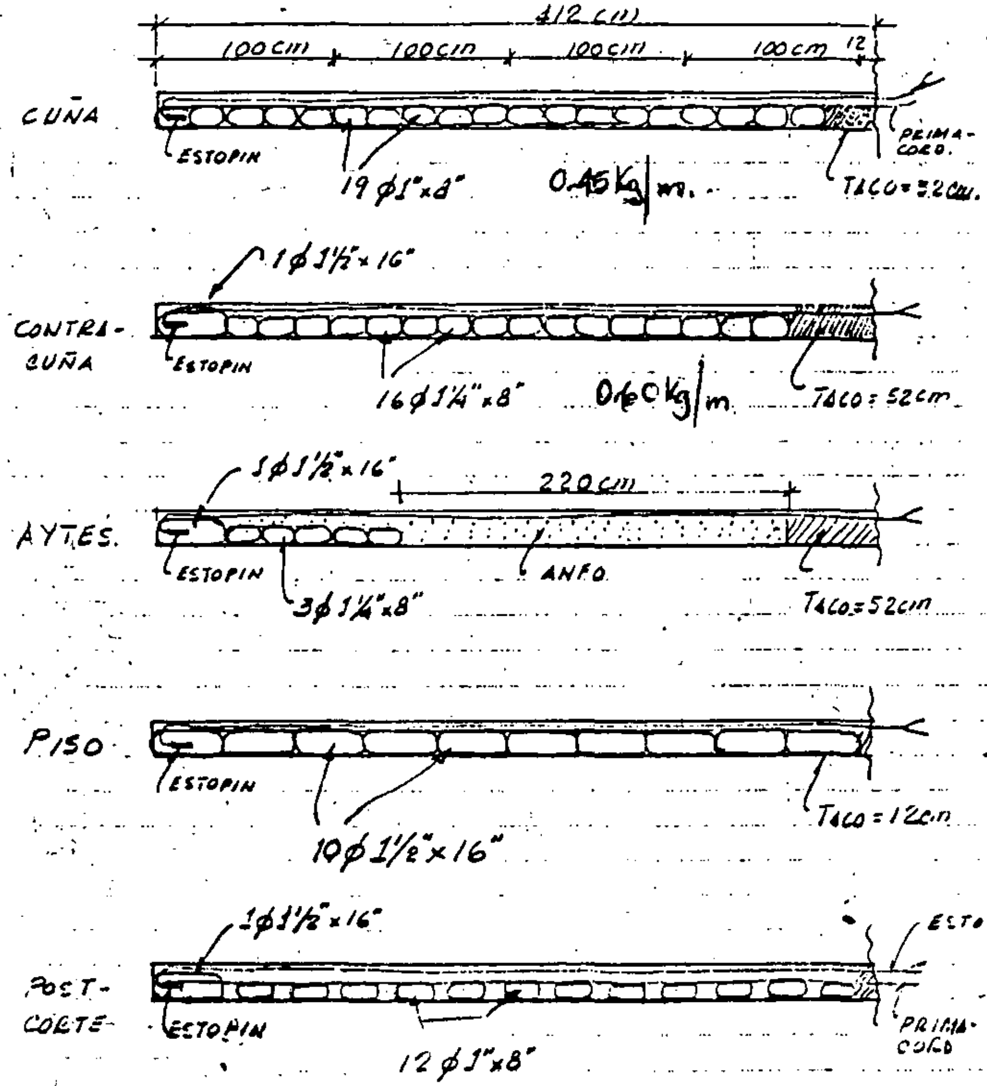
P. H. ZIMAPAN

TUNEL DE DESVIO

DISTRIBUCION DE CARGAS EXPLOSIVAS

C.R.T.

DIC. 12-1989.



ϕ BARRENO = 1¾" = 4.45 cm.
AREA = 15.52 cm² ; = 1.55 Lt/m.

{ ϕ 1" x 8" / 19 Pz. } Estopin + Cordón detonante

Peso = 2.356 Kg

{ 1 ϕ 1½" x 16" / 16 ϕ 1¼" x 8" } Estopin

Peso = 0.556 + 3.088 = 3.644 Kg

1 ϕ 1½" x 16" } 0.556
5 ϕ 1¼" x 8" } 0.965 } 1.521 Kg

ANFO γ = 0.7 Kg/Lt

Peso = 1.55 Lt/m × 0.7 Kg/Lt × 2.2 m
= 2.387 Kg

Σ = 3.91 Kg

{ 10 pz. ϕ 1½" x 16" / Peso = 5.56 Kg }

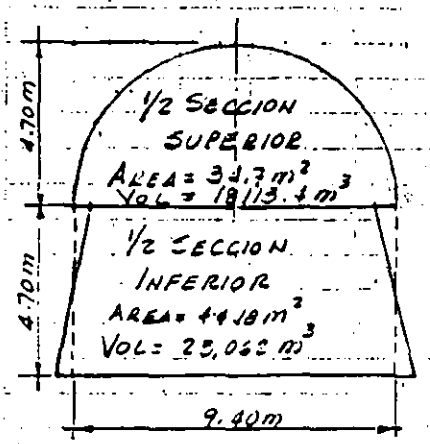
{ 1 ϕ 1½" x 16" / 12 ϕ 1" x 8" }

Peso = 0.556 + 1.488 = 2.04

MCCSA
TUNEL DE DESVIO
LONGITUD = 522 m.

