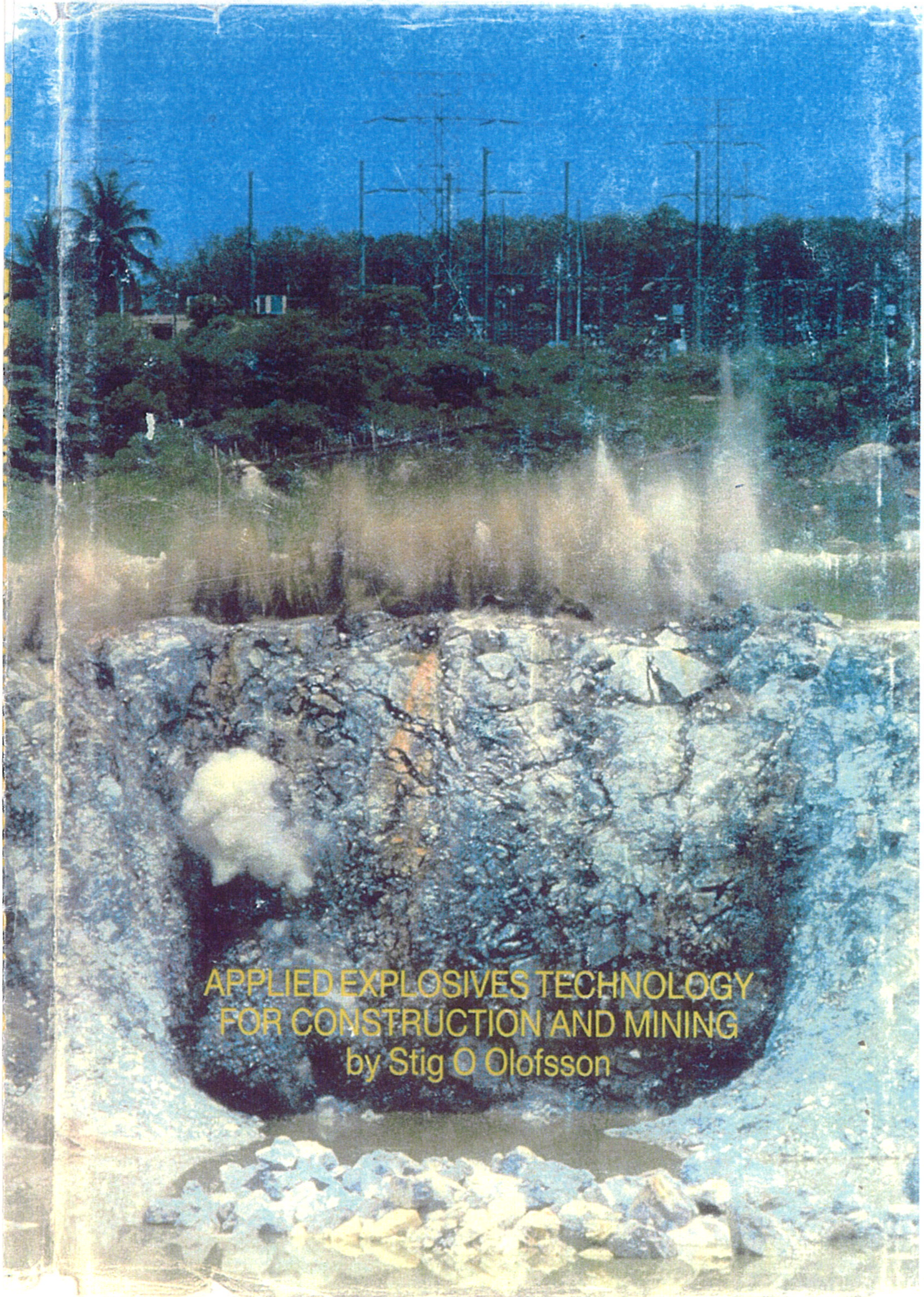

# APPLIED EXPLOSIVES TECHNOLOGY FOR CONSTRUCTION AND MINING

by Stig O Olofsson

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# CONTENTS

# 1. FOREWORD

The writing of this book was first started during my stay in Malaysia at the beginning of the 1980's and the intention was to take away some of the mysticism which surrounds rock blasting and rock blasters in the world.
I had then been using the Swedish rock blasting technique on three continents, and found that the theories concerning rock blasting that were worked out in Sweden in the 1940's and 1950's functioned well also in other environments and in other rocks than those I was used to. The blasting works I performed confirmed the theories.

In blasting technology Sweden became the forerunner due to the research of Dr. Ulf Langefors. His scientific approach to rock blasting transformed it from an occupation in which personal experience, skill and intuition formed the dominating part to technological science. Dr. Langefors' pioneering work opened the way for controlled blasting operations and bigger rounds could be blasted in a safer and more economic way.
It is no overstatement to say that the works of Dr. Langefors have had the same significance for the development of the blasting technology as Alfred Nobel's research for the development of civil high explosives.

This manual is, from blasting technology point of view, based on the book "The Modern Technique of Rock Blasting" which is written jointly by Ulf Langefors and Björn Kihlström. Formulas have been simplified or transformed to graphs for easier accessibility to the reader.
The method of calculation and design of blasting operations such as bench blasting, tunnel blasting and underwater blasting has been simplified, and the calculation procedure made easy to follow step by step.
Most of the problems that may occur in a blasting operation are dealt with and I believe that the reader will have use for this book for both reference and guidance.

Ärla, October 1988
Stig O Olofsson

**Foreword to the second edition.**

The first edition of this book was sold out rapidly so a new edition seems to be of interest. The content is mainly the same as in the first edition, but misprints have been corrected and in the chapter about cautious blasting new instruments for measurement of ground vibrations, air shock waves and water shock waves are presented. The development in this field has been very progressive the last year.
On request from many readers I have added a conversion table with conversion factors from metric to American and English units.

Ärla June 1990
Stig O Olofsson

# 2. INTRODUCTION

In the beginning of the 17th century black powder was introduced and blasting replaced fire setting as the principal method of loosening rock in the mining industry in Europe. The ancient method of building a wood fire against the face of ore-bearing rock and keeping it burning until the rock was heated through, and then pouring water on the heated rock to cause it to break by tension was too slow, and became too costly as the forests around the mines were clean-cut and transport of wood became a problem.

The introduction of black powder into the mining industry was relatively fast and by the end of the 17th century most of the European miners used black powder to loosen rock. By the turn of the century black powder became widely used in construction work.

The initiation of black powder was hazardous until William Bickford, an Englishman, patented the "Miners Safety Fuse" in 1831, thus giving the blasters a reliable and safe initiating device for black powder.

The demand for more powerful tools to break rock engaged many in developing new explosives. In 1846, Ascanio Sobrero, an Italian, discovered nitroglycerin, but he considered it too unpredictable and hazardous for anyone to manufacture and use.

The new invention was tried out in Sweden by Alfred Nobel and his brothers. They found it excellent for blasting the hard Swedish granite. Subsequently Alfred Nobel in 1864, formed his first explosives company, "Nitroglycerin Aktiebolaget", for the manufacture of nitroglycerin.

The main problem with the nitroglycerin was to get it to shoot consistently. Alfred Nobel solved this problem by the invention of the fulminate of mercury blasting cap in 1867, which when used together with safety fuse made an excellent initiating system for nitroglycerin.

Nitroglycerin conquered the world rapidly and factories were erected all over the world. However, disastrous explosions in Europe and America made people aware of the risks in the manufacture and use of the new explosive and subsequently laws were passed against its use in many countries.

In his efforts to make nitroglycerin safer to handle, Alfred Nobel in 1866 discovered that Kieselguhr ( a diatomaceous earth) not only absorbed three times its own weight of nitroglycerin, but also rendered it less sensitive to shock. After kneading and shaping it into a cartridge, it was wrapped in paper and the **DYNAMITE** was invented. It was an explosive with a brisance of power twenty times greater than that of black powder. The development of dynamite continued, and in 1875 Alfred Nobel dissolved nitrocellulose into nitroglycerin, thus introducing blasting gelatine which is still one of the most powerful explosives for civil use.

In the 1920's nitroglycol was added to dynamite, thus lowering its freezing point (+13°C) considerably. In 1964, a new manufacturing process of dynamite was introduced in Sweden, where the nitroglycerin was flegmatized (DNT was added) making it both safer to manufacture and handle the explosive which was marketed under the trade name of Dynamex. Dynamex was classified as a safety explosive.

Together with the development of dynamite new methods were searched for to initiate the explosives. In the beginning of the 20th century the electric initiation was introduced, and by 1922 the first electric delay detonator (with 1 sec. delay) came into practical use. The introduction of the short delay detonator 10–100 milliseconds) in the late 1940's has had the greatest importance in the development of modern blasting techniques.

In the late 1970's we saw new non-electrical initiating systems like Nonel being developed.

The development of blasting techniques in the United States and Europe were more or less similar until 1955.

In 1955, Robert W. Akre presented a paper on "Akremite" on the year's Coal Show in the U.S.A. Akremite was a mixture of prilled ammonium nitrate and black carbon forming a "do-it-yourself" blasting agent. The mixture had to be put into polyethylene bags to be kept dry in the blast hole.

The following year, 1956, ANFO (Ammonium Nitrate and Fuel Oil) was introduced to the U.S. market (89 years after the patent of AN with various sensitizers was given to Johan Norrbin and Johan V. Ohlsson in Sweden). The success of the ANFO in U.S.A. is indisputable, from a consumption rate of almost nil in 1956, the consumption had increased to over 1,000,000 tons by 1975. The consumption of dynamites has, during the same time, declined from 340,000 tons to 135,000 tons. In Europe the ANFO was not so widely accepted and the dynamites were prevailing into the 1980's. In 1985 a new, more water resistant ANFO, Akvanol was presented by Nitro Nobel, which will increase the versatility of this explosive.

In the 1960's, we have seen the development of water gels and slurries and in the 1970's the development of emulsion explosives (EMULITE) and the 1980's we will see new powered ANFO, (EMULAN). In this manual I will emphasize on the use of Dynamex, Emulite, Emulan and ANFO. The latter three are the explosives which will be used in the future and Dynamex M will be on the market for many years to come.

## 2.1 Terminology

Terms used in explosives and blasting.

**ABUTMENT.** – The point in a tunnel section where wall and roof meet.
**ABUTMENT HEIGHT.** – Height from tunnel floor to abutment.
**ACCELERATION.** – Unit of ground vibration i g (1 g=9.81 m/sec$^2$)
**ACCESS TUNNEL.** – Tunnel from surface to underground work site.
**ADIT.** – Horizontal entrance to mine.
**ADVANCE.** – Excavated length of tunnel per blasting round.
**AIRBLAST.** – Airborne shockwave resulting from the detonation of explosives. Can be caused by rock movement or by release of expanding gas into the air.
**AIR OVERPRESSURE.** – See airblast.
**ALUMINUM.** – Metal commonly used as fuel or sensitizing agent in explosives and blasting agents. Increases the energy content.
**AMPLITUDE.** – See displacement.
**AMMONIUM NITRATE (AN).** – The most commonly used oxidizer in explosives and blasting agents.
**ANFO.** – Powder form explosive consisting of **A**mmonium **N**itrate and **F**uel **O**il. The most commonly used blasting agent.
**ARCH HEIGHT.** – Height from abutment to the highest point of the tunnel roof.
**AXIAL PRIMING.** – System for priming blasting agents in which the core of priming material extends through the column of the blasting agent.

**BACK BREAK.** – Rock broken beyond the limits of the last row of holes.
**BASE CHARGE.** – Main explosive charge in a detonator.
**BENCH.** – Horizontal rock shelf.
**BENCH BLASTING.** – Blasting of a bench with at least two free faces.
**BLAST.** – Detonation of explosives to break rock.
**BLAST AREA.** – Area close to a blast, which may be influenced by flyrock and/or concussion.
**BLASTER.** – Qualified person in charge of a blast.
**BLASTHOLE.** – Hole drilled in rock for the placement of explosives.
**BLASTING AGENT.** – Explosive that meets prescribed criteria for insensitivity to initiation.
**BLASTING CAP.** – See detonator.
**BLASTING CIRCUIT.** – Electric circuit used to fire one or more electric detonators.
**BLASTING DIARY.** – Diary to be kept on work site containing all information about each blast.
**BLASTING MACHINE.** – Machine expressly built for initiating electric detonators or other types of initiators.
**BLASTING MAT.** – Covering placed over a blast to hold down flyrock. Usually made of scrap tires, logs, ropes or wire cables.
**BLASTING PLAN.** – Plan indicating planned drilling, charging, initiation and safety measures for blasting operation.

**BLOCKHOLE.** – Hole drilled into a boulder for the placement of a small charge to break the boulder.
**BOOSTER.** – Charge of high explosive used to improve detonation stability and to intensify the explosive reaction.
**BOREHOLE.** – See blasthole.
**BOTTOM BENCH.** – Underground bench blasted after the excavation of the top heading.
**BOTTOM CHARGE.** – Concentrated charge in the bottom part of the blasthole.
**BOULDER.** – Oversized blocks from blasting.
**BRIDGE WIRE.** – Very fine filament wire embedded in the ignition element of an electric detonator. The heat from a current passing through the bridge wire initiates a pyrotechnic element, which in turn initiates the detonator.
**BRISANCE.** – Property of an explosive approximately equivalent to the velocity of detonation (VOD). An explosive with high VOD has high brisance.
**BUBBLE ENERGY.** – Energy of expanding gases of an explosive, as measured in an underwater test.
**BULK EXPLOSIVE.** – Explosive material prepared for use without packing.
**BULK STRENGTH.** – Strength of a given volume of an explosive compared with the equivalent volume of Blasting Gelatine.
**BULL HOLE.** – Large empty center hole in a parallel hole cut.
**BURDEN.** – Distance from an explosive charge in a blasthole to the nearest free or open face.
**BURN CUT.** – Parallel hole cut with closely spaced boreholes. One or several of the holes are left uncharged.
**BUTT.** – Portion of a blasthole that remains relatively intact after a blast. A butt may contain explosives and is thus considered hazardous.

**CAPPED FUSE.** – Safety fuse to which a blasting cap has been attached.
**CAP SENSITIVITY.** – Sensitivity of an explosive to initiation by a #8 detonator or fraction thereof.
**CARBON MONOXIDE.** – Poisonous gas created by the detonation of explosives. Inadequate amount of oxygen in the explosive causes excessive carbon monoxide content in the "after the blast" fumes.
**CARTRIDGE.** – Container of explosive. Can be rigid or semi-rigid.
**CAST PRIMER.** – High velocity explosive used to initiate blasting agents.
**CAUTIOUS BLASTING.** – Blasting with respect to surrounding areas, controlling flyrock, ground vibrations and air shock waves.
**CHAMBERING.** – Process of enlarging the bottom part of the blasthole by firing small explosive charges, enabling a larger final charge in the hole.
**CIRCUIT TESTER.** –Measuring instrument used to check that the electric circuit of an electric blasting round is unbroken. Should not be used for larger rounds.
**COLLAR.** – Opening of a blasthole. The act of collaring the hole means to start drilling the hole.
**COLLAR DISTANCE.** – Distance from top of explosive to collar of blasthole.

Usually filled with stemming.

**COLUMN CHARGE.** – Charge of explosive or blasting agent in the column section of the blasthole, above the bottom charge.

**CONCUSSION CHARGE.** – Surface charge used to blast boulders.

**CONFINED DETONATION VELOCITY.** – Velocity of detonation (VOD) of an explosive or blasting agent under confinement, such as in a blasthole or an iron pipe.

**CONNECTING WIRE.** – Wire used to connect the detonator circuit with the firing cable or to extend leg wires from one blasthole to another.

**CONTOUR HOLES.** – Holes drilled along the perimeter of the excavation.

**CONTROLLED BLASTING.** – Technique to control overbreak and damage to remaining rock surface.

**COVERING.** – See blasting mat.

**CRITICAL DIAMETER.** – Minimum diameter of an explosive for propagation of stable detonation.

**CURRENT LEAKAGE.** – Arcing of ignition current to earth (water) instead of going through the electric blasting circuit.

**CURRENT LEAKAGE TESTER.** – Instrument to detect current leakage.

**CUSHION BLASTING.** – Technique to produce competent slopes in bench blasting.

**CUT.** – Opening part of a tunnel blast to provide a free face for the remainder of the round.

**CUT EASER HOLES.** – In tunneling, the holes closest to the cut used to enlarge the opening formed by the cut.

**CUTOFFS.** – Part of a charged blasthole where the explosive has failed to detonate, often due to influence of detonation from blastholes with lower delay number.

**CUTTINGS.** – Dust of rock created by drilling.

**DEAD PRESSING.** – Desensitization of an explosive caused by pressurization.

**DECIBEL.** – Unit of sound pressure used to measure airblast.

**DEFLAGRATION.** – Subsonic but rapid explosive reaction.

**DELAY BLASTING.** – Use of delay detonators or relay connectors to cause separate charges to detonate at different times.

**DELAY CONNECTOR.** – Short interval delay device used with detonating cord for short delay blasting.

**DELAY DETONATOR.** – Detonator, electric or non-electric, with a built-in delay element creating a delay between the input of energy and the explosion of the detonator.

**DELAY TIME.** – Time between initiation and detonation of a detonator.

**DENSITY.** – Specific weight of an explosive expressed in grams per cubic centimeter (gr/c.c.).

**DETONATING CORD.** – Plastic covered core of high velocity explosive, used to initiate explosive charges.

**DETONATING RELAY.** – Device inserted in a line of detonating cord to produce a delay between incoming and outgoing sides.

**DETONATION.** – Supersonic explosive reaction which creates a high pressure shock wave, heat and gases.
**DETONATION PRESSURE.** – Pressure created by the detonation proceeding through the explosive column.
**DETONATION VELOCITY.** – See velocity of detonation, VOD.
**DETONATOR.** – Device containing a detonating charge that is used to initiate an explosive.
**DISPLACEMENT.** – Unit of ground vibration (height of deflection in mm).
**DOWNLINE.** – Line of detonating cord in a blasthole transmitting the initiation from the trunkline to the explosive down in the hole.
**DRILL DUST.** – See cuttings.
**DRILLING PATTERN.** – Plan of holes laid out on a tunnel face or a bench which are to be drilled for blasting. The burden and spacing are usually expressed in meters while the diameter of the blastholes is expressed in millimeters.
**DRILL SERIES.** – Series of integral steels in which the diameter of the drillbit decreases 1 mm for every increase of 0.8 m in the length of the steel.
**DROP BALL.** – Steel weight suspended on a wire which is dropped from a height onto large boulders to break them into smaller pieces.
**DYNAMITE.** – High explosive invented by Alfred Nobel. Any high explosive containing nitroglycerin as a sensitizer is considered a dynamite.

**EASER HOLES.** – See cut easer holes.
**EARTH FAULT.** – See current leakage.
**ELECTRIC DETONATOR.** – Detonator designed to be initiated by an electric current.
**ELECTRIC STORM.** – Atmospheric disturbance creating hazards in blasting operations with electric detonators.
**EMULSION.** – Explosive where the oxidizers are dissolved in water and surrounded by immiscible fuels.
**EXPLODER.** – See blasting machine.
**EXPLOSION.** – Thermochemical process in which mixtures of gases, solids or liquids react with almost instantaneous formation of gaseous pressures and heat release. Also see detonation.
**EXPLOSIVE.** – Chemical mixture that releases gases and heat at high velocity, causing very high pressures.
**EXTRANEOUS ELECTRICITY.** – Electrical energy, other than the firing current, being a hazard to blasting with electric detonators. Includes stray current, static electricity, lightning, radiofrequency energy and inductive and capacitive energy.

**FACE.** – Rock surface against which a blast can be executed.
**FAN CUT.** – Cut for tunnel blasting where the opening holes are spread in the form of a fan.
**FAULT.** – Natural crack formation in the rock.
**FIRING CABLE.** – Cable connecting the blasting round with the blasting machine.

**FLASH OVER.** – Sympathetic detonation between explosive charges or between charged blastholes.
**FLYROCK.** – Undesirable throw of rock from the blast.
**FRAGMENTATION.** – Act of breaking the rock. Also the distribution of the particle size of the blasted rock.
**FREQUENCY.** – Unit of ground vibration characteristics (periods per second).
**FUEL OIL.** – Fuel, usually diesel fuel, in ANFO.
**FUMES.** – Noxious or poisonous gases occurring from a blast.
**FUSE.** – See safety fuse.
**FUSE LIGHTER.** – Pyrotechnic device for lighting of safety fuse.

**GALVANOMETER.** – Also called blasters galvanometer, see ohm meter.
**GAP SENSITIVITY.** – Distance across which an explosive can propagate a detonation.
**GRAINS.** – System of weight measurement in which 7000 grains equal 1 lb. (15400 grains=1 kg). The measurement is normally used in connection with detonating cord where the core load is expressed in grains per foot. A core load of 50 grains per foot is approximately 10 grams per meter according to the metric system.
**GROUND VIBRATION.** – Shock wave emanating from a blast transmitted through the surrounding ground.

**HALF-SECOND DETONATOR.** – Delay detonator with approximately 0.5 sec. delay between subsequent numbers.
**HEAD.** – See tunnel face.
**HEADING.** – Horizontal underground excavation in rock.
**HERTZ.** – Term used to express the frequency of ground vibrations and airblast.
**HIGH EXPLOSIVE.** – Any explosive which is sensitive to a #8 detonator and reacts supersonically.
**HOLE SPACING.** – See spacing.

**IGNITER CORD.** – Flexible line containing a core of composition which burns with an intense flame. The cord is designed to be used together with connectors for firing with plain detonators and fuses, when the number of fuses is greater than can be lit safely with a fuse lighter.
**IGNITION CABLE.** – See firing cable.
**INITIATION.** – Act of detonating an explosive by means of a detonator or a primer.
**INITIATING MACHINE.** – See blasting machine.
**INSTANTANEOUS DETONATOR.** – Detonator not containing any delay element.
**INSULATION METER.** – See current leakage tester.
**INTERVAL.** – Difference in delay time between detonators with different numbers.

**JOINTS.** – Planes within the rock mass which separate solid rock masses from each other.

**LEADS.** – See leg wires.
**LEAD WIRE.** – See firing cable.
**LEG WIRES.** – Cables supplied with the electric detonator.
**LEVELING.** – Blast of low benches, where the bench height is less than twice the burden (2×B).
**LIFTERS.** – Blastholes in a tunnel round breaking upwards.
**LINE DRILLING.** – Method to control overbreak. A series of very closely spaced holes are drilled at the perimeter of the excavation. These holes are not charged with explosives.
**LOADING DENSITY.** – Expression of explosive density in terms of kilograms of explosive per meter of charge length of a specific diameter.
**LOADING POLE.** – See tamping rod.
**LOOK-OUT.** – Angling of the contour holes in a tunnel outside the theoretical contour to provide space for the drilling equipment when drilling the following round.

**MAGAZINE.** – Structure specially constructed for storing explosives and other explosive materials.
**MICROBALLOONS.** – Tiny hollow spheres of glass or plastic which are added to explosive materials to enhance sensitivity by assuring adequate content of entrapped air. In emulsion explosives like EMULITE, the microballoons act as sensitizer.
**MILLISECOND.** – Unit of measurement of short delay intervals, equal to 1/1000 of a second.
**MILLISECOND DETONATOR.** – Short delay detonator with less than 100 ms delay between subsequent numbers.
**MISFIRE.** – Charge, or part of charge, which has failed to fire as planned.
**MUCKPILE.** – Pile of broken rock which is the result of the blast.
**MUD CAP.** – See concussion charge.

**NITROGLYCERIN.** – Explosive oil originally used as sensitizer in dynamites.

**OD-BLASTING.** – Drilling and charging through the overburden. Mainly used in underwater blasting. (**OD** – **O**verburden **D**rilling).
**OHM METER.** – Used to check the resistance of a single electric detonator, detonators in series and parallel and to check the final round. Has to be approved by the authorities for use in blasting operations.
**OVERBREAK.** – Excessive breakage of rock beyond the theoretical contour.
**OVERBURDEN.** – Useless material laying on top of a deposit of useful material. Also referred to as the earth laying on top of the rock in construction blasting, road cuts etc.
**OXIDIZER.** – Ingredient in an explosive or blasting agent which supplies oxygen, which combines with the fuel to form gaseous products of detonation.
**OXYGEN BALANCE.** – State of equilibrium in a mixture of fuels and oxidizers

in which the fumes from the detonation are mainly carbon dioxide, water vapor and free nitrogen. (Harmless fumes.)

**PARALLEL HOLE CUT.** – Tunnel cut with all holes parallel and perpendicular to the rock face. The uncharged hole (holes) is (are) normally larger than the blastholes.

**PARTICLE VELOCITY.** – Measure of ground vibration. The velocity at which a particle of ground vibrates when hit by a seismic wave.

**PLASTER SHOT.** – See concussion charge.

**PLAIN DETONATOR.** – Detonator designed to be fired by the flash from a safety fuse, and used only with safety fuse.

**POWDER FACTOR.** – See specific charge.

**PREMATURE.** – Charge detonating earlier than intended.

**PRESHEARING.** – See presplitting.

**PRESPLITTING.** – Blasting of closely spaced holes along the perimeter of the excavation. The presplit is fired before the main blast. One of the methods for controlled blasting.

**PRILL.** – Small porous sphere of ammonium nitrate used for the manufacture of ANFO.

**PRIMARY EXPLOSIVE.** – Explosive, sensitive to spark, friction, impact or flame, which is used in a detonator to initiate the explosion.

**PRIMER.** – Cap-sensitive cartridge of high explosive which is used to initiate blasting agents.

**PRIMER CARTRIDGE.** – Cartridge in which the detonator is placed.

**PROPAGATION.** – Detonation of explosives charges by impulse from explosive charge located nearby. See flash over.

**PROPAGATION VELOCITY.** – Velocity of the ground shock wave.

**PULL.** – See advance.

**RAISE SHAFT.** – Tunnel or shaft excavated from a lower to a higher level with an inclination of at least 45°.

**RELIEVERS.** – See cut easer holes.

**ROUND.** – Group or set of blastholes forming a blast, when connected to each other.

**SAFETY FUSE.** – Core of black powder covered by textile and water-proofing material, which is used to initiate plain detonators.

**SCALED DISTANCE.** – Ratio used to predict ground vibrations.

**SCALING.** – Cleaning the rock surface from loose rock after blasting.

**SECONDARY BLASTING.** – Blasting of oversized boulders from previous blast.

**SENSITIVENESS.** – Explosive's ability to propagate a detonation.

**SENSITIVITY.** – Explosive's susceptibility to detonation upon receiving an external impulse such as impact, flame or friction.

**SENSITIZER.** – Ingredient used in an explosive to ease initiation and propagation of detonation.

**SHELF LIFE.** – Length of time an axplosive can be stored without losing its performance properties.

**SHOT FIRER.** – Person who actually fires the blast. He/she is assigned to control the blasting operation with authority to decide charge weights, delay patterns, etc.

**SINK SHAFT.** – Underground shaft excavated vertically downwards.

**SLURRY EXPLOSIVE.** – High density aqueous explosive containing ammonium nitrate, sensitized with a fuel, thickened and crosslinked to a gelatinous consistency. Also called watergel.

**SMOOTH BLASTING.** – Method of controlled blasting in which closely spaced holes are drilled at the perimeter of the excavation, charged with low charges to reduce overbreak. The perimeter holes are fired with a higher delay number than the rest of the round.

**SOCKET.** – See butt.

**SPACING.** – Distance between blastholes in a row.

**SPECIFIC CHARGE.** – Explosives consumption per cubic meter of rock.

**SPECIFIC DRILLING.** – Drilled meters per cubic meter of rock.

**STEMMING.** – Inert material used in the collar part of the blasthole to confine the gases from the detonation.

**STRIPPING.** – Removal of overburden.

**STUMP.** – Unbroken rock within the final contour.

**SUBDRILLING.** – Part of the blasthole below the planned grade or floor level.

**SWELLING.** – Difference in volume of a material in its solid state and when it is broken.

**SYMPATHETIC PROPAGATION.** – See flash over.

**TAMPING.** – Act of compressing the explosive in a blasthole.

**TAMPING ROD.** – Rod of wood or plastic used to introduce and tamp explosives in a blasthole.

**TRUNKLINE.** – Detonating cord line used to connect the downlines in a blasting round.

**TUNNEL.** – Horizontal underground excavation.

**TUNNEL FACE.** – Rock face at the end of the tunnel.

**UNCONFINED VELOCITY OF DETONATION.** – Velocity of detonation of an explosive not confined in a blasthole or other confining medium.

**V-CUT.** – Tunnel cut with the holes in V-layout. Also called wedge cut.

**VELOCITY OF DETONATION.** – VOD, velocity at which the detonation wave travels through the explosives column. May be measured confined or unconfined.

**VIBRATION VELOCITY.** – Unit of ground vibration in mm/sec.

**VOLUME STRENGTH.** – See bulk strength.

**WATERGEL EXPLOSIVE.** – See slurry.
**WATER RESISTANCE.** – Ability of an explosive to withstand water exposure without becoming deteriorated or desensitized.
**WEB.** – Rock mass between presplitting holes.
**WEIGHT STRENGTH.** – Strength of a given weight of an explosive compared with the equivalent weight of Blasting Gelatine.
**WINZE.** – See sink shaft.

# 3. BLASTING PRODUCTS

## 3a. Explosives.

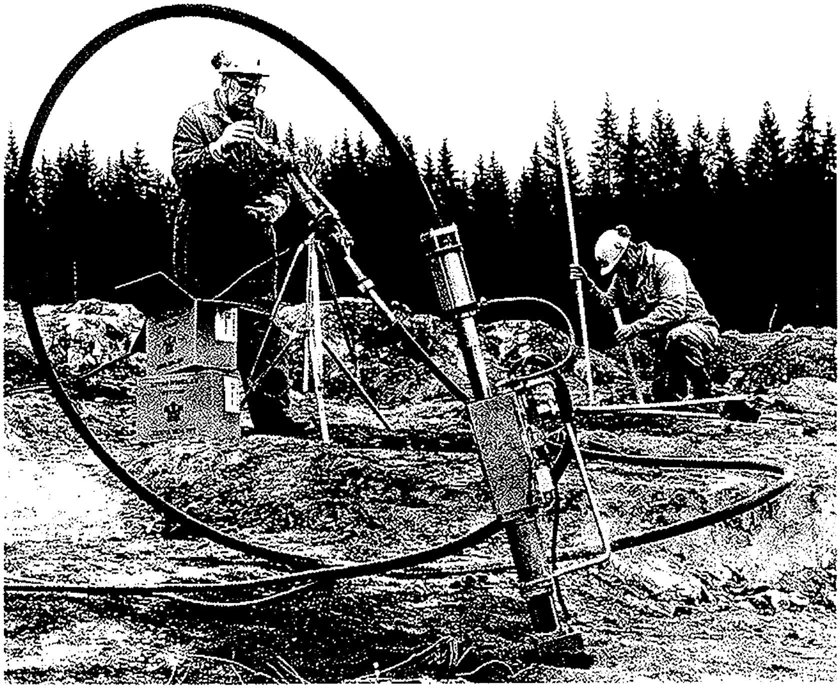


*Fig. 3a.1 Charging of bench with Dynamex M.*

### 3a.1 Properties.

Different working conditions have made it necessary to manufacture explosives with differing properties appropriate to the varying purposes for which they are used.

In the ideal conditions of dry blastholes a simple explosive can be used, while under wet conditions, more sophisticated products are called for.
To be able to select the ideal explosive for each purpose the basic properties of the available explosives have to be considered.
The most important characteristics of an explosive are:

* velocity of detonation (VOD)
* strength
* detonation stability
* sensitiveness (propagation ability)
* density
* water resistance
* sensitivity
* safety in handling

* environmental properties
* resistance to freezing
* oxygen balance
* shelf life

**The velocity of detonation** (VOD) is the speed at which the detonation travels through the explosive.
The velocity of detonation of an explosive is higher when the explosive is confined than unconfined.
Recent research in the U.S.A. confirms the old assumption that the detonation velocity should be equal to the velocity of the seismic shock wave through the rock. High velocity of detonation would thus be favorable in the case of hard rock. Explosives like Dynamex M and Emulite are suitable for hard rocks like granite, gneiss and basalt while ANFO is suitable for softer rocks like limestone and sandstone.

High velocity of detonation is of the greatest importance for boulder blasting with concussion charges and underwater blasting with shaped charges. The high velocity is necessary to give the powerful impact at detonation which is required to cause the tensile stresses which break the rock.

**The strength** of an explosive is in most cases expressed as a percentage of the strength of blasting gelatine. The **weight strength** denotes the strength of any weight of an explosive compared with the same weight of blasting gelatine. The bulk or **volume strength** denotes the comparison of any volume of an explosive with the same volume of blasting gelatine. Blasting gelatine was chosen as a standard, as it is well known all over the world and is the most powerful civil explosive. Lately the manufacturers have started to compare the weight and volume strengths with those of ANFO. It is a natural development as blasting gelatine has been phased out as a civil explosive in nearly all countries and ANFO has become the most widely used and well known explosive. In order to measure the strength of an explosive, different tests can be carried out, e.g.:

* Lead block test
* Ballistic mortar test
* Bubble energy test
* "Nitrodyn"

**The lead block test** is the oldest test method and is still used. A small amount of explosive is detonated in a hole in a lead block. By measuring the volume of the cavity produced by the detonation, an indication of the blasting effect can be obtained.

**The ballistic mortar test.** A small amount of explosive is detonated in a steel cylinder which is fixed to a pendulum. The pendulum will swing away from the detonation and the deflection angle of the swing indicates the blasting effect.

**The bubble energy test** was proposed some years ago as a means of comparing the relative strength of different explosives. By detonating an amount of explosive under water and measuring the shock energy and the bubble energy the strength of the explosive can be calculated.

**"Nitrodyn"** is a theoretical evaluation of the available energy in an explosive to obtain an indication of its strength.
In Sweden, the strength of an explosive is compared to the strength of Dynamex M, which has a weight strength of 78 percent compared to that of blasting gelatine.

**Detonation stability** means that the detonation goes through the entire explosives column.

An explosive's **sensitiveness** or propagating ability is expressed in the length of the air-gap over which a donar cartridge of an explosive will detonate a receptor cartridge under unconfined conditions.
The sensitiveness is an important property which has to be considered in blasting operations. If the sensitiveness is low, there can be interruptions in the detonation if the column of explosive in the charged blasthole is not continuous or some obstacle has come between the various charges. An explosive with too high a sensitiveness can cause propagation between adjacent blastholes if the holes are closely spaced. Especially in faulty rock and in underwater blasting the risk of propagation between the blastholes is great. The propagation ability is higher in confined conditions than in unconfined ones.
**The density** of an explosive is its specific weight expressed as kilograms per liter (kg/l) or grams per cubic centimeter (g/c.c.). The density determines the possible charge concentration in the blasthole. The density of an explosive is one of the most important properties to be considered when designing blasting operations. The drilling pattern will be considerably more widely spaced if the high density Dynamex M is used instead of the low density ANFO.

**The water resistance** is an explosive's ability to withstand water penetration and is normally expressed as the time the product can be underwater and still detonate reliably. The water resistance of an explosive depends on the packing as well as its inherent ability to repel water. An explosive can be affected by water in two different ways. Salts can be dissolved in water and leak out of the explosive and the water pressure can reduce the size and amount of air bubbles, which act as "hot spots", resulting in the explosive becoming desensitized.
Plastic explosives normally have high resistance to water. Emulsion explosives like Emulite have excellent water resistance properties, as the salts are protected by an oil/wax film and the "hot spots" are produced from air filled microballoons.
Explosives which have no inherent water resistance properties can be used in water filled blastholes if proper packaging material, such as plastic bags, are used.
Except for Emulite, with its outstanding water resistance properties, Dynamex AM (Nitro Nobel's underwater explosive) is guaranteed to withstand water

pressure for one week and Dynamex M for 24 hours. On the other hand, powder-form nitroglycerin explosives in paper cartridges are not guaranteed to last in water for more than one hour. ANFO poured into water filled blastholes will deteriorate very quickly.
With the wide range of explosives available, it is a matter of planning to choose the explosive with the water resistance properties most suitable for the specific job.

The **sensitivity** of an explosive is expressed as the minimum energy needed to initiate the explosive.

Civil explosives are divided into:

* cap sensitive explosives
* non-cap sensitive explosives

The cap sensitive explosives can be initiated by a #6 or #8 blasting cap. The manufacturer indicates the sensitivity of his product.
The non-cap sensitive explosives need to be primed with an amount of high explosives in order to obtain initiation and stable detonation.

**Safety in handling** is of the utmost importance as the transportation and usage of an explosive should be carried out without any risks for the personnel involved. Before an explosive is approved by the authorities it is subjected to extensive tests.

1) **The drop hammer test** determines the height from which a weight must fall on the explosive in order to create a detonation.
2) **The friction test,** is a test in which friction under increased pressure is applied to a small amount of explosive. When a reaction in the explosive is obtained, the pressure is recorded.
3) **The projectile impact test** determines the bullet velocity needed to create a reaction in the explosive.
4) **The heat test** determines how much heat an explosive can withstand before a reaction starts.

The tests form the basis for the authorities to classify the various explosives from the point of view of handling and transport.

**The environmental properties** are more and more taken into consideration. The aim is to minimize the toxic fumes and such negative side-effects as headaches and skin irritation when handling nitroglycerin explosives.
The gases produced from a detonation of a civil explosive are principally carbon dioxide, nitrogen and water vapor, which are all non-toxic. Various toxic gases are also produced like carbon monoxide, oxides of nitrogen, and nitroglycerin vapors. The fumes' characteristics differ greatly between different kinds of explosives. No matter which explosive is used, some noxious gases will be produced in the detonation.
For "open-pit" operations the toxic fumes rarely create any problem, but for

underground operations, it is essential that noxious gases are kept to an acceptable level. Increased amounts of fumes can be produced if explosives with insufficient water resistance are used. Inadequate priming, poor confinement, use of wooden spacers, and incomplete explosion are other causes of an increased production of noxious fumes.

It is important that time for sufficient ventilation is allowed in underground operations as some of the toxic gases are odorless. Too early a return to the blasting site may be fatal.

The headache caused by nitroglycerin explosives is a side effect that causes a lot of inconvenience to many of those exposed to the explosive. It is difficult to protect oneself from nitroglycerin vapors, which enter the blood system via the respiratory organs or by direct contact with the skin, thus lowering the blood pressure.

The new water based explosives, like Emulite, have very good fume characteristics and the advantage of "non-headache" properties.

**Resistance to freezing** is important in countries where the temperature falls below 0° C. Dynamites and watergels become stiffer in low temperatures and lose their good tamping characteristics while emulsion explosives retain their excellent tamping characteristics even at the lowest temperatures. Modern explosives will not freeze under normal exposure to the lowest temperatures encountered in normal working conditions. The explosives will thus work in the coldest weather, in which it is possible to work, without the hazards of thawing them before use.

**The oxygen balance** must be considered in underground applications. An excess of oxygen in the explosive can form nitrogen oxides ($NO$ and $NO_2$) and a deficit of oxygen will form carbon monoxide ($CO$).

Those gases are toxic and exposure to them may be fatal. In open air blasting these gases rarely cause any problem as the blasting fumes are rapidly dispersed after the detonation.

**The shelf life** of the explosive is very important as the explosive frequently has to be kept for a long time in storage, often under unfavorable conditions.

Plastic nitroglycerin explosives undergo a normal aging process during storage. The air bubbles in the explosive disappear partly or wholly, thus decreasing its sensitivity to initiation and its propagation ability but not its energy content.

Plastic nitroglycerin explosives should not be stored in high temperatures, as they tend to soften and the salts in the explosive penetrate the paper wrapping of the cartridges, thus deforming them. Storage temperatures around +32° C should be avoided, especially if the temperature fluctuates around that figure. The ammonium nitrate in the explosive undergoes a physical rearrangement making the explosive in the cartridge swell, deforming the cartridge. The blasting effect is not affected. Powder type explosives in cartridges are sensitive to moisture. In a humid environment, the salt in the explosive tend to form deposits on the cartridge, thus hardening it.

ANFO is sensitive to humidity and cakes easily when stored under such conditions.

### 3a.2 Classification.

The explosives used in civil engineering and mining can nowadays be classified as:

* High explosives
* Blasting agents

**High explosives** are characterized by high velocity of detonation (VOD), high pressure shock wave, high density and by being capsensitive.

**Blasting agents** are mixtures consisting of a fuel and oxidizer system, where none of the ingredients are classified as an explosive and when unconfined cannot be detonated by means of a #8 test blasting cap. Blasting agents have to be initiated by a primer. ANFO is a typical blasting agent.

### 3a.3 Description of explosives presently in civil use.

The development of high explosives has undergone three generation alternations towards safer products:

* 1st generation. Dynamites, sensitized by nitroglycerin.
* 2nd generation. Watergels, sensitized by TNT, methylamin nitrate (MAN) or other explosive components.
* 3rd generation. Emulsion explosives, sensitized by plastic or glass micro spheres.

The sensitization of the explosive is the most critical operation in the manufacturing process and the selected sensitizer influences not only the manufacture but also the handling of the finished product. During a full century of manufacture, nitroglycerin-based explosives have been the subject of many disastrous accidents. The manufacture and transport of MAN has also proved to have certain unpredictable elements in it causing unexpected explosions.
It was a gigantic step towards a safer product when emulsion explosives were invented* and thus the sensitizing agent for the first time in history was a substance which in itself was not an explosive. That the emulsion explosives also have strengths comparable to that of the preceding two generations and better tamping characteristics make them an equal alternative for all kinds of rock blasting.

*By Atlas Powder U.S.A.

**Nitroglycerin based explosives.**

The invention of dynamite in 1866 tamed nitroglycerin and made it possible to handle it in a reasonably safe way. However, the miners who had got used to the stronger nitroglycerin complained, and when Alfred Nobel in 1875 discovered that nitroglycerin could be dissolved in nitro cellulose, they got the new strong explosive they had been asking for. Nobel called the nitro-cotton explosive blasting gelatine.

The industrial use of blasting gelatine has rapidly declined since WWII due to high cost, sensitivity to friction and shock and its high inflammability properties. Over the years ammonium nitrate has become a more important ingredient in dynamite replacing a large portion of nitroglycerin. In Sweden blasting gelatine, which originally contained 92 % nitroglycerin and 8 % nitro cellulose, has gradually been replaced with explosives with a lower nitroglycerin content. In today's Dynamex M, ammonium nitrate has replaced much of the nitroglycerin which now makes up only 20 % of the ingredients. To add further safety to the product, the nitroglycerin in Dynamex M is flegmatized.

**Nitroglycerin based products:**

**Dynamex AM** is specially intended for underwater blasting. The cartridges can lie in water for a week without being damaged.
Dynamex AM is delivered in paper cartridges, suitable for mechanized charging and in screw mount plastic tubes for through-the-rod charging.

**Dynamex M** is the most versatile explosive in the Nitro Nobel range of nitro-

*Fig. 3a. 2 Dynamex M.*

glycerin explosives. Its good fume characteristics make it suitable for severe environmental conditions.

Dynamex M is delivered in:

* paper cartridges
* plastic hoses
* plastic pipes

The **paper cartridges** are suitable for all types of surface and underground operations where small diameter blastholes are used. Dynamex M in paper cartridges is suited for mechanized charging, for better packing and utilization of the blasthole.

The **plastic hose** charge is a rational alternative for all kinds of bench blasting.

The **plastic pipe** charge provides a high charging capacity with a well balanced charge concentration. In tunnel blasting, a complete round can be charged very quickly using pipe charges, thus eliminating over-charging.

*Fig. 3a.3 Gurit.*

**Gurit** is a nitroglycerin sensitized powder explosive, which was developed exclusively for contour blasting. It is manufactured in firm plastic tubes with small diameters (11, 17, and 22 mm). The tubes are fitted with connector sleeves, which enable fast and easy assembly of the charges when charging. The Gurit charge connection method and the small diameters employed, give a low and evenly distributed charge concentration throughout the blasthole, which is of paramount importance in contour blasting.

Applications:

* smooth blasting of rock contours
* presplitting of rock contours
* cautious blasting when there are ground vibration problems
* trench blasting, where high quality trench walls are required
* demolition of buildings and structures

**Primex.**

Primex charges were developed for blasting operations where extreme caution was required. Primex is designed mainly for mini-hole blasting.
The charges have a length of 150 mm and a diameter of 17 mm. The weight of the cartridge is 52 gr. The suitable length/weight is obtained by cutting the tube. A special sheath may be adapted to the cut piece of tubing in order to hold the detonator in the correct position.
The mini-hole method may with great advantage be used for the following blasting tasks:

* boulder blasting
* leveling
* pipe and cable trenching
* pylon and pole footings
* demolition blasting

**Concussion charge M S1M.**

M S1M contains the same explosive as Primex. Its plastic consistancy and high detonation velocity makes it ideal for plaster shots where the charges are moulded to the surface of the object to be blasted and high velocity is needed for best concussion effect.
Concussion charge M S1M is principally used for the blasting of natural boulders and for secondary blasting.

**Watergel explosives** have a gel-like consistency obtained by adding thickening agents to water. Their main explosives base is ammonium nitrate which is an oxidizer. This type of explosives contains 10 to 30 per cent water and is sensitized by carbonaceous fuels, TNT, aluminum or certain organic compounds like methylamin nitrate. Both cap sensitive and non-cap sensitive watergel explosives are available.
Watergel explosives are also called slurry explosives.

*Fig. 3a.4 Concussion charge M 51M.*

**Emulsion explosives** are composed of separate, very small drops of ammonium nitrate solution and other oxidizers, densely dispersed in a continuous phase, which is composed of a mixture of mineral oil and wax.
The oil/wax mixture, which is the fuel, is in this way given a very large contact surface to the oxidizer, the ammonium nitrate solution. What distinguishes the emulsion explosives from other liquid and plastic explosives, is that they can be made to detonate without the addition of a sensitizer which in itself is an explosive. To make the emulsion initiable, small cavities are mixed in, i.e. in form of microballoons with a diameter of about one tenth of a millimeter. These collapse under the influence of the initiating shock wave from the blasting cap, creating a multitude of local hot spots where the temperature is sufficiently high to start a fast explosive combustion of the explosive.
The density of the explosive and its capacity of initiation can be adjusted with the amount of microballoons in the emulsion.
The strength is regulated by the amount of the additive fuel, aluminum, that is added. Furthermore, there are considerable possibilities to vary the consistency

to fit different purposes. This can be determined mainly by the proportion oil/wax. With a high percentage of wax, a margarine-like consistency is obtained. With more oil, grease-like pumpable qualities can be obtained.
The margarine-like type of emulsion explosives is best suited for cartridged explosives. The tamping characteristics are excellent making it possible to utilize the blasthole volume to almost 100 %.
Because of the physical nature and properties of emulsion explosives, they retain their consistency over a wide temperature range, the tamping and pumping characteristics are virtually unchanged from −20° to +35° C.
The stability of emulsions is outstanding, compared with other civil explosives. The detonation properties remain unchanged over long periods of time under normal storage conditions.
The velocity of detonation is high for emulsion explosives but may decrease somewhat if the diameter is decreased or aluminum is added.

Thanks to the fact that the water soluble drops of ammonium nitrate in the emulsion are completely surrounded by an oil/wax film, the explosive becomes water-repellent, thus being highly water resistant.
The sensitivity of emulsion explosives may vary from a high explosive which may be initiated by a #8 strength detonator to blasting agent products requiring a primer for initiation.
From a handling point of view, the emulsion explosives are very safe and a high degree of impact is needed for accidental initiation.

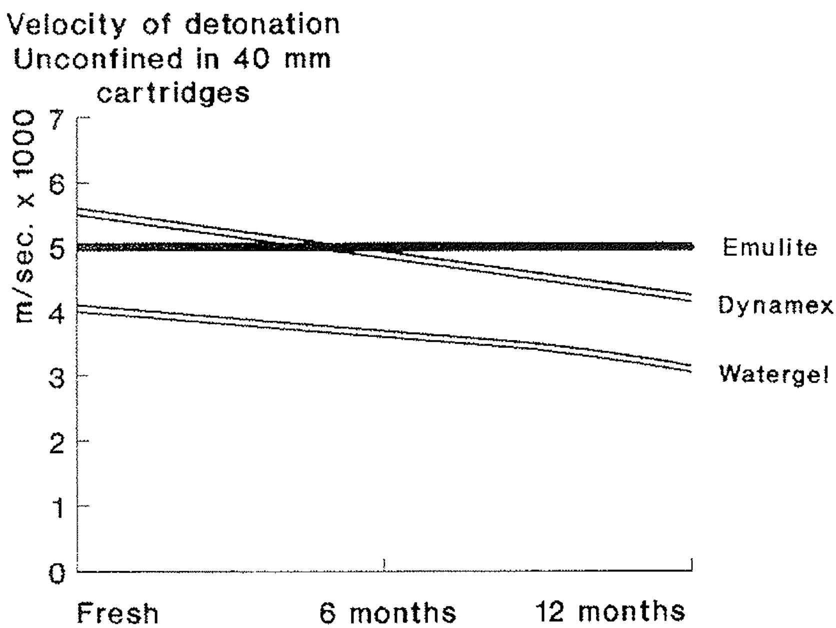



*Fig. 3a.5 Comparison of velocity of detonation with respect to storage time.*

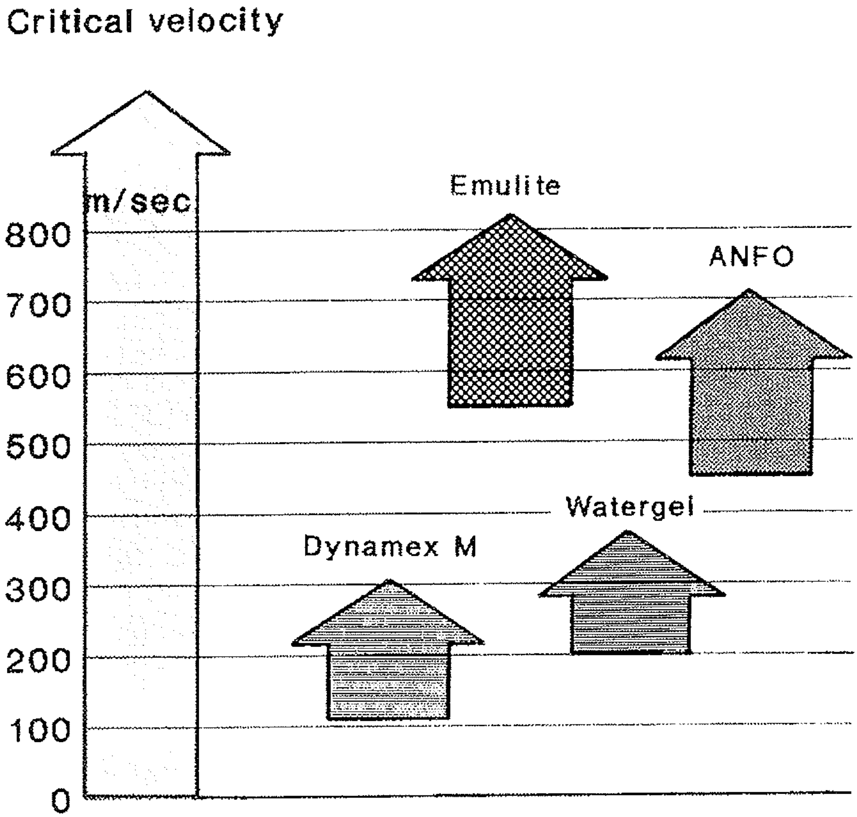



*Fig. 3a.6 Projectile impact test.*

**Emulsion explosives products.**

By using different percentages of microballoons and aluminum, a wide range of emulsion explosives can be manufactured.
The cap sensitive range of Emulite 100 and 150 are intended for small and medium diameter blastholes and are delivered in papershells and plastic bags. The non-cap sensitive range of Emulite 200 and 300 are intended for medium and large diameter blastholes in bench blasting, and are delivered in plastic hoses. Of real interest are the pumpable bulk emulsions Emulite 1200 and 1300 which are economical alternatives to ANFO in quarrying and mining.
**EMULITE** explosives are at present manufactured in four qualities.
**Emulite 100,** a cap sensitive emulsion explosive which is manufactured in paper cartridges or plastic hoses. Its good fume characteristics and water resistance makes it an excellent all-round explosive. It is suitable for mechanized charging.

*Fig. 3a.7 Emulsion explosives.*

Emulite 100 can also be supplied in plastic pipes with dimensions 20×500 mm for presplitting and smooth blasting.

**Emulite 150** is similar to Emulite 100 but aluminum is added to increase the energy content. It is manufactured in paper cartridges, plastic hoses and plastic pipes.

**Emulite 200** is a non-cap sensitive emulsion explosive which is intended for bench blasting with medium and large size blastholes. It is supplied in plastic hoses. A cartridge of Emulite 100 or 150 will serve well as a primer. The Emulite 200 is not compatible with detonating cord.

**Emulite 300** is the cord compatible variety of non-cap sensitive emulsion explosives.

All types of EMULITE are suitable for underwater blasting. EMULITE can be manufactured in a pumpable version upon request.

## ANFO.

**ANFO** is the most widely used civil explosive in the world. It is considered a blasting agent and has to be initiated by a primer. ANFO is a mixture of prilled **A**mmonium **N**itrate and **F**uel **O**il at a ratio of 94/6.

The primer used to initiate the ANFO should have a diameter which is close to the blasthole diameter and a length long enough to ensure stable detonation. (See Chapter 8.2.1 Priming of ANFO.)

A velocity of detonation less than 2000 m/sec. is not considered stable. Tests

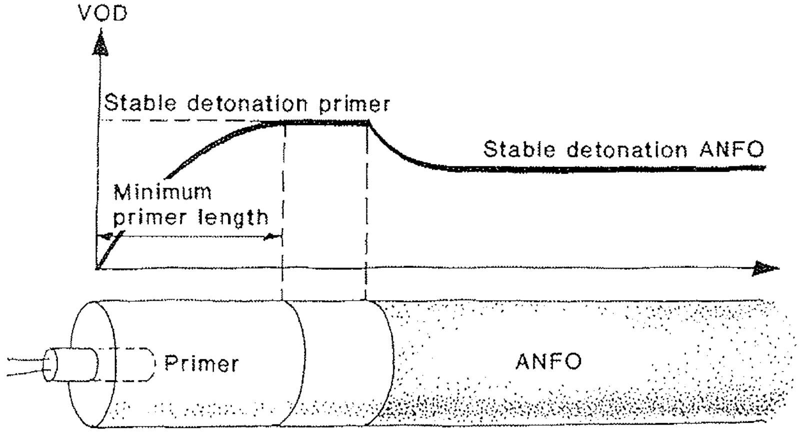



*Fig. 3a.8 Effect of the primer on velocity of detonation.*

made by SVEDEFO* show that a Dynamex M primer cartridge initiates ANFO directly to its full velocity. The same result will be obtained with an Emulite 100 primer, provided that its diameter is close to the blasthole diameter.

The velocity of detonation changes with the diameter of the blasthole and reaches its highest velocity of 4400 m/sec. in a 250 mm blasthole. The velocity of detonation decreases with the diameter of the blasthole and when the diameter is less than 25 mm, the detonation will not be stable. ANFO is most suitable in middle and large diameter blastholes (75 to 250 mm) under dry conditions.

Initiation of ANFO should not be made with detonating cord in small and medium size blastholes (25 to 100 mm). The detonating cord will initiate the ANFO diametrically (axial priming) and as the ANFO will not reach a stable velocity of detonation (2000 to 4400 m/sec.), the chemical reaction will be incomplete. That this is the case is confirmed by the use of ANFO and detonating cord for smoothblasting purposes which has been reported from different projects.

ANFO has poor water resistance and should, where water is present in the blastholes, be protected by plastic hoses.

The apperance of orange-brown fumes upon detonation is a sign of water deterioration and an indication that a more water resistant product should be used or that the ANFO should be packed in plastic bags of better quality.

When the ANFO is packed in plastic hoses, consideration has to be given to the design of the drilling pattern as the explosives column will have smaller diameter in hoses than if poured into the blasthole. A narrower drilling pattern is needed.

***Swedish Detonic Research Foundation.**


*Fig. 3a.9 Prilled ANFO*

**Water resistant ANFO.**

As mentioned above, one of the main problems with ANFO is its poor water resistance properties. Nitro Nobel has developed a water resistant ANFO which is marketed under the tradename **AKVANOL.**

AKVANOL is manufactured from different types of AN mixed with fuel oil. In addition, a component is mixed in, which forms a gel when it comes in contact with water. The water resistance properties of AKVANOL depend upon the thickening agents ability to swell and form a gel. AKVANOL should preferably be charged into the blasthole with a charging machine, starting to fill the hole from the bottom.

**EMULET**

ANFO is used in many underground applications, but has been found to be too powerful for the contour, with overbreak as a result. Sometimes it is impractical to use special smoothblasting explosives or smaller hole diameter in the peri-

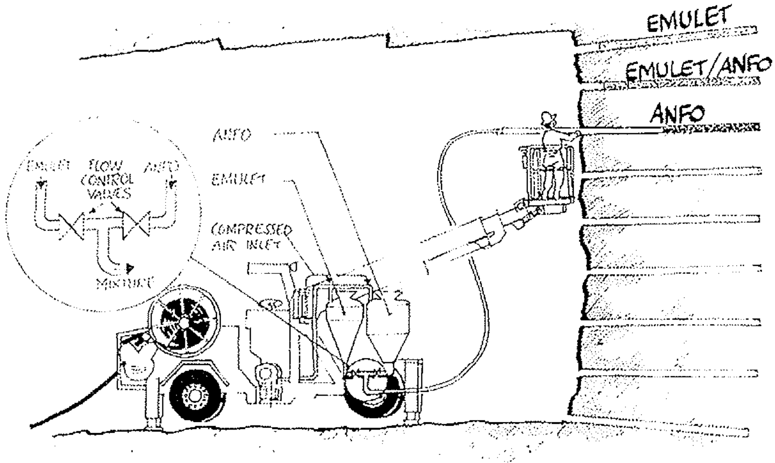



*Fig. 3a.10 With Emulet and ANFO, the density and bulk strength can be varied at the worksite.*

meter holes. It has therefore been a demand for an ANFO with reduced strength. Different methods of mixing the ANFO with inert material have been tried, but today the most commonly used material for reduction is expanded polystyrene spheres. Due to the difference in density, ANFO 0.8 g/c.c. and polystyrene 0.02 g/c.c., the two components in the mixture tend to separate during the charging operation and the polystyrene spheres are blown out of the blasthole.

By adding bulk emulsion to the mixture, a homogeneous blasting agent is obtained, which does not separate and is chargeable with penumatic charging machines, its tradename is Emulet.

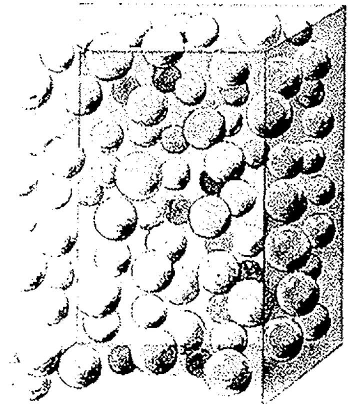
Four blends of Emulet are presently available, Emulet 20, 30, 40 and 50. The figures denote the bulk strength in percentage of ANFO.

Emulet 20 has a bulk strength comparable to that of Gurit 17 mm in a 38 mm blasthole and Emulet 30 is comparable to Gurit 22 mm in a 51 mm blasthole.

For efficient charging of the round, two charging containers should be used, one for ANFO and the other for Emulet. By a control device, the blaster can switch from one container to the other depending on which part of the round he is charging, the contour or the main round. It is also possible to mix ANFO and Emulet in the same hole, if an explosive with a bulk strength between ANFO and Emulet is required, e.g. in the row closest to the contour or in a tunnel cut. See Fig. 3a. 10.

**EMULAN.**

Emulan is a hybrid product, which is a mixture of ANFO and bulk Emulite. In the mixed product, the air spaces between the prills in the ANFO are filled with emulsion explosive, resulting in a sound increase in both energy and density.



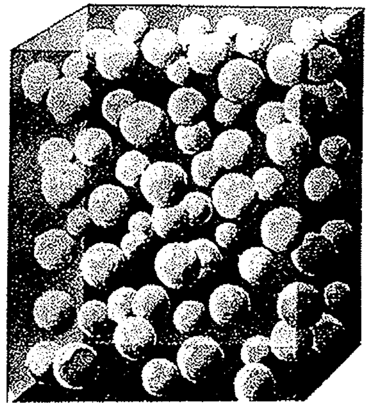


*Fig. 3a.11 ANFO* *EMULAN*

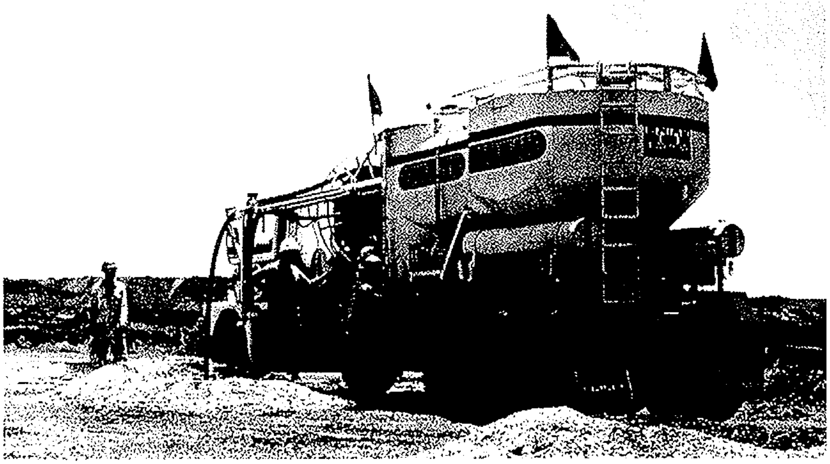


*Fig. 3a.12 Mobile EMULAN plant.*

As the airspaces between the ANFO prills are filled with Emulite with excellent water resistance properties, blasting agents with good to excellent water resistance can be manufactured.

The ratio Emulite/ANFO may be varied from 20/80 in dry conditions to 80/20 in extremely wet conditions.

Another advantage of filling up the airspaces between the ANFO prills is that density is increased with an explosive material, thus drastically increasing the energy content.

Due to the higher density and higher energy content, up to 40 % more rock can be blasted per drillmeter compared with ANFO. The burden and spacing can both be increased by 20 %.

In most cases EMULAN has proved to be an economic alternative to ANFO, especially in severe water conditions.

**Special products.**

**FRAGMEX** is a shaped charge, specially developed for underwater and seismic operations. Using shaped charges, underwater rock blasting can be carried out with better technical and economical results at bench heights of less than 1.5 m.

*Fig. 3a.13 Fragmex.*

**Nobel Prime** is a primer with very high velocity especially developed to initiate ANFO.

*Fig. 3a.14 Nobel Prime.*

### 3a.4 Explosives products

The strength of the explosives are in comparison with ANFO.

| PRODUCT | Density gr/c.c. | Weight strength per cent of ANFO | Bulk strength per cent of ANFO | Velocity of detonation m/sec confined | Water resistance |
|---|---|---|---|---|---|
| Dynamex AM | 1.40 | 127 | 192 | 6.000 | Excellent |
| Dynamex M | 1.40 | 121 | 181 | 5.000 | Excellent |
| Gurit | 1.00 | 85 | – | 4.000 | Fair |
| Emulite 100 | 1.20 | 78 | 110 | 5.300 | Excellent |
| Emulite 150 | 1.21 | 113 | 145 | 5.100 | Excellent |
| Emulite 200 | 1.25 | 78 | 115 | 4.900 | Excellent |
| Emulite 300 | 1.28 | 76 | 100 | 4.900 | Excellent |
| Emulite 1200 | 1.25 | 78 | 130 | 4.900 | Excellent |
| EMULAN 5000 | 1.30 | 88 | 140 | 5.000 | Good |
| ANFO | 0.80 | 100 | 100 | 2.500 | Poor |
| Emulet 20 | 0.22 | 74 | 20 | 1.850 | Poor |
| Emulet 30 | 0.33 | 81 | 30 | 2.000 | Poor |
| Emulet 40 | 0.40 | 86 | 40 | 2.200 | Poor |
| Emulet 50 | 0.50 | 89 | 50 | 2.650 | Poor |
| Primex | 1.50 | 127 | 208 | 6.000 | Excellent |

Special products.
Fragmex.
Nobel Prime.

Dynamex AM, Dynamex M, Gurit, Primex, Fragmex and Nobel Prime are trademarks of Nitro Nobel AB, Sweden.

Emulite, EMULAN and Emulet are trademarks of Nitro Nobel AB, Sweden and its licensees.

Dimensions, weights and packing:

| EXPLOSIVE | SIZE mm | WEIGHT APPROX. | PACKING |
|---|---|---|---|
| Dynamex AM | 29×200 | 175 g | Paper cartridge |
| | 40×200 | 330 g | –"– |
| Dynamex M | 22×200 | 100 g | –"– |
| | 25×200 | 125 g | –"– |
| | 29×200 | 175 g | –"– |
| | 40×200 | 320 g | –"– |
| Dynamex M | 50×550 | 1.4 kg | Plastic hoses |
| | 55×550 | 1.7 kg | –"– |
| | 65×550 | 2.4 kg | –"– |
| | 80×400 | 2.7 kg | –"– |
| | 90×375 | 3.0 kg | –"– |
| | 125×375 | 5.4 kg | –"– |
| Dynamex M | 25×1110 | 0.74 kg | Plastic pipes |
| | 29×1110 | 0.98 kg | –"– |
| | 32×1110 | 1.2 kg | –"– |
| | 39×1110 | 1.6 kg | –"– |
| Gurit | 11×460 | 50 g | Plastic pipes |
| | 17×500 | 115 g | –"– |
| | 22×725 | 310 g | –"– |
| Emulite 150 | 22×200 | 90 g | Paper cartridges |
| | 25×200 | 110 g | –"– |
| | 29×200 | 150 g | –"– |
| | 40×200 | 280 g | –"– |
| Emulite 150 | 25×1110 | 0.64 kg | Plastic tubes |
| | 29×1110 | 0.86 kg | –"– |
| | 32×1110 | 1.05 kg | –"– |
| | 39×1110 | 1.54 kg | –"– |
| Emulite 150 | 43×550 | 0.90 kg | Plastic hoses |
| | 50×550 | 1.30 kg | –"– |
| | 55×550 | 1.50 kg | –"– |
| | 60×550 | 1.86 kg | –"– |
| | 65×550 | 2.20 kg | –"– |
| | 75×550 | 2.70 kg | –"– |
| Fragmex 4 | | 4.2 kg | Plastic cover |
| 8 | | 8.2 kg | –"– |
| Nobel prime | 25×150 | 100 g | Plastic tube |
| | 32×150 | 165 g | –"– |
| Primex | 17×150 | 52 g | Plastic tube |
| Concussion charge M | S1M | 0.5 kg | Plastic hoses |

## 3b. Firing devices.

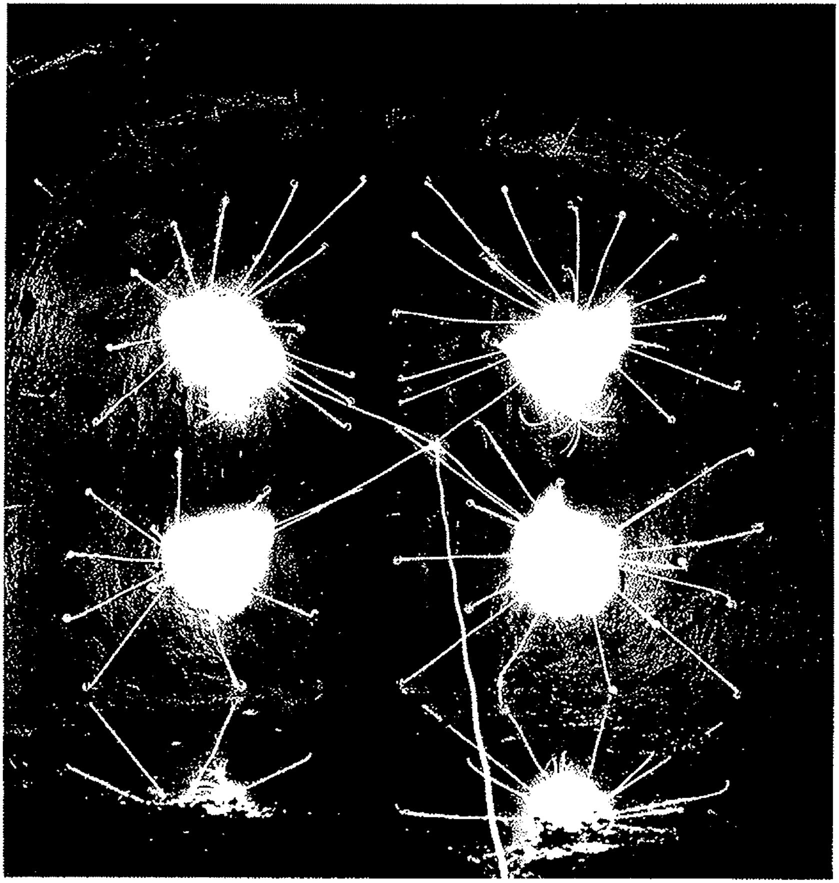


*Fig. 3b.1 Firing of tunnel round with NONEL.*

### 3b.1 General.

Before William Bickford invented the "Miners Safety Fuse" in 1831, the initiation of blasts was a hazardous task. A lot of more or less dangerous methods were applied to initiate black powder. With the invention of the safety fuse, the blaster was given the facility to initiate black powder with reasonable precision and reliability.

The increased use of nitroglycerin in the 1850s and of dynamite in the 1860s, made it necessary to supplement the safety fuse with a detonator, as the safety fuse alone could not initiate the new explosives.

Alfred Nobel's invention of the fulminate of mercury blasting cap in 1867 made the initiation of all explosives safer and more efficient.
Detonators in civil use have all been developed from Nobel's basic ideas.

The introduction of the electric milli-second detonator has been of utmost importance for the development of new blasting techniques, where throw, fragmentation and ground vibrations can be controlled in large blasting rounds. Electric initiation has been more widely accepted in Europe than in U.S.A., where non-electric firing methods are more commonly used.

Lately, a new non-electric milli-second detonator, NONEL, has come into wide use. Its inherent short delay characteristics are the same as for its electric counterpart, but the electric hazards have been eliminated by replacing the electric wires with a shock tube.

The firing methods can be divided into two main groups:

* Non-electric
  Safety fuse with plain detonator
  Detonating cord
  NONEL
* Electric detonators

### 3b.2 Firing methods.

The firing methods will be presented in historical order, starting with the safety fuse and ending with the sophisticated NONEL system.

### 3b.2.1 Safety fuse and plain detonator.

Initiation with safety fuse is a method which is increasingly being replaced by less time consuming and more sophisticated initiating methods. However, the method is still used in small operations, in secondary blasting and for stone and stump blasting, where it is still the simplest and most economical method.

The safety fuse consists of a black powder core which is tightly wrapped with coverings of textile and insulated against moisture by waterproofing materials like asphalt and plastics. The coverings act as protection for the black powder core against water, oil and other materials which can change the burning speed or desensitize the powder. The covering also prevents "side-spit" which can cause premature detonation if it sets fire to the explosives charge.

The safety fuse has a steady and well controlled burning speed, but as there are many brands of safety fuse in the world, the burning speed may differ between different brands. The burning rate of most brands in U.S.A. is 130 seconds per meter (120 sec/yard) at sea level with allowable variation of 10 seconds from standard. In Europe, the standard burning rate is 120 seconds per meter with the same variation.

It should be taken into account that the safety fuse will burn faster if it is subject to confinement or pressure, and that the use at high altitudes slows down the burning speed. Therefore, some occurrences that are referred to as "premature" or "delayed" detonations may not be due to inherent properties of the fuse, but to conditions on the work-site or poor storage or handling of the fuse.

As the burning speed does vary, the speed should be verified by accurate timing of a sample from each coil just before use.

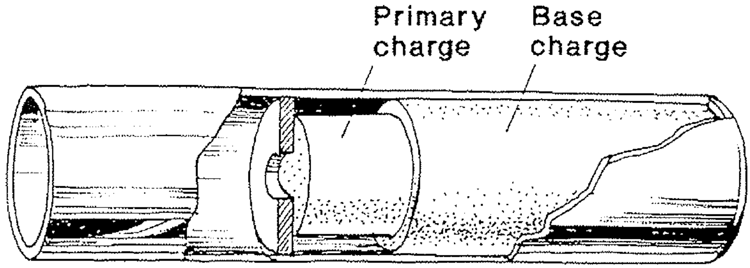



*Fig. 3b.2 Plain detonator.*

To initiate the explosive, a plain detonator has to be attached to the safety fuse. Due to varying sensitivity of different explosives, detonators of different strengths are available. The strength of the detonator is expressed in numbers, of which #6 and #8 are presently available on the market. The #8 detonator contains approximately 1.0 gram of high explosives and the #6 contains approximately 0.8 gram. As explosives have become safer to handle and thus less sensitive to impact, the #8 detonator has become more widely used.

The plain detonator consists of an aluminum or copper cylinder which is closed at one end. A charge of high explosive, like hexytol, tetryl or similar, is placed in the base of the cylinder. On top of the base charge, a primary charge is placed normally lead azide. The primary charge is sensitive to initiation by the endspit of the safety fuse and subsequently initiates the base charge.

Assembly of plain detonators to a safety fuse:

* cut the fuse so the black powder core is visible.
* cut the end of the fuse squarely and introduce it gently into the detonator against the primary charge – leave no airgap. Slanting cuts must be avoided as the tapered end can fold over and block the endspit.
* crimp the detonator thoroughly to the fuse with a crimper. Two types are available, hand crimpers and bench crimpers.
* in wet conditions, insulate the crimp with grease.

The length of the safety fuse should generally not fall below 1.0 m, but for single shots a length of 0.6 m may be allowed. However, the fuse should have sufficient length to exceed the collar of the blasthole with at least 0.1 m.

A safety fuse may be lit by using matches or, better, special igniter torches. When several fuses are lit, a control fuse, the length of which is 0.6 m shorter than the shortest fuse of the round, may be lit and carried around as an extra safety measure. When the control fuse has burnt out, the blasting crew should evacuate the blasting site immediately.

When a large amount of fuses are to be lit in the same area, it may be practical to use igniter cord and bean-hole connectors. The bean-hole connector, which

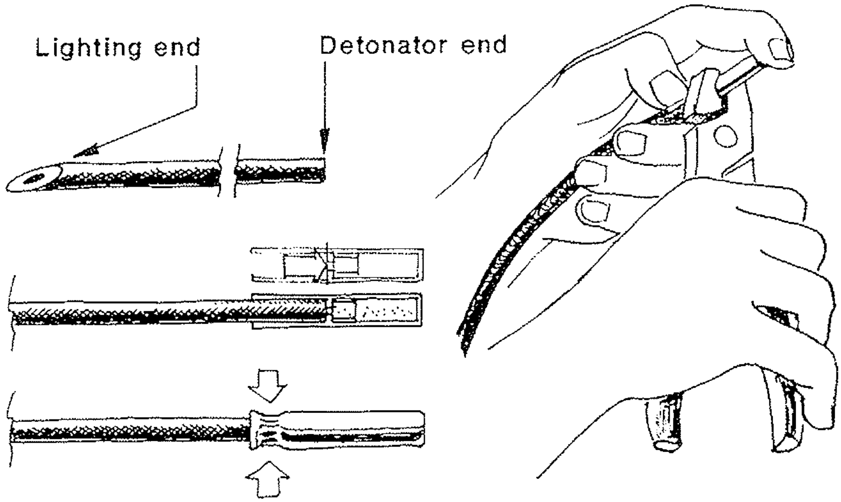



*Fig. 3b.3 Assembly of plain detonator and a safety fuse.*

contains a pyrochemical compound, is crimped to the end of the safety fuse and the igniter cord is inserted into a slot in the bean-hole connector. When the igniter cord is lit, it ignites the connector, which in turn lights the safety fuse.

### 3b.2.2 Detonating cord.

Detonating cord is a very common firing device throughout the world. It has especially been adopted in countries with difficult climatic conditions, with frequent thunderstorms, which disallows the use of electric firing systems.

Apart from being used in difficult electric conditions, it is also used when an exact simultaneous detonation of several holes is desired, as in presplitting. Detonating cord is also used as a supplement to other firing methods in blast-holes that are ragged and difficult to charge.

The detonating cord consists of a PETN core, which is wrapped in coverings of textiles, waterproofing materials and plastics.

The detonating cord may be initiated with a #6 strength detonator and detonates along its entire length with a velocity of about 7,000 meters per second. It initiates most explosives, but care must be taken when detonating cord is used together with ANFO in small and medium size blastholes, where cords with low core load tend to give inclomplete initiation and sometimes cause dead pressing of the ANFO.

Detonating cord is manufactured with core loads ranging from 3 grams per meter to 80 grams per meter. The most widely used cord has a core load of 10 grams per meter (50 grains per foot). The powerful detonating cords with core loads of 40 and 80 grams per meter are mainly used for seismic prospecting and other special purposes.

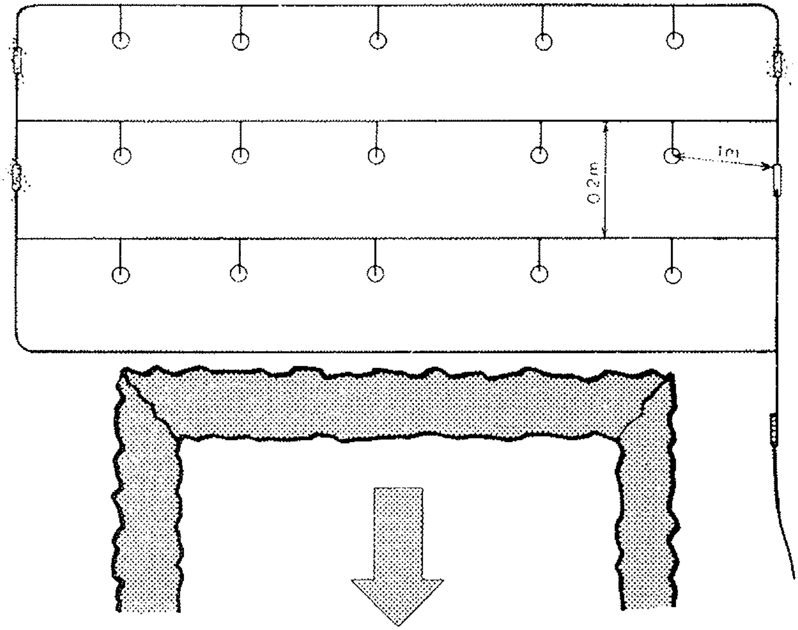



*Fig. 3b.4 Firing pattern for a detonating cord blast.*

Multiple-row blasting can be carried out with detonating cord, where the delays between the rows are obtained by means of relay connectors. A wide range of relays with delay times from 5 ms to 50 ms are available from different manufacturers.

Connection of detonating cord:

* keep each connection at a right angle. Plastic connectors are convenient and reliable.
* the distance between parallel cords should be no less than 0.2 m.
* the distance between relay connector and parallel cord should be at least 1.0 m.
* no kinks or loops are permitted in the round.
* the initiating detonator should always be pointed in the desired direction of the detonating cord detonation.

**3b.2.3 Electric firing.**

The introduction of electric firing gave a higher degree of safety for the people involved in blasting operations.

The blaster became able to fire the blast from a protected area and could have the moment of firing completely under control.

As it became possible to check with instruments that all the detonators were connected, the risk of misfires decreased.

The introduction of short delay blasting revolutionized the rock blasting technique, making it possible to overcome the problems with ground vibrations and increase the size of the rounds.

Electric detonators can be divided into three different classes due to their inherent timing properties:

* instantaneous detonators
* milli-second detonators
* half-second detonators

The instantaneous detonator is a development of the plain detonator, where the safety fuse has been replaced by electric legwires and a fusehead which burns and ignites the primary charge when the bridge wire receives an electric current. Instantaneous detonators are used for stone and boulder blasting, presplitting etc., where no delay between the different charges is needed nor desired.

The millisecond delay detonator has a built-in millisecond delay element which delays the detonation a predetermined time. To be considered a millisecond delay detonator, the delay between each interval in the series should not exceed 100 ms (0.1 sec). The Nitro Nobel millisecond delay series has 20 intervals with 25 ms delay between each interval.
The millisecond series may be prolonged with decisecond (100 ms) delays for tunneling.
Millisecond delay detonators are mainly used for bench and trench blasting.

The half-second delay detonator has a 500 ms (0.5 sec) delay between the intervals. It is intended exclusively for tunnel blasting where longer delays are required to prepare space for the movement of the blasted rock masses.

The electric detonators available on the market may roughly be divided into:

* conventional detonators
* high safety detonators

The classification is based on the detonators' inherent capacity to withstand extraneous electric hazards.

A couple of examples:
High safety detonators of VA type can be used under a 70 kV powerline while the safety distance for a conventional detonator is 200 m.

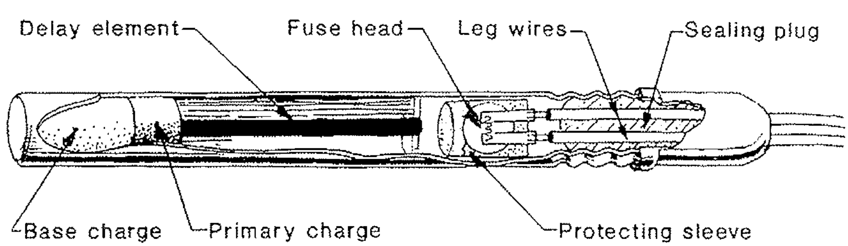



*Fig. 3b.5 Delay detonator.*

Portable walkie-talkie equipment and mobile radio transmitters can be used in the immediate vicinity of high safety VA detonators, but a safety distance of at least 10 meters is the general rule for conventional detonators.

Even though the VA detonators are 30 times safer for unintentional initiation by electric hazards than conventional detonators, safety precautions have to be taken in the neighborhood of strong radio transmitters, powerlines over 120 kV and during thunderstorms.

**WARNING!**

**Detonators of different brands should NOT be used in the same round. Due to different properties of the detonators, this would definitely cause MISFIRES.**

The following types of VA detonators are available:

| TYPE | Colours on legwires | Interval numbers | Delay time | Standard legwire length |
|---|---|---|---|---|
| Milli-second VA/MS | gray-green | 1–20<br>24–80* | 25 ms<br>100 ms | 2, 4, 6, 10, 16, 24 and 35 m |
| | (No. 24–80 manufactured only to order) | | | |
| Half-second VA/HS | gray-red<br>(manufactured only to order) | 1–12 | 500 ms | 4 and 6 m |

* The MS series of the VA-system is prolonged with 100 ms intervals from No. 20 to No. 44 (20, 24, 28 etc.) and with 150 ms intervals from No. 44 to No. 80 (44, 50 etc.)

The VA-OD detonator has a specially strong plastic cover on the legwires. The detonator has double aluminum capsules for protection in heavy duty applications. The VA-OD detonator is developed for underwater blasting where the detonators are subject to stress from rough handling.

The characteristic of the VA-detonator is thus its greater degree of safety against unintentional initiation. The high resistance, which is 3.6 ohms irrespective of the legwire length, makes it necessary to use a 30 times higher firing impulse than for conventional detonators. As the legwires form part of the VA-detonators safety properties, they should never be cut. For further safety, the VA-detonator is supplied with an attached connection sleeve which gives rational and safe connection.

### 3b.2.3a Connecting the electric round.

The electric detonators can be connected in series or in parallel, depending on the amount of detonators in the round and the electric data of the blasting machine available.

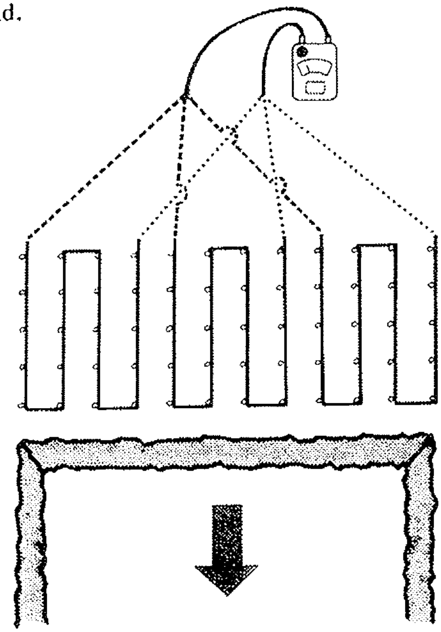


*Fig. 3b.6 Parallel-series circuit.*

### 3b.2.3b Testing the round.

The testing instruments for blasting circuits have to be specially designed for their purpose and be approved by the authorities concerned.

The simplest instrument for testing purposes is **Circuit Tester LP4.** It has been designed to test small numbers of detonators in one series. The **Circuit Tester LP4** only shows if the circuit is closed and no break occurs. The LP4 must not be used in multiple series blasting. The **Ohm-meter GM 2** is more suitable for that purpose.

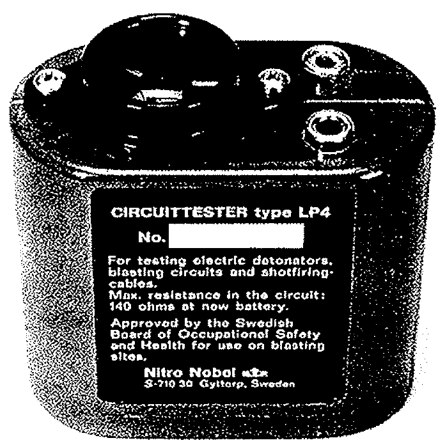



*Fig. 3b.7 Circuit tester LP4.*

The **Ohm-meter GM2** is used to control the resistance of single electric detonators, detonators in series and in parallel-series and for the final check before firing.

Procedure for testing:

* Calibrate the instrument before measuring.
* Divide into series in accordance with the instructions on the blasting machine.
* Measure the resistance in Ohms for each series.
* The resistance must not vary more than ±5 percent between each series in the round.
* The series are connected in parallel and subsequently measured. The resistance of the parallel connection is in accordance with Kirckhoff's law:

$$\frac{1}{R} = \frac{1}{R_1} + \frac{1}{R_2} + \frac{1}{R_3} + \cdots\cdots\cdots \frac{1}{R_n}$$

As the difference in resistance between the series must not exceed ±5 percent, the resistance of the parallel connection will be:

$$R = \frac{\text{Resistance/series}}{\text{Number of series}}$$

Example:

Assume a blast of 250 VA-detonators with a resistance of 3.6 Ohms each. (The resistance is always 3.6 Ohms independent of legwire length.)

The firing cable has a resistance of 5 Ohms and a CID 330 VA blasting machine is used.

In accordance with the instructions on the blasting machine, the round may be connected in 5 parallel series.

Number of detonators in each series: 50.

Resistance per series: $50 \times 3.6 = 180$ Ohms.

Resistance after parallel connection:

$$\frac{\text{Resistance/series}}{\text{Number of series}} = \frac{180}{5} = 36 \text{ Ohms}$$

Resistance at the firing point is the resistance of the parallel-series connection plus the resistance of the firing cable.

$36 + 5 = 41$ Ohms.

**Possible errors during measuring:**

Resistance too high:

- Larger number of detonators than calculated.
- Sub-division into series wrongly carried out.
- Poor contact in some connection or detonator.