J.R.T.
DIC 12-1989

RESUMEN:



SECCION TRANSVERSAL

MEDIA SECCION SUPERIOR EN TUNEL

a) BARRENACION

Diámetro $\phi\ 1\frac{3}{4}''$

Barrenación específica B.E. = 2.09 m/m^3

b) EXPLOSIVOS

Factor de carga F.C. = 1.57 Kg/m^3

Alto Explosivo = 86.55 %
ANFO = 13.45 %

c) Volumen de roca = 34.7 m^2 × 522 = 18113.4 m^3

d) Cantidad de explosivos

Alto explosivo = 1.57 × 18113.4 × 0.8655 = 24,613.12 Kg.

ANFO : 1.57 × 18113.4 × 0.1345 = 3,824.92 Kg.

ACCESORIOS: PARA TODA LA LONG. DEL TUNEL

ESTOPINES MS = 18 × $\frac{522}{3.91}$ = 2403 pz
ESTOPINES RET = 51 × $\frac{522}{3.91}$ = 6809 pz

PRIMACORD = 31 BARR × (4.13+0.5) × $\frac{522}{3.91}$ = 19120 m.

MCCEA

P. H. ZIMAPAN, HGO. — C.R.T.

TUNEL DE DESVIO — Dic 12-1989

RESUMEN... CONTINUACION — MEDIA SECCION INFERIOR — BANQUEO

a) BARRENACION → Diametro barrenos: ϕ 2 1/2"

Barrenación específica (incluyendo precorte) $= 0.5714\ m/m^3$

Volumen de roca $= 44.18\ m^2 \times 522\ m = 23{,}062\ m^3$

b) EXPLOSIVOS

Factor de carga (Incluyendo Precorte): $F.C. = 0.6346\ Kg/m^3$

Banqueo: Alto Explosivo $= 12\ Barr \times 3\ Kg = 36\ Kg$

ANFO $= 12\ Barr \times 8\ Kg = 96\ Kg$

Precorte: Alto Explosivo $= 16\ Barr \times 1.737\ Kg = 27.79\ Kg.$

$\Sigma = 159.79\ Kg$

Alto Explosivo $= 39.92\ \%$

ANFO $= 60.08\ \%$

c) CANTIDAD DE EXPLOSIVOS

Alto Explosivo $= 0.6346 \times 23{,}062\ m^3 \times 0.3992 = 5842.35\ Kg$

ANFO $= 0.6346 \times 23062\ m^3 \times 0.6008 = 8,793\ Kg$

d) ACCESORIOS: PARA TODA LA LONG. DEL TUNEL

Estopines MS para Banqueo $= \frac{F.C. \times Vol}{Long.\ Barr.} = \frac{0.2664\ m/m^3 \times 23062\ m^3}{5.59\ m/barr} = 1099\ Barr = 1099\ Pzas$

Estopines MS para Precorte (1 pza. @ 20 m) $= \frac{2 \times Long.\ tunel}{0.70\ m \times 20} = \frac{2 \times 522}{0.70 \times 20} = 75\ pz.$

Total $= 1099 + 75 = 1174\ pz.$

J.R.T.

Dic. 12-1989.

RESUMEN ..... CONTINUACION ..... MEDIA SECCION INFERIOR ..... BLINQUEO

d) ACCESORIOS

Primacord = Nº Barr. de precorte × long. Barr. = $\frac{2 \times 522}{0.7} \times (4.8 + 0.5) = 7905$ m.

TUNEL DE DESVIO Long = 522 m

RESUMEN DE BARRENACION Y MATERIALES PARA TODA LA LONGITUD DEL TUNEL (522 m)

SECCION	BARR. ϕ 1 3/4" c/Jumbo	BARR. ϕ 2 1/2" c/Trackdrill	ALTO EXPLOS. Kg	ANFO Kg	ESTOPIN. MS Pz.	ESTOPIN BET. Pz	P. CORD. m
1/2 Sec. Sup.	2.0910 m/m³ (37,857 m)	—	24,613	3,825	2,403	6,809	18,293
1/2 Sec. Inf.	—	0.5714 m/m³ (13,178 m)	5,842	8,793	1,174	—	7,905

SUMA   37,857 m   13,178 m   30,455 Kg   12,618 Kg   3,577 Pz   6809 Pz   26,198 m

P.U.   ×(   )   ×(   )   ×(   )   ×(   )   ×(   )   ×(   )   ×(   )

COSTO DTO. →

COSTO DIRECTO $= \frac{\Sigma(\quad)}{522\,m} = \$ \frac{(\quad)}{m.}$