Resistance too low:

- All detonators are not connected into the circuit.
- Sub-division into series wrongly carried out.
- Some part of the round not connected into the circuit.

Infinite resistance:

- Interruption in series through incomplete connection.
- Faulty detonator (usually torn off legwire).

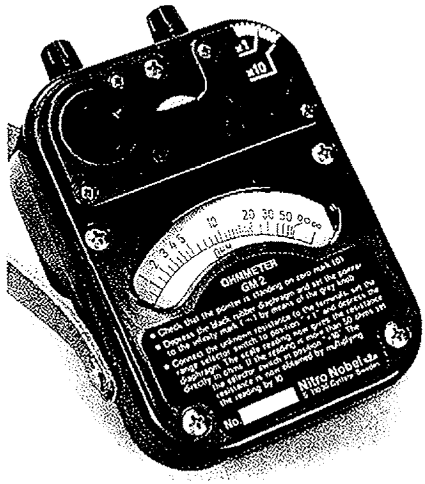



*Fig. 3b.8 Ohm meter GM2.*

**RIM 1.**

Resistance and Insulation Meter. **RIM 1** is a combination instrument for measurement of resistance and insulation in electrically initiated blasting rounds. The instrument is provided with three terminal clips for connection of the objects to be measured.

The center and left clips are used for measurement of resistance. The center and right clips are used for insulation testing. The instrument is auto-starting and auto-ranging. All the operator has to do is to connect the round (or part of the round) to the correct terminals and read the value at the digital display.

The displayed value is automatically rounded off to the accuracy needed for practical blasting work.

*Fig. 3b.9 RIM 1.*

The instrument is provided with a built-in self-test function, which gives a display reading of 5.0 to 5.3 when the test button is pressed. The test consists of an insulation measurement of an internal resistor.

The resistance is measured with a direct current (DC) of max. 2 mA.

The insulation is tested with alternating current (AC) to reduce the influence from possible voltaic cells created by the metal in the blasting round wires and salt solution in the ground. Such voltaic cells can influence measurement with direct current (DC) to such degree that actual earth faults escapes detection. The instrument is powered with a 9V battery size 6F22 and indicates when there is time to exchange battery.

### 3b.2.3c Firing the round.

Blasting machines used to fire electric blasting rounds have to be approved by the appropriate authorities. The use of batteries and accumulators is strictly prohibited.

Capacitor blasting machines have proved to be very reliable even under severe working conditions. The introduction of high safety electric detonators has led to the demand for more powerful blasting machines.

The following range of blasting machines are manufactured by Nitro Nobel Sweden.

CI 50 (2VA):

This is the smallest blasting machine in the range and it is designed for the firing of maximum 2 VA-detonators or 50 conventional detonators. It is charged by means of an induction generator driven by a hand crank. The charge current is 340 V and it takes about 5 seconds to charge the machine. The firing current is automatically released, which means that you crank until firing occurs.

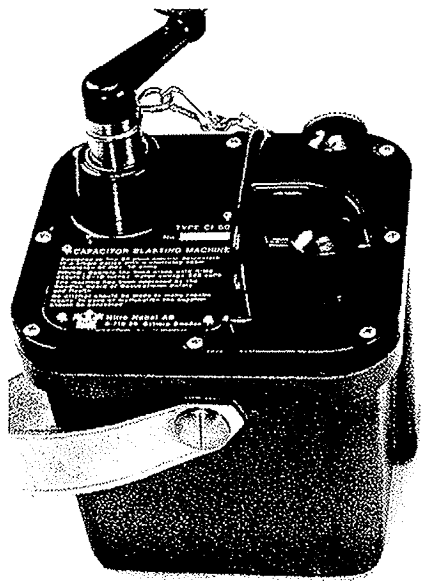



*Fig. 3b.10 CI 50 (2VA).*

### CI 15 VA:

This is a somewhat larger capacitor blasting machine with an inductor generator driven by a hand crank.
It can be charged to 600 V in about 10 seconds. A dial shows when the blasting machine is fully charged. The charge current is high, therefore the blasting machine is two-hand operated to avoid electrical accidents and reduce the risk of accidental firing.
This blasting machine is designed for the blasting of up to 15 VA-detonators in one series with 10 Ohms firing cable resistance. The lowest connection resistance to the blasting machine should be not less than 3.5 Ohms.

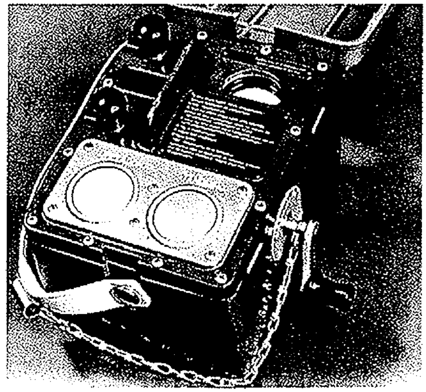


*Fig. 3b.11 CI 15 VA.*

### CB 20 VA:

**CB 20 VA** is a capacitor blasting machine designed for the firing of up to 20 VA detonators connected in one series and with a firing cable resistance of 5 Ohms. The blasting machine is battery powered and of a two-hand operation type to minimize the risk of accidental firing. The four NiCd-accumulators powering the machine are placed in the handle. The accumulators are recharged by connecting the terminals of the blasting machine to a 12 to 14 V DC power source, e.g. the cigarette lighter socket of a car.

If necessary, the accumulators may be replaced by four alkaline batteries.

The control panel has three signal lamps indicating:

1. If accumulator charge is sufficient.
2. That the resistance of a ONE SERIES round is within the capacity of the blasting machine.
3. That the capacitor is charged and ready for firing.

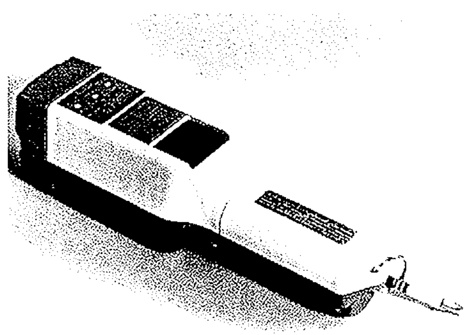


*Fig. 3b.12 CB 20 VA.*

**CI 160 VA:**

With the **CI 160 VA** it is possible to adjust the accumulated energy, and thereby the firing pulse, to suit the round to be fired, even when the round contains considerably less detonators than can be handled by the blasting machine.
Full energy accumulation is only used when it is actually required. The voltage is kept as low as possible to minimize the risk of current leakage. The machine is charged up by an efficient hand cranked generator. The maximum voltage (1950 V) is reached in approx. 20 sec. The charge level is indicated on a meter which has four divisions (I–IV). The required charge levels are shown on a table on the machine's instruction plate.

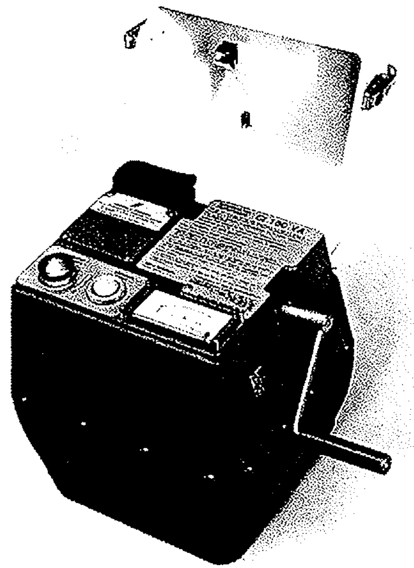


*Fig. 3b.13 CI 160 VA.*

CAPACITY OF **CI 160 VA.**

Resistance of shotfiring cable 5 Ohms.

| Detonator type | Charging level | Series in parallel | Detonators/ series | Detonators in the round |
|---|---|---|---|---|
| VA | I | 1 | 1–10 | 1–10 |
| | II | 1 | 11–40 | 11–40 |
| | III | 1 | 41–70 | 41–70 |
| | | 2 | 21–45 | 42–90 |
| | IV | 1 | 71–100 | 71–100 |
| | | 2 | 46–70 | 92–140 |
| | | 3 | 31–50 | 93–150 |

**CI 330 VA:**

The **CI 330 VA** blasting machine is similar to CI 160 VA but is designed for the blasting of up to 330 VA detonators in one round. The machine is charged by means of a hand-cranked generator and is of a two-hand operation type to minimize the risk of accidental firing. The CI 330 VA is very robust and fitted with heat and cold resistant capacitors.

*Fig. 3b.14 CI 330 VA.*

CAPACITY of **CI 330 VA:**

The round can consist of between 1 and 300 VA detonators in 1 to 5 series in parallel using a shot firing cable of 5 Ohms resistance.
If a shot firing cable of 2.5 Ohms resistance is used, the CI 330 VA blasting machine will fire up to 330 VA detonators divided into 6 parallel series at full charging level. (IV.)
Like the CI 160 VA blasting machine, the CI 330 VA machine may be charged to four levels which provides the possibility to choose the suitable energy level for any size of the round up to 330 VA detonators.

**Test instruments for the blasting machines.**

Special test instruments are available for all Nitro Nobel's blasting machines, with which the blasting machines can be checked to see that they give the effect they are designed for. The instrument is attached to the blasting machine's terminals and the machine is charged and fired as usual. The tester pointer should then respond and move into the red area. This indicates that the blasting machine is giving full effect.

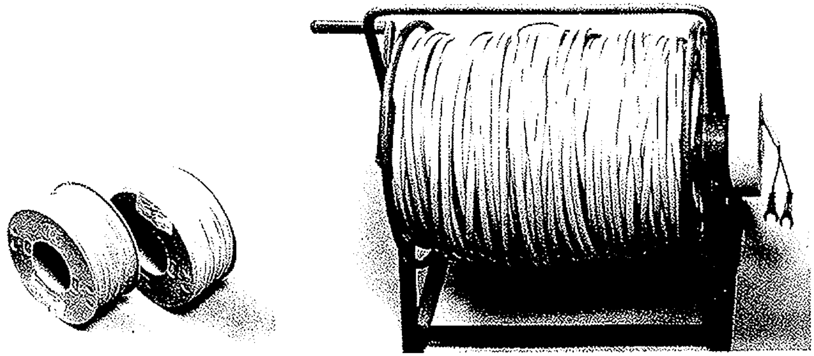


*Fig. 3b.15 Connecting wire and firing cable.*

**Connecting wire.**

Connecting wire is used to connect the different series in parallel and to connect the blasting round to the firing cable. Connecting wire may also be used to extend the length of the legwires.
The connecting wires must be properly insulated and have a low resistance. If the connecting wires have high resistance the capacity of the blasting machine will be reduced. Worn or used connecting wires must not be used as they may cause misfires.

**Firing cable.**

The firing cable is used to connect the blasting round with the blasting machine. The firing cable must be twin wired to reduce the risk of induction. Induction currents may arise due to lightning, radio energy, snow or sand storms. The twin wire cables to be used must be approved for use as firing cable.
The firing cable must be completely insulated and should not be spliced by persons other than electricians.

The firing cable should have as low resistance as possible. High resistance in the firing cable leads to reduced effectiveness of the blasting machine. The cable should be kept on a cable winch and wound up to protect it from damage. It should be specially colored so it is not mistaken for other electric cables.

**3b.2.3d Lightning detection.**

Whenever there is a risk of a thunderstorm, it is necessary to evacuate worksites where electric firing is used. Nitro Nobel has developed a lightning detector, V S L 2, to evaluate the actual on-site risk, registering not only distant lightning but also the electrostatic field which arises during thunderstorms.

In order to provide effective warning of lightning, flashes and high field intensities, three different types of thunderstorms must be taken into account:

1. Thunderstorms starting to develop in the warning area. The very first lightning discharge may then occur within the area concerned.
2. Electrostatically charge precipitation in the form of rain and snow. This type of precipitation often terminates with a single flash of lightning.

3. Thunderstorms that are already fully developed when they enter the warning area.

Types 1 and 2 are registered through a field mill which accurately measures the field intensity in the air. Type 3 is registered, not only by the field mill but also by a radio-wave antenna which can detect the electro-magnetic radiation from lightning flashes at distances of approx. 15 kilometers.

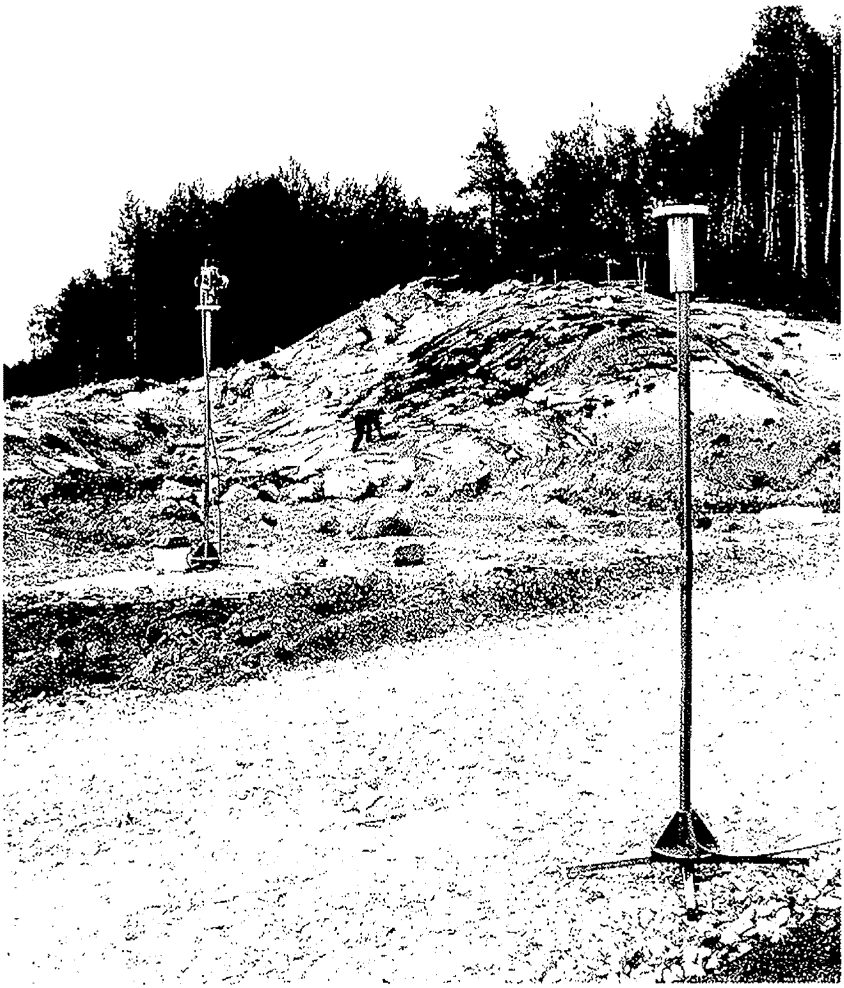


*Fig. 3b.16 V S L 2 Lightning forecasting system.*

### 3b.2.4 NONEL.

With the invention of NONEL, blasters were provided with a long sought replacement for the electric detonator which possesses the advantages of the electric detonator but none of its disadvantages.

NONEL is completely immune to any electric hazard and is thus ideal when electric firing is neither possible nor permitted.

The NONEL detonator functions as an electric delay detonator, but the legwires and the fusehead have been replaced by a plastic tube through which a shock wave is transmitted. The endspit of the shock wave from the plastic tube initiates the delay element in the detonator.

The plastic tube, which has an outer diameter of 3 mm is coated on the inside with a thin layer of reactive material, which transmits the shock wave with a velocity of approx. 2.000 meters per second. The plastic is unaffected by the shock wave and will consequently not initiate any explosives column it goes through.

Two NONEL systems are available:

- NONEL GT
- NONEL UNIDET

### 3b.2.4a NONEL GT.

The range of NONEL GT timing offers both short delays and deci-second and half-second delays. The short delay periods, NONEL GT/MS, meet the needs of bench blasting and the deci-second and half-second delay periods, NONEL GT/T are intended for tunnel blasting.

For the connection and initiation of the NONEL GT detonators, NONEL group igniters UB O or detonating cord are used.

The group igniter NONEL UB O consist of a NONEL tube with one end sealed and a transmitter cap in the other. The transmitter cap is protected by a plastic block.

The initiation with detonating cord should be done with a cord with low core load (E-cord 5 gr/m). In bench blasting the NONEL tube is connected to the cord with a MULTICLIP and in tunneling the NONEL tubes are collected in bunches and then fired with cord. These methods will be dealt with more thoroughly later.

NONEL GT/MS is an all-round firing system which may be used in most situations when millisecond delay is needed like bench blasting, trench blasting, underwater blasting etc.

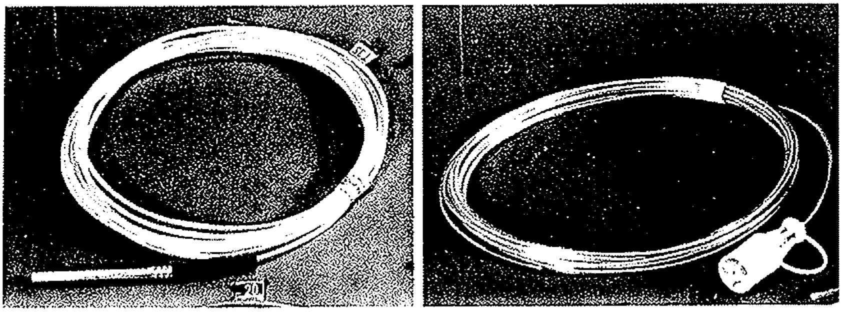


*Fig. 3b.17*

*NONEL GT/MS detonator* — *NONEL UB O*

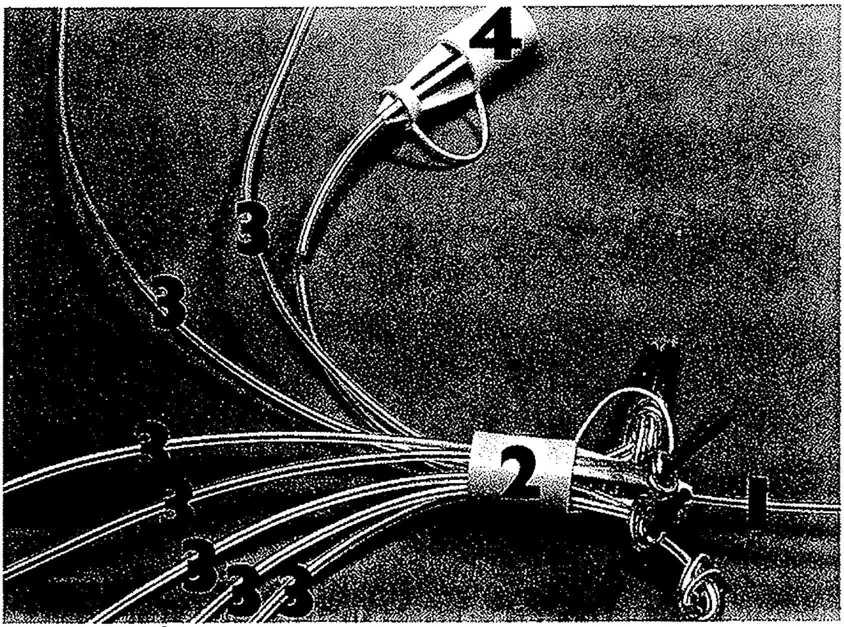



*Fig. 3b.18 NONEL GT/MS hooked up to UB O connector unit.*

The UB O connector unit includes a transmitter cap with a strength corresponding to 1/3 of a #8 blasting cap. It is designed solely to initiate the NONEL tube. The connector unit is so designed that the connected NONEL tube is in close contact with the transmitter cap, thus reducing the risk of discontinuance in the round. When the shock wave, which goes through the NONEL tube (1), reaches the connecting block (2) the transmitter cap explodes and initiates all the NONEL tubes connected to the block. In this way, the initiating impulse is transmitted to one or more detonators (3) and also to the next connector unit (4) where the procedure is repeated. The NONEL UB O connector unit is principally designed for bench blasting and up to 8 NONEL tubes may be hooked up in each connector unit.

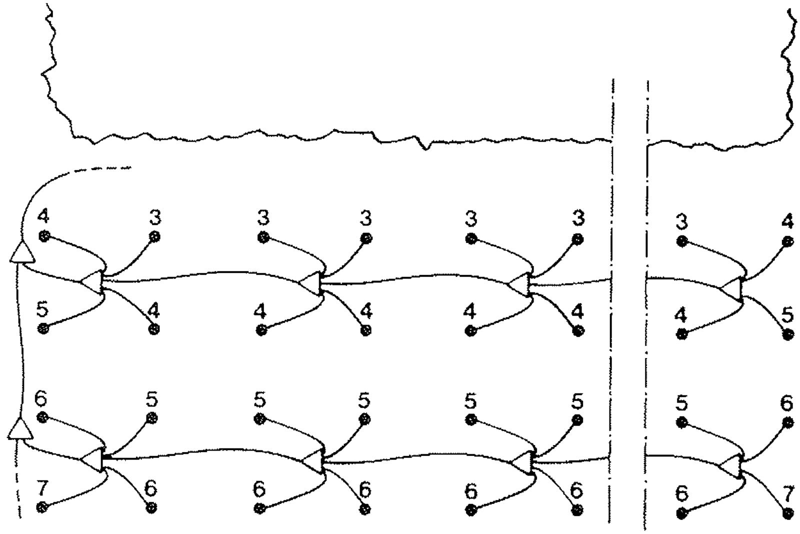



*Fig. 3b.19 NONEL GT/MS connected to UB O in bench blasting.*

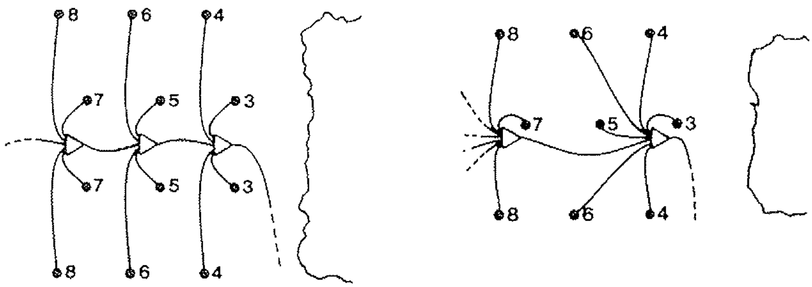



*Fig. 3b.20 NONEL GT/MS connected to UB O in trench blasting.*

The NONEL GT/MS detonators may also be connected to detonating cord if noise is no problem. The NONEL tubes should be connected to the detonating cord with a MULTICLIP as the NONEL tubes ought to be perpendicular to the cord. Each MULTICLIP can take up to 2 NONEL tubes together with the cord.

*Fig. 3b.21 MULTICLIP.*

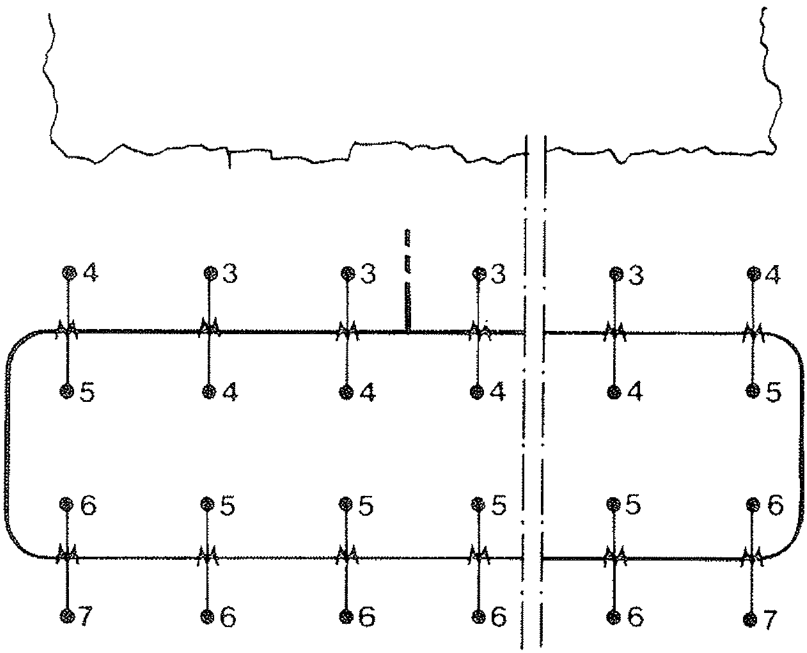



*Fig. 3b.22 NONEL GT/MS connected to detonating cord.*

The **NONEL GT/T** is designed specially for tunnel blasting. The delays are longer than in the NONEL GT/MS system as more time is needed for breakage and movement of the rock in the constricted tunnel blasts.
The delay times between the periods vary from 75 ms to 500 ms and 25 periods are available.
The standard tube lengths are 6.0 and 7.8 m but other lengths are available on special order.

The simplest way of connecting NONEL GT/T is by using the NONEL Bunch Connector, which consists of a loop of detonating cord (E-cord) connected to a UB O connector unit.

Connection procedure:

The NONEL tubes, which should be around 2 m longer than the hole depth, are collected in bunches with a maximum of 20 tubes in each. The bunch is secured with insulation tape. A NONEL Bunch Connector is thoroughly tightened around the bunch. Immediately before evacuation of the blasting site, the bunch connectors are connected to a connector unit UB O or a Starter after which the round is ready for blasting.

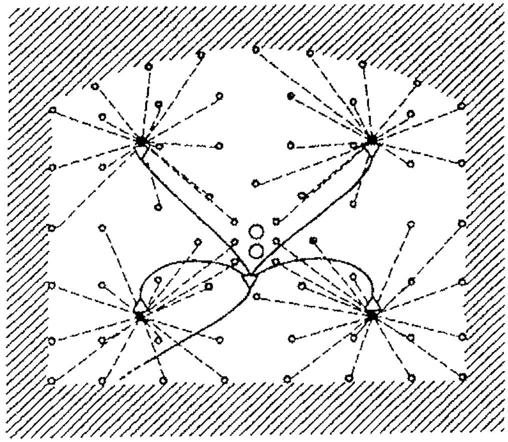


*Fig. 3b.23 Connection of tunnel round with NONEL Bunch Connectors.*

The bunches may also be tied up and connected together with 5 gr detonating cord. As the amount of cord on the rock surface is greater in this case, there is an increased risk of cut-offs and subsequent misfires.

Range of NONEL GT detonators:

| Description | Interval number | Number of intervals | Delay range (ms) | Delay time between intervals |
|---|---|---|---|---|
| NONEL GT/MS detonator | 3–20 | 18 | 75–500 | 25 |
| NONEL GT/T detonator | 0 | 1 | 25 | 25 |
| | 1–12 | 12 | 100–1200 | 100 |
| | 14, 16, 18, 20 | 4 | 1400–2000 | 200 |
| | 25, 30, 35, 40, 45, 50, 55, 60 | 8 | 2500–6000 | 500 |

The standard tube lengths of NONEL GT/MS detonators are 4.8, 7.8 and 15.0 m. Other lengths can be ordered from 2.4 m and upwards in intervals of 0.6 m. NONEL GT/T is only manufactured in standard lengths of 6.0 and 7.8 m.

Range of connectors:

| Description | Delay (ms) | Standard tube length (m) |
|---|---|---|
| UB O | 0 | 1.8, 3.0, 4.8, 6.0 |
| Starter | 0 | 30, 50, 100 |

### 3b.2.4b NONEL UNIDET.

Many applications require more period numbers than are available in any previous firing system. In the case of rounds where the length of the blast is great in comparison with the width, e.g. trench blasting, the blasts become too small with MS detonators with its limited number of periods. In cautious blasting, ground vibration limitations may restrict the use of several charges within the same period number. With the NONEL UNIDET system an unlimited number of exactly timed delay periods are obtained.

The NONEL UNIDET system consists of a NONEL downline with a detonator with 500 ms delay and NONEL surface connectors with 17, 25 and 42 ms delay.

Each hole has the same base delay, 500 ms, and the actual delay between the holes is obtained by the surface delay connectors.

As the actual time between initiation and detonation of the blasthole is greater than the surface activation time, the risk of cut-off downlines or trunklines due to rock movement is minimized.

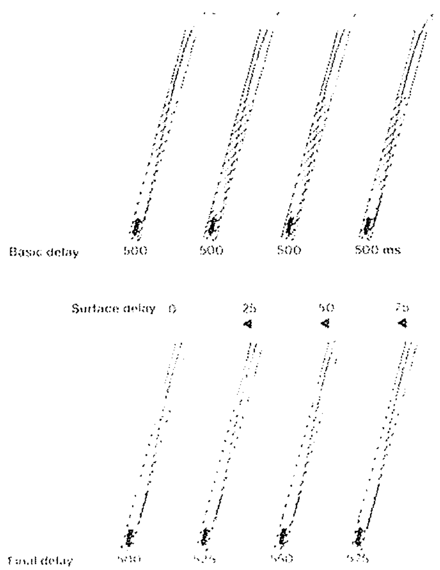



*Fig. 3b.24 Principle of function.*

Denominations:

The basis of the UNIDET system is the detonator with 500 ms delay which is known as U 500. The standard lengths of the tube are 4.8, 6.0, 7.8 and 15.0 m, but other lengths may be ordered from 2.4 m and upwards in 0.6 m intervals.

For the connection of the round, four different kinds of connecting units are available.

Three with a built-in delay of 17 ms, 25 ms and 42 ms respectively, the fourth is the connector with no delay which is mentioned in the NONEL GT system. The standard tube lengths are 2.4 and 4.8 m but, as in the case of the detonators, they may be manufactured in any length in 0.6 m intervals.

| NONEL UNIDET | |
|---|---|
| Denomination | U 500 |
| Delay ms | 500 |

| Connector | | | | |
|---|---|---|---|---|
| Color | Yellow | Blue | Red | Green |
| Denomination | UB 0 | UB 17 | UB 25 | UB 42 |
| Delay ms | 0 | 17 | 25 | 42 |

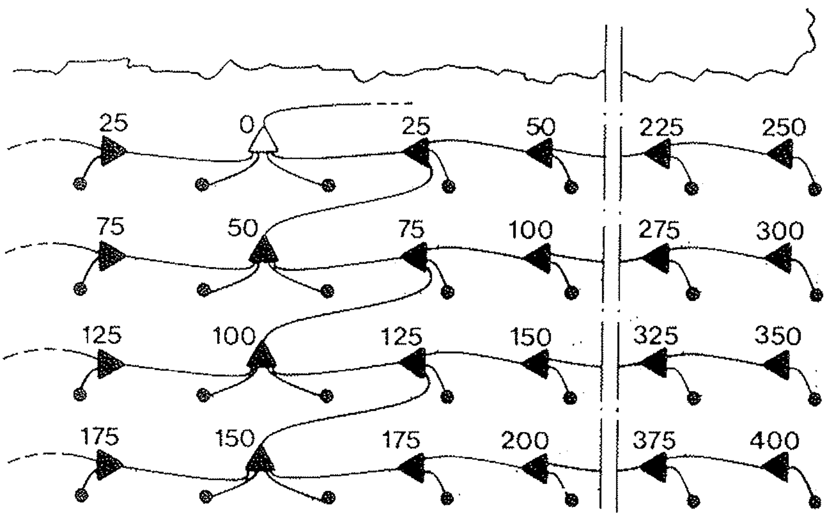



*Fig. 3b.25 Connection of bench blasting round with NONEL UNIDET.*

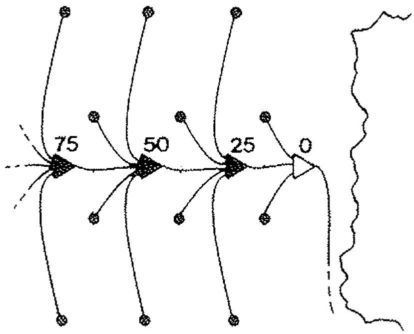



*Fig. 3b.26 Connection of trench blasting round with NONEL UNIDET.*

### 3b.2.4c NONEL System Blasting Machines.

A NONEL round may be fired by safety fuse and a plain detonator in locations where time precision is not necessary e.g. in underground blastings. In blastings where the moment of initiation must be totally under control, like surface blasting and blastings with ground vibration control, electric firing may be used if the firing point is extended outside the hazardous area.
However, the safest and most reliable way of initiating a NONEL round is to use NONEL System Blasting Machines.
Nitro Nobel has developed two types of blasting machines to initiate NONEL rounds, the manually actuated HN 1 and the pneumatic PN 1. Both machines use "Shot Shell Primers No. 20" primer caps.

**HN 1:**

**HN 1** is a simple and highly effective hand held blasting machine, robustly constructed in tough metal alloys and stainless steel. Its slender form and a weight of only approx. 450 g makes it easy to carry in a pocket.
It has an integral safety device and is a complete blasting machine, no other equipment being needed to fire the NONEL round.

*Fig. 3b.27 HN 1.*

**PN 1:**

**PN 1** is a pneumatically operated blasting machine designed to stand in the vicinity of the round and be remotely actuated by compressed air. Thus the blasters safety is assured and savings are made in NONEL starter tubes.
Snap-on air-line couplings allow the machine to be moved between locations and a safety lock is provided to prevent premature firing.

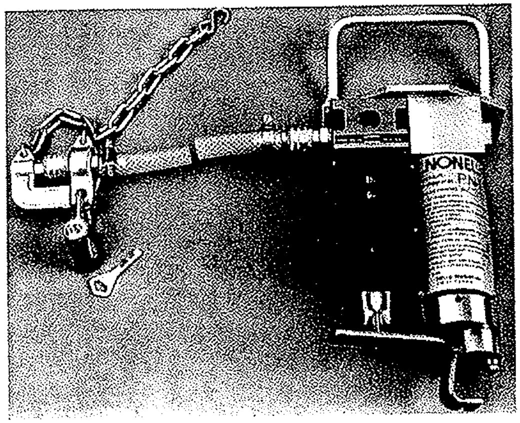


*Fig. 3b.28 PN 1.*

NONEL is invented by Nitro Nobel of Sweden, which is manufacturer and trademark holder.
NONEL is also manufactured under license by licenseholders.

## 3c. ACCESSORIES.

### Electric Warning Siren for Blasting ESS 1.

The ESS 1 is a specially designed warning siren for blasting operations. The siren is powered by 12 V DC and has a powerful magnet in its base, which enables it to be firmly attached to for example a car roof.
ESS 1 is designed to suit the "one-off" and smaller worksites.

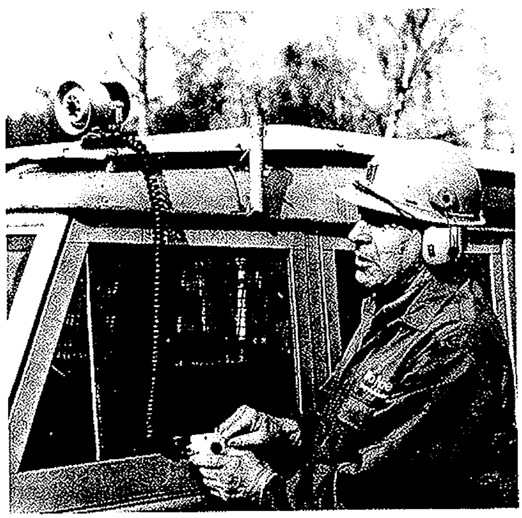


*Fig. 3c.1 ESS 1.*

**Drill hole plugs.**

Drill hole plugs prevent the finished drill hole from becoming blocked by dirt, gravel, drill cuttings etc. entering at the collar of the hole.

The plug's vivid red coloring also serves as an effective marker.

| Type | Plug dia. min/max mm | Drill hole dia. min/max mm |
|---|---|---|
| #1 | 29–50 | 34–45 |
| #2 | 39–60 | 45–57 |
| #3 | 57–115 | 64–110 |

*Fig. 3c.2 Drill hole plugs.*

**Charge locks.**

Charge locks prevent the charge from being blown out of the blast hole during firing. Two sizes are available:

| Type | Blast hole dia. min/max mm |
|---|---|
| #1 | 31–43 |
| #2 | 43–64 |

See Fig. 8.11.

**Tamping rods.**

Plastic tamping rods are essential for a good standard of packing during charging with paper cartridges.

Standard range:

| Diameter | Length |
|---|---|
| 25 mm | 3, 5 and 8 m |
| 40 mm | 3 and 6 m. |

Tamping rods may also be made of wood or bamboo. It is important that its diameter is close to the diameter of the blasthole.

# 4. BLASTING THEORY

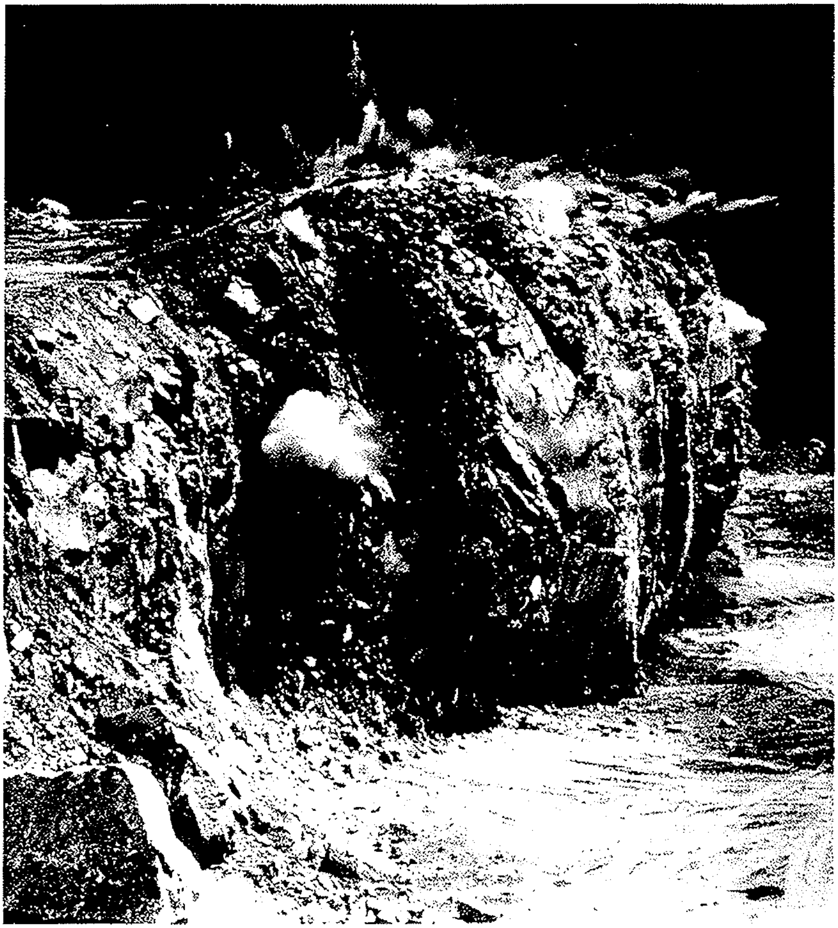


*Fig. 4.1 Bench blasting with EMULITE and NONEL*

The rock is affected by a detonating explosive in three principal stages.

In the first stage, starting from the initiation point, the blasthole expands by crushing the blasthole walls. This is due to the high pressure upon detonation.

In the second stage, compressive stress waves emanate in all directions from the blasthole with a velocity equal to the sonic wave velocity in the rock.
When these compressive stress waves reflect against a free rock face, they cause tensile stresses in the rock mass between the blasthole and the free face. If the tensile strength of the rock is exceeded, the rock breaks in the burden area, which is the case in a correctly designed blast.

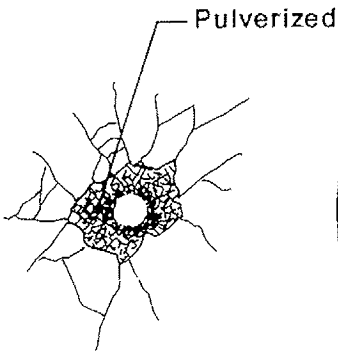



*Fig. 4.2 Radial crack formation.*

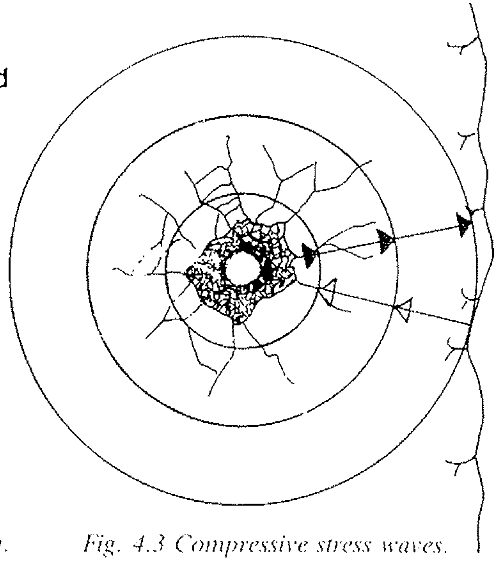


*Fig. 4.3 Compressive stress waves.*

In the third stage, the released gas volume "enters" the crack formation under high pressure, expanding the cracks. If the distance between the blasthole and the free face is correctly calculated, the rock mass between the blasthole and the free face will yield and be thrown forward.

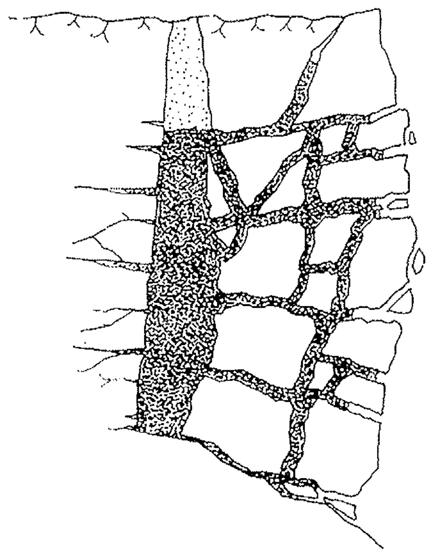


*Fig. 4.4 Gas penetration of crack formation.*

The explosives reaction in the blasthole is very fast and the effective work of the explosive is considered completed when the blasthole volume has expanded to 10 times its original volume which takes approx. 5 ms.
The following graph shows how the expansion of the blasthole is related to time.

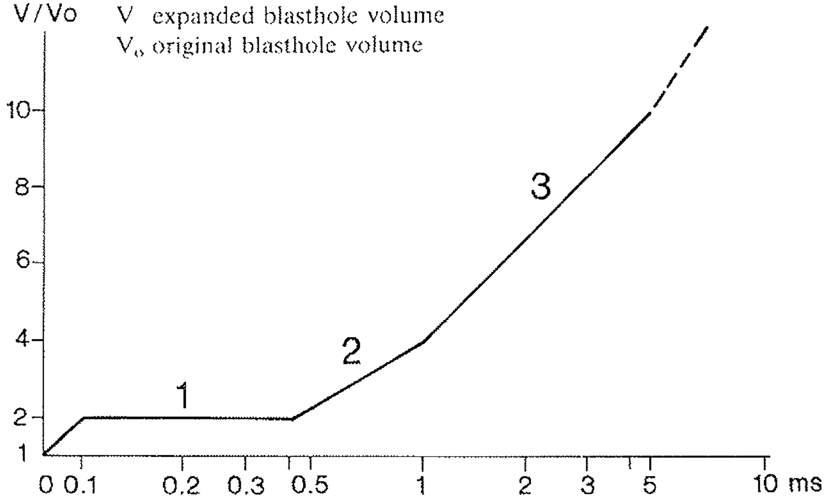



*Fig. 4.5 Blasthole expansion in relation to time.*

1. Initiation of shockwave in rock crushing. The blasthole expands to double its original volume ($2V_0$). The blasthole will stay at this volume for relatively long time (0.1 to 0.4 ms) before radial cracks start to open.
2. Besides the natural cracks are new cracks formed mainly by interaction between the stress field around the blasthole and tensile stresses formed by reflection of the outgoing shockwave at the free face. Reaction products expand from blasthole (which volume now is quadrupled) into the cracks. Fragmentation starts.
3. Gas expands further and accelerates the rock mass.

# 5. BENCH BLASTING

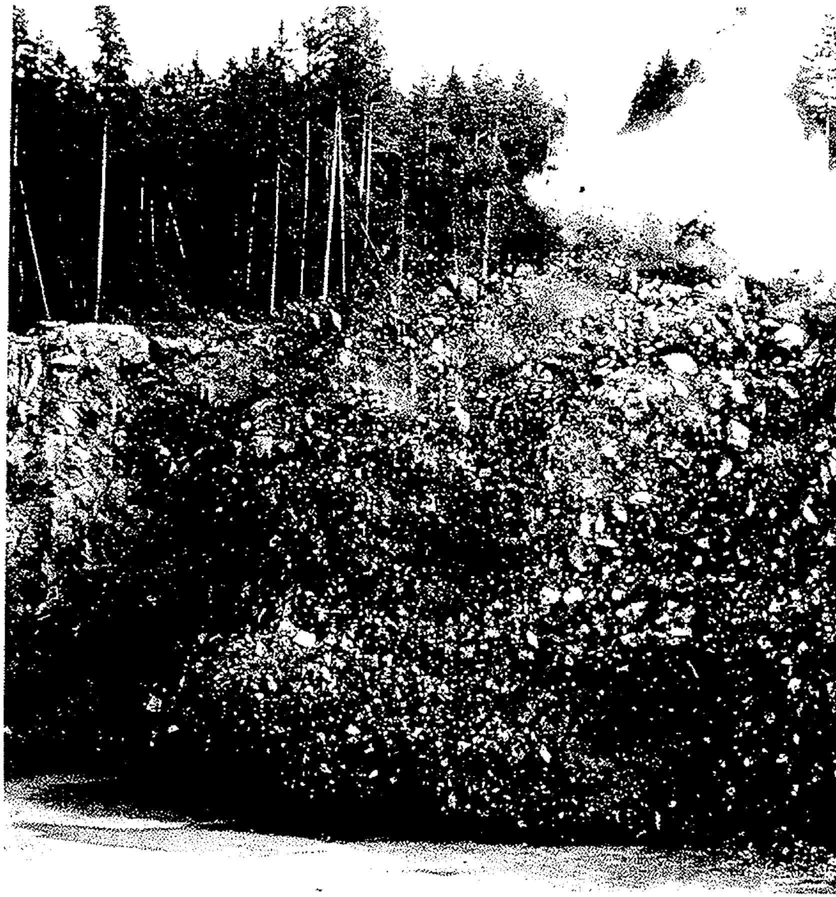


*Fig. 5.1 Quarry blasting in Sweden.*

## 5.1 General

Bench blasting is the most common kind of blasting work. It can be defined as blasting of vertical or close to vertical blastholes in one or several rows towards a free surface. The blastholes can have free breakage or fixed bottom.

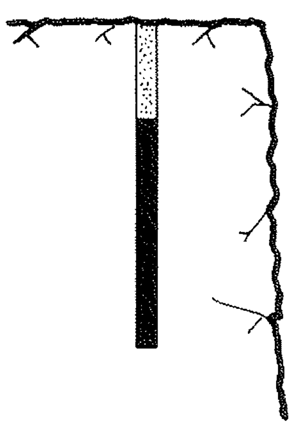


*Fig. 5.2 Free breakage.*

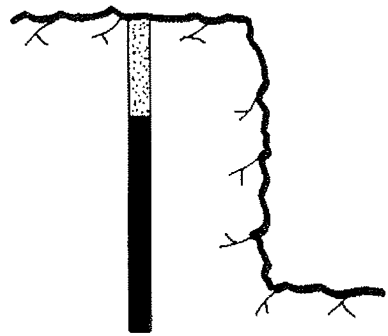


*Fig. 5.3 Fixed bottom.*

Most types of blasting can be considered as bench blasting.

Trench blasting for pipelines is also a kind of bench blasting, but as the rock is more constricted, it requires a higher specific charge and more closely spaced drilling.

In tunneling, after the cut has been blasted, the stoping towards the cut is a type of bench blasting.

Rock is a material with widely varying properties. Its tensile, compressive and shearing strengths vary with different kinds of rock and may vary within the same blast. As the rock's tensile strength has to be exceeded in order to break the rock, its geological properties will affect its blastability.

Rock formations are rarely homogeneous. The rock formation in the blast area may consist of different types of rock. Furthermore, faults and dirt-seams may change the effect of the explosive in the blast. Faulty rock containing voids, where the gases penetrate without giving full effect, may be difficult to blast even though the rock may have a relatively low tensile strength.

The requisite specific charge, (kg/cu.m.) provides a first-rate measure of the blastability of the rock. By using the specific charge as a basis for the calculation, it is possible to calculate the charge which is suitable for the rock concerned.

The distribution of the explosives in the rock is of the utmost importance. A closely spaced round with small diameter blastholes gives much better fragmentation of the rock than a round of widely spaced large diameter blastholes, provided that the same specific charge is used. (See Chapter 5.6 Fragmentation.)

The following calculations are based on a specific charge of 0.4 kg/cu.m. of EMULITE 150 in the bottom part of the round. In the constricted bottom part of the blasthole, this specific charge is needed to shatter the burden, but in the column part of the hole considerably less explosives are needed to break the rock. The average specific charge of the round (hole) will be less than 0.4 kg/cu.m.

The value applies to burdens between 1.0 and 10.0 m and can be used for most kinds of rock. The basis of the computations of bench blasting will be Langefors'* formula:

$$B_{max} = \frac{d}{33}\sqrt{\frac{p \cdot s}{\bar{c} \cdot f \cdot S/B}}$$

where

| | | |
|---|---|---|
| $B_{max}$ | = maximum burden | (m) |
| d | = diameter in the bottom of the blasthole | (mm) |
| p | = packing degree (loading density) | (kg/liter) |
| s | = weight strength of the explosive | (EMULITE 150=0.95) |
| c | = rock constant | (kg/cu.m.) |
| $\bar{c}$ | = c + 0.05 for $B_{max}$ between 1.4 and 15.0 meters | |
| f | = degree of fixation, 1.0 for vertical holes and 0.95 for holes with inclination 3:1 | |
| S/B | = ratio of spacing to burden | |

* The Modern Technique of Rock Blasting, Langefors/Kihlström.

In the following calculations, Langefors' formula is simplified to:

$B_{max} = 1.47\sqrt{l_b}$ for Dynamex M
$B_{max} = 1.45\sqrt{l_b}$ for Emulite 150
$B_{max} = 1.36\sqrt{l_b}$ for ANFO

where $l_b$ is the requisite charge concentration (kg/m) of the selected explosive in the bottom part of the blasthole.

The hole inclination is assumed to be 3:1 and the rock constant c is 0.4. The bench height K is $\geq 2 \times B_{max}$.
For other values of hole inclination and rock constant correction factors are used.
The charge concentration depends on the diameter of the blasthole and the utilization of the hole.

Explosives in paper cartridges, which are normally tamped with a tamping rod in small diameter blastholes, can be tamped to an utilization of up to 90 % of the blasthole if tamping is carried out after the introduction of each cartridge. If tamping is carried out after every two or three cartridges, the charge concentration will be considerably lower. Pneumatic charging machines give good tamping of paper cartridges with high utilization of the blasthole volume.

Explosives in plastic hoses were introduced for the convenience of fast charging and easy handling. Dropped into the blasthole, they are intended to fill up the hole well. However, different tamping characteristics of different explosives give varying results. Emulite cartridges in plastic hoses, which are cut along the side, fill up the hole almost completely by impact, while dynamites and watergels with their stiffer consistency do not fill up the hole that well, especially in the winter. It is important when charging wet blastholes that the holes are flushed and cleaned before charging. If the blastholes contain water, the packing of the explosive will be almost nil and the charge concentration of the cartridges should be used for the calculations. Bulk explosives which are pumped, augered or poured into the blasthole utilize the blasthole volume to 100 %.

The calculations that follow will involve the following explosives:

Dynamex M
Emulite 150
ANFO

which are explosives with differing characteristics regarding weight strength and density.

As the maximum burden, $B_{max}$, is also dependent on the fixation degree at the bottom part of the blasthole, the computations will involve drilling with inclination 3:1, which decreases the constriction in the bottom part of the hole. For other inclinations correction factors can be used.

The packing degree (utilization of the blasthole) of the explosive in the bottom part of the blasthole is assumed to be 95 % for Emulite 150 in plastic hoses and 90 % for Dynamex M. Poured ANFO and pumped Emulite fill up the hole to 100 %.

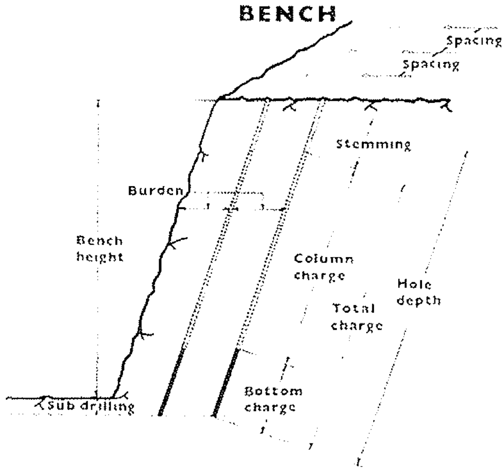
It is very important for the blasting result that the charge concentration obtained by the calculations is achieved in practice.

The formulae used in the calculations are empirical, but are based on information from thousands of blasts. The experience of Langefors' calculations is so good that it could be considered unnecessary in most blasting operations to make trial blasts. However, local conditions may make it necessary for the practical operator to test the theoretical calculations in the field.

## 5.2 Charge calculations.

Bench height $\geq 2 \times B_{max}$.

| | | | |
|---|---|---|---|
| d | = | Diameter of the blasthole in the bottom | (mm) |
| K | = | Bench height | (m) |
| $B_{max}$ | = | Maximum burden | (m) |
| U | = | Subdrilling | (m) |
| H | = | Hole depth | (m) |
| E | = | Error in drilling | (m) |
| B | = | Practical burden | (m) |
| S | = | Practical spacing | (m) |
| b | | Specific drilling | (m/cu m ) |
| $l_b$ | = | Concentration of bottom charge | (kg/m) |
| $h_b$ | = | Height of bottom charge | (m) |
| $Q_b$ | = | Weight of bottom charge | (kg) |
| $h_0$ | = | Height of stemming | (m) |
| $l_c$ | = | Concentration of column charge | (kg/m) |
| $h_c$ | = | Height of column charge | (m) |
| $Q_c$ | = | Weight of column charge | (m) |
| $Q_{tot}$ | = | Total charge weight per hole | (kg) |
| q | = | specific charge | (kg/cu m ) |




The following assumptions are made for the calculations:

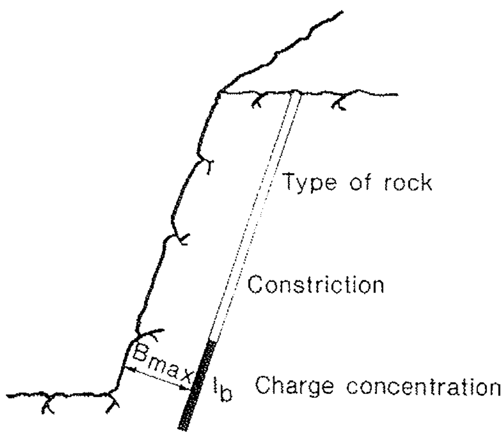
K is equal to, or more than $2 \times B_{max}$.

| | | | |
|---|---|---|---|
| Explosive: | Emulite 150 | Dynamex M | ANFO |
| Packing degree: | 95 % | 90 % | 100 % |
| | 1.15 kg/l | 1.25 kg/l | 0.8 kg/l |
| Rock constant c: | 0.4 | 0.4 | 0.4 |
| Hole inclination: | 3:1 | 3:1 | 3:1 |

Calculation procedure:

The **maximum burden** in the bottom of the blasthole depends on:

* weight strength of the actual explosive (s)
* charge concentration ($l_b$)
* rock constant (c)
* constriction of the blasthole ($R_1$)



As mentioned before, the maximum burden $B_{max}$ is calculated from Langefors' formula, which has been simplified to:

for Dynamex M $\quad B_{max} = 1.47 \sqrt{l_b} \times R_1 \times R_2 \quad (m)$
for Emulite 150 $\quad B_{max} = 1.45 \sqrt{l_b} \times R_1 \times R_2 \quad (m)$
for ANFO $\quad B_{max} = 1.36 \sqrt{l_b} \times R_1 \times R_2 \quad (m)$

where $l_b$ = charge concentration, kg/m as per point 1, as follows
$R_1$ = correction for hole inclination other than 3:1 as per point 2, as follows
$R_2$ = correction for rock constant other than 0.4 as per point 3, as follows

1. Determination of charge concentration, $l_b$.
a/ For tamped cartridges and bulk explosives

$$l_b = 7.85\ d^2 \times P$$

where d = blasthole diameter, dm
P = packing degree, kg/liter

Charge concentration for different blasthole diameters and different explosives:

| Blasthole diameter (mm) | 51 | 64 | 76 | 89 | 102 | 127 | 152 |
|---|---|---|---|---|---|---|---|
| ANFO, kg/m | 1.6 | 2.6 | 3.6 | 5.0 | 6.5 | 10.1 | 14.5 |
| Emulite 150 (cut and dropped into dry blastholes), kg/m | 2.3 | 3.7 | 5.0 | 7.1 | 9.3 | — | — |
| Bulk emulite, kg/m | 2.4 | 3.9 | 5.3 | 7.5 | 9.9 | 15.3 | 21.9 |
| Dynamex M (charged with pneumatic charging machine and ROBOT), kg/m | 2.6 | 4.0 | 5.6 | 7.8 | 10.2 | — | — |

Charge concentration, kg/m, for drill series 11 and 12. Tamped explosives.

| | Blasthole diameter, mm | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | 27 | 28 | 29 | 30 | 31 | 32 | 33 | 34 | 35 | 36 | 37 | 38 | 39 |
| Em150 | 0.66 | 0.71 | 0.76 | 0.81 | 0.87 | 0.92 | 0.98 | 1.04 | 1.11 | 1.17 | 1.24 | 1.30 | 1.37 |
| Dx M | 0.69 | 0.74 | 0.79 | 0.85 | 0.91 | 0.96 | 1.03 | 1.09 | 1.16 | 1.22 | 1.29 | 1.36 | 1.43 |

b/ Charge concentration, $l_b$, for explosives in plastic hoses at different degrees of compression.

| Hose diameter (mm) | DYNAMEX M | | | EMULITE 150 | | | |
|---|---|---|---|---|---|---|---|
| | Charge concentration kg/m at a compression of | | | Charge concentration kg/m at a compression of | | | |
| | 0 % | 5 % | 10 % | 0 % | 10 % | 20 % | 25% |
| 43 | 1.95 | 2.05 | 2.15 | 1.75 | 1.90 | 2.10 | 2.20 |
| 50 | 2.65 | 2.80 | 2.90 | 2.35 | 2.60 | 2.80 | 2.90 |
| 55 | 3.20 | 3.35 | 3.50 | 2.85 | 3.10 | 3.40 | 3.55 |
| 60 | — | — | — | 3.40 | 3.75 | 4.05 | 4.25 |
| 65 | 4.40 | 4.60 | 4.80 | 4.00 | 4.40 | 4.80 | 5.00 |
| 75 | — | — | — | 5.30 | 5.80 | 6.35 | 6.60 |
| 80 | 6.50 | 6.80 | 7.10 | — | — | — | — |
| 90 | 8.00 | 8.40 | 8.80 | — | — | — | — |
| 125 | 14.50 | 15.20 | 16.00 | — | — | — | — |

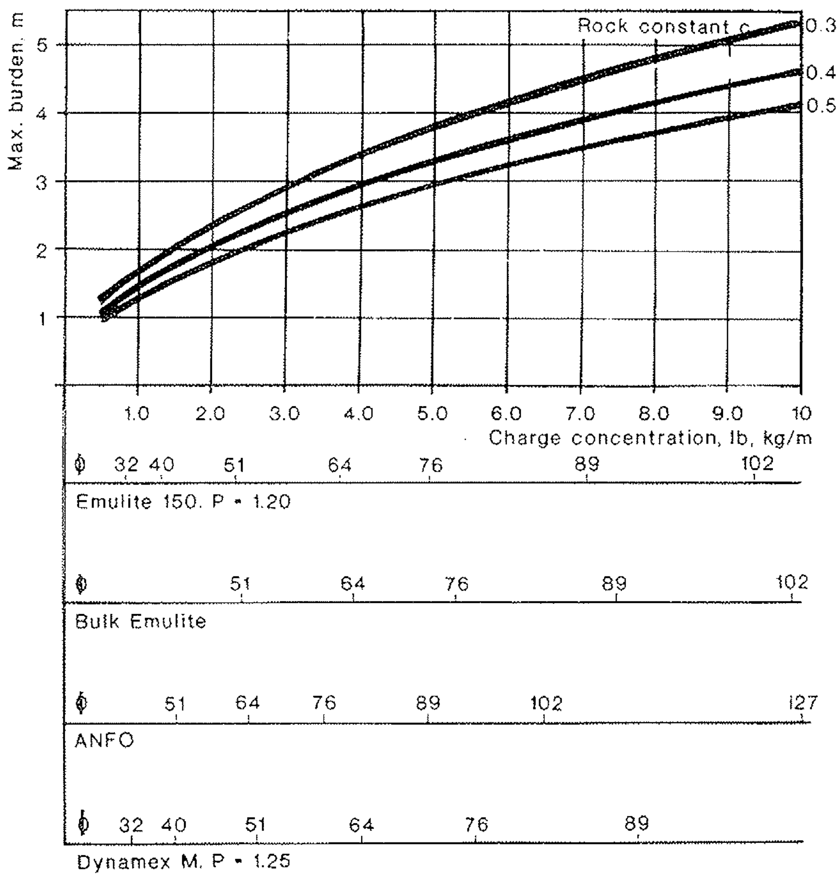
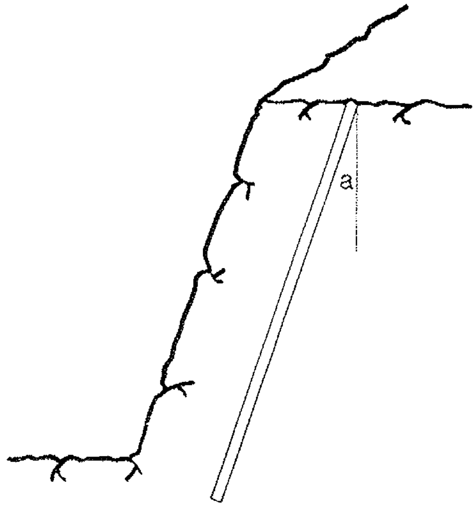



*Fig. 5.4 The influence of charge concentration on maximum burden, $B_{max}$.*

2. Correction of $B_{max}$ for different hole inclinations.

| Incli-nation | Vertical | 10:1 | 5:1 | 3:1 | 2:1 | 1:1 |
|---|---|---|---|---|---|---|
| $R_1$ | 0.95 | 0.96 | 0.98 | 1.00 | 1.03 | 1.10 |



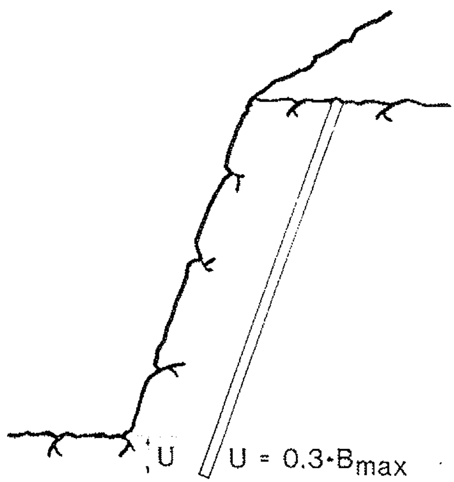

3. Correction of $B_{max}$ for different rock constant c.

| c | 0.3 | 0.4 | 0.5 |
|---|---|---|---|
| $R_2$ | 1.15 | 1.00 | 0.90 |

**Subdrilling** = 0.3 × maximum burden, at least 10 × d.

$$U = 0.3 \times B_{max} \qquad \text{(m)}$$

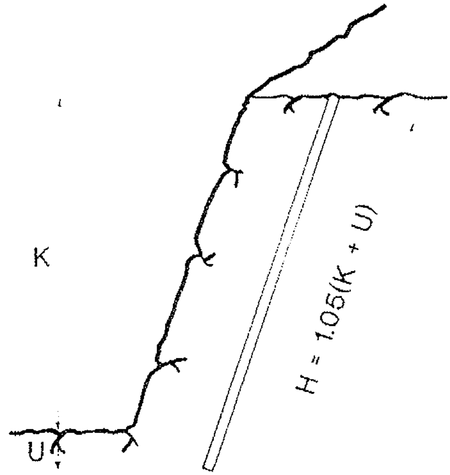
The subdrilling is necessary to avoid stumps above the theoretical grade.



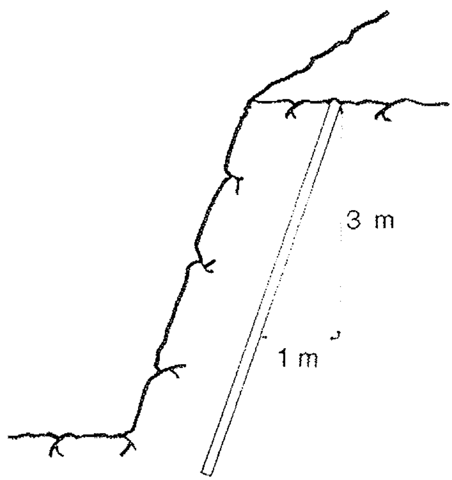
**Depth of the hole** = bench height + subdrilling + 5 cm/m of the depth of the blasthole due to 3:1 inclination.

$$H = K+U+0.05(K+U)$$

$$\mathbf{H = 1.05(K+U)} \qquad \text{(m)}$$




Inclined holes have a favorable angle of breakage between the holes and the intended bottom, thus decreasing the constriction in the bottom part of the holes.




**Faulty drilling** consists of:

* Collaring error = d (in mm)
* Alignment error = 0.03 m/m of the blasthole depth.

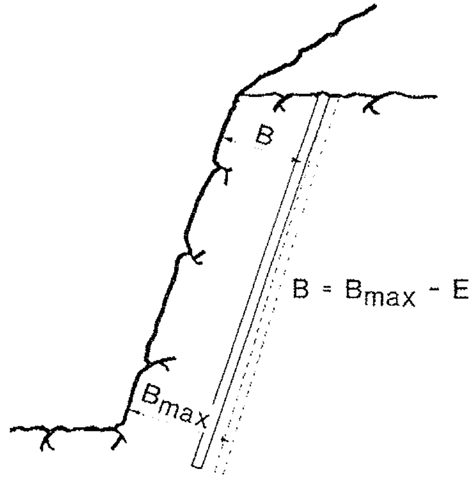
$$E = \frac{\mathbf{d}}{\mathbf{1000}} + \mathbf{0.03 \times H} \qquad \text{(m)}$$

It has to be taken into account that it is impossible to drill a hole exactly in accordance with theoretical computations.
Both the machines used and the skill of the operator affect the accuracy of the drilling.
The error should not be allowed to exceed E as calculated in accordance with the above formula.

When the **practical burden** is calculated, the error in drilling has to be deducted.

$B = B_{max} - E$ (m)

The rule of thumb, B = d, where B (burden) is expressed in meters and d (blasthole diameter) is expressed in inches, can be used to check the calculations.




The burden is the distance from the blasthole to the nearest free face at the instant of detonation. In multiple row blasts new faces are created at each detonation.

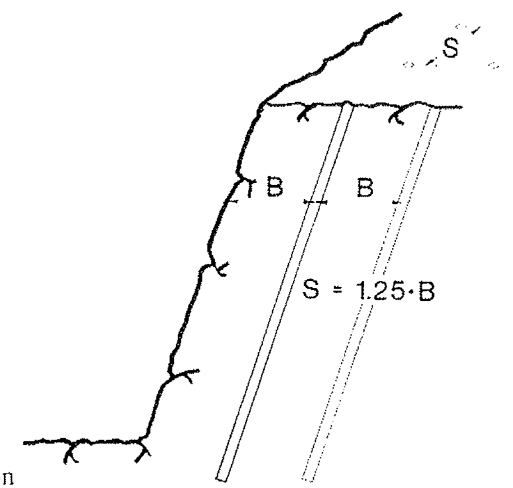
**Practical hole spacing S** is calculated from the relation

$S = 1.25 \times B$ (m)

Spacing is the distance between the adjacent blastholes in a row.

If the ratio S/B is changed without the specific drilling or the specific charge being changed it will result in the following:

S/B>1.25, finer fragmentation
S/B<1.25, coarser fragmentation



**Specific drilling** is the drilling needed to blast 1 cu.m. of rock and can be expressed as:

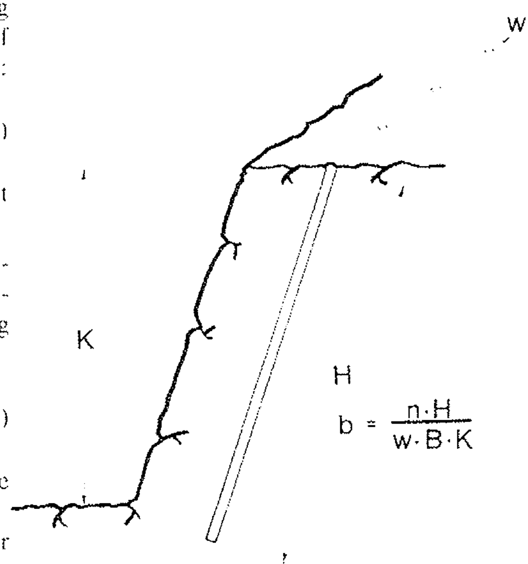
$$b = \frac{n \times H}{n \times B \times S \times K} \quad \text{(m/cu.m)}$$

for quarries and open pit mines.

In road cuts etc. where blasting is performed within a limited area, the specific drilling is calculated per row:

$$b = \frac{n \times H}{w \times B \times K} \quad \text{(m/cu.m.)}$$

where w is the width of the round.
The latter value will be higher due to the influence of the edge holes.



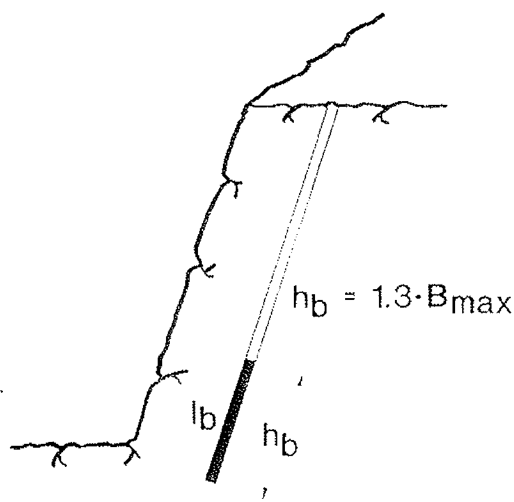

**Charging the blasthole.**

In order to loosen and break the rock in the constricted bottom part of the blasthole, the charge concentration used for the calculation of $B_{max}$ should be used:

$l_b$ = **charge concentration used for determination of** $B_{max}$.

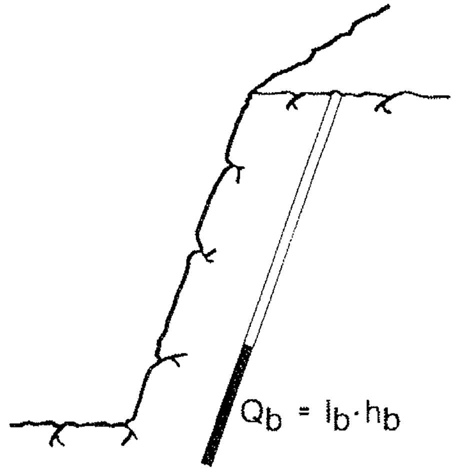
**Height of the bottom charge:**

$h_b = 1.3 \times B_{max}$ (m)



The **bottom charge** will then be:

$$Q_b = l_b \times h_b \qquad \text{(kg)}$$



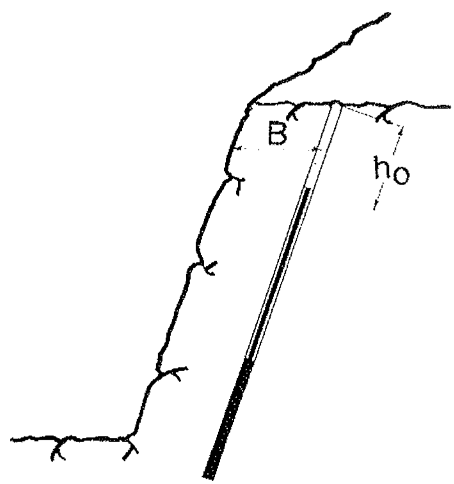

**Stemming.**

The unloaded part of the blasthole, the stemming, is normally equal to the burden:

$$h_o = B \qquad \text{(m)}$$

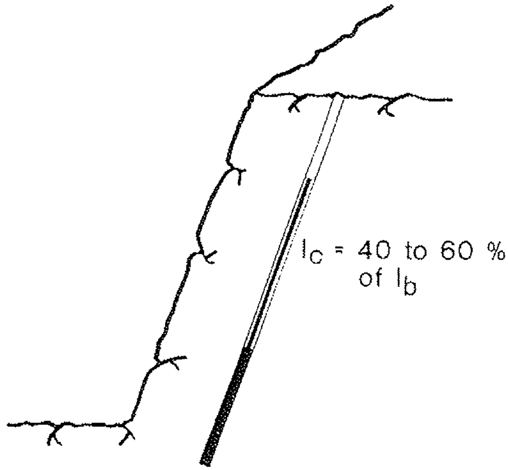
The stemming should consist of sand or gravel with a particle size of 4 to 9 mm. Research has shown that this size gives the best confinement of the explosives gases. Drillfines should be avoided.

If $h_o < B$, the risk of flyrock from the upper surface increases, but the amount of boulders decreases. On the other hand, $h_o > B$, it will give more boulders but superficial throw will be less or eliminated.



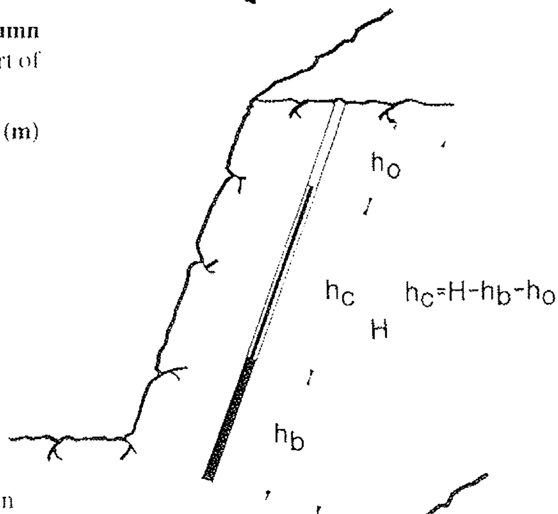
To break the rock above the bottom charge, a **column charge** is applied. As this part of the blasthole is less constricted, the **charge concentration** may be less.

$\mathbf{l_c = 40\ to\ 60\ \%\ of\ l_b}$ **(kg/m)**



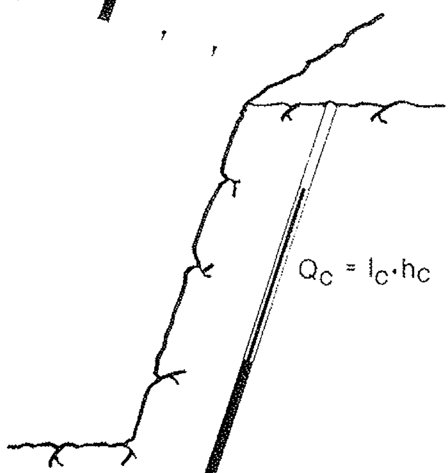

The **height of the column charge** is the remaining part of the blasthole.

$\mathbf{h_c = H - h_b - h_o}$ **(m)**



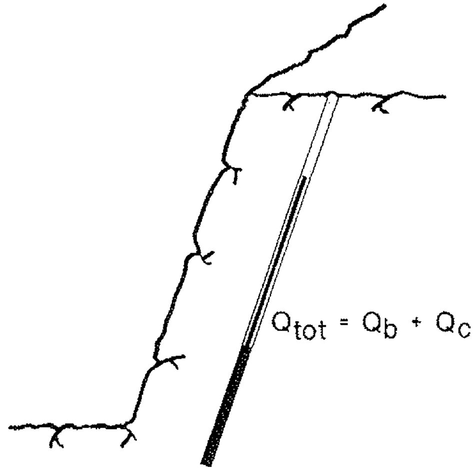

The **column charge** is then

$\mathbf{Q_c = l_c \times h_c}$ **(kg)**



The **total charge** per hole is the bottom charge plus the column charge.

$$Q_{tot} = Q_b + Q_c \qquad \text{(kg)}$$




The **specific charge** may be calculated in the same manner as specific drilling (b).

$$q = \frac{n \times Q_{tot}}{n \times B \times S \times K} \quad \text{(kg/cu.m.)}$$

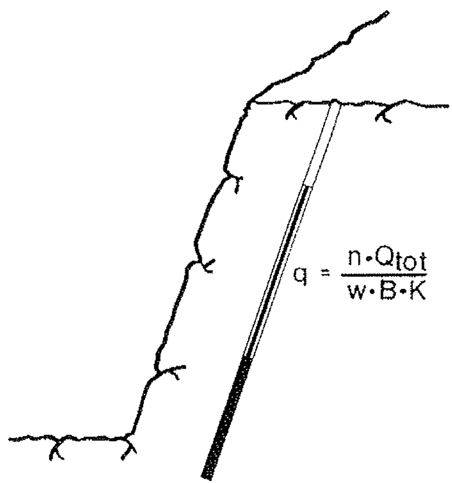
for quarries and open pit mines.

In road cuts etc. where blasting is performed within a limited area, the specific charge is calculated per row:

$$q = \frac{n \times Q_{tot}}{w \times B \times K} \qquad \text{(kg/cu.m.)}$$

where w is the width of the round.

The value of the specific charge will be higher due to the influence of the edge holes.



## EXAMPLE SHOWING HOW THE CALCULATION FOR BENCH BLASTING IS CARRIED OUT:

Conditions:

| | |
|---|---|
| Bench height: | $K = 15$ m |
| Width of the round: | $w = 26$ m |
| Blasthole diameter: | $d = 76$ mm |
| Rock constant: | $c = 0.4$ |
| Hole inclination: | 3:1 |
| Explosive: | Emulite 150 in 65 mm plastic hoses dropt into the hole. |
| Charging condition: | Dry holes |

**Calculation of drilling pattern.**

**1. Maximum burden.**

$B_{max} = 1.45\sqrt{l_b}$

Charge concentration, $l_b$, is found in table 1a and is in this case 5.0 kg/m. No correction for hole inclination or rock constant.
$B_{max} = 1.45\ \sqrt{5.0} = 3.24$ m

**2. Subdrilling.**

$U = 0.3 \times B_{max}$ $\qquad$ $U = 0.3 \times 3.24 = 0.97$ m

**3. Depth of blasthole.**

$H = 1.05(K+U)$ $\qquad$ $H = 1.05(15.0+0.97) = 16.76$ m

**4. Error in drilling.**

$$E = \frac{d}{1000} + 0.03 \times H \qquad E = \frac{76}{1000} + 0.03 \times 16.76 = 0.58 \text{ m}$$

**5. Practical burden.**

$B = B_{max} - E$ $\qquad$ $B = 3.24 - 0.58 = 2.66$ m

**6. Practical spacing.**

$S = 1.25 \times B$ $\qquad$ $S = 1.25 \times 2.66 = 3.32$ m

**7. Adjustment for width of the round.**

$$\frac{w}{s} \qquad \frac{26.0}{3.32} = 7.83 = 8 \text{ spaces}$$

$$S_{adj} = \frac{\text{width}}{\text{No. of spaces/row}} \qquad S_{adj} = \frac{26.0}{8} = 3.25 \text{ m}$$

Note that the number of holes in a row is the number of spaces +1.

**8. Specific drilling.**

$$b = \frac{n \times H}{B \times K \times w} \qquad b = \frac{9 \times 16.76}{2.66 \times 15.0 \times 26.0} = 0.145 \text{ m/cu.m.}$$

**Calculation of charges.**

**9. Concentration of bottom charge.**

$l_b = l_b$ for the determination of $B_{max}$. In accordance with table 1 a. $\qquad l_b = 5.0$ kg/m

**10. Height of the bottom charge.**

$h_b = 1.3 \times B_{max} \qquad h_b = 1.3 \times 3.24 = 4.20$ m

**11. Weight of bottom charge.**

$Q_b = l_b \times h_b \qquad Q_b = 5.0 \times 4.20 = 21.0$ kg

**12. Stemming.**

$h_o = B \qquad h_o = 2.66$ m

**13. Concentration of column charge.**

$l_c = 40$ to $60\ \%$ of $l_b \qquad l_c = 0.5 \times 5.0 = 2.50$ kg/m

**14. Height of column charge.**

$h_c = H - h_b - h_o \qquad h_c = 16.76 - 4.20 - 2.66 = 9.90$ m

**15. Weight of column charge.**

$Q_c = l_c \times h_c \qquad Q_c = 2.50 \times 9.90 = 24.75$ kg

**16. Total charge.**

$Q_{tot} = Q_b + Q_c \qquad Q_{tot} = 21.00 + 24.75 = 45.75$ kg

**17. Specific charge.**

$$q = \frac{n \times Q_{tot}}{B \times K \times w} \qquad q = \frac{9 \times 45.75}{2.66 \times 15.0 \times 26.0} = 0.40 \text{ kg/cu.m.}$$

If the blast is not limited to a certain area, the specific drilling and specific charge will be lower.
In the above example, the specific charge will then be:

$$q = \frac{Q_{tot}}{B \times S \times K} \qquad q = \frac{45.75}{2.66 \times 3.32 \times 15.0} = 0.35 \text{ kg/cu.m.}$$

In quarrying there is no need to adjust the spacing between the holes and number of holes in accordance with the width of the cut.

Summary of important data:

| Bench height m | Hole depth m | Burden m | Spacing m | Bottom charge kg | Column charge kg | Specific drilling m/cu.m | Specific charge kg/cu.m |
|---|---|---|---|---|---|---|---|
| 15.0 | 16.8 | 2.65 | 3.25 | 21.0 | 24.8 | 0.145 | 0.40 |

**Drilling and charging tables.**

The following tables give the computed key data for different blasthole diameters.

The tables give the values for bench heights higher than $2 \times B_{max}$, lower benches will be dealt with separately (See Leveling).

The first two tables give the key data for drill series 11 and 12. These series are developed mainly for manual drilling.
The series is a series of drill rods with carbide-tipped bits, increasing 0.8 m in length and decreasing 1 mm in diameter between one drill rod and the next one.

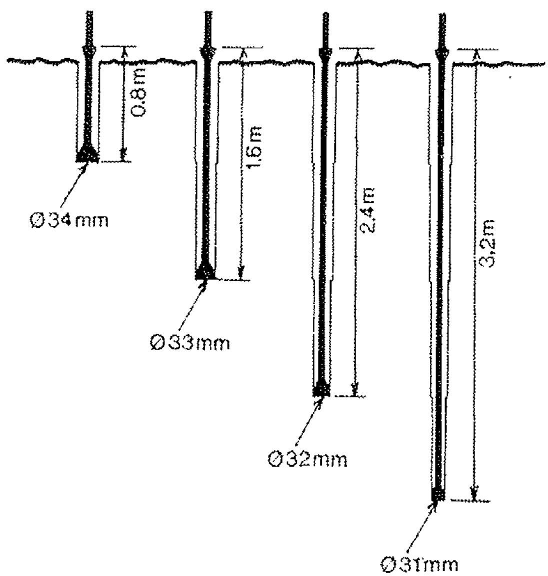



*Fig. 5.5 Drill series 11.*

The drilling and charging tables for Emulite 150 may also be used for Dynamex M.

Drilling and charging table for drill series 11.
Blasthole diameter 34–26 mm.

Explosive: Emulite 150
Hole inclination: 3:1

| | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|
| Bench height | K (m) | 2.0 | 2.5 | 3.0 | 3.5 | 4.0 | 4.5 | 5.0 | 5.5 |
| Hole diameter | d (mm) | 31 | 31 | 30 | 29 | 29 | 28 | 28 | 27 |
| Hole depth | H (m) | 2.50 | 3.05 | 3.55 | 4.10 | 4.60 | 5.10 | 5.60 | 6.15 |
| Practical burden | B (m) | 1.10 | 1.20 | 1.15 | 1.10 | 1.10 | 1.05 | 1.00 | 0.95 |
| Practical spacing | S (m) | 1.35 | 1.50 | 1.45 | 1.40 | 1.35 | 1.30 | 1.25 | 1.20 |
| Stemming | $h_o$ (m) | 1.10 | 1.20 | 1.15 | 1.10 | 1.10 | 1.05 | 1.00 | 0.95 |
| Bottom charge: | | | | | | | | | |
| Concentration | $l_b$ (kg/m) | 0.87 | 0.87 | 0.81 | 0.76 | 0.76 | 0.71 | 0.71 | 0.66 |
| Height | $h_b$ (m) | 1.40 | 1.70 | 1.70 | 1.65 | 1.65 | 1.60 | 1.60 | 1.55 |
| Weight | $Q_b$ (kg) | 1.20 | 1.50 | 1.40 | 1.25 | 1.25 | 1.15 | 1.15 | 1.00 |
| Column charge: | | | | | | | | | |
| Concentration | $l_c$ (m) | 0.44 | 0.44 | 0.41 | 0.38 | 0.38 | 0.36 | 0.36 | 0.33 |
| Height | $h_c$ (m) | 0.00 | 0.15 | 0.70 | 1.35 | 1.85 | 2.45 | 3.00 | 3.65 |
| Weight | $Q_c$ (kg) | 0.00 | 0.10 | 0.30 | 0.50 | 0.70 | 0.90 | 1.05 | 1.20 |
| Total charge | $Q_{tot}$ (kg) | 1.20 | 1.60 | 1.70 | 1.75 | 1.95 | 2.05 | 2.20 | 2.20 |
| Specific drilling | b (m/cu.m) | 0.842 | 0.678 | 0.710 | 0.761 | 0.776 | 0.830 | 0.896 | 0.980 |
| Specific charge | q (kg/cu m) | 0.40 | 0.36 | 0.34 | 0.33 | 0.33 | 0.33 | 0.35 | 0.35 |

The reduction of the diameter of the blasthole for each drill rod used has to be taken into account in the drill and charge calculations.

Drilling and charging table for drill series 12.
Blasthole diameter 40–29 mm.

Explosive: Emulite 150
Hole inclination: 3:1

| | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|
| Bench height | K (m) | 3.0 | 3.5 | 4.0 | 4.5 | 5.0 | 5.5 | 6.0 | 6.5 |
| Hole diameter | d (mm) | 36 | 35 | 35 | 34 | 33 | 33 | 32 | 32 |
| Hole depth | H (m) | 3.65 | 4.20 | 4.70 | 5.20 | 5.70 | 6.25 | 6.75 | 7.25 |
| Practical burden | B (m) | 1.40 | 1.35 | 1.35 | 1.30 | 1.25 | 1.20 | 1.15 | 1.15 |
| Practical spacing | S (m) | 1.75 | 1.75 | 1.70 | 1.60 | 1.55 | 1.50 | 1.45 | 1.40 |
| Stemming | $h_o$ (m) | 1.40 | 1.35 | 1.35 | 1.30 | 1.25 | 1.20 | 1.15 | 1.15 |
| Bottom charge: | | | | | | | | | |
| Concentration | $l_b$ (kg/m) | 1.17 | 1.11 | 1.11 | 1.04 | 0.98 | 0.98 | 0.92 | 0.92 |
| Height | $h_b$ (m) | 2.00 | 2.00 | 2.00 | 1.90 | 1.90 | 1.90 | 1.80 | 1.80 |
| Weight | $Q_b$ (kg) | 2.30 | 2.20 | 2.20 | 2.00 | 1.90 | 1.90 | 1.70 | 1.70 |
| Column charge: | | | | | | | | | |
| Concentration | $l_c$ (m) | 0.59 | 0.56 | 0.56 | 0.52 | 0.49 | 0.49 | 0.46 | 0.46 |
| Height | $h_c$ (m) | 0.25 | 0.85 | 1.35 | 2.00 | 2.55 | 3.15 | 3.80 | 4.30 |
| Weight | $Q_c$ (kg) | 0.15 | 0.50 | 0.75 | 1.05 | 1.25 | 1.55 | 1.75 | 2.00 |
| Total charge | $Q_{tot}$ (kg) | 2.45 | 2.70 | 2.95 | 3.05 | 3.15 | 3.45 | 3.45 | 3.70 |
| Specific drilling | b (m/cu.m) | 0.497 | 0.508 | 0.511 | 0.556 | 0.589 | 0.631 | 0.675 | 0.700 |
| Specific charge | q (kg/cu.m) | 0.33 | 0.33 | 0.33 | 0.33 | 0.33 | 0.35 | 0.35 | 0.36 |

If excavation is not carried out between the rounds, it may be necessary to increase the specific charge.

This can be done either by denser drilling or by increasing the concentration of the column charge.

Normally the latter alternative will suffice.

For hole inclinations other than 3:1, the correct burden B and spacing S are obtained by multiplying by the appropriate reduction factor in table 2, page 69.

Drilling and charging table for blasthole diameter 51 mm.

Explosive: Emulite 150
Hole inclination: 3:1

| | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|
| Bench height | K (m) | 4.0 | 4.5 | 5.0 | 5.5 | 6.0 | 6.5 | 7.0 | 7.5 |
| Hole diameter | d (mm) | 51 | 51 | 51 | 51 | 51 | 51 | 51 | 51 |
| Hole depth | H (m) | 4.90 | 5.40 | 5.90 | 6.50 | 7.00 | 7.50 | 8.00 | 8.60 |
| Practical burden | B (m) | 2.00 | 2.00 | 1.95 | 1.95 | 1.90 | 1.90 | 1.90 | 1.90 |
| Practical spacing | S (m) | 2.50 | 2.45 | 2.45 | 2.40 | 2.40 | 2.40 | 2.35 | 2.35 |
| Stemming | $h_o$ (m) | 2.00 | 2.00 | 1.95 | 1.95 | 1.90 | 1.90 | 1.90 | 1.90 |
| Bottom charge: | | | | | | | | | |
| Concentration | $l_b$ (kg/m) | 2.30 | 2.30 | 2.30 | 2.30 | 2.30 | 2.30 | 2.30 | 2.30 |
| Height | $h_b$ (m) | 2.90 | 2.90 | 2.90 | 2.90 | 2.90 | 2.90 | 2.90 | 2.90 |
| Weight | $Q_b$ (kg) | 6.70 | 6.70 | 6.70 | 6.70 | 6.70 | 6.70 | 6.70 | 6.70 |
| Column charge: | | | | | | | | | |
| Concentration | $l_c$ (m) | 1.15 | 1.15 | 1.15 | 1.15 | 1.15 | 1.15 | 1.15 | 1.15 |
| Height | $h_c$ (m) | 0.00 | 0.50 | 1.05 | 1.65 | 2.20 | 2.70 | 3.20 | 3.80 |
| Weight | $Q_c$ (kg) | 0.00 | 0.60 | 1.20 | 1.90 | 2.50 | 3.10 | 3.70 | 4.40 |
| Total charge | $Q_{tot}$ (kg) | 6.70 | 7.30 | 7.90 | 8.60 | 9.20 | 9.80 | 10.40 | 11.10 |
| Specific drilling | b (m/cu.m) | 0.245 | 0.245 | 0.247 | 0.252 | 0.256 | 0.253 | 0.256 | 0.257 |
| Specific charge | q (kg/cu.m) | 0.33 | 0.33 | 0.33 | 0.33 | 0.33 | 0.33 | 0.33 | 0.33 |

Drilling and charging table for blasthole diameter 64 mm.

Explosive: Emulite 150
Hole inclination: 3:1

| | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|
| Bench height | K (m) | 5.0 | 6.0 | 7.0 | 8.0 | 9.0 | 10.0 | 11.0 | 12.0 |
| Hole diameter | d (mm) | 64 | 64 | 64 | 64 | 64 | 64 | 64 | 64 |
| Hole depth | H (m) | 6.10 | 7.20 | 8.20 | 9.30 | 10.30 | 11.40 | 12.40 | 13.50 |
| Practical burden | B (m) | 2.55 | 2.50 | 2.45 | 2.45 | 2.40 | 2.40 | 2.35 | 2.30 |
| Practical spacing | S (m) | 3.15 | 3.15 | 3.10 | 3.05 | 3.00 | 2.95 | 2.95 | 2.90 |
| Stemming | $h_o$ (m) | 2.55 | 2.50 | 2.45 | 2.45 | 2.40 | 2.40 | 2.35 | 2.30 |
| Bottom charge: | | | | | | | | | |
| Concentration | $l_b$ (kg/m) | 3.70 | 3.70 | 3.70 | 3.70 | 3.70 | 3.70 | 3.70 | 3.70 |
| Height | $h_b$ (m) | 3.60 | 3.60 | 3.60 | 3.60 | 3.60 | 3.60 | 3.60 | 3.60 |
| Weight | $Q_b$ (kg) | 13.30 | 13.30 | 13.30 | 13.30 | 13.30 | 13.30 | 13.30 | 13.30 |
| Column charge: | | | | | | | | | |
| Concentration | $l_c$ (m) | 1.85 | 1.85 | 1.85 | 1.85 | 1.85 | 1.85 | 1.85 | 1.85 |
| Height | $h_c$ (m) | 0.00 | 1.10 | 2.15 | 3.25 | 4.30 | 5.40 | 6.45 | 7.60 |
| Weight | $Q_c$ (kg) | 0.00 | 2.00 | 4.00 | 6.00 | 8.00 | 10.00 | 12.00 | 14.00 |
| Total charge | $Q_{tot}$ (kg) | 13.30 | 15.30 | 17.30 | 19.30 | 21.30 | 23.30 | 25.30 | 27.30 |
| Specific drilling | b (m/cu.m) | 0.152 | 0.152 | 0.154 | 0.156 | 0.159 | 0.161 | 0.162 | 0.169 |
| Specific charge | q (kg/cu.m) | 0.33 | 0.32 | 0.32 | 0.32 | 0.33 | 0.33 | 0.33 | 0.34 |

For hole inclinations other than 3:1, the correct burden B and spacing S are obtained by multiplying by the appropriate reduction factor in table 2, page 69.

Drilling and charging table for blasthole diameter 76 mm.

Explosive: Emulite 150
Hole inclination: 3:1

| | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|
| Bench height | K (m) | 6.0 | 8.0 | 10.0 | 12.0 | 14.0 | 15.0 | 16.0 | 18.0 |
| Hole diameter | d (mm) | 76 | 76 | 76 | 76 | 76 | 76 | 76 | 76 |
| Hole depth | H (m) | 7.30 | 9.40 | 11.50 | 13.60 | 15.70 | 16.80 | 17.80 | 19.90 |
| Practical burden | B (m) | 2.95 | 2.85 | 2.80 | 2.75 | 2.70 | 2.65 | 2.60 | 2.55 |
| Practical spacing | S (m) | 3.65 | 3.60 | 3.55 | 3.45 | 3.35 | 3.30 | 3.30 | 3.20 |
| Stemming | $h_o$ (m) | 2.95 | 2.85 | 2.80 | 2.75 | 2.70 | 2.65 | 2.60 | 2.55 |
| Bottom charge: | | | | | | | | | |
| Concentration | $l_b$ (kg/m) | 5.00 | 5.00 | 5.00 | 5.00 | 5.00 | 5.00 | 5.00 | 5.00 |
| Height | $h_b$ (m) | 4.20 | 4.20 | 4.20 | 4.20 | 4.20 | 4.20 | 4.20 | 4.20 |
| Weight | $Q_b$ (kg) | 21.00 | 21.00 | 21.00 | 21.00 | 21.00 | 21.00 | 21.00 | 21.00 |
| Column charge: | | | | | | | | | |
| Concentration | $l_c$ (m) | 2.50 | 2.50 | 2.50 | 2.50 | 2.50 | 2.50 | 2.50 | 2.50 |
| Height | $h_c$ (m) | 0.15 | 2.35 | 4.50 | 6.65 | 8.80 | 9.95 | 11.00 | 13.15 |
| Weight | $Q_c$ (kg) | 0.40 | 5.90 | 11.30 | 16.60 | 22.00 | 24.90 | 27.50 | 32.90 |
| Total charge | $Q_{tot}$ (kg) | 21.40 | 26.90 | 32.30 | 37.60 | 43.00 | 45.90 | 48.50 | 53.90 |
| Specific drilling | b (m/cu.m) | 0.113 | 0.115 | 0.116 | 0.119 | 0.124 | 0.128 | 0.130 | 0.135 |
| Specific charge | q (kg/cu.m) | 0.33 | 0.33 | 0.33 | 0.33 | 0.34 | 0.35 | 0.35 | 0.37 |

Drilling and charging table for blasthole diameter 89 mm.

Explosive: Emulite 150
Hole inclination: 3:1

| | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|
| Bench height | K (m) | 8.0 | 10.0 | 12.0 | 14.0 | 15.0 | 16.0 | 18.0 | 20.0 |
| Hole diameter | d (mm) | 89 | 89 | 89 | 89 | 89 | 89 | 89 | 89 |
| Hole depth | H (m) | 9.60 | 11.70 | 13.80 | 15.90 | 17.00 | 18.00 | 20.10 | 22.20 |
| Practical burden | B (m) | 3.45 | 3.40 | 3.35 | 3.30 | 3.25 | 3.20 | 3.15 | 3.10 |
| Practical spacing | S (m) | 4.30 | 4.30 | 4.20 | 4.10 | 4.10 | 4.05 | 4.00 | 3.90 |
| Stemming | $h_o$ (m) | 3.45 | 3.40 | 3.35 | 3.30 | 3.25 | 3.20 | 3.15 | 3.10 |
| Bottom charge: | | | | | | | | | |
| Concentration | $l_b$ (kg/m) | 7.10 | 7.10 | 7.10 | 7.10 | 7.10 | 7.10 | 7.10 | 7.10 |
| Height | $h_b$ (m) | 5.00 | 5.00 | 5.00 | 5.00 | 5.00 | 5.00 | 5.00 | 5.00 |
| Weight | $Q_b$ (kg) | 35.50 | 35.50 | 35.50 | 35.50 | 35.50 | 35.50 | 35.50 | 35.50 |
| Column charge: | | | | | | | | | |
| Concentration | $l_c$ (m) | 3.55 | 3.55 | 3.55 | 3.55 | 3.55 | 3.55 | 3.55 | 3.55 |
| Height | $h_c$ (m) | 1.15 | 3.30 | 5.45 | 7.60 | 8.75 | 9.80 | 11.95 | 14.10 |
| Weight | $Q_c$ (kg) | 4.10 | 11.70 | 19.40 | 27.00 | 31.00 | 34.80 | 42.40 | 50.00 |
| Total charge | $Q_{tot}$ (kg) | 39.60 | 47.20 | 54.90 | 62.50 | 66.50 | 70.30 | 77.90 | 85.50 |
| Specific drilling | b (m/cu.m) | 0.081 | 0.081 | 0.082 | 0.084 | 0.085 | 0.087 | 0.089 | 0.092 |
| Specific charge | q (kg/cu.m) | 0.33 | 0.33 | 0.33 | 0.33 | 0.33 | 0.34 | 0.34 | 0.35 |

**EXAMPLE OF CHARGE CALCULATION FOR BENCH BLASTING WITH ANFO IN VERTICAL HOLES.**

Conditions:

| | |
|---|---|
| Bench height: | $K = 18$ m |
| Width of the round: | $w = 40$ m |
| Blasthole diameter: | $d = 102$ mm |
| Hole inclination: | Vertical |
| Explosive: | ANFO, poured into the blasthole |
| Charging condition: | Dry holes |

**Calculation of drilling pattern.**

**1. Maximum burden.**

$B_{max} = 1.36\sqrt{l_b} \times R_1$

Charge concentration, $l_b$, is found in table 1a and is in this case 6.5 kg/m. Correction for vertical drilling is found in table 2 and is 0.95. No correction for rock constant.

$B_{max} = 1.36 \times \sqrt{6.5} \times 0.95 = 3.29$ m

**2. Subdrilling.**

$U = 0.3 \times B_{max}$ $\qquad$ $U = 0.3 \times 3.29 = 0.99$ m

**3. Depth of blasthole.**

$H = K + U$ $\qquad$ $H = 18.0 + 0.99 = 19.00$ m

**4. Error in drilling.**

$$E = \frac{d}{1000} + 0.03 \times H \qquad E = \frac{102}{1000} + 0.03 \times 19.00 = 0.67 \text{ m}$$

**5. Practical burden.**

$B = B_{max} - E$ $\qquad$ $B = 3.29 - 0.67 = 2.62$ m

**6. Practical spacing.**

$S = 1.25 \times B$ $\qquad$ $S = 1.25 \times 2.62 = 3.27$ m

**7. Adjustment for width of round.**

$$\frac{w}{s} \qquad \frac{40}{3.27} = 12.23 = 13 \text{ spaces}$$

$$S_{adj} = \frac{\text{Width}}{\text{No. of spaces/row}} \qquad S_{adj} = \frac{40}{13} = 3.07 \text{ m}$$

Note that the number of holes in a row is the number of spaces +1.

**8. Specific drilling.**

$$b = \frac{n \times H}{B \times K \times w} \qquad b = \frac{14 \times 19.00}{2.62 \times 18 \times 40} = 0.141 \text{ m/cu.m.}$$

**Calculation of charge.**

In the case of ANFO the charge cannot be divided into bottom charge and column charge as it consists of only one column of charge with the same charge concentration.

**9. Stemming.**

$h_o = B$ $\qquad$ $h_o = 2.62$ m

**10. Charge concentration.**

$l_b = l_b$ for determination of $B_{max}$. According to table 1a. $\qquad$ $l_b = 6.5$ kg/m

**11. Height of charge.**

$h = H - h_o$ $\qquad$ $h = 19.00 - 2.62 = 16.38$ m

**12. Weight of charge.**

$Q = l_b \times h$ $\qquad$ $Q = 6.5 \times 16.38 = 106.5$ kg

**13. Specific charge.**

$$q = \frac{n \times Q}{B \times K \times w} \qquad q = \frac{14 \times 106.5}{2.62 \times 18 \times 40} = 0.79 \text{ kg/cu.m.}$$

Summary of important data:

| Bench height m | Hole depth m | Burden m | Spacing m | Charge kg | Specific drilling m/cu.m. | Specific charge kg/cu.m. |
|---|---|---|---|---|---|---|
| 18.0 | 19.0 | 2.62 | 3.07 | 106.5 | 0.141 | 0.79 |

The high specific charge depends to a large extent on the overcharged column part of the blast.

Drilling and charging tables for blastholes charged with ANFO.

For hole inclinations other than 3:1, the correct hole burden B and spacing S are obtained by multiplying by the appropriate reduction factor in table 2, page 69.

Drilling and charging table for blasthole diameter of 64 mm

Explosive: ANFO
Hole inclination: 3:1

| | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|
| Bench height | K (m) | 6.0 | 7.0 | 8.0 | 9.0 | 10.0 | 11.0 | 12.0 | 14.0 |
| Hole diameter | d (mm) | 64 | 64 | 64 | 64 | 64 | 64 | 64 | 64 |
| Hole depth | H (m) | 7.00 | 8.00 | 9.10 | 10.10 | 11.20 | 12.20 | 13.30 | 15.40 |
| Practical burden | B (m) | 1.90 | 1.90 | 1.85 | 1.80 | 1.80 | 1.75 | 1.70 | 1.65 |
| Practical spacing | S (m) | 2.40 | 2.35 | 2.30 | 2.30 | 2.25 | 2.20 | 2.20 | 2.10 |
| Stemming | $h_o$ (m) | 1.90 | 1.90 | 1.85 | 1.80 | 1.80 | 1.75 | 1.70 | 1.65 |
| Charge, ANFO: | | | | | | | | | |
| Concentration | $l_b$ (kg/m) | 2.60 | 2.60 | 2.60 | 2.60 | 2.60 | 2.60 | 2.60 | 2.60 |
| Height | h (m) | 4.60 | 5.60 | 6.75 | 7.80 | 8.90 | 9.95 | 11.10 | 13.25 |
| Weight | Q (kg) | 12.00 | 14.60 | 17.60 | 20.30 | 23.10 | 25.90 | 28.90 | 34.50 |
| Primer: Emulite 150. | 50×550 mm | 1.30 | 1.30 | 1.30 | 1.30 | 1.30 | 1.30 | 1.30 | 1.30 |
| Total charge | $Q_{tot}$ (kg) | 13.30 | 15.90 | 18.90 | 21.60 | 24.40 | 27.20 | 30.20 | 35.80 |
| Specific drilling | b (m/cu.m) | 0.256 | 0.256 | 0.267 | 0.271 | 0.277 | 0.288 | 0.296 | 0.317 |
| Specific charge | q (kg/cu.m) | 0.49 | 0.51 | 0.56 | 0.58 | 0.60 | 0.64 | 0.67 | 0.74 |

The specific charge increases with the hole depth as the uncharged part of the hole (stemming) is long compared to the bench height in lower benches. The use of primer will not effect the burden or spacing unless the primer is of considerable size.

Drilling and charging table for blasthole diameter of 76 mm.

Explosive: ANFO
Hole inclination: 3:1

| | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|
| Bench height | K (m) | 8.0 | 10.0 | 12.0 | 14.0 | 15.0 | 16.0 | 18.0 | 20.0 |
| Hole diameter | d (mm) | 76 | 76 | 76 | 76 | 76 | 76 | 76 | 76 |
| Hole depth | H (m) | 9.20 | 11.30 | 13.40 | 15.50 | 16.60 | 17.60 | 19.70 | 21.80 |
| Practical burden | B (m) | 2.20 | 2.20 | 2.10 | 2.05 | 2.00 | 2.00 | 1.90 | 1.85 |
| Practical spacing | S (m) | 2.80 | 2.70 | 2.65 | 2.55 | 2.50 | 2.45 | 2.40 | 2.30 |
| Stemming | $h_o$ (m) | 2.20 | 2.20 | 2.10 | 2.05 | 2.00 | 2.00 | 1.90 | 1.85 |
| Charge, ANFO: | | | | | | | | | |
| Concentration | $l_b$ (kg/m) | 3.60 | 3.60 | 3.60 | 3.60 | 3.60 | 3.60 | 3.60 | 3.60 |
| Height | h (m) | 6.50 | 8.60 | 10.80 | 12.95 | 14.10 | 15.10 | 17.30 | 19.45 |
| Weight | Q (kg) | 23.40 | 31.00 | 38.90 | 46.60 | 50.80 | 54.40 | 62.30 | 70.00 |
| Primer: Emulite 150. | 65×550 mm | 2.20 | 2.20 | 2.20 | 2.20 | 2.20 | 2.20 | 2.20 | 2.20 |
| Total charge | $Q_{tot}$ (kg) | 25.60 | 33.20 | 41.10 | 48.80 | 53.00 | 56.60 | 64.50 | 72.20 |
| Specific drilling | b (m/cu.m) | 0.187 | 0.190 | 0.200 | 0.212 | 0.221 | 0.224 | 0.240 | 0.256 |
| Specific charge | q (kg/cu.m) | 0.52 | 0.56 | 0.61 | 0.67 | 0.71 | 0.72 | 0.79 | 0.85 |

Drilling and charging tables for blastholes charged with ANFO.

For hole inclinations other than 3:1, the correct burden B and spacing S are obtained by multiplying by the appropriate reduction factor in table 2, page 69.

Drilling and charging table for blasthole diameter of 89 mm.

Explosive: ANFO
Hole inclination: 3:1

| | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|
| Bench height | K (m) | 8.0 | 10.0 | 12.0 | 14.0 | 15.0 | 16.0 | 18.0 | 20.0 |
| Hole diameter | d (mm) | 89 | 89 | 89 | 89 | 89 | 89 | 89 | 89 |
| Hole depth | H (m) | 9.40 | 11.50 | 13.60 | 15.70 | 16.70 | 17.80 | 19.90 | 22.00 |
| Practical burden | B (m) | 2.70 | 2.60 | 2.55 | 2.50 | 2.45 | 2.40 | 2.35 | 2.30 |
| Practical spacing | S (m) | 3.30 | 3.30 | 3.15 | 3.10 | 3.10 | 3.00 | 2.95 | 2.85 |
| Stemming | $h_o$ (m) | 2.70 | 2.60 | 2.55 | 2.50 | 2.45 | 2.40 | 2.35 | 2.30 |
| Charge, ANFO: | | | | | | | | | |
| Concentration | $l_b$ (kg/m) | 5.00 | 5.00 | 5.00 | 5.00 | 5.00 | 5.00 | 5.00 | 5.00 |
| Height | h (m) | 6.20 | 8.40 | 10.55 | 12.70 | 13.75 | 14.90 | 17.05 | 19.20 |
| Weight | Q (kg) | 31.00 | 42.00 | 52.80 | 63.50 | 68.70 | 74.50 | 85.30 | 96.00 |
| Primer: Emulite 150, | 65×550 mm | 2.20 | 2.20 | 2.20 | 2.20 | 2.20 | 2.20 | 2.20 | 2.20 |
| Total charge | $Q_{tot}$ (kg) | 33.20 | 44.20 | 55.00 | 65.70 | 70.90 | 76.70 | 87.50 | 98.20 |
| Specific drilling | b (m/cu.m) | 0.132 | 0.134 | 0.141 | 0.145 | 0.147 | 0.155 | 0.159 | 0.168 |
| Specific charge | q (kg/cu.m) | 0.47 | 0.52 | 0.57 | 0.61 | 0.62 | 0.67 | 0.70 | 0.75 |

Drilling and charging table for blasthole diameter of 102 mm.

Explosive: ANFO
Hole inclination: 3:1

| | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|
| Bench height | K (m) | 10.0 | 12.0 | 14.0 | 15.0 | 16.0 | 18.0 | 20.0 | 22.0 |
| Hole diameter | d (mm) | 102 | 102 | 102 | 102 | 102 | 102 | 102 | 102 |
| Hole depth | H (m) | 11.60 | 13.70 | 15.80 | 16.80 | 17.90 | 20.00 | 22.10 | 24.20 |
| Practical burden | B (m) | 3.00 | 2.95 | 2.90 | 2.85 | 2.85 | 2.75 | 2.70 | 2.65 |
| Practical spacing | S (m) | 3.80 | 3.70 | 3.60 | 3.60 | 3.55 | 3.50 | 3.40 | 3.30 |
| Stemming | $h_o$ (m) | 3.00 | 2.95 | 2.90 | 2.85 | 2.85 | 2.75 | 2.70 | 2.65 |
| Charge, ANFO: | | | | | | | | | |
| Concentration | $l_b$ (kg/m) | 6.50 | 6.50 | 6.50 | 6.50 | 6.50 | 6.50 | 6.50 | 6.50 |
| Height | h (m) | 8.10 | 10.25 | 12.40 | 13.45 | 14.55 | 16.75 | 18.90 | 21.05 |
| Weight | Q (kg) | 53.00 | 67.00 | 81.00 | 88.00 | 95.00 | 109.00 | 123.00 | 137.00 |
| Primer: Emulite 150, | 75×550 mm | 2.70 | 2.70 | 2.70 | 2.70 | 2.70 | 2.70 | 2.70 | 2.70 |
| Total charge | $Q_{tot}$ (kg) | 55.70 | 69.70 | 83.70 | 90.70 | 97.70 | 111.70 | 125.70 | 139.70 |
| Specific drilling | b (m/cu.m) | 0.102 | 0.104 | 0.108 | 0.109 | 0.111 | 0.115 | 0.120 | 0.126 |
| Specific charge | q (kg/cu.m) | 0.49 | 0.53 | 0.57 | 0.59 | 0.60 | 0.64 | 0.68 | 0.73 |

Drilling and charging tables for blastholes charged with ANFO.

For hole inclinations other than 3:1, the correct burden B and spacing S are obtained by multiplying by the appropriate reduction factor in table 2, page 69.

Drilling and charging table for blasthole diameter of 127 mm.

Explosive: ANFO
Hole inclination: 3:1

| | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|
| Bench height | K (m) | 10.0 | 12.0 | 14.0 | 15.0 | 16.0 | 18.0 | 20.0 | 22.0 |
| Hole diameter | d (mm) | 127 | 127 | 127 | 127 | 127 | 127 | 127 | 127 |
| Hole depth | H (m) | 11.90 | 14.00 | 16.10 | 17.10 | 18.20 | 20.30 | 22.40 | 24.50 |
| Practical burden | B (m) | 3.85 | 3.80 | 3.70 | 3.70 | 3.65 | 3.60 | 3.50 | 3.45 |
| Practical spacing | S (m) | 4.80 | 4.70 | 4.65 | 4.60 | 4.55 | 4.50 | 4.40 | 4.35 |
| Stemming | $h_o$ (m) | 3.85 | 3.80 | 3.70 | 3.70 | 3.65 | 3.60 | 3.50 | 3.45 |
| Charge, ANFO: | | | | | | | | | |
| Concentration | $l_b$ (kg/m) | 10.10 | 10.10 | 10.10 | 10.10 | 10.10 | 10.10 | 10.10 | 10.10 |
| Height | h (m) | 7.55 | 9.70 | 11.90 | 12.90 | 14.05 | 16.20 | 18.40 | 20.55 |
| Weight | Q (kg) | 76.00 | 98.00 | 120.00 | 130.00 | 142.00 | 164.00 | 186.00 | 208.00 |
| Primer: Emulite 150, | 75×550 mm | 2.70 | 2.70 | 2.70 | 2.70 | 2.70 | 2.70 | 2.70 | 2.70 |
| Total charge | $Q_{tot}$ (kg) | 78.70 | 100.70 | 122.70 | 132.70 | 144.70 | 166.70 | 188.70 | 210.70 |
| Specific drilling | b (m/cu.m) | 0.064 | 0.065 | 0.067 | 0.067 | 0.069 | 0.070 | 0.073 | 0.074 |
| Specific charge | q (kg/cu.m) | 0.43 | 0.47 | 0.51 | 0.52 | 0.54 | 0.57 | 0.61 | 0.64 |

Drilling and charging table for blasthole diameter of 152 mm

Explosive: ANFO
Hole inclination: 3:1

| | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|
| Bench height | K (m) | 12.0 | 14.0 | 15.0 | 16.0 | 18.0 | 20.0 | 22.0 | 24.0 |
| Hole diameter | d (mm) | 152 | 152 | 152 | 152 | 152 | 152 | 152 | 152 |
| Hole depth | H (m) | 14.20 | 16.30 | 17.40 | 18.40 | 20.50 | 22.60 | 24.70 | 26.80 |
| Practical burden | B (m) | 4.60 | 4.55 | 4.50 | 4.45 | 4.40 | 4.35 | 4.30 | 4.20 |
| Practical spacing | S (m) | 5.75 | 5.65 | 5.60 | 5.60 | 5.50 | 5.45 | 5.35 | 5.30 |
| Stemming | $h_o$ (m) | 4.60 | 4.55 | 4.50 | 4.45 | 4.40 | 4.35 | 4.30 | 4.20 |
| Charge, ANFO: | | | | | | | | | |
| Concentration | $l_b$ (kg/m) | 14.50 | 14.50 | 14.50 | 14.50 | 14.50 | 14.50 | 14.50 | 14.50 |
| Height | h (m) | 9.10 | 11.25 | 12.40 | 13.45 | 15.60 | 17.75 | 19.90 | 22.10 |
| Weight | Q (kg) | 132.00 | 163.00 | 180.00 | 195.00 | 226.00 | 257.00 | 289.00 | 320.00 |
| Primer: 2×Emulite 150, | 75×550 mm | 5.40 | 5.40 | 5.40 | 5.40 | 5.40 | 5.40 | 5.40 | 5.40 |
| Total charge | $Q_{tot}$ (kg) | 137.40 | 168.40 | 185.40 | 200.40 | 231.40 | 262.40 | 294.40 | 325.40 |
| Specific drilling | b (m/cu.m) | 0.045 | 0.045 | 0.046 | 0.046 | 0.047 | 0.048 | 0.049 | 0.050 |
| Specific charge | q (kg/cu.m) | 0.43 | 0.47 | 0.49 | 0.50 | 0.53 | 0.55 | 0.58 | 0.61 |

The main problem when blasting with ANFO is the breakage in the bottom part of the blast as the bottom part of the blasthole, more often than not contains water. If the ANFO is contained in plastic hoses, the diameter of the explosives column will be reduced and the charge concentration will not be big enough to break the constricted bottom part. If the ANFO is poured into the wet blasthole it will deteriorate rapidly and the effect will be the same.
A shortened bottom charge to reinforce the priming of the ANFO has shown to be effective. The height of the reinforced priming should be $0.4 \times B_{max}$, giving a bottom charge which is extended above the theoretical grade. The reinforced primer makes it possible to increase the burden and spacing with 7 % each. The savings in costs of drilling and secondary blasting of the toe are sufficient to justify the higher consumption of high explosives in the blast.

The following charging tables give the guide lines for blasting with Emulite 150 as reinforced primer and ANFO as column charge.

For hole inclinations other than 3:1, the correct burden B and spacing S are obtained by multiplying by the appropriate reduction factor in table 2, page 69.

Drilling and charging table for blasthole diameter of 64 mm.

Explosive: Reinforced primer Emulite 150, height $0.4 \times B_{max}$
Column charge ANFO

Hole inclination: 3:1

| | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|
| Bench height | K (m) | 6.0 | 7.0 | 8.0 | 9.0 | 10.0 | 11.0 | 12.0 | 14.0 |
| Hole diameter | d (mm) | 64 | 64 | 64 | 64 | 64 | 64 | 64 | 64 |
| Hole depth | H (m) | 7.10 | 8.10 | 9.20 | 10.20 | 11.30 | 12.30 | 13.40 | 15.50 |
| Practical burden | B (m) | 2.10 | 2.05 | 2.05 | 2.00 | 1.95 | 1.95 | 1.90 | 1.85 |
| Practical spacing | S (m) | 2.65 | 2.60 | 2.55 | 2.50 | 2.50 | 2.40 | 2.40 | 2.30 |
| Stemming | $h_o$ (m) | 2.10 | 2.05 | 2.05 | 2.00 | 1.95 | 1.95 | 1.90 | 1.85 |
| Primer Emulite 150: | | | | | | | | | |
| Concentration | $l_b$ (kg/m) | 3.70 | 3.70 | 3.70 | 3.70 | 3.70 | 3.70 | 3.70 | 3.70 |
| Height | $h_b$ (m) | 1.20 | 1.20 | 1.20 | 1.20 | 1.20 | 1.20 | 1.20 | 1.20 |
| Weight | $Q_b$ (kg) | 4.40 | 4.40 | 4.40 | 4.40 | 4.40 | 4.40 | 4.40 | 4.40 |
| Charge, ANFO: | | | | | | | | | |
| Concentration | $l_c$ (kg/m) | 2.60 | 2.60 | 2.60 | 2.60 | 2.60 | 2.60 | 2.60 | 2.60 |
| Height | $h_c$ (m) | 3.80 | 4.85 | 5.95 | 7.00 | 8.15 | 9.15 | 10.30 | 12.45 |
| Weight | $Q_c$ (kg) | 9.90 | 12.60 | 15.50 | 18.20 | 21.20 | 23.80 | 26.80 | 32.40 |
| Total charge | $Q_{tot}$ (kg) | 14.30 | 17.00 | 19.90 | 22.60 | 25.60 | 28.20 | 31.20 | 36.80 |
| Specific drilling | b (m/cu.m) | 0.213 | 0.217 | 0.220 | 0.227 | 0.232 | 0.239 | 0.245 | 0.260 |
| Specific charge | q (kg/cu.m) | 0.43 | 0.46 | 0.48 | 0.50 | 0.53 | 0.55 | 0.57 | 0.62 |

For hole inclinations other than 3:1, the correct burden B and spacing S are obtained by multiplying by the appropriate reduction factor in table 2, page 69.

Drilling and charging table for blasthole diameter of 76 mm.

Explosive: Reinforced primer Emulite 150, height $0.4 \times B_{max}$
Column charge ANFO

Hole inclination: 3:1

| | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|
| Bench height | K (m) | 8.0 | 10.0 | 12.0 | 14.0 | 15.0 | 16.0 | 18.0 | 20.0 |
| Hole diameter | d (mm) | 76 | 76 | 76 | 76 | 76 | 76 | 76 | 76 |
| Hole depth | H (m) | 9.30 | 11.40 | 13.50 | 15.60 | 16.65 | 17.70 | 19.80 | 21.90 |
| Practical burden | B (m) | 2.45 | 2.40 | 2.30 | 2.25 | 2.20 | 2.20 | 2.10 | 2.05 |
| Practical spacing | S (m) | 3.05 | 2.95 | 2.90 | 2.80 | 2.75 | 2.70 | 2.65 | 2.50 |
| Stemming | $h_0$ (m) | 2.45 | 2.40 | 2.30 | 2.25 | 2.20 | 2.20 | 2.10 | 2.05 |
| Primer Emulite 150: | | | | | | | | | |
| Concentration | $l_b$ (kg/m) | 5.00 | 5.00 | 5.00 | 5.00 | 5.00 | 5.00 | 5.00 | 5.00 |
| Height | $h_b$ (m) | 1.35 | 1.35 | 1.35 | 1.35 | 1.35 | 1.35 | 1.35 | 1.35 |
| Weight | $Q_b$ (kg) | 6.80 | 6.80 | 6.80 | 6.80 | 6.80 | 6.80 | 6.80 | 6.80 |
| Charge, ANFO: | | | | | | | | | |
| Concentration | $l_c$ (kg/m) | 3.60 | 3.60 | 3.60 | 3.60 | 3.60 | 3.60 | 3.60 | 3.60 |
| Height | $h_c$ (m) | 5.50 | 7.65 | 9.85 | 12.00 | 13.10 | 14.15 | 16.35 | 18.50 |
| Weight | $Q_c$ (kg) | 19.80 | 27.50 | 35.50 | 43.20 | 47.20 | 51.00 | 59.90 | 66.60 |
| Total charge | $Q_{tot}$ (kg) | 26.60 | 34.30 | 42.30 | 50.00 | 54.00 | 57.80 | 66.70 | 73.40 |
| Specific drilling | b (m/cu.m) | 0.156 | 0.161 | 0.169 | 0.177 | 0.184 | 0.186 | 0.198 | 0.214 |
| Specific charge | q (kg/cu.m) | 0.45 | 0.48 | 0.53 | 0.57 | 0.60 | 0.61 | 0.67 | 0.72 |

Drilling and charging table for blasthole diameter of 89 mm.

Explosive: Reinforced primer Emulite 150, height $0.4 \times B_{max}$
Column charge ANFO

Hole inclination: 3:1

| | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|
| Bench height | K (m) | 8.0 | 10.0 | 12.0 | 14.0 | 15.0 | 16.0 | 18.0 | 20.0 |
| Hole diameter | d (mm) | 89 | 89 | 89 | 89 | 89 | 89 | 89 | 89 |
| Hole depth | H (m) | 9.50 | 11.60 | 13.70 | 15.80 | 16.80 | 17.90 | 20.00 | 22.10 |
| Practical burden | B (m) | 2.90 | 2.85 | 2.80 | 2.75 | 2.70 | 2.65 | 2.60 | 2.50 |
| Practical spacing | S (m) | 3.70 | 3.60 | 3.50 | 3.40 | 3.35 | 3.35 | 3.20 | 3.15 |
| Stemming | $h_0$ (m) | 2.90 | 2.85 | 2.80 | 2.75 | 2.70 | 2.65 | 2.60 | 2.50 |
| Primer Emulite 150: | | | | | | | | | |
| Concentration | $l_b$ (kg/m) | 7.10 | 7.10 | 7.10 | 7.10 | 7.10 | 7.10 | 7.10 | 7.10 |
| Height | $h_b$ (m) | 1.60 | 1.60 | 1.60 | 1.60 | 1.60 | 1.60 | 1.60 | 1.60 |
| Weight | $Q_b$ (kg) | 11.40 | 11.40 | 11.40 | 11.40 | 11.40 | 11.40 | 11.40 | 11.40 |
| Charge, ANFO. | | | | | | | | | |
| Concentration | $l_c$ (kg/m) | 5.00 | 5.00 | 5.00 | 5.00 | 5.00 | 5.00 | 5.00 | 5.00 |
| Height | $h_c$ (m) | 5.00 | 7.15 | 9.30 | 11.45 | 12.50 | 13.65 | 15.80 | 18.00 |
| Weight | $Q_c$ (kg) | 25.00 | 35.80 | 46.50 | 57.30 | 62.50 | 68.30 | 79.00 | 90.00 |
| Total charge | $Q_{tot}$ (kg) | 36.40 | 47.20 | 57.90 | 68.70 | 73.90 | 79.70 | 90.40 | 101.40 |
| Specific drilling | b (m/cu.m) | 0.111 | 0.113 | 0.117 | 0.121 | 0.124 | 0.126 | 0.134 | 0.140 |
| Specific charge | q (kg/cu.m) | 0.42 | 0.46 | 0.49 | 0.52 | 0.54 | 0.56 | 0.60 | 0.64 |

For hole inclinations other than 3:1, the correct burden B and spacing S are obtained by multiplying by the appropriate reduction factor in table 2, page 69.

Drilling and charging table for blasthole diameter of 102 mm.

| | |
|---|---|
| Explosive: | Reinforced primer Emulite 150, height $0.4 \times B_{max}$<br>Column charge ANFO |
| Hole inclination: | 3:1 |

| | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|
| Bench height | K (m) | 10.0 | 12.0 | 14.0 | 15.0 | 16.0 | 18.0 | 20.0 | 22.0 |
| Hole diameter | d (mm) | 102 | 102 | 102 | 102 | 102 | 102 | 102 | 102 |
| Hole depth | H (m) | 11.70 | 13.80 | 15.90 | 17.00 | 18.00 | 20.10 | 22.20 | 24.20 |
| Practical burden | B (m) | 3.30 | 3.25 | 3.20 | 3.15 | 3.10 | 3.05 | 3.00 | 2.90 |
| Practical spacing | S (m) | 4.15 | 4.05 | 4.00 | 3.90 | 3.90 | 3.80 | 3.70 | 3.60 |
| Stemming | $h_0$ (m) | 3.30 | 3.25 | 3.20 | 3.15 | 3.10 | 3.05 | 3.00 | 2.90 |
| Primer Emulite 150: | | | | | | | | | |
| Concentration | $l_b$ (kg/m) | 9.30 | 9.30 | 9.30 | 9.30 | 9.30 | 9.30 | 9.30 | 9.30 |
| Height | $h_b$ (m) | 1.85 | 1.85 | 1.85 | 1.85 | 1.85 | 1.85 | 1.85 | 1.85 |
| Weight | $Q_b$ (kg) | 17.20 | 17.20 | 17.20 | 17.20 | 17.20 | 17.20 | 17.20 | 17.20 |
| Charge, ANFO: | | | | | | | | | |
| Concentration | $l_c$ (kg/m) | 6.50 | 6.50 | 6.50 | 6.50 | 6.50 | 6.50 | 6.50 | 6.50 |
| Height | $h_c$ (m) | 6.55 | 8.70 | 10.85 | 12.00 | 13.05 | 15.20 | 17.35 | 19.45 |
| Weight | $Q_c$ (kg) | 42.60 | 56.60 | 70.60 | 78.00 | 85.00 | 99.00 | 113.00 | 126.00 |
| Total charge | $Q_{tot}$ (kg) | 59.80 | 73.80 | 87.80 | 95.20 | 102.20 | 116.20 | 130.20 | 143.20 |
| Specific drilling | b (m/cu.m) | 0.085 | 0.087 | 0.089 | 0.092 | 0.093 | 0.096 | 0.100 | 0.105 |
| Specific charge | q (kg/cu.m) | 0.44 | 0.47 | 0.49 | 0.52 | 0.53 | 0.56 | 0.59 | 0.62 |

Drilling and charging table for blasthole diameter of 127 mm

| | |
|---|---|
| Explosive: | Reinforced primer Emulite 150, height $0.4 \times B_{max}$<br>Column charge ANFO |
| Hole inclination: | 3:1 |

| | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|
| Bench height | K (m) | 10.0 | 12.0 | 14.0 | 15.0 | 16.0 | 18.0 | 20.0 | 22.0 |
| Hole diameter | d (mm) | 127 | 127 | 127 | 127 | 127 | 127 | 127 | 127 |
| Hole depth | H (m) | 12.00 | 14.10 | 16.20 | 17.30 | 18.30 | 20.40 | 22.50 | 24.60 |
| Practical burden | B (m) | 4.20 | 4.15 | 4.10 | 4.05 | 4.00 | 3.95 | 3.90 | 3.80 |
| Practical spacing | S (m) | 5.25 | 5.20 | 5.10 | 5.05 | 5.00 | 4.90 | 4.80 | 4.75 |
| Stemming | $h_0$ (m) | 4.20 | 4.15 | 4.10 | 4.05 | 4.00 | 3.95 | 3.90 | 3.80 |
| Primer Emulite 150: | | | | | | | | | |
| Concentration | $l_b$ (kg/m) | 14.40 | 14.40 | 14.40 | 14.40 | 14.40 | 14.40 | 14.40 | 14.40 |
| Height | $h_b$ (m) | 2.30 | 2.30 | 2.30 | 2.30 | 2.30 | 2.30 | 2.30 | 2.30 |
| Weight | $Q_b$ (kg) | 33.00 | 33.00 | 33.00 | 33.00 | 33.00 | 33.00 | 33.00 | 33.00 |
| Charge, ANFO: | | | | | | | | | |
| Concentration | $l_c$ (kg/m) | 10.10 | 10.10 | 10.10 | 10.10 | 10.10 | 10.10 | 10.10 | 10.10 |
| Height | $h_c$ (m) | 5.50 | 7.65 | 9.80 | 10.95 | 12.00 | 14.15 | 16.30 | 18.50 |
| Weight | $Q_c$ (kg) | 56.00 | 77.00 | 99.00 | 111.00 | 121.00 | 143.00 | 165.00 | 187.00 |
| Total charge | $Q_{tot}$ (kg) | 89.00 | 110.00 | 132.00 | 144.00 | 154.00 | 176.00 | 198.00 | 220.00 |
| Specific drilling | b (m/cu.m) | 0.054 | 0.054 | 0.055 | 0.056 | 0.057 | 0.059 | 0.060 | 0.062 |
| Specific charge | q (kg/cu.m) | 0.40 | 0.42 | 0.45 | 0.47 | 0.48 | 0.51 | 0.53 | 0.55 |

As can be observed in the charge calculations, more drilling and explosives are needed in the case of blasting with ANFO compared with blasting operations with Emulite 150.

The higher consumption of explosives does not affect the overall economy of the blasting operation, as ANFO is an inexpensive blasting agent, but the increased use of accessories (cord, detonators, primers etc.) has to be considered.
The drilling cost will also increase considerably due to denser drilling pattern.

It is always a good habit to sit down and analyze the blasting operation before selecting the explosive, taking into consideration the following parameters:

* The explosive, cost per ton or cu.m. of blasted rock
* Detonators, –"–
* Cord, –"–
* Primers, –"–
* Drilling, –"–
* Blasting, –"–
* Secondary blasting, –"–
* Mucking, –"–
* Hauling, –"–
* Crushing, –"–

Besides the above parameters, consideration has to be given to weather conditions and ground water levels. Blasting agents of ANFO type have poor water resistance properties.

## 5.3 Low benches, leveling.

Leveling is the kind of blasting where the bench height is below $2 \times B_{max}$, but normally applies to bench heights under 1.0 m.

The blastholes for leveling generally have a small diameter, drill series 11 (34, 33, 32 .... mm), or smaller.

The design of the firing pattern is important in leveling. Due to the low bench heights and small burdens, the rock moves faster than in normal bench blasting, which calls for shorter delay times between the holes. If the delay time is too long, the protective effect of short delay firing will not occur, increasing the risk of flyrock.

The risk of flyrock is however always great in leveling and this is why the round should be given a heavy cover and several layers of splinter protective covering on top. (See also Chapter 5.9 Covering.)

To reduce drilling costs, 22 mm blastholes are now more widely used. This is because blasting with small diameter holes decreases the drilling cost and also the explosives cost due to better utilization of energy of the explosive compared with the conventional technique.
Using small diameter blastholes also reduces the risk of flyrock and ground vibrations.
For the charging of small diameter blastholes, Nitro Nobel produces a tube charge, Primex 17×150 mm which can be cut in suitable lengths.

Leveling with drill series 11.

Drilling and charging table for drillseries 11.
Blasthole diameters 34–26 mm.

Explosive: Emulite 150.
Hole inclination: 3:1

| | | | | | | | |
|---|---|---|---|---|---|---|---|
| Bench height | K (m) | 0.20 | 0.30 | 0.40 | 0.60 | 0.80 | 1.00 |
| Hole diameter | d (mm) | 34 | 34 | 34 | 34 | 33 | 33 |
| Hole depth | H (m) | 0.60 | 0.60 | 0.60 | 0.90 | 1.10 | 1.40 |
| Practical burden | B (m) | 0.40 | 0.40 | 0.40 | 0.50 | 0.60 | 0.80 |
| Practical spacing | S (m) | 0.50 | 0.50 | 0.50 | 0.65 | 0.75 | 1.00 |
| Bottom charge: | | | | | | | |
| Concentration | $l_b$ (kg/m) | 1.00 | 1.00 | 1.00 | 1.00 | 1.00 | 1.00 |
| Height | $h_b$ (m) | 0.05 | 0.05 | 0.05 | 0.10 | 0.20 | 0.40 |
| Weight | $Q_b$ (kg) | 0.05 | 0.05 | 0.05 | 0.10 | 0.20 | 0.40 |
| Column charge: | $Q_c$ (kg) | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 |
| Total charge | $Q_{tot}$ (kg) | 0.05 | 0.05 | 0.05 | 0.10 | 0.20 | 0.40 |
| Stemming | $h_o$ (m) | 0.50 | 0.50 | 0.50 | 0.80 | 0.90 | 1.00 |
| Specific drilling | b (m/sq.m) | 3.00 | 3.00 | 3.00 | 2.80 | 2.45 | 1.75 |
| Specific charge | q (kg/sq.m) | 0.25 | 0.25 | 0.25 | 0.31 | 0.45 | 0.50 |
| Specific charge | q' (kg/cu.m) | 1.25 | 0.83 | 0.63 | 0.51 | 0.56 | 0.50 |

The specific drilling and specific charge are calculated in relation to the blasted area expressed in m/sq.m. and kg/sq.m.

The consumption of detonators and explosives is high in low bench blasting. The cost of drilling is high due to the close drilling pattern.

If the bottom of the cut is not restricted to a certain level, it is economically favorable to increase the subdrilling and subsequently increase the spacing between the blastholes.
As a consequence of the deeper drilling, the risk of flyrock will be reduced.

A suitable drill depth is 1.6 m which is the length of the second drillrod in the integral series No. 11. The diameter of the drillbit is 33 mm.

Comparison of consumption of explosives and detonators as well as specific drilling for a bench height of 0.4 m when drilled conventionally and with increased subdrilling.

| | Conventional | Increased subdrilling |
|---|---|---|
| Bench height | 0.4 m | 0.4 m |
| Hole depth | 0.6 m | 1.6 m |
| Practical burden | 0.4 m | 0.9 m |
| Practical spacing | 0.5 m | 1.1 m |
| Charge of explosives per hole | 0.05 kg | 0.5 kg |
| Charge of explosives per sq.m. | 0.25 kg | 0.5 kg |
| Number of detonators per sq.m. | 5 pcs | 1 pc |
| Drilling per sq.m. | 3.0 m | 1.6 m |

The higher specific charge in the case of increased subdrilling is well compensated by the lesser consumption of detonators and the decreased specific drilling.

In certain cases larger blasthole sizes must be used for low benches due to availability of equipment and drillbits.

The following tables are recommendations for blasting of low benches with blasthole diameters of 51, 64 and 76 mm.

Drilling and charging table for low benches
Blasthole diameter 51 mm.

Explosive: Emulite 150
Hole inclination: 3:1

| | | | | | |
|---|---|---|---|---|---|
| Bench height | K (m) | 1.00 | 1.50 | 2.00 | 2.50 |
| Hole diameter | d (mm) | 51 | 51 | 51 | 51 |
| Hole depth | H (m) | 1.40 | 2.00 | 2.60 | 3.20 |
| Practical burden | B (m) | 0.80 | 1.00 | 1.30 | 1.50 |
| Practical spacing | S (m) | 1.00 | 1.20 | 1.60 | 1.90 |
| Bottom charge: | | | | | |
| Concentration | $l_b$ (kg/m) | 1.50 | 1.50 | 1.50 | 1.50 |
| Height | $h_b$ (m) | 0.20 | 0.50 | 1.00 | 1.60 |
| Weight | $Q_b$ (kg) | 0.30 | 0.75 | 1.50 | 2.40 |
| Explosives dimension | (mm) | 40 | 40 | 40 | 40 |
| Column charge: | $Q_c$ (kg) | 0.10 | 0.10 | 0.20 | 0.30 |
| Total charge | $Q_{tot}$ (kg) | 0.40 | 0.85 | 1.70 | 2.70 |
| Stemming | $h_o$ (m) | 1.10 | 1.20 | 1.30 | 1.50 |
| Specific drilling | b (m/cu.m) | 1.75 | 1.11 | 0.63 | 0.45 |
| Specific charge | q (kg/cu.m) | 0.50 | 0.47 | 0.41 | 0.38 |

Drilling and charging table for low benches.
Blasthole diameter 64 mm.

Explosive: Emulite 150
Hole inclination: 3:1

| | | | | | |
|---|---|---|---|---|---|
| Bench height | K (m) | 1.00 | 2.00 | 3.00 | 4.00 |
| Hole diameter | d (mm) | 64 | 64 | 64 | 64 |
| Hole depth | H (m) | 1.40 | 2.70 | 3.80 | 4.90 |
| Practical burden | B (m) | 0.80 | 1.30 | 1.60 | 2.10 |
| Practical spacing | S (m) | 1.00 | 1.60 | 2.00 | 2.60 |
| Bottom charge: | | | | | |
| Concentration | $l_b$ (kg/m) | 2.20 | 2.20 | 2.20 | 2.20 |
| Height | $h_b$ (m) | 0.15 | 0.80 | 1.50 | 2.70 |
| Weight | $Q_b$ (kg) | 0.30 | 1.80 | 3.30 | 6.00 |
| Explosives dimension | (mm) | 50 | 50 | 50 | 50 |
| Column charge: | $Q_c$ (kg) | 0.10 | 0.10 | 0.50 | 0.50 |
| Total charge | $Q_{tot}$ (kg) | 0.40 | 1.90 | 3.80 | 6.50 |
| Stemming | $h_o$ (m) | 1.10 | 1.50 | 1.60 | 2.00 |
| Specific drilling | b (m/cu.m) | 1.75 | 0.65 | 0.40 | 0.22 |
| Specific charge | q (kg/cu.m) | 0.50 | 0.46 | 0.40 | 0.30 |

Drilling and charging table for low benches.
Blasthole diameter 76 mm.

Explosive: Emulite 150
Hole inclination: 3:1

| | | | | | | | |
|---|---|---|---|---|---|---|---|
| Bench height | K (m) | 1.00 | 2.00 | 3.00 | 4.00 | 5.00 | 6.00 |
| Hole diameter | d (mm) | 76 | 76 | 76 | 76 | 76 | 76 |
| Hole depth | H (m) | 1.60 | 2.60 | 3.80 | 5.00 | 6.20 | 7.40 |
| Practical burden | B (m) | 1.10 | 1.30 | 1.50 | 1.70 | 2.00 | 2.60 |
| Practical spacing | S (m) | 1.30 | 1.60 | 1.80 | 2.10 | 2.50 | 3.20 |
| Bottom charge: | | | | | | | |
| Concentration | $l_b$ (kg/m) | 1.51 | 1.50 | 2.40 | 2.40 | 3.50 | 4.50 |
| Height | $h_b$ (m) | 0.38 | 1.00 | 1.10 | 2.00 | 2.00 | 3.00 |
| Weight | $Q_b$ (kg) | 0.57 | 1.50 | 2.60 | 4.80 | 7.00 | 13.50 |
| Explosives dimension | (mm) | 40 | 40 | 50 | 50 | 60 | 70 |
| Column charge: | $Q_c$ (kg) | 0.00 | 0.20 | 0.60 | 0.80 | 3.00 | 4.60 |
| Total charge | $Q_{tot}$ (kg) | 0.57 | 1.70 | 3.20 | 5.60 | 10.00 | 18.10 |
| Stemming | $h_o$ (m) | 1.20 | 1.30 | 1.50 | 1.70 | 2.00 | 2.60 |
| Specific drilling | b (m/cu.m) | 1.12 | 0.63 | 0.47 | 0.35 | 0.25 | 0.15 |
| Specific charge | q (kg/cu.m) | 0.40 | 0.41 | 0.40 | 0.39 | 0.40 | 0.36 |

**Leveling with mini-hole technique.**

Nitro Nobel has developed a new technique for rock blasting in sensitive environments. The new technique, blasting with 22 mm blastholes and small charges, is presented in depth in Chapter 12.10 Mini-hole blasting.

## 5.4 Secondary blasting.

Secondary blasting means the treatment of boulders that are still considered big enough to cause obstruction in subsequent operations like excavation, transport and crushing.

The handling of boulders is normally very expensive and the aim in all blasting operations must be to avoid secondary blasting. Careful planning and execution of a blast may decrease the need of secondary blasting to a minimum. (See Chapter 5.6 Fragmentation.)

As it is impossible to completely avoid boulders in blasting operations, the problem has to be taken care of by blasting.

The most widely used method of blasting boulders is by drilling one or more blastholes in them. The blasthole/s is/are drilled so that the explosive can be placed in the center of the mass of the boulder, which requires a hole slightly deeper than half the thickness of the boulder.

As boulders resulting from blasting have been exposed to stresses during the blast, they contain a lot of microscopic cracks, thus making them relatively easy to blast compared with natural stones. (See Chapter 12.1 Blasting of natural boulders.)

For blasting of boulders resulting from blasts, a specific charge of approx. 0.06 kg/cu.m. is needed.

**Secondary blasting.**

| Size of boulder cu.m. | Thickness of boulder m | Depth of blasthole m | Number of blastholes | Charge kg/hole |
|---|---|---|---|---|
| 0.5 | 0.8 | 0.45 | 1 | 0.03 |
| 1.0 | 1.0 | 0.55 | 1 | 0.06 |
| 2.0 | 1.0 | 0.55 | 2 | 0.06 |
| 3.0 | 1.5 | 0.80 | 2 | 0.09 |

When several blastholes are used in a boulder, the initiation should be carried out with instantaneous detonators or detonators with the same interval number.

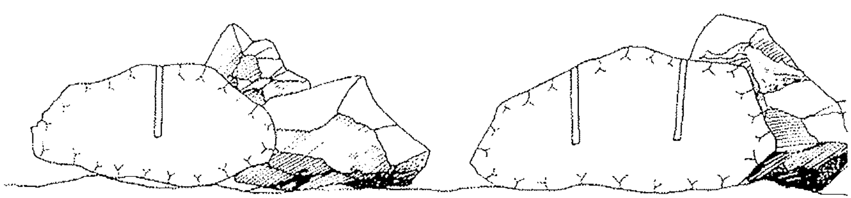


*Fig. 5.6 Secondary blasting.*

## 5.5 Opening of the bench.

Generally, bench blasting is understood to be blasting of vertical, or close to vertical, blastholes towards a free face.
Occasionally, no free face is available and the conditions for bench blasting have to be created.

The simplest way to do this is by using a fan cut of the type shown below.

The drilling pattern and charging of the blasthole depend upon the blasthole diameter and explosive used. Guide values for drilling and charge calculations are found in Chapter 5.2 Charge calculations.

Note that the burden must be calculated in relation to the charge concentration in the bottom of the hole.
The fan cut creates a certain risk of flyrock. Care must therefore be taken close to inhabited areas, especially if the blasthole diameter is greater than 40 mm.

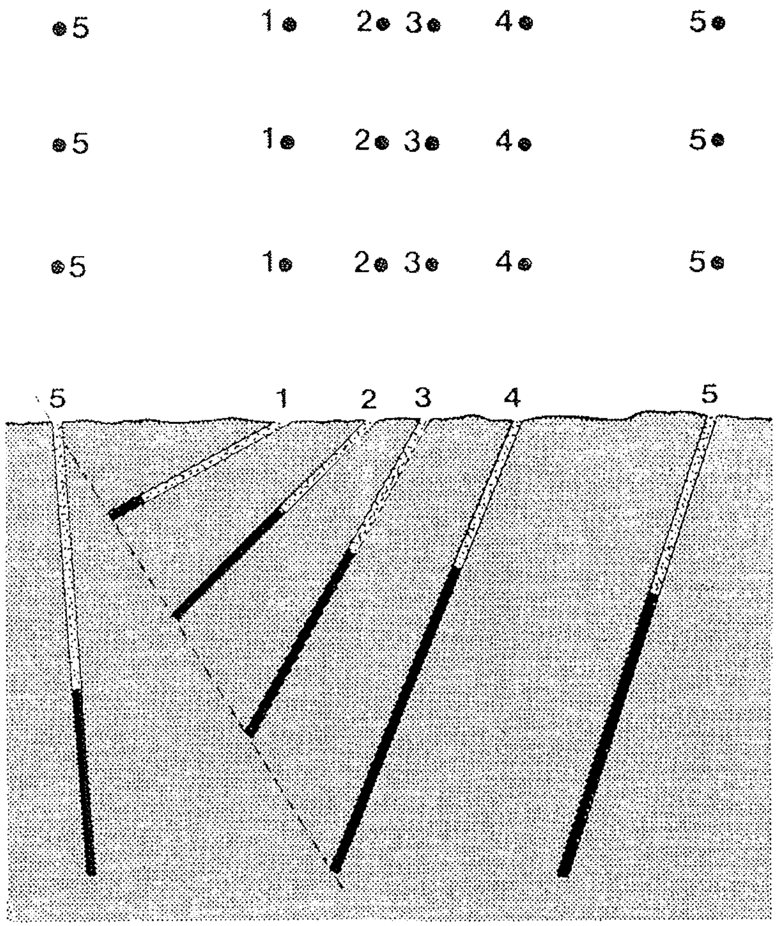



*Fig. 5.7 Opening of the bench.*

## 5.6 Rock fragmentation.

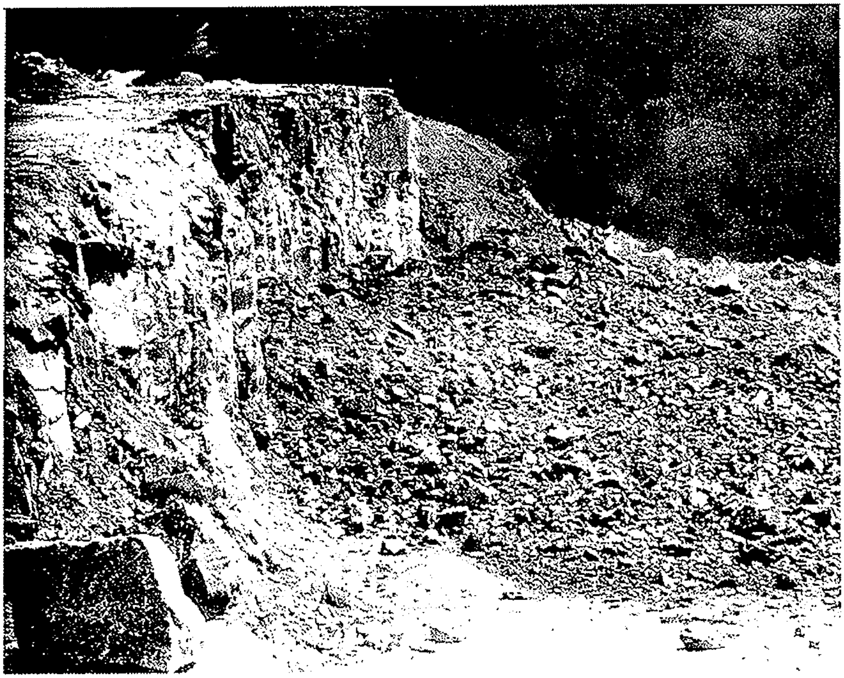


*Fig. 5.8 Blast with good rock fragmentation, result of blast in Fig. 4.1.*

### 5.6.1 Small rock fragmentation.

Good rock fragmentation is a subjective matter and depends on the end use of the rock.

The desired degree of fragmentation also depends upon the type and size of equipment which is used for the subsequent handling of the rock.

Large loaders, trucks and crushers can allow larger fragmentation, but it is a common misconception that larger fragmentation can be allowed because large loading, transport and crushing equipment is used. The large size equipment is designed to handle large volumes of material, not large size material.

The ideally fragmented rock is the rock that needs no further treatment after the blast. Therefore, the parameters for the subsequent operations are the guide lines for deciding on the desired fragmentation of the rock. If the rock is just to be transported to a dumping area, it should be easy to load and transport. If the rock is intended for crushing, the size of the largest boulders should not exceed 75 per cent of the length of the shortest side of the opening of the primary crusher, thus allowing a free flow through the plant.

Since the size of the broken rock is of the utmost importance for the subsequent operations, all possible efforts have to be made to keep the size down.

In bench blasting, the fragmentation is influenced by the following factors:

* The geology of the rock (faults, voids etc.)
* Specific drilling
* Specific charge
* Drilling pattern
* Firing pattern
* Hole inclination
* Hole deviation
* Size of the round

By considering the above factors during the drilling and blasting operation, it is possible to influence the fragmentation. However, it is not possible to make a completely reliable calculation beforehand. Test blasting of some rows is a good way to obtain some impression of the blasting characteristics of the rock.

The **geology** of the rock frequently affects the fragmentation more than the explosive used in the blast. The properties that influence the result of the blast are compression strength, tensile strength, density, propagation velocity, hardness and structure.
Most rocks have a **tensile strength** which is 8 to 10 times lower than the **compression strength.** This property is an important factor in rock blasting. The rock's tensile strength has to be exceeded, otherwise the rock will not break.

**Compression and tensile strengths of different rocks.**

| | Compression strength kg/sq.cm. | Tensile strength kg/sq.cm. |
|---|---|---|
| Granite | 2000–3600 | 100–300 |
| Diabase | 2900–4000 | 190–300 |
| Marble | 1500–1900 | 150–200 |
| Limestone | 1300–2000 | 170–300 |
| Sandstone (hard) | approx. 3000 | approx. 300 |

Rock with high **density** is normally harder to blast than a low density rock because the heavier rock masses require more explosives for the displacement of the rock.

The **propagation velocity** varies with different kinds of rock. Field tests have shown that hard rocks with high propagation velocity are best fragmented by an explosive with high velocity of detonation (VOD). In consequence a rock with low propagation velocity may be blasted with explosives with low VOD. EMULITE and DYNAMEX with a VOD of 5000 to 6000 m/sec. are suitable for blasting granite, marble and diabase (propagation velocity 4000 to 7000 m/sec.) while ANFO is suitable for limestone, sandstone etc. with low propagation velocities.

The **hardness** or brittleness of the rock can have a great effect on the blasting result. Soft rock is more "forgiving" than hard rock. If soft rock is somewhat undercharged, it will still be muckable and if it is somewhat overcharged, excessive throw rarely occurs. On the other hand, undercharging of hard rock frequently results in a tight and blocky muckpile that is tough to excavate. Overcharging of hard rock may cause flyrock and airblast. The design of blasts in hard rock requires tighter control than in soft rock.

Granite, gneiss and marble represent the hard rock while soft limestone and shale are considered soft.

The structure of the rock should be documented before the blasting works start. The direction, severity and spacing between the joint sets should be mapped out so that drilling and firing patterns can be adjusted to the prevailing conditions.

The planning of the drilling with respect to the direction of the joints is very important.

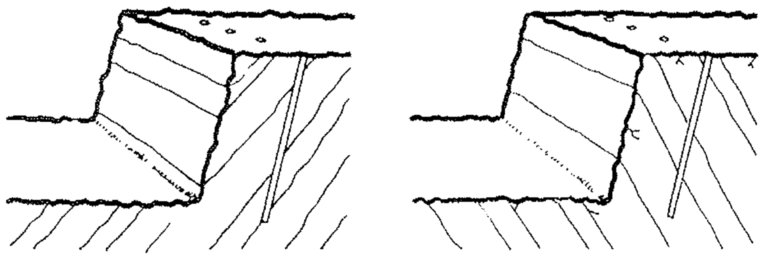


*Fig. 5.9*

**Advantages**
Effectively exploits the energy in the explosive. Good displacement of the blasted rock giving good digging conditions. Normally no stumps in the bottom part.
(no toe-problem)

**Disadvantages**
Back break

**Advantages**
Reduced over break.

**Disadvantages**
Bad displacement with tighter muckpile. Risk for stumps in the bottom part. Overhang may occur in the back row.

When the rock is full of faults and incompetent zones, much of the explosive's energy is lost in the faults instead of being used to break the rock. Alternate zones of competent and incompetent rock normally result in too blocky fragmentation. Higher specific charge will rarely correct this problem; it will only increase the risk of flyrock. The best way to lessen the problem is to use smaller blastholes with a closer drilling pattern in order to obtain better distribution of the explosives in the rock. The explosive charges should be concentrated in the competent rock while the faults and incompetent zones should be stemmed if possible.

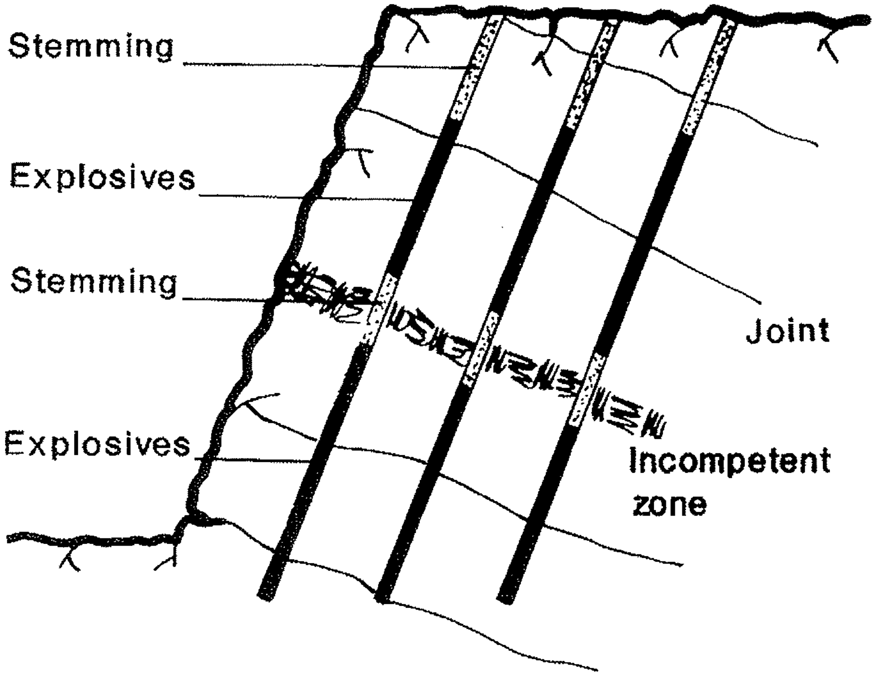



*Fig. 5.10 Stemming of incompetent zones.*

The collar part of the blasthole, which contains the stemming, has an unfavorable effect on the rock fragmentation.
As a general rule a collar distance equal to the burden is left uncharged. In tough rock and rock with horizontal planes, the uncharged part of the hole will cause problems in form of an increased amount of boulders. To improve the blasting result, the following steps may be taken:

* Shorten the collar distance, thus charging higher in the blasthole.
* Drilling of relievers in the stemming section of the blast.

Higher charges in the blasthole can only be recommended when a large area can be evacuated and no delicate structures are in the vicinity of the blast, due to the increased risk of flyrock.
Occasionally a small charge in the stemming section of the blasthole can improve the result of the blast.

The drilling of relievers between the main blastholes helps to break the upper part of the round.

The relievers are short holes, normally with smaller diameter than the main blastholes.
The drilling of relievers between the blastholes does not usually make economic sense. More often than not, it is advisable to tolerate a certain amount of boulders from the blast and break them by secondary blasting or other method.

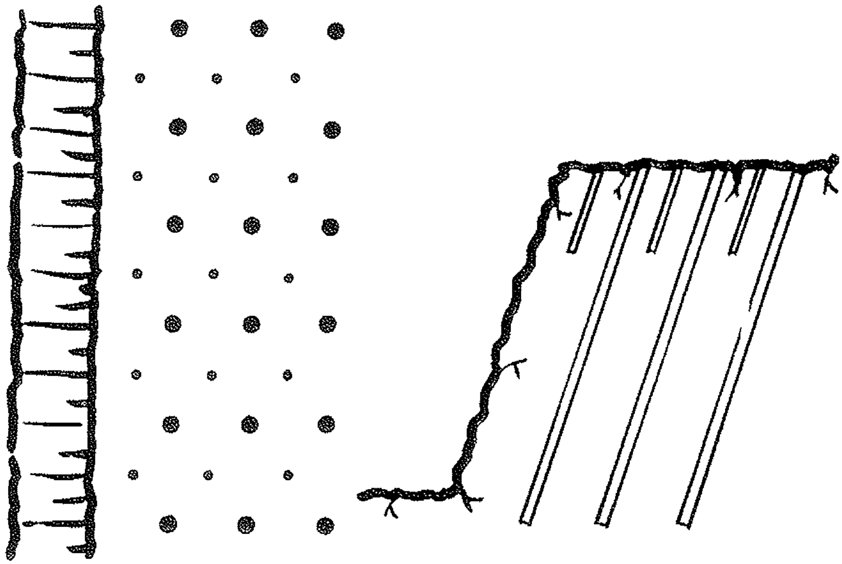


*Fig. 5.11 Relievers between the main blastholes.*

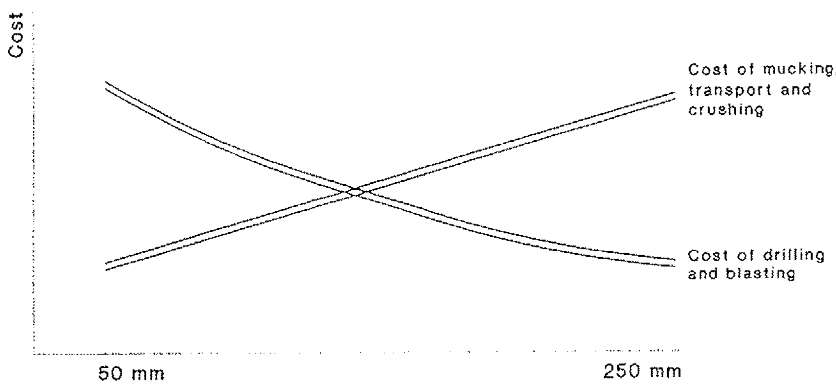



*Fig. 5.12 Effect of small and large diameter blastholes on cost.*

**Specific drilling.**

The size of the blasthole is the first consideration of any blast design. The blasthole diameter together with the explosive used will determine burden, spacing and hole depth.
Practical hole diameters for bench blasting range from 30 mm to 400 mm.

Generally the cost of large diameter drilling is cheaper per cubic meter rock than small diameter drilling. Furthermore, cheaper blasting agents can be used in large diameter blastholes.

The large diameter blasthole pattern gives a relatively low drilling and blasting cost. However, in geologically difficult situations the blasted material will be blocky, resulting in high mucking, transport and crushing cost as well as requiring more secondary breakage.

The geological structure is a major factor in determining the blasthole diameter. Joints and planes tend to isolate large blocks of rock in the burden area. The larger the drilling pattern, the greater the risk that these are left unbroken.

Higher specific drilling with smaller diameter blastholes distributes the explosives better in the rock resulting in better rock fragmentation.

Lately, button bits have replaced insert bits to a great extent thanks to their excellent drilling characteristics and convenience in use (easier sharpening of the bit and longer intervals between the grindings). Added to the above advantages is a life span which is twice that of an insert bit.
Because of the construction of the button bit, the diameter decreases with wear and when the bit is worn out, the diameter is up to 15 mm smaller than that of a new bit.

If the same burden and spacing is maintained during the entire life span of the bit, the fragmentation tends to be blockier at the end of the life span, due to the smaller diameter resulting in a smaller specific charge.

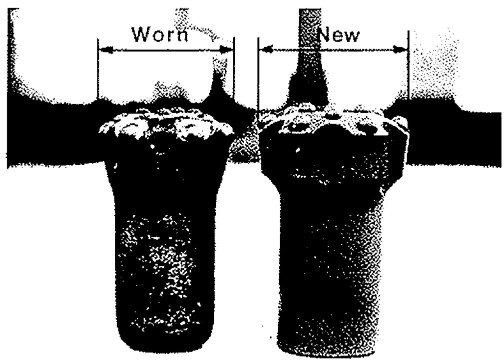



*Fig. 5.13 Comparison of diameter of new and worn button bit.*

**Specific charge.**

Rock will be broken up more if the specific charge is increased and the drilling pattern maintained.

The bottom part of the blast usually has the optimal specific charge and the fragmentation in this part is normally satisfactory.

The increase in specific charge can only be done in the column and stemming parts of the blast. The fragmentation will then be better but a greater forward movement of the rock has to be expected as well as an increased risk of flyrock.

**Drilling pattern.**

The typical drilling pattern has a spacing/burden ratio of 1.25 (S/B=1.25), which has proved to give good rock fragmentation in multiple row blasting.

In the 70s, tests were carried out in Sweden with wide-space hole blasting with S/B ratios greater than 1.25. The results of the tests showed improved fragmentation up to an S/B ratio of 8. The method is now common practice in Swedish quarries.

The burden and spacing must be normal in the first row, otherwise the burden will be too small, increasing the risk of flyrock.

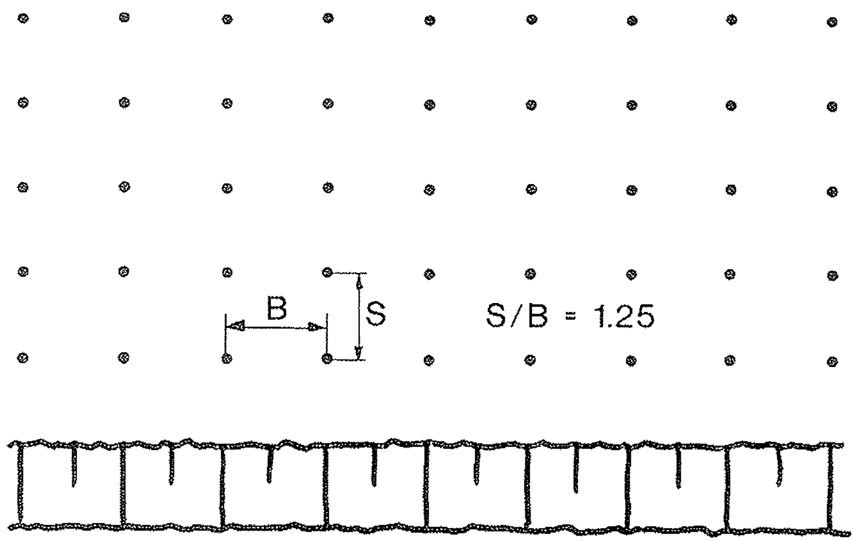



*Fig. 5.14 Normal drilling pattern.*

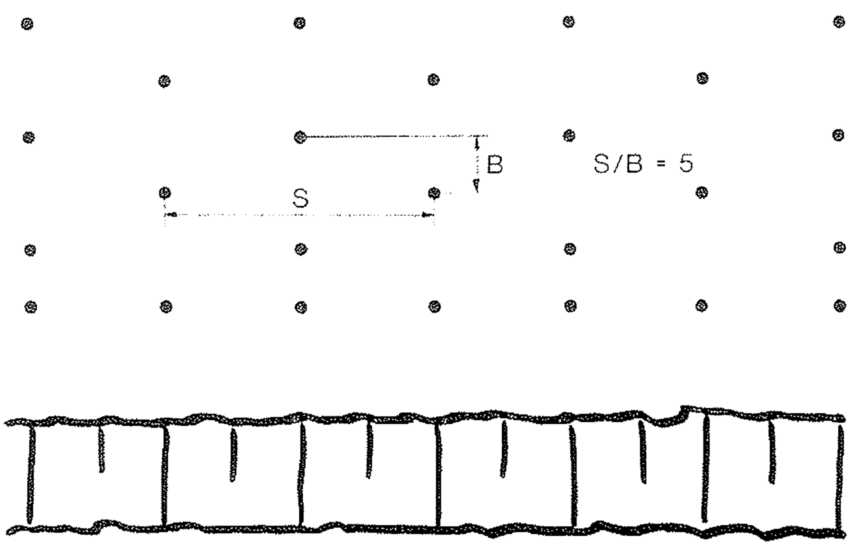



*Fig. 5.15 Wide-space blasting.*

**Firing pattern.**

Bench blasting is normally carried out as short delay blasting. The firing pattern has to be designed so that each blasthole has free breakage.

The delay time between blastholes and between rows has to be long enough to create space for the blasted rock from the succeeding rows.

Bernt Larsson of Nitro Nobel has studied the effect of the delay time on multiple row blastings. He states that the rock must be allowed to move 1/3 of the burden distance before the next row is allowed to detonate. The delay time between the rows may vary from 10 ms/m (hard rock) to 30 ms/m (soft rock) but generally 15 ms/m of the burden distance is a good guide value.

This length of delay gives good fragmentation and controls flyrock. It also gives the burden from the previously fired holes enough time to move forward to accommodate the broken rock from subsequent rows.

If the delay between the rows is too short, the rock from the back rows tends to take an upward direction instead of a horizontal. On the other hand, too long a delay may cause flyrock, airblast and boulders, as the protection from previously fired rows disappears due to too great a rock movement between detonations. The increase in boulders is due to the fact that the blast in this case may be compared with a single row blast.

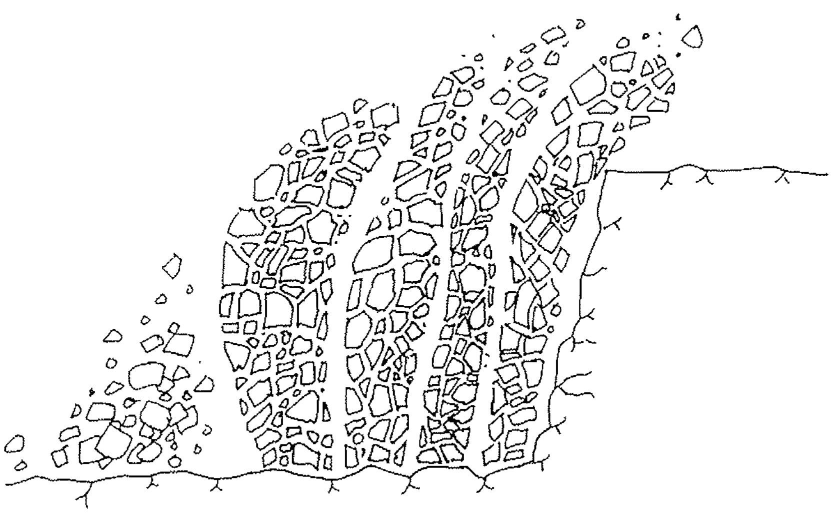


*Fig. 5.16 Too short a delay between rows.*

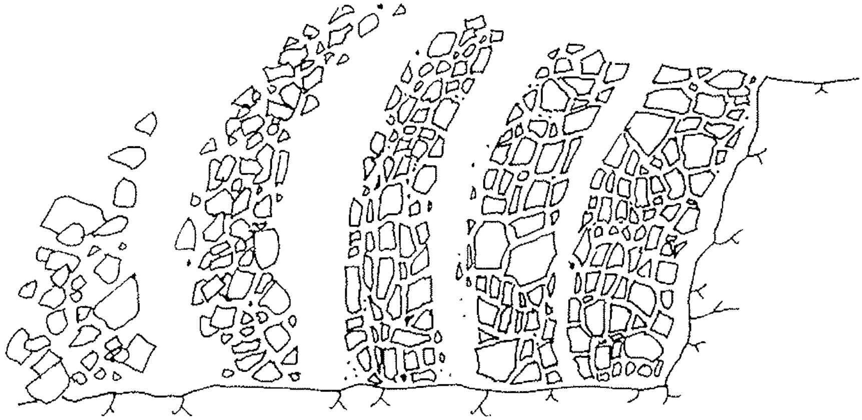


*Fig. 5.17 Perfect delay between rows.*

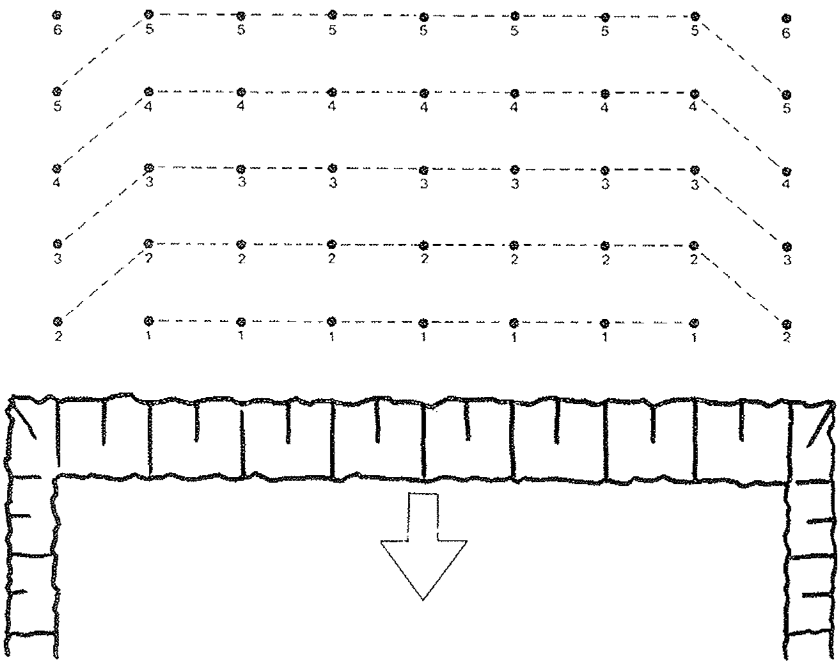



*Fig. 5.18 Firing pattern, multiple row blasting.*

Simple firing pattern for a laterally constricted multiple row round. All holes in the row have the same delay except the perimeter holes, which are delayed one interval number to avoid excessive overbreak outside the limits of the excavation.

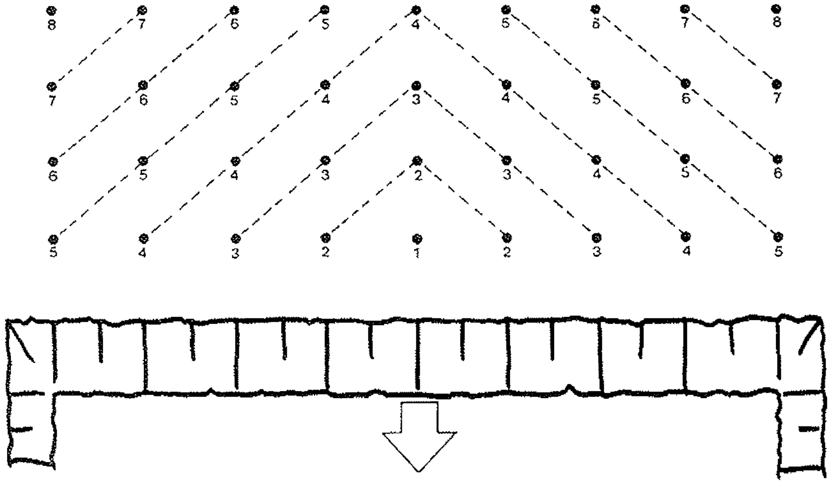



*Fig. 5.19 Firing pattern.*

This firing pattern gives better fragmentation. The ratio between true spacing and true burden, S/B, becomes more favorable. (See wide-space drilling pattern.)

One disadvantage with the above firing pattern is the risk that the center hole in the second row of the blast may detonate before the detonators in the front row with the same delay number, due to the scatter within the delay interval. The hole will then be quite constricted causing incomplete breakage which will form boulders and possible butts above the theoretical grade.

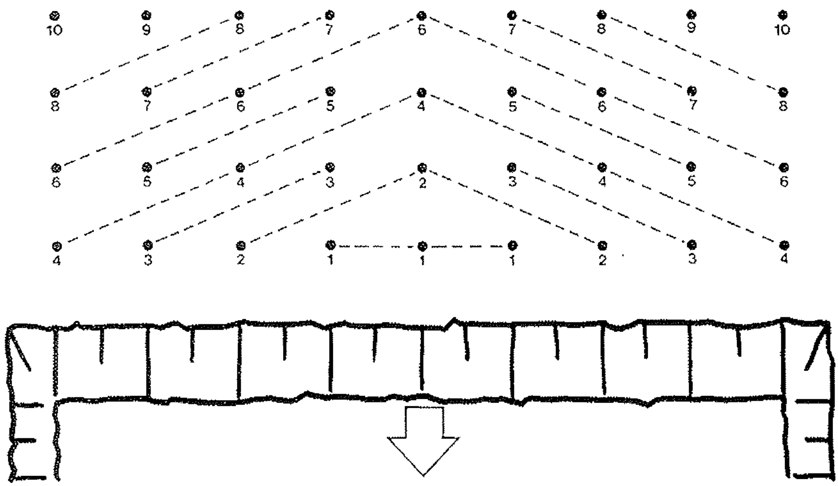



*Fig. 5.20 Firing pattern.*

This firing pattern provides separate delay time for practically all blastholes and gives good fragmentation as well as good breakage in the bottom part of the round.

**Hole inclination.**

Inclined holes with an inclination of approx. 3:1 reduce the back break and the amount of boulders from the upper part of the blast.

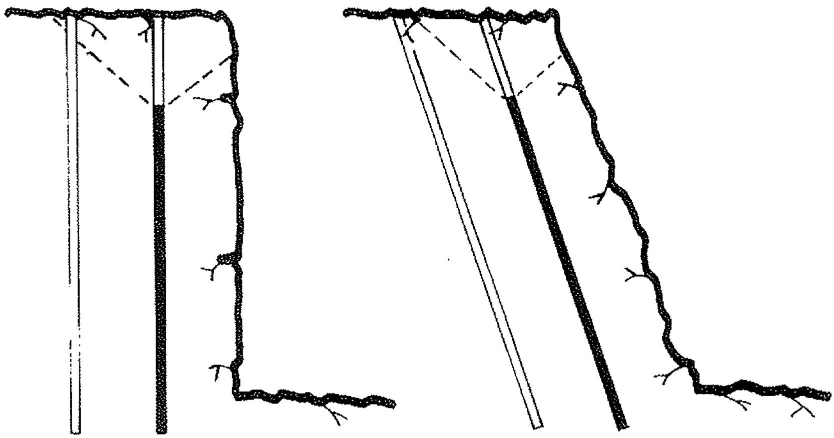


*Fig. 5.21*

**Hole deviation.**

Precision in drilling is important for the blasting result.

Poor precision in drilling will form boulders due to irregular burdens and spacings.

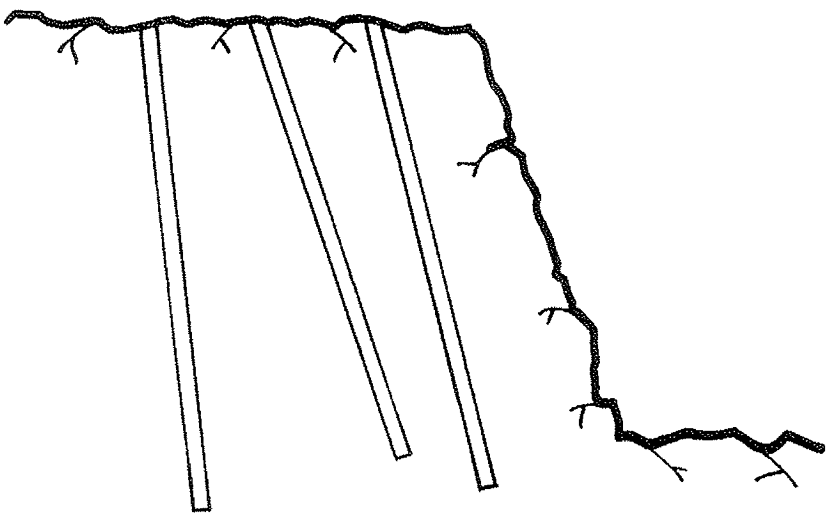


*Fig. 5.22*

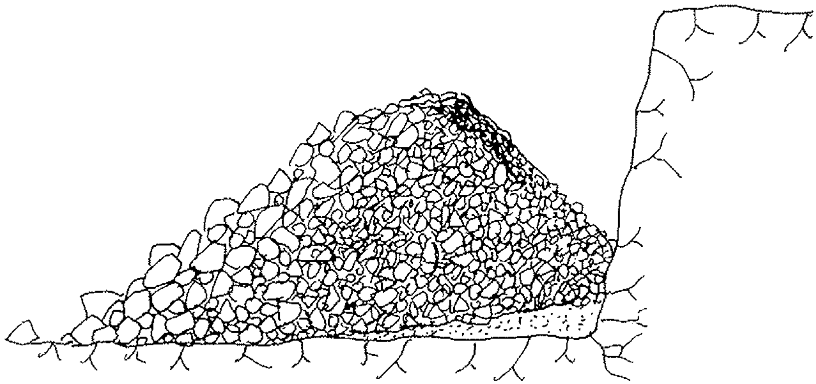


*Fig. 5.23*

**Size of the round.**

It is a known fact that most of the boulders in a blast come from the front row. Therefore, multiple row blasts give proportionally fewer boulders than single row blasts.
However, the length of the blast should not be greater than 50 % of the width.

### 5.6.2 Large size fragmentation.

Frequently large size fragmentation is required. In the construction of ports, large size rock is used for the construction of breakwaters.

The blasting to produce large size rock may be as difficult as producing small size fragmentation. The geology of the rock may form the greatest obstacle to obtain a good blasting result. A homogenous rock is preferable in large fragmentation blasting to a fissured rock.

The method of blasting large sized blocks is somewhat different from normal bench blasting.

* The specific charge has to be low.
* The spacing/burden ratio (S/B) should be less than 1.
* Blasting of one row at a time, preferably instantaneously.

The specific charge should be decreased down to 0.20 kg per cubic meter (and occasionally lower) which may be sufficient to loosen the rock but not move it forward. Rock fragmentation will not occur to any large extent. The charge should be well distributed in the blasthole with a reasonable bottom charge although smaller than normal. Due to the smaller bottom charge, a certain secondary blasting of the bottom has to be tolerated.

The choice of larger burden than spacing will definitely give blockier fragmentation with an optimum blasting result when the S/B ratio is between 0.5 and 1.0.

The instantaneous firing results in larger fragmentation than short delay blasting, as the tearing between the blastholes becomes less.

A combination of low specific charge, S/B ratio of 0.5 to 1.0, and single row instantaneous blasting, normally results in large fragmentation.

The following sketch gives an example how to blast to obtain large blocks.

| | |
|---|---|
| * Blasthole diameter | 76 mm |
| * Bench height | 15 m |
| * Burden | 3.2 m |
| * Spacing | 1.6 m |
| * Charge | |
| Bottom | 5.0 kg |
| Column | 11.0 kg |
| * Specific charge | 0.21 kg/cu.m. |

Emulite 150, 50×550 mm as bottom charge.

Emulite 150, 32×550 mm taped on a 10 gram/meter detonating cord as column charge.

Initiation: Detonating cord.

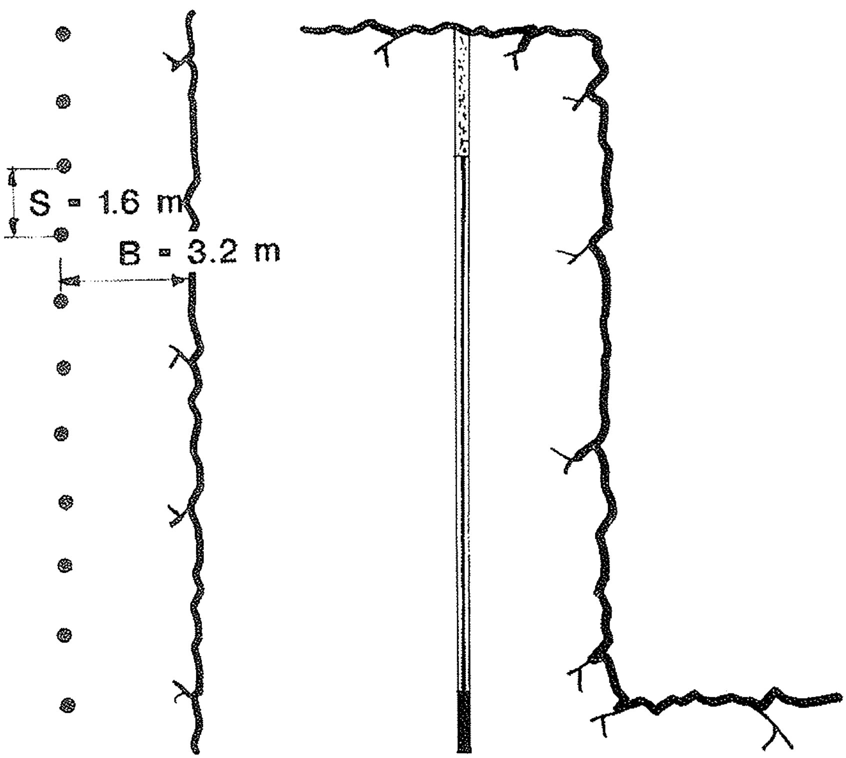



*Fig. 5.24*

## 5.7 Swelling.

When the rock is fragmented by a blast, its volume increases considerably, up to 50 %, this is known as the swelling.

The increased rock volume needs more space and if there is not space enough in front of the round, the rock must move upwards.

The same applies to long blasts where the rock piles up in front of the round as the blast proceeds row by row.

As mentioned in Chapter 5.2 Charge calculation, the specific charge should be increased if the blastings are carried out without mucking between the blasts.

According to Langefors, in his book Rock blasting, the requisite extra specific charge to compensate for the elevation of the blasted rock masses is 0.04×K (bench height) if the inclination of the blasthole is 2:1. If the hole inclination is steeper, the compensation of the specific charge has to be increased and is 0.08×K at a hole inclination of 3:1.

When no excavation is carried out between the blasts, the hole inclination must not be less than 3:1. Furthermore, the bench must not be too high. High benches must have such a high specific charge to compensate for the swelling that the risk of flyrock makes it prohibitive.

For long blasts, the rule of thumb is that the elevation of the swelling has to be considered when the length of the blast exceeds 50 % of the width. Experienced blasters usually compensate for the swelling by increasing the charge in the back rows.

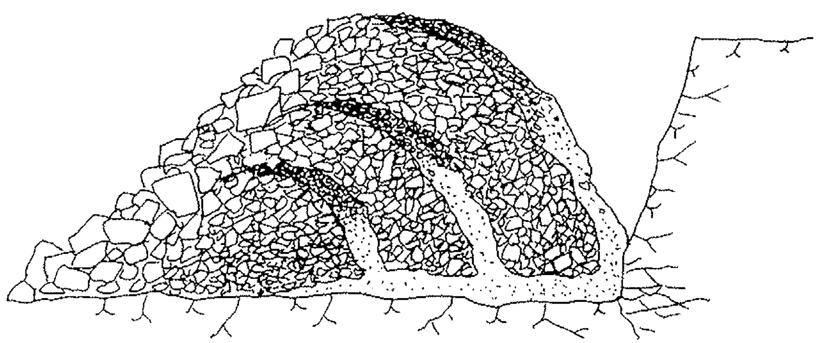


*Fig. 5.25*

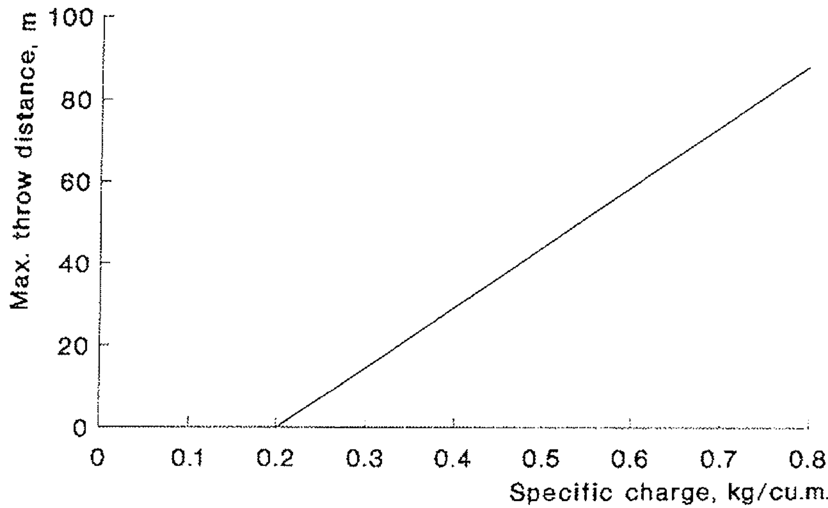



*Fig. 5.26 Maximum forward movement as a function of specific charge.*

## 5.8 Throw, flyrock.

In bench blasting we have to differentiate between two types of rock movement.

Firstly we have the forward movement of the entire rock mass which is mainly horizontal.
Secondly, the flyrock, which is scatter from the rock surface and the front of the blast.

The forward movement of the rock mass is dependent on the specific charge. Research performed by SVEDEFO* shows that a specific charge of 0.2 kg/cu.m. does not cause any forward movement of the rock, but only breaks the rock.

A normal specific charge of 0.4 kg/cu.m. moves the rock forward 20 to 30 m, which is the expected normal displacement of the rock mass.
Too short a forward movement causes a tight muckpile which is hard to excavate, while excessive forward movement spreads the muckpile, resulting in higher loading costs.

The forward movement of the rock mass rarely represents any hazard in the blasting operation but may cause inconvenience when miscalculated.

Flyrocks are rocks ejected from the blast. They tend to travel long distances and are the main cause of on-site fatalities and damage to equipment.

Flyrock is mostly caused by an improperly designed or improperly charged blast.

Research performed by SVEDEFO shows that the maximum ejection of flyrock is a function of the blasthole diameter.

$$L_{max} = 260\left(\frac{d}{25}\right)^{2/3} \quad \begin{array}{ll} L_{max} & \text{in meter} \\ d & \text{in millimeter} \end{array}$$

which is valid for a given specific charge. Fig. 5.27 shows the ejection distances

* Swedish Detonic Research Foundation.

of flyrock as a function of the specific charge at blasthole diameters of 25 to 100 mm.

A burden of less than 30 times the diameter of the blasthole gives too high a specific charge, especially if the explosive is poured or pumped into the blasthole. The excessive explosives content in the blasthole may result in rocks traveling long distances.
Too large a burden may cause flyrock if the explosive cannot break the burden and the gases vent through the collar of the hole creating crater effects.

An inappropriate firing pattern may cause the same effect as a too large burden. The gases from the blast tend to vent through the collar if the blasthole does not have free breakage.

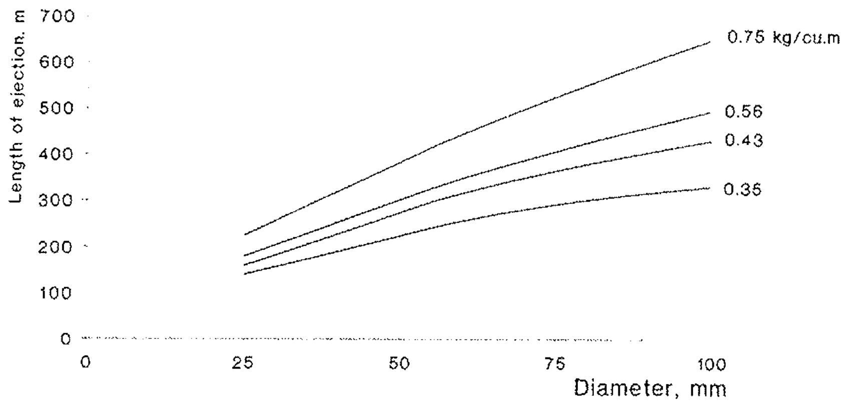



*Fig. 5.27 Maximum traveling distance of flyrock as a function of blasthole diameter for different specific charges.*

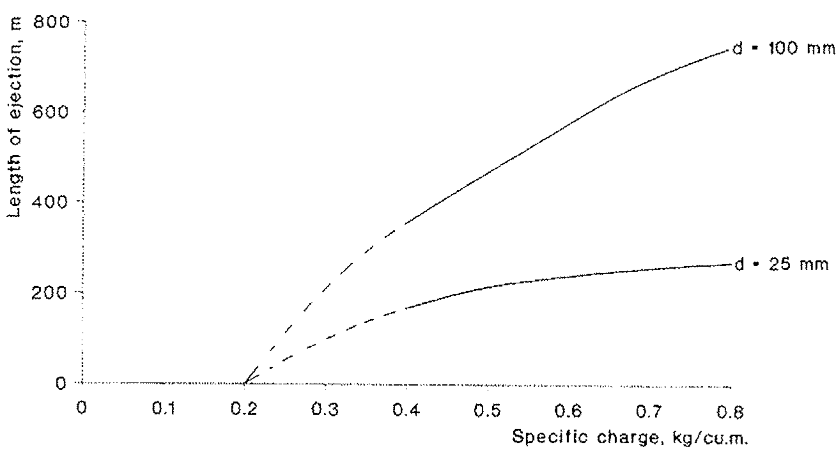
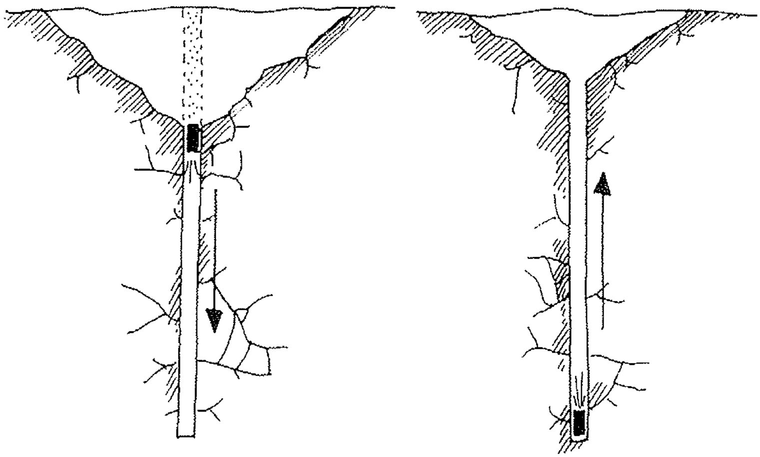



*Fig. 5.28 Maximum traveling distance of flyrock as a function of the specific charge for 50 and 100 mm blasthole diameters.*



*Top-initiation* *Bottom-initiation*

*Fig. 5.29 Crater effect, vertical holes without free breakage.*

Too short or too long a delay time between the blastholes may also cause flyrock. Too short a delay time creates an effect shown in fig. 5.16. The traveling distance is relatively limited. A more serious hazard appears when the delay time between the blastholes is too long. In a correctly designed firing pattern, the rock is held together and the rock from the front rows acts as a protection when the charges in the following rows detonate. If the delay between the rows or single blastholes is too long, the protective effect is not achieved.

**Delay times between adjacent blastholes must not exceed 100 ms if the burden is less than 2 m.**

When large diameter blastholes are used, longer delay between rows/holes must be used due to the sluggish movement of the large rock mass between the rows/holes.

Blasting of low benches, leveling, normally causes flyrock because of the fast movement of the rock mass. The low benches and the short burdens make it necessary to use short delay times between the blastholes. Leveling blasts should always be covered with heavy cover as well as light splinter-protective covering.

Various investigations, both in the U.S.A. and Sweden indicate that flyrock is more frequent when the blasthole is top-initiated than when it is bottom-initiated. (See Fig. 5.29.) Detonating cord with high core load top-initiates the explosive and tends to blow part of the stemming material out of the hole thus lowering the confinement of the explosive.

As mentioned in Chapter 5.2 Charge calculations, the stemming should have a particle size of 4 to 9 mm for best confinement. The best material for stemming is crusher run.

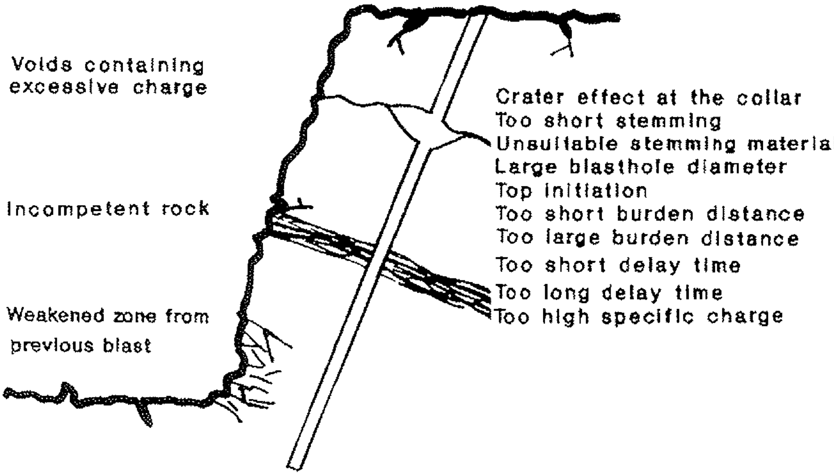



*Fig. 5.30 Causes of flyrock.*

Flyrock is often caused by incompetent rock where the gases may break through easily due to less resistance than in the more competent parts of the rock. Necessary care must be taken during the charging of the blast, especially the first row.

The incompetent zones may be natural but also caused by the previous blast, especially in the heavily charged bottom part of the hole.

If the blastholes are drilled with poor precision so that the burden is considerably smaller than the calculated one in the first row, the specific charge will locally be high. A deviation of 1 m for a 76 mm blasthole decreases the burden from 2.70 m (Emulite 150) to 1.70 m. The specific charge will locally be increased from 0.40 kg/cu.m. to 0.63 kg/cu.m. as an average for the hole. The bottom part of the hole will have even higher specific charge. The possible result of such an overcharge may be evaluated from Figures 5.27 and 5.28.

**How to avoid flyrock.**

* Clean the rock surface from loose stones which may eject easily if gases vent through the collar of the blasthole.
* Avoid stemming shorter than the burden distance. Too short stemming may create crater effects.
* Use good stemming material. No drillfines.
* Check that the drilling pattern is correct and that the blastholes are drilled with correct inclination.
* Design the firing pattern so that each blasthole has free breakage and adequate delay time between each other.
* Look out for incompetent zones and voids and charge with care, stem incompetent parts of the holes.
* Charge the first row carefully. Look out for back break which shortens the burden.

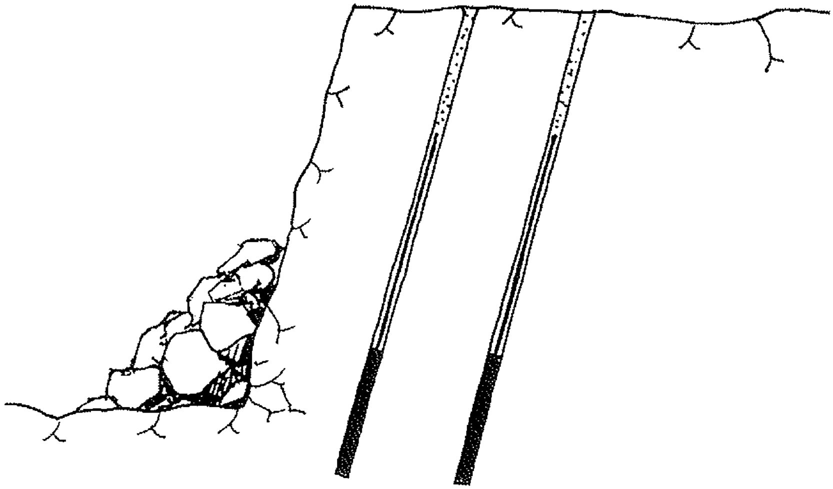


*Fig. 5.31 Rock from previous blast in front of the face prevents flyrock from the heavily charged bottom part of the hole.*

* Check that the right amount of explosives is used. When flyrock is a problem, do NOT use free flowing explosives unless confined in plastic hoses and weighted.
* Leave rock from the previous blast in front of the face, up to 1/3 of the bench height.
* In built-up areas, cover the blast.

## 5.9 Covering.

To further protect against flyrock, the blast may be covered by energy-absorbing coverings which are placed on the top of the blast. This measure can be used for smaller blasts with small diameter blastholes, less than 76 mm.
Blasts with larger diameter blastholes are practically impossible to cover efficiently. However, large diameter blastholes are rarely used where flyrock is a problem, i.e. close to populated areas. Other limiting factors such as ground vibration levels and airblast levels will restrict the amount of explosive that may be detonated in each blasthole thus making large diameter blastings impractical.

The general rule for covering a blast is that the covering material should have the same weight as the blasted rock. This is valid for low benches, leveling, when small rock masses are loosened and the distance from the charge to the rock surface is short.
For normal bench blasting, where the bench height is more than twice the maximum burden ($K \geqslant 2 \times B_{max}$), it is hardly possible to use such a heavy covering.
What we have to strive for in this case is to shorten the forward movement of the rock mass and to avoid flyrock.

The forward movement may be shortened by well-balanced charging of the blast and by leaving blasted rock from the previous blast in front of the rock face. The flyrock can be stopped by placing covering material over the blast and by well-poised stemming.

Two types of covering are used and should be used together:

* Heavy covering.
* Splinter protective covering.

The heavy covering is intended to hold the blast together so that no part of it escapes when the round is fired.

The splinter protective covering is intended to prevent flyrock from the surface section of the round.

Heavy covering material:

* Rubber blasting mats made of scrap tires which are cut into sections and twined together with steel wires.
  The size of the blasting mat should be approximately 3×4 meters and have a weight of around 1 ton. Smaller mats which can be connected together to larger units can also be used.
* Mats made of logs shackled together.

These blasting mats are heavy and energy absorbing (at least the rubber mats). The gases produced by the blast vent through the mats without displacing them to any greater extent.

Splinter protective covering material:

* Industrial felt.
* Tarpaulins.
* Mesh nets.

The heavy covering should be placed closest to the rock surface with the lighter splinter protective covering on top.

The covering with heavy mats should start from the back of the blast and work forward, each mat overlapping the previous
When the blast is fired, the mats ripple and do not follow the blast forward, which may happen if the blast is covered from the opposite direction, leaving the back rows without cover.

The covering work with heavy covering material has to be carried out with a crane or a retro excavator. For smaller-scale works small rubber mats may be used, but must be connected together with hooks to form covering units that are large enough.

The splinter protective material is then placed on top of the heavy covering, starting from the back and working forward.

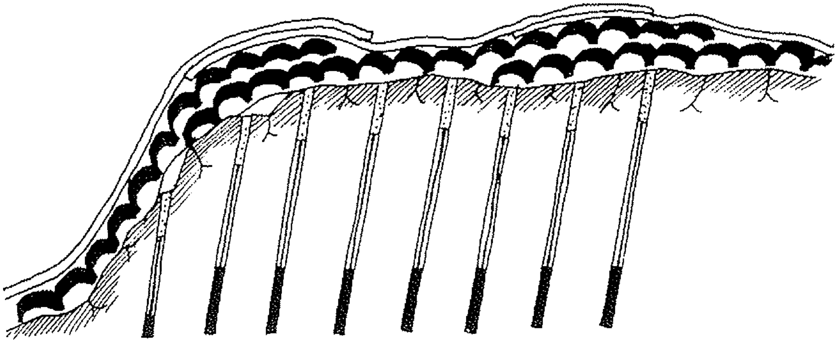


*Fig. 5.32 Principle of covering.*

*Fig. 5.33 Covering with heavy tire mats.*

*Fig. 5.34 Blasting with heavy covering in Abha, Saudi Arabia.*

Note that no rock travels any great distance. All rock remains within 5 m of the blast.

## 5.10 Blasting economy.

Economic aspects on blasting operations.

In order to evaluate the cost of the blasting operation, it is not rational to isolate the drilling and blasting operations from the subsequent operations in the work cycle.

All operations in the work cycle have to be considered:

* drilling
* charging and blasting
* boulder blasting
* mucking (excavation)
* transport
* crushing

If the cost of the drilling and blasting operation is minimized, there is a great risk of increased costs in subsequent operations, which may give an increased total cost.

The factor that affects most on the operations following the blasting operation is that of **rock fragmentation,** which has to be considered when the cost of drilling and blasting is calculated.
To quantify the rock fragmentation in relation to the blasting operation and the subsequent operations (boulder breakage, loading, transport and crushing) is quite a problem.
Harries and Mercer have visualized the relationship between blasting cost, transport and crushing to rock fragmentation in the following way:

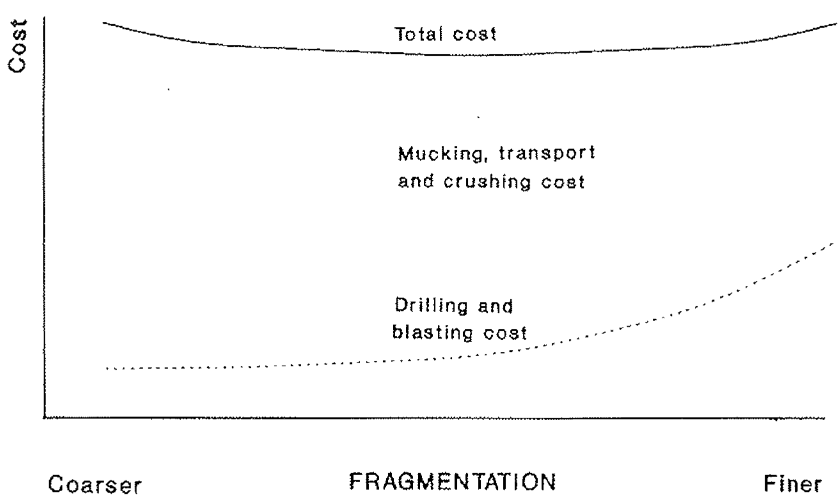



*Fig. 5.35 Relationship between blasting cost and subsequent costs to rock fragmentation.*

Despite the above, we will look into the cost of the drilling and blasting operation and ways of lowering the blasting costs.

The drilling and blasting costs may be lowered by using bigger blasthole diameters.

The cost per meter blasthole increases with the diameter.

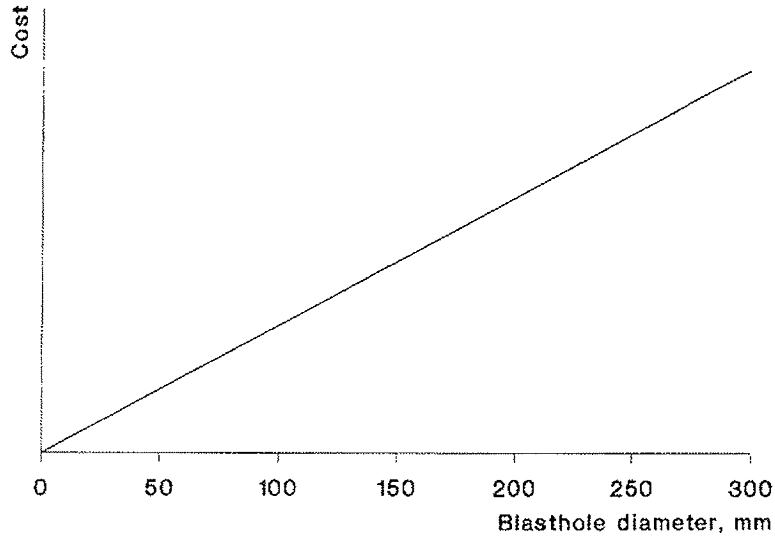



*Fig. 5.36 Relative drill cost per meter blasthole.*

The cost per volume of the blasthole decreases with larger diameter blastholes.

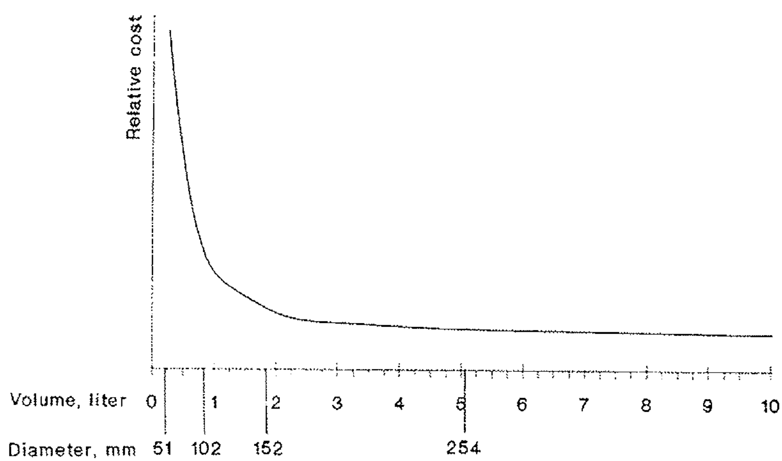



*Fig. 5.37 Relative drill cost per liter of blasthole.*

When large diameter blastholes are used, less expensive blasting agents may be used in the blastholes, lowering the total cost of the drilling and blasting operation.

As mentioned in Chapter 5.7 Rock fragmentation, large diameter blastholes tend to give a tight and blocky muckpile, resulting in more difficult loading and transport. Furthermore, the handling of boulders has a detrimental effect on the work cycle.

More often than not, the crushing operation is the bottleneck of the total workcycle. The free flow through the crusher can in such cases have such a great influence, that extra expenditure on the drilling and blasting operation may be the only way to assure full capacity in the plant and improved overall economy.

The most effective utilization of the explosive charge is obtained when the maximum burden is 35 (ANFO) to 45 (Dynamex M) times the blasthole diameter. As the blasthole diameter is increased, the burden approaches the dimensions of the bench height, making the explosives work less efficiently. Such large burdens also increase the risk of flyrock and airblast as a stemming distance with the same dimension as the burden is impossible to obtain.

In the 1980s it has been a trend among the rock producers to abandon the use of large diameter blastholes to medium size holes. The trend is opposite in crushing equipment, the manufacturers are going for larger equipment. Larger equipment has been installed, but as mentioned earlier: **"Big crushing equipment is designed to handle large amounts of rock material, not large size material"**.

The bench height will also influence the economics of the blasting operation. We know from experience that drilling deviates from the theoretical drill line. The magnitude of the deviation depends on the skill and care of the operator as well as the drilling equipment used. The geological characteristics of the rock may also influence the drilling precision.

In normal blasting operations we calculate with a deviation in drilling of 3 cm per meter of blasthole depth. The deviation has to be compensated for in the drilling pattern.

**The higher the bench — the more closely spaced the drilling pattern.**

The higher specific drilling increases the costs of the operation.

If the drilling pattern is not adjusted for the deviation in drilling, the bottom part of the blast will most certainly not be blasted to the intended level (toe-problem). This will add extra costs to the subsequent mucking operation, as the loading equipment will have a problem loading because of stumps. The secondary blasting of the toe adds unnecessary cost to the operation.

It is of vital importance to the economics of the blasting operation that the right explosive is used on each occasion. In dry conditions, cheap blasting agents can be used with excellent results. ANFO has become the most used blasting agent in the world due to its availability and economy. In dry conditions no explosive beats ANFO in overall economy including drilling, blasting, secondary blasting, loading, transport and crushing.

When the blastholes contain water, ANFO should not be used, at least not in the bottom part of the hole. ANFO deteriorates fast and should be replaced by Emulite 150 or Dynamex M which have excellent water resistance properties. If the bottom part of the hole is charged up over the theoretical grade with Emulite 150 or Dynamex M instead of ANFO, the risk of stumps in the blast will be practically eliminated. Furthermore, the rock fragmentation will be improved because of greater effectiveness of the explosive.
Tests carried out by Langefors (The modern technique of Rock Blasting) show that it is possible to increase the burden and spacing with 7 % each if the bottom charge of ANFO is replaced by a more potent explosive such as Emulite 150 to a height of $0.4 \times B_{max}$.

There are overall economic benefits from such a potent charge in the bottom part of the blast. The charge should be considered a reinforced primer rather than a reduced bottom charge. The advantages are clear:

* Reliable initiation of the blasting agent.
* Water resistant explosive in the water contaminated bottom part of the blast.
* Good breakage in the constricted bottom part of the blast.
* Less stumps above the intended grade, simplifying the loading operation.
* Less drilling.
* Lower consumption of firing devices.

# 6. TRENCH BLASTING

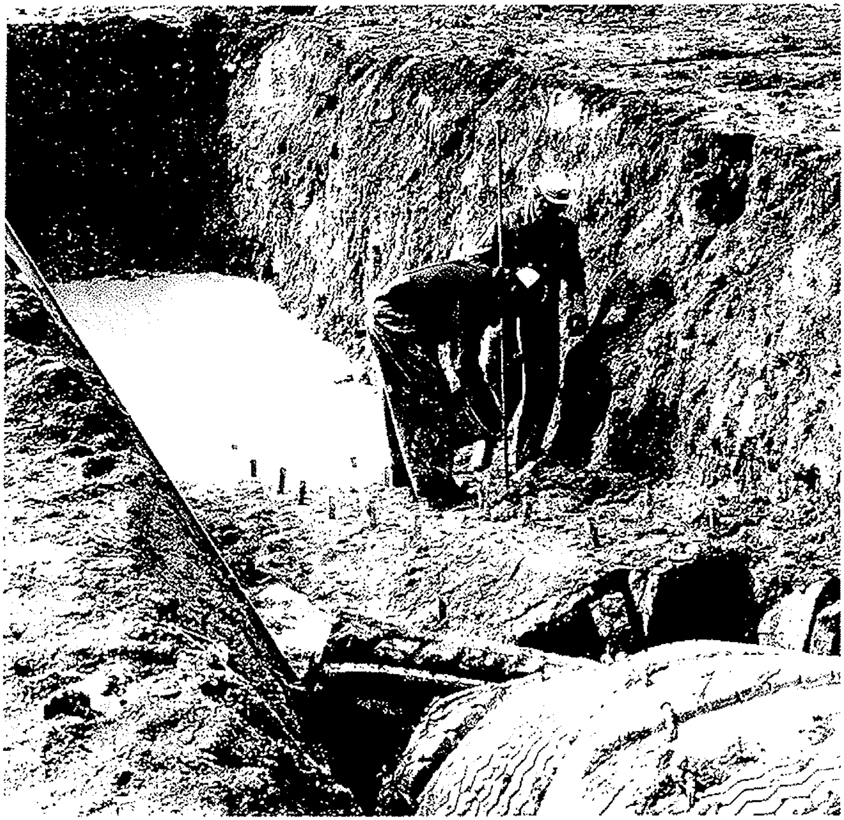


*Fig. 6.1 Charging trench blast.*

## 6.1 General.

Trench blasting is an important part of today's blasting activities. Pipeline trenches are blasted across continents for the distribution of oil and gas. Furthermore, the growing cities require increased excavation of trenches for water supply, sewerage, cables etc.

Trench blasting is a form of bench blasting, but the bench is narrower. Normally the blasting is called trench blasting if the width of the bench is less than 4 meters.

The characteristic feature of trench blasting is that the width of the bench is small in comparison to its height. The rock will be more constricted than in normal bench blasting, thus requiring a higher specific charge and higher specific drilling.
The friction against the sides of the blast is great and extra charge is also needed to compensate for the swelling.

The hole inclination is of utmost importance in trench blasting. It decreases

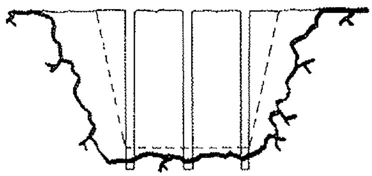


*Fig. 6.2*
*Expected result of trench blast with medium size blastholes. (50 to 75 mm).*

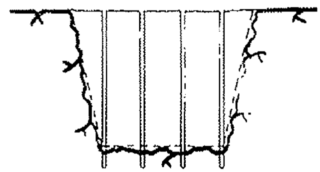


*Fig. 6.3*
*Expected result of trench blast with small size blastholes. (Drill series 11).*

the fixation in the bottom part of the blast and makes the swelling of the blasted rock easier, especially in deep trenches. The hole inclination should not be less than 3:1.
Vertical holes must be avoided as they will most probably leave stumps above the theoretical grade.

The hole diameter must be carefully considered in the planning of trench blasting. Medium size blastholes (50 to 75 mm) increase the overbreak and the risk of flyrock. Due to larger charges in the blastholes ground vibrations will also increase.

The use of medium size blastholes is economically favorable from a drilling and blasting point of view, but the cost reduction must be weighed against the increased cost of such subsequent operations as excavation and transport of excessive rock (overbreak) and the increased refill with stonefree material.

Normally the choice of the blasthole diameter is a compromise between high production and final cost.

As a rule of thumb, the blasthole diameter should be chosen in relation to the width of the trench:

$$d = \frac{w}{60} \text{ for normal trench blasting (d and w in mm).}$$

Two main trench blasting methods are used:

* Traditional trench blasting.
* Smoothwall trench blasting.

## 6.2 Traditional trench blasting.

In the traditional trench blasting method the middle hole/holes is/are placed in front of the lateral holes. All the holes have the same charge, but the column charge concentration is lower than in normal bench blasting. (The main part of the charge is placed in the constricted bottom part of the blast.)

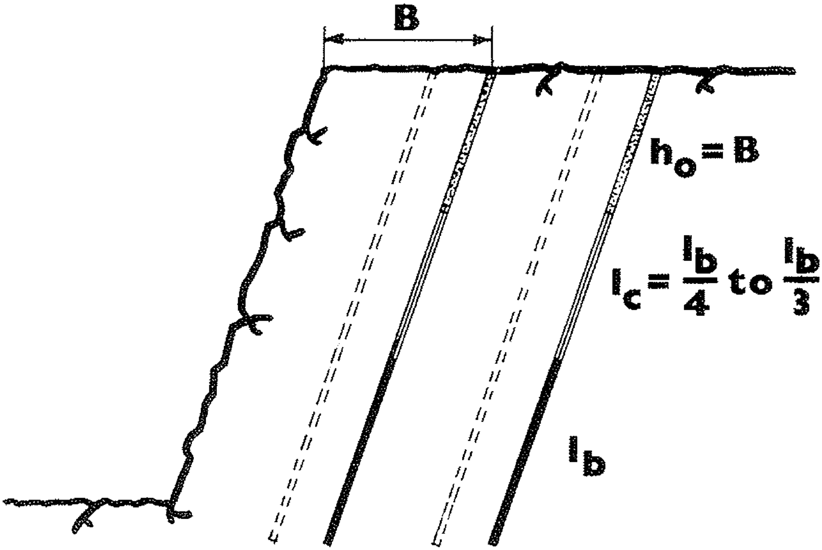



*Fig. 6.4 Charge distribution in the blasthole.*

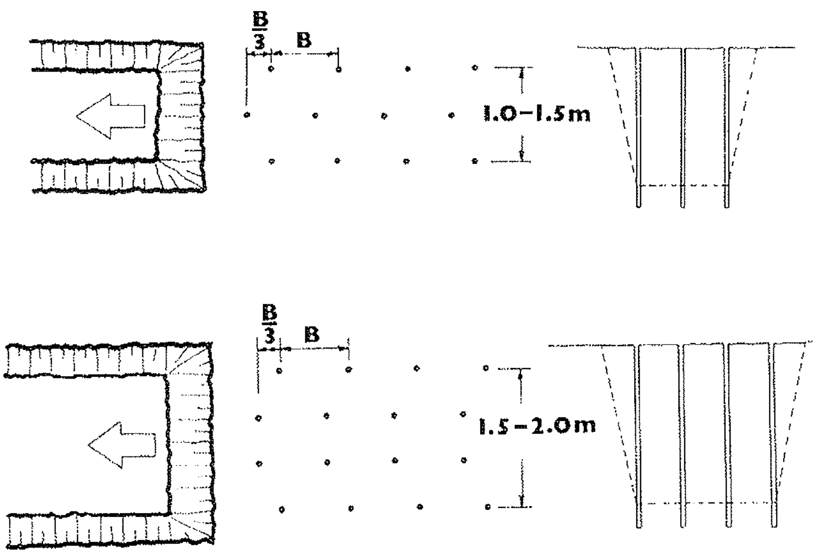



*Fig. 6.5 Traditional trench blasting.*

Charging table for traditional trench blasting.

| | |
|---|---|
| Explosive: | Emulite 150. |
| Width of the trench: | 1.0 to 1.5 m. |
| Blasthole diameter: | 34 to 26 mm (drill series 11) |
| Number of holes in the row: | 3 |
| Hole inclination: | 3:1 |

| | | | | | | | | |
|---|---|---|---|---|---|---|---|---|
| Trench depth | K (m) | 1.0 | 1.5 | 2.0 | 2.5 | 3.0 | 3.5 | 4.0 |
| Hole depth | H (m) | 1.6 | 2.1 | 2.6 | 3.1 | 3.7 | 4.2 | 4.7 |
| Hole diameter | d (mm) | 33 | 32 | 32 | 31 | 30 | 30 | 29 |
| Practical burden | B (m) | 0.8 | 0.8 | 0.8 | 0.8 | 0.7 | 0.7 | 0.6 |
| Bottom charge: | | | | | | | | |
| Concentration | $l_b$ (kg/m) | 0.9 | 0.9 | 0.9 | 0.9 | 0.8 | 0.8 | 0.7 |
| Height | $h_b$ (m) | 0.4 | 0.5 | 0.6 | 0.7 | 0.8 | 0.8 | 0.9 |
| Weight | $Q_b$ (kg) | 0.4 | 0.5 | 0.6 | 0.6 | 0.6 | 0.6 | 0.6 |
| Column charge: | | | | | | | | |
| Concentration | $l_c$ (kg/m) | 0.3 | 0.3 | 0.3 | 0.3 | 0.3 | 0.3 | 0.3 |
| Height | $h_c$ (m) | 0.3 | 0.7 | 1.1 | 1.5 | 2.0 | 2.5 | 3.0 |
| Weight | $Q_c$ (kg) | 0.1 | 0.2 | 0.3 | 0.5 | 0.6 | 0.8 | 0.9 |
| Total charge | $Q_{tot}$ (kg) | 0.5 | 0.7 | 0.9 | 1.1 | 1.2 | 1.4 | 1.5 |
| Stemming | $h_o$ (m) | 0.9 | 0.9 | 0.9 | 0.9 | 0.9 | 0.9 | 0.9 |
| Spec.charge (1.5 m) | q (kg/cu.m.) | 1.3 | 1.2 | 1.1 | 1.1 | 1.1 | 1.1 | 1.1 |

Charging table for traditional trench blasting.

| | |
|---|---|
| Explosive: | Emulite 150. |
| Width of the trench: | 2.0 m. |
| Blasthole diameter: | 34 to 26 mm (drill series 11) |
| Number of holes in the row: | 4 |
| Hole inclination: | 3:1 |

| | | | | | | | | |
|---|---|---|---|---|---|---|---|---|
| Trench depth | K (m) | 1.0 | 1.5 | 2.0 | 2.5 | 3.0 | 3.5 | 4.0 |
| Hole depth | H (m) | 1.6 | 2.1 | 2.6 | 3.1 | 3.7 | 4.2 | 4.7 |
| Hole diameter | d (m) | 33 | 32 | 32 | 31 | 30 | 30 | 29 |
| Practical burden | B (m) | 0.9 | 1.0 | 1.0 | 1.0 | 0.9 | 0.9 | 0.8 |
| Bottom charge: | | | | | | | | |
| Concentration | $l_b$ (kg/m) | 0.9 | 0.9 | 0.9 | 0.9 | 0.8 | 0.8 | 0.7 |
| Height | $h_b$ (m) | 0.3 | 0.5 | 0.5 | 0.6 | 0.8 | 0.9 | 0.9 |
| Weight | $Q_b$ (kg) | 0.3 | 0.5 | 0.5 | 0.6 | 0.6 | 0.7 | 0.6 |
| Column charge: | | | | | | | | |
| Concentration | $l_c$ (kg/m) | 0.3 | 0.3 | 0.3 | 0.3 | 0.3 | 0.3 | 0.3 |
| Height | $h_c$ (m) | 0.4 | 0.6 | 1.1 | 1.6 | 2.0 | 2.4 | 2.9 |
| Weight | $Q_c$ (kg) | 0.1 | 0.2 | 0.3 | 0.5 | 0.6 | 0.7 | 0.9 |
| Total charge | $Q_{tot}$ (kg) | 0.4 | 0.7 | 0.8 | 1.1 | 1.2 | 1.4 | 1.5 |
| Stemming | $h_o$ (m) | 0.9 | 1.0 | 1.0 | 0.9 | 0.9 | 0.9 | 0.9 |
| Spec.charge | q (kg/cu.m.) | 0.9 | 0.8 | 0.8 | 0.8 | 0.9 | 0.9 | 0.9 |

### 6.3 Smoothwall trench blasting.

To reduce overbreak it is advantageous to move the blastholes in the middle backwards so that all holes in the row are in line. The angle of breakage for the lateral holes increases, thus reducing the constriction. This makes it possible to reduce the charge concentration in the perimeter holes further, but the charge concentration in the middle hole/holes has/have to be increased. The lower charge concentration in the lateral holes reduces the overbreak.

The specific charge will be the same as in traditional trench blasting, it is just a redistribution of the explosives in the round.

The low charge concentration in the lateral holes means that it is possible to reduce the stemming distance in those holes, giving a better cut along the perimeter which reduces overbreak.
However, the short stemming distance in the perimeter holes increases the risk of flyrock.

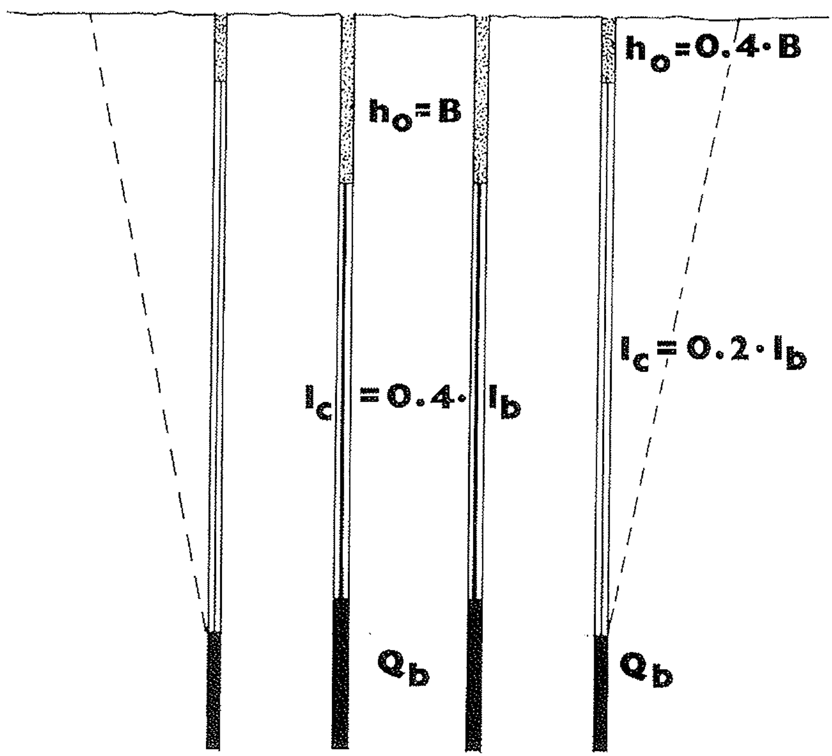



*Fig. 6.6 Charge distribution in blasthole, smoothwall trench blasting.*

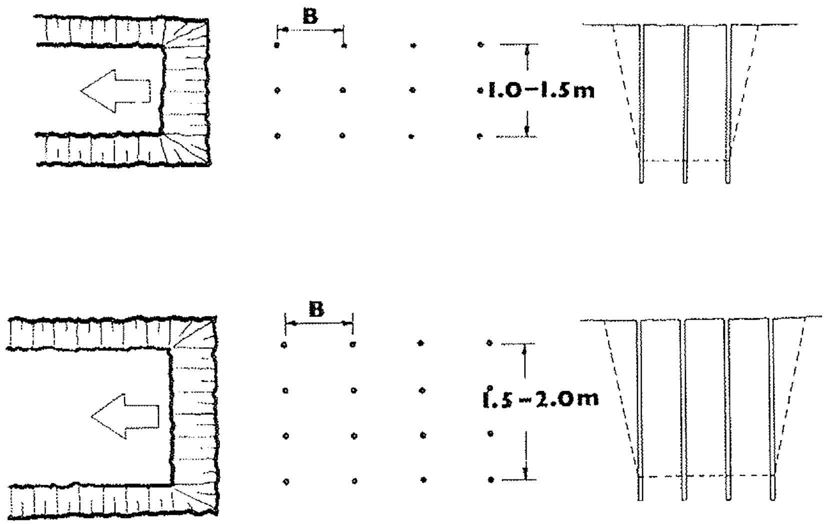



*Fig. 6.7 Smoothwall trench blasting.*

Charging table for smoothwall trench blasting.

| | |
|---|---|
| Explosive: | Emulite 150 |
| Width of the trench: | 1.0 to 1.5 m. |
| Blasthole diameter: | 34 to 26 mm (drill series 11) |
| Number of holes in the row: | 3 |
| Hole inclination: | 3:1 |

| | | | | | | | | |
|---|---|---|---|---|---|---|---|---|
| Trench depth | K (m) | 1.0 | 1.5 | 2.0 | 2.5 | 3.0 | 3.5 | 4.0 |
| Hole depth | H (m) | 1.6 | 2.1 | 2.6 | 3.1 | 3.7 | 4.2 | 4.7 |
| Hole diameter | d (mm) | 33 | 32 | 32 | 31 | 30 | 30 | 29 |
| Practical burden | B (m) | 0.7 | 0.7 | 0.7 | 0.7 | 0.6 | 0.6 | 0.6 |
| Bottom charge: | | | | | | | | |
| Middle hole | $Q_b$ (kg) | 0.4 | 0.4 | 0.5 | 0.5 | 0.6 | 0.7 | 0.7 |
| Perimeter holes | $Q_b$ (kg) | 0.3 | 0.3 | 0.4 | 0.4 | 0.5 | 0.6 | 0.6 |
| Column charge: | | | | | | | | |
| Middle hole | $Q_c$ (kg) | 0.2 | 0.3 | 0.5 | 0.7 | 0.9 | 1.0 | 1.2 |
| Perimeter holes | $Q_c$ (kg) | 0.1 | 0.2 | 0.3 | 0.4 | 0.5 | 0.6 | 0.7 |
| Total charge: | | | | | | | | |
| Middle hole | $Q_{tot}$ (kg) | 0.6 | 0.7 | 1.0 | 1.2 | 1.5 | 1.7 | 1.9 |
| Perimeter holes | $Q_{tot}$ (kg) | 0.4 | 0.5 | 0.7 | 0.8 | 1.0 | 1.2 | 1.3 |
| Stemming: | | | | | | | | |
| Middle hole | $h_o$ (m) | 0.7 | 0.7 | 0.7 | 0.7 | 0.7 | 0.7 | 0.7 |
| Perimeter holes | $h_o$ (m) | 0.3 | 0.3 | 0.3 | 0.3 | 0.3 | 0.3 | 0.3 |
| Spec. charge (1.0 m) | q (kg/cu.m.) | 2.0 | 1.7 | 1.7 | 1.7 | 1.9 | 1.9 | 1.9 |

Charging table for smoothwall trench blasting.

| | |
|---|---|
| Explosive: | Emulite 150. |
| Width of the trench: | 2.0 m |
| Blasthole diameter: | 34 to 26 (drill series 11). |
| Number of holes in the row: | 4 |
| Hole inclination: | 3:1 |

| | | | | | | | | |
|---|---|---|---|---|---|---|---|---|
| Trench depth | K (m) | 1.0 | 1.5 | 2.0 | 2.5 | 3.0 | 3.5 | 4.0 |
| Hole depth | H (m) | 1.6 | 2.1 | 2.6 | 3.1 | 3.7 | 4.2 | 4.7 |
| Hole diameter | d (mm) | 33 | 32 | 32 | 31 | 30 | 30 | 29 |
| Practical burden | B (m) | 0.8 | 0.8 | 0.8 | 0.8 | 0.7 | 0.7 | 0.7 |
| Bottom charge: | | | | | | | | |
| Middle holes | $Q_b$ (kg) | 0.4 | 0.5 | 0.6 | 0.7 | 0.8 | 0.9 | 0.9 |
| Perimeter holes | $Q_b$ (kg) | 0.3 | 0.4 | 0.5 | 0.6 | 0.6 | 0.7 | 0.7 |
| Column charge: | | | | | | | | |
| Middle holes | $Q_c$ (kg) | 0.2 | 0.3 | 0.4 | 0.6 | 0.8 | 0.9 | 1.1 |
| Perimeter holes | $Q_c$ (kg) | 0.2 | 0.2 | 0.3 | 0.3 | 0.4 | 0.5 | 0.6 |
| Total charge: | | | | | | | | |
| Middle holes | $Q_{tot}$ (kg) | 0.6 | 0.8 | 1.0 | 1.3 | 1.6 | 1.8 | 2.0 |
| Perimeter holes | $Q_{tot}$ (kg) | 0.5 | 0.6 | 0.8 | 0.9 | 1.0 | 1.2 | 1.3 |
| Stemming: | | | | | | | | |
| Middle holes | $h_o$ (m) | 0.8 | 0.8 | 0.8 | 0.8 | 0.8 | 0.7 | 0.7 |
| Perimeter holes | $h_o$ (m) | 0.3 | 0.3 | 0.3 | 0.3 | 0.3 | 0.3 | 0.3 |
| Spec.charge | q (kg/cu.m.) | 1.0 | 0.8 | 0.8 | 0.8 | 0.8 | 0.8 | 0.8 |

Breakage patterns:

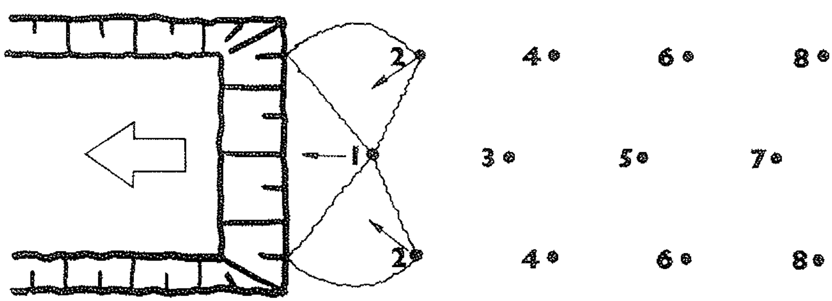



*Fig. 6.8 Traditional trench blasting.*

Advantages:

* Uniform charging of all holes.
* Less ground vibration.

Disadvantages:

* Asymmetrical drilling pattern.
* Overbreak is normal and can be considerable.

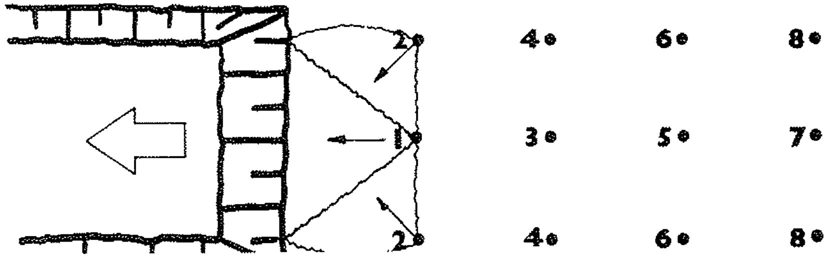



*Fig. 6.9 Smoothwall trench blasting.*

Advantages:
- * Symmetrical drilling pattern.
- * Reduced overbreak.

Disadvantages:
- * Varying amounts of explosives in middle and perimeter holes.
- * Higher ground vibrations due to higher amount of explosives in the middle hole/s.

## 6.4 High productivity trench blasting.

Trench blasting for oil and water pipelines frequently requires high productivity blasting. In these cases larger diameter blastholes are used, normally 51 to 64 mm. The disadvantages with high ground vibrations and excessive overbreak are compensated by big advance per day. Saudi Chemical Co has in the recent years blasted trenches over the Saudi Arabia peninsula and kept up a speed of 3 kilometers per day. In these cases it is necessary to use firing devices like detonating cord with relays or NONEL UNIDET. Careful planning is also needed.

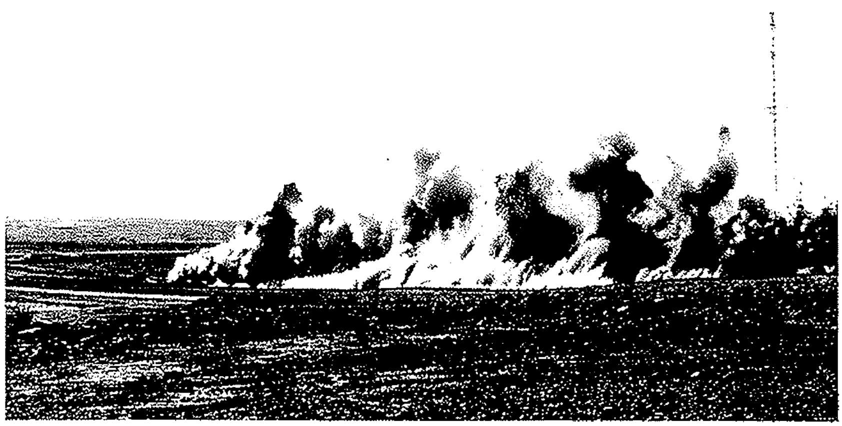


*Fig. 6.10 Pipeline blasting in Saudi Arabia*

Charging table for high productivity trench blasting.

| | |
|---|---|
| Explosive: | Emulite 150 (primer)<br>ANFO (bulk) |
| Width of the trench: | 3.0 m |
| Blasthole diameter: | 64 mm |
| Number of holes in the row: | 4 |
| Hole inclination: | 3:1 |

| | | | | | | | | |
|---|---|---|---|---|---|---|---|---|
| Trench depth | K (m) | 2.0 | 2.5 | 3.0 | 3.5 | 4.0 | 4.5 | 5.0 |
| Hole depth | H (m) | 2.6 | 3.2 | 3.7 | 4.2 | 4.7 | 5.3 | 5.8 |
| Hole diameter | d (mm) | 64 | 64 | 64 | 64 | 64 | 64 | 64 |
| Practical burden | B (m) | 1.6 | 1.6 | 1.6 | 1.6 | 1.5 | 1.5 | 1.5 |
| Primer: | | | | | | | | |
| Emulite 150, 50×550 or 40×550 mm | | | | | | | | |
| Weight | Q (kg) | 1.25 | 1.25 | 1.25 | 1.25 | 1.25 | 1.25 | 1.25 |
| Charge: | | | | | | | | |
| ANFO | | | | | | | | |
| Concentration | l (kg/m) | 2.6 | 2.6 | 2.6 | 2.6 | 2.6 | 2.6 | 2.6 |
| Height | h (m) | 0.6 | 1.2 | 1.7 | 2.2 | 2.7 | 3.3 | 3.8 |
| Weight | $Q_c$ (kg) | 1.55 | 3.10 | 4.40 | 5.70 | 7.00 | 8.60 | 9.90 |
| Total charge | $Q_{tot}$ (kg) | 2.80 | 3.35 | 5.65 | 6.95 | 8.25 | 9.85 | 11.15 |
| Stemming | $h_0$ | 1.5 | 1.5 | 1.5 | 1.5 | 1.5 | 1.5 | 1.5 |
| Spec. charge | q (kg/cu.m.) | 1.2 | 1.2 | 1.6 | 1.6 | 1.8 | 1.8 | 1.8 |

In the case of hard cap rock, 22 mm pipecharges may be used in the stemming area to break up the surface rock and avoid "tunneling".

# 7. UNDERGROUND BLASTING

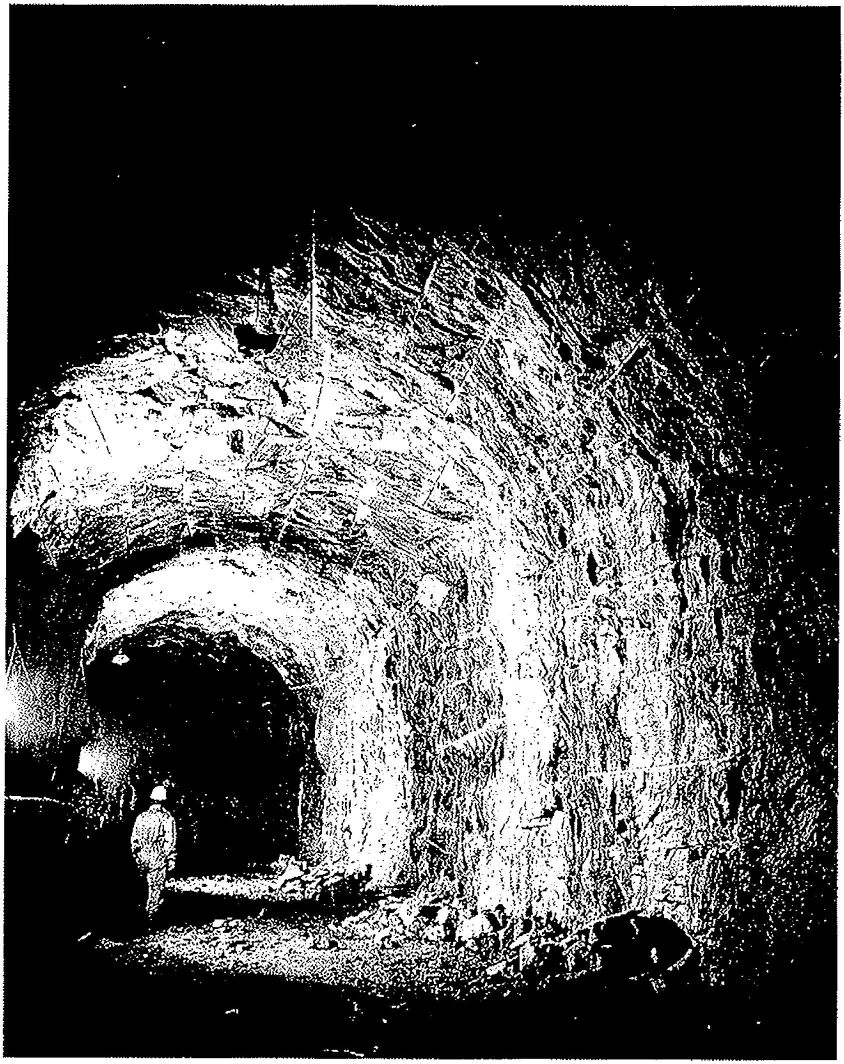


*Fig. 7.1 Tunneling.*

## 7.1 Tunneling.

There are two reasons to go underground and excavate:

- to use the excavated space, e.g. for storage, transport etc.
- to use the excavated material, e.g. mining operations.

In both cases tunneling forms an important part of the entire operation. In underground construction it is necessary to gain access to the construction site by

tunneling, but the tunnel can be a purpose in itself e.g. road, water, cable tunnels etc.
In mining operations tunnels are used as adits to the mining site and for preparatory work as well as for internal communication.
Tunnels are driven mainly in horizontal or close to horizontal directions but also inclined, from vertically upwards to vertically downwards. In the following, tunneling, raise shafts and sink shafts will be dealt with in detail while storage in rock caverns and mining will be dealt with more briefly.

Tunneling is the most frequently ocurring underground operation which also forms part of the construction of rock chambers etc. and is normally an integral part of mining operations.
The development of tunnel driving techniques has been tremendous during the last few years. The drilling techniques have developed from pneumatic drilling machines to electro-hydraulic drilling jumbos with a very high capacity. The charging of the blastholes can be carried out quickly either manually with plastic pipe charges or mechanically with pneumatic charging equipment.
The development of explosives has moved in the direction of safer products with better fumes characteristics. Modern explosives like Emulite and Dynamex M are well oxygen-balanced with a minimum of noxious fumes.
Initiating systems like NONEL have shortened the charging time and added further safety to the blasting operation due to their insusceptibility to electrical hazards.

The modern drilling equipment has shortened the drilling time, the NONEL system has made connecting of the detonators safer and faster and Emulite, with its excellent fumes characteristics, has shortened the ventilation time.
All the above contribute to a faster work cycle:

- drilling
- charging
- blasting
- ventilation
- scaling
- grouting (if necessary)
- loading and transport
- setting out for the new blast

The shorter work cycle calls for better work planning as well as better precision and accuracy in the different operations of the work cycle.
In the following, the drilling, charging and blasting operations will be dealt with.
It is obvious that it is of the utmost importance that the holes should be drilled at the right locations and with the right inclination. The marking of the holes on the rock face as well as collaring and drilling must be carried out accurately.
Langefors in "The modern technique of Rock Blasting", says about drilling precision: **"The scattering of the drill holes as a quantitative factor is often disregarded. It is included quite indefinitely in the technical margin together with the rock factor. In discussing blasting as a whole it would be a great advantage if**

**attention could be paid to the drilling precision in calculating the charges and in constructing the drilling pattern; for the blasting of the cut it is essential.''**

The main difference between tunnel blasting and bench blasting is that tunnel blasting is done towards one free surface while bench blasting is done towards two or more free surfaces. The rock is thus more constricted in the case of tunneling and a second free face has to be created towards which the rock can break and be thrown away from the surface. This second face is produced by a cut in the tunnel face and can be either a parallel hole cut, a V-cut, a fan-cut or other ways of opening up the tunnel face.
After the cut opening is made, the stoping towards the cut will begin. The stoping can be compared with bench blasting, but it requires a higher specific charge due to higher drilling deviation, desire for good fragmentation, and absence of hole inclination. In addition, overcharge of a tunnelblast does not have the same disastrous effect as in an open air blast, where high precision in calculation is a must.

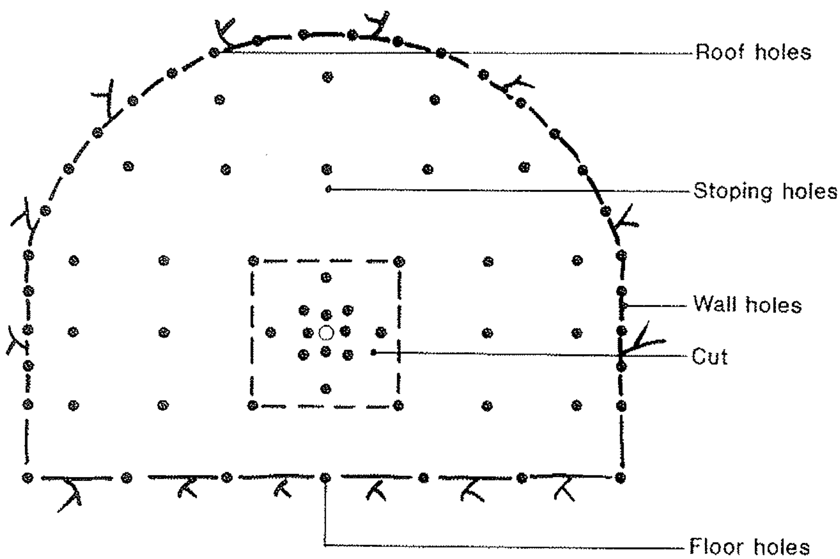



*Fig. 7.2 Nomenclature.*

In the case of V-cuts and fan cuts, the cut holes will occupy the major part of the width of the tunnel.

The contour holes – roof holes, wall holes and floor holes – have to be angled out of the contour, "look-out", so the tunnel will retain its designed area. The "look-out" should only be big enough to allow space for the drilling equipment for the coming round. As a guide value, the "look-out" should not exceed:

10 cm + 3 cm/m holedepth

which keeps the "look-out" to around 20 cm.

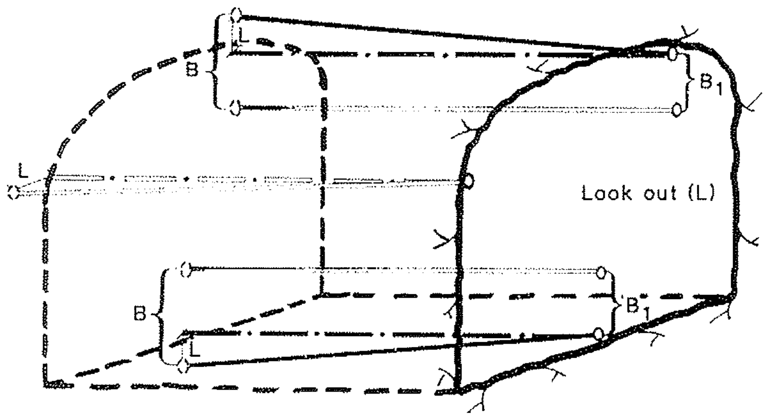



*Fig. 7.3 Look-out.*

The consumption of explosives in tunnel blasting is higher than in bench blasting. The specific charge is 3 to 10 times higher than that for bench blasting, depending mainly on reasons mentioned above like large drilling scatter, higher fixation of the holes, heave of lower rock upwards to ensure swell and lack of cooperation between adjacent blastholes.

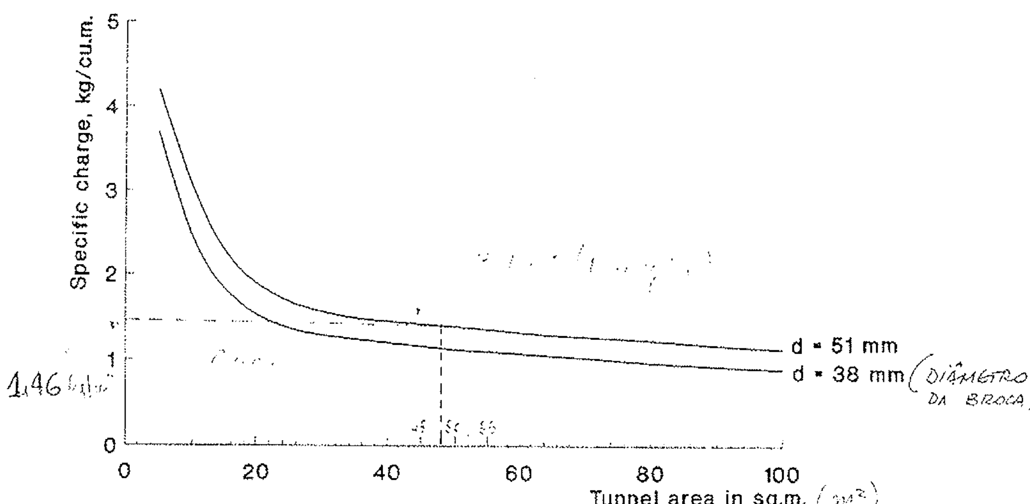



*Fig. 7.4 Specific charge for different tunnel areas.*

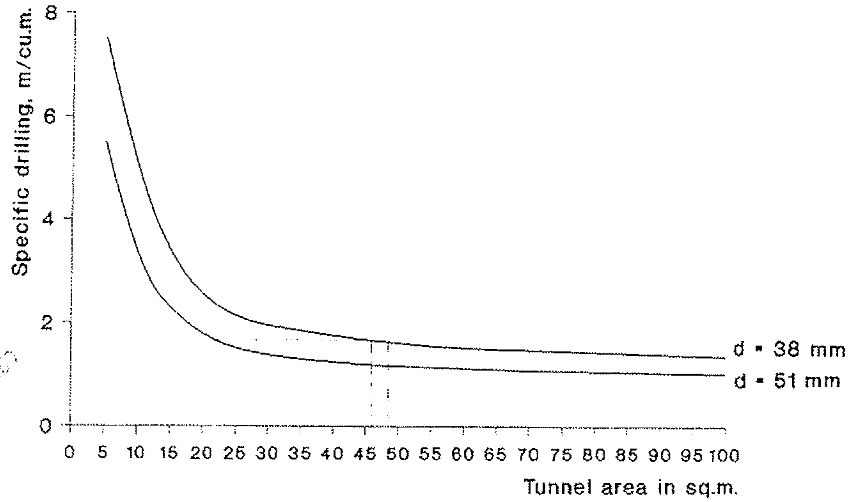



*Fig. 7.5 Specific drilling for different tunnel areas.*

The consumption of explosives will be greatest in the cut area of the blast. A 1×1 m area around the empty hole/s in a parallel cut will consume approx. 7 kg/cu.m. and the specific charge will decrease with the distance from the cut until it reaches a minimum value of about 0.9 kg/cu.m.

### 7.1.1 The cut.

The most commonly used cut in tunneling today is the **circular cut** or **large hole cut** as most of the modern drilling equipment is designed for horizontal drilling perpendicular to the rock face. (Other cuts will be dealt with in the end of this chapter.)
All cut holes in the large hole cut are drilled parallel to each other and the blasting is carried out towards an empty large drill hole which acts as an opening. The parallel hole cut is a development of the **burn cut,** where all the holes are parallel and normally of the same diameter. One hole in the middle is given a heavy charge and the four holes around it are left uncharged, in other cases the middle hole is left uncharged and the four holes are charged.

However, the burn cuts generally result in less advance than the large hole cuts. The burn cut will therefore be disregarded and only the **large hole cuts** will be dealt with.
The cut may be placed at any location on the tunnel face, but the location of the cut influences the throw, the explosives' consumption and generally the number of holes in the round.

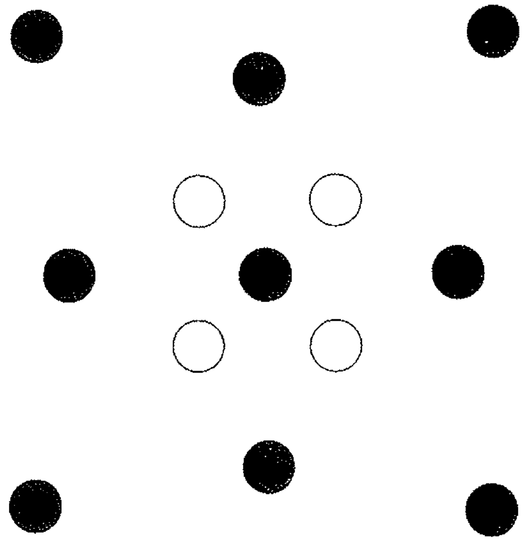


*Fig. 7.6 Burn cut.*

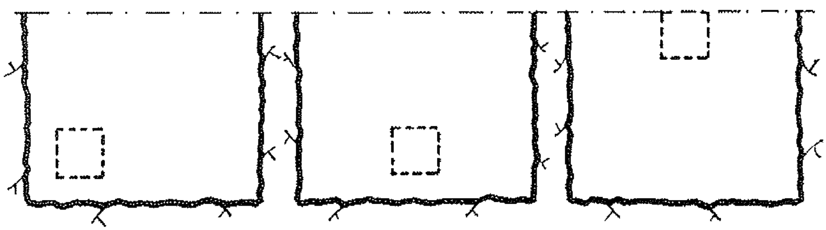


*Fig. 7.7 Location of the cut.*

If the cut is placed close to a wall, there is a probability of better exploitation of the drilling pattern with less holes in the round. Furthermore, the cut may be placed alternatively on the right or left side thus placing the cut in relatively undisturbed rock. To obtain good forward movement and centering of the muckpile, the cut may be placed approximately in the middle of the cross section and quite low down. This position will give less throw and less explosives' consumption because of more stoping downwards. A high position of the cut gives an extended and easily loaded muckpile, but higher explosives' consumption and normally more drilling due to more upwards stoping.

The normal location of the cut is on the first helper row above the floor.

As mentioned before, the large hole cut is the most common cut today. The cut is composed of one or more uncharged large diameter holes which are surrounded by small diameter blastholes with small burdens to the large hole/s. The blastholes are placed in squares around the opening.

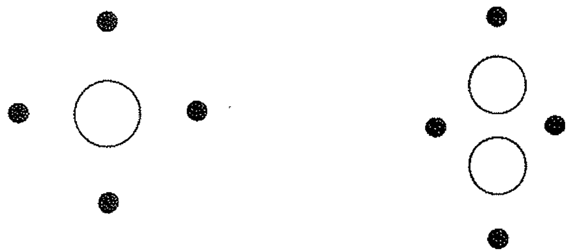


*Fig. 7.8 Typical designs of large hole cuts.*

The number of squares in the cut is limited by the fact that the burden in the last square must not exceed the burden of the stoping holes for a given charge concentration in the hole.

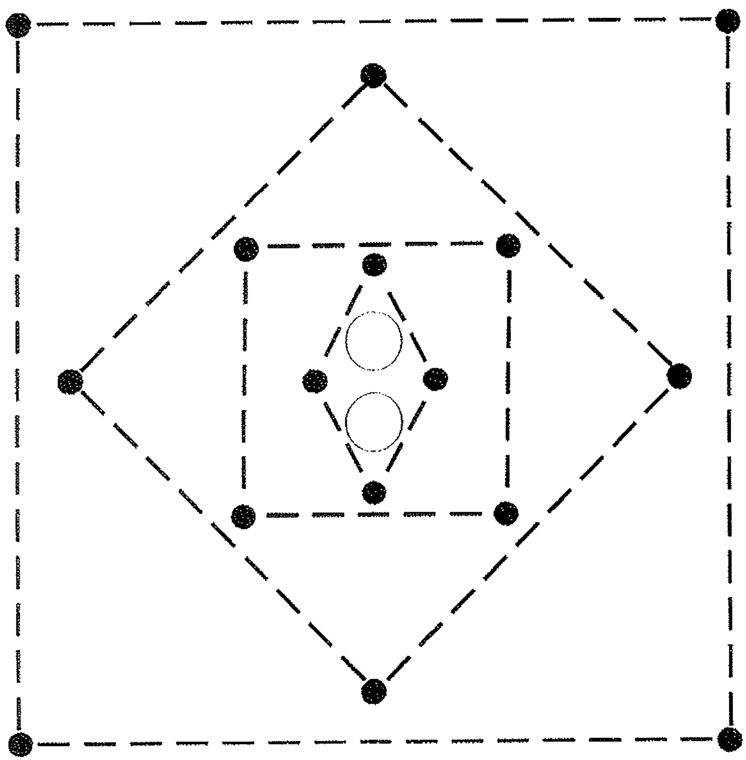


*Fig. 7.9 The complete cut.*

The cut holes occupy an area of approx. 2 sq.m. (Small tunnel areas, as a matter of fact, consist only of cut holes and contour holes.)

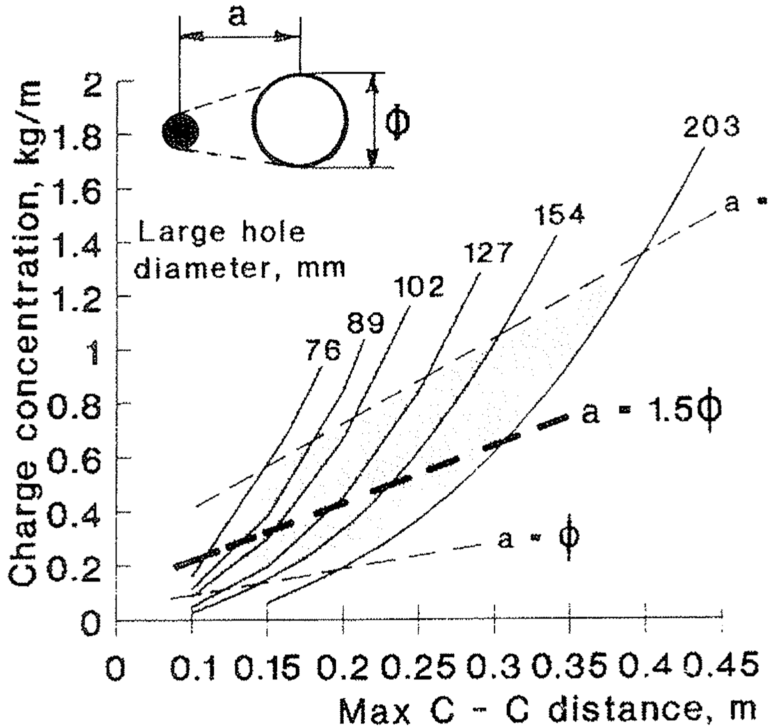



*Fig. 7.12 The minimum required charge concentration (kg/m) and maximum C–C distance (m) for different large hole diameters.*

The requisite charge concentration for different C–C distances between the large hole and the nearest blasthole/s may be found in graph 7.12 for different large hole diameters. The normal relation for the distance is **a=1.5 Ø**. An increase in the C–C distance between the holes will cause subsequent increment of the charge concentration.
The cut is often somewhat overcharged to compensate for error in drilling which may cause too small an angle of breakage. However, too high a charge concentration may cause recompaction in the cut.

**Calculation of the remaining squares of the cut.**

The calculation method for the remaining squares of the cut is essentially the same as for the 1st square, with the difference that the breakage is towards a rectangular opening instead of a circular.
As is the case of the 1st square, the angle of breakage must not be too acute as small angles of breakage can only be compensated to a certain extent with higher charge concentration.
Normally the burden (B) for the remaining squares of the cut is equal to the width (W) of the opening. **B=W.**

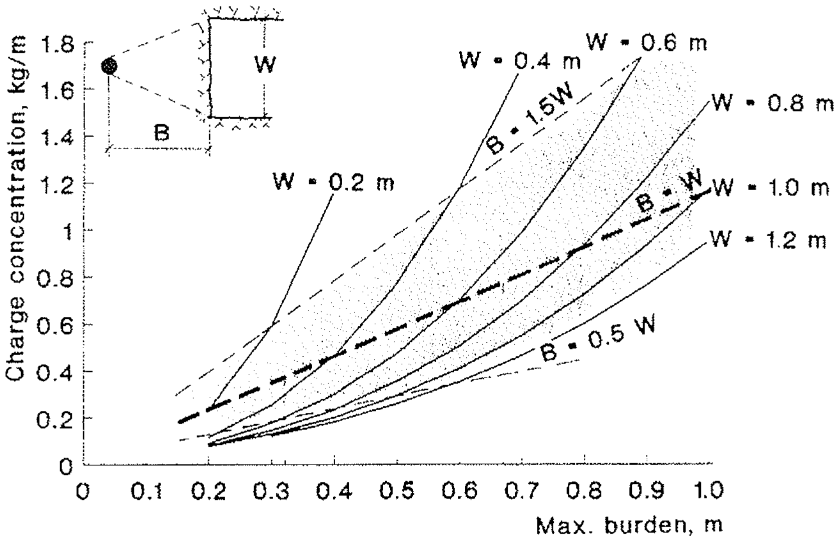



*Fig. 7.13 The required minimum charge concentration (kg/m) and maximum burden (m) for different widths of the opening.*

The charge concentration obtained in graph 7.12 is that of the column of the hole. In order to break the constricted bottom part, a bottom charge with twice the charge concentration and a height of 1.5×B should be used. The stemming part of the hole has a length of 0.5×B.

**Design of cut.**

The following formulae are used for the geometric design of the cut area:

**The cut:**

**1st square:**

$$a = 1.5\ \varnothing$$

$$W_1 = a\sqrt{2}$$

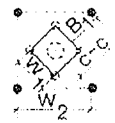
| Ø mm | = | 76 | 89 | 102 | 127 | 154 |
|---|---|---|---|---|---|---|
| a mm | = | 110 | 130 | 150 | 190 | 230 |
| $W_1$ mm | = | 150 | 180 | 210 | 270 | 320 |


**2nd square:**

$$B_1 = W_1$$

$$C-C = 1.5W_1$$

$$W_2 = 1.5W_1\sqrt{2}$$




| Ø mm | = | 76 | 89 | 102 | 127 | 154 |
|---|---|---|---|---|---|---|
| $W_1$ mm | = | 150 | 180 | 210 | 270 | 320 |
| C−C | = | 225 | 270 | 310 | 400 | 480 |
| $W_2$ mm | = | 320 | 380 | 440 | 560 | 670 |

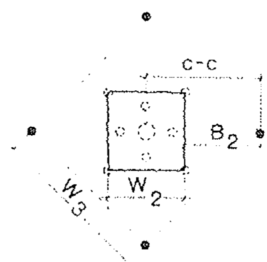
**3rd square:**

$$B_2 = W_2$$

$$C-C = 1.5W_2$$

$$W_3 = 1.5W_2\sqrt{2}$$

| Ø mm | = | 76 | 89 | 102 | 127 | 154 |
|---|---|---|---|---|---|---|
| $W_2$ mm | = | 320 | 380 | 440 | 560 | 670 |
| C−C | = | 480 | 570 | 660 | 840 | 1000 |
| $W_3$ mm | = | 670 | 800 | 930 | 1180 | 1400 |



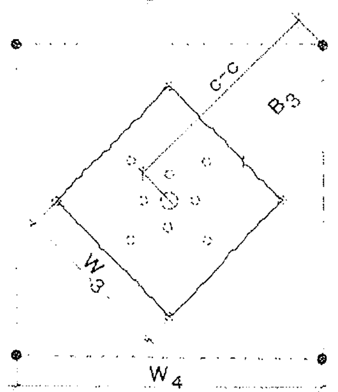

**4th square:**

$$B_3 = W_3$$

$$C-C = 1.5W_3$$

$$W_4 = 1.5W_3\sqrt{2}$$

| Ø mm | = | 76 | 89 | 102 | 127 |
|---|---|---|---|---|---|
| $W_3$ mm | = | 670 | 800 | 930 | 1180 |
| C−C | = | 1000 | 1200 | 1400 | 1750 |
| $W_4$ mm | = | 1400 | 1700 | 1980 | 2400 |




The above distances apply to 38 mm blastholes. If larger blastholes are used which can accommodate more explosives, the values can be adjusted. However, an increased amount of explosives in the cut holes may not increase the burden to any greater extent.

### 7.1.2 Stoping.

When the cut holes have been calculated, the rest of the tunnel round may be calculated.

The round is divided into:

* floor holes
* wall holes
* roof holes
* stoping holes with breakage upwards and horizontally
* stoping holes with breakage downwards

To calculate burdens (B) and charges for the different parts of the round the following graph (7.14) may be used as a basis.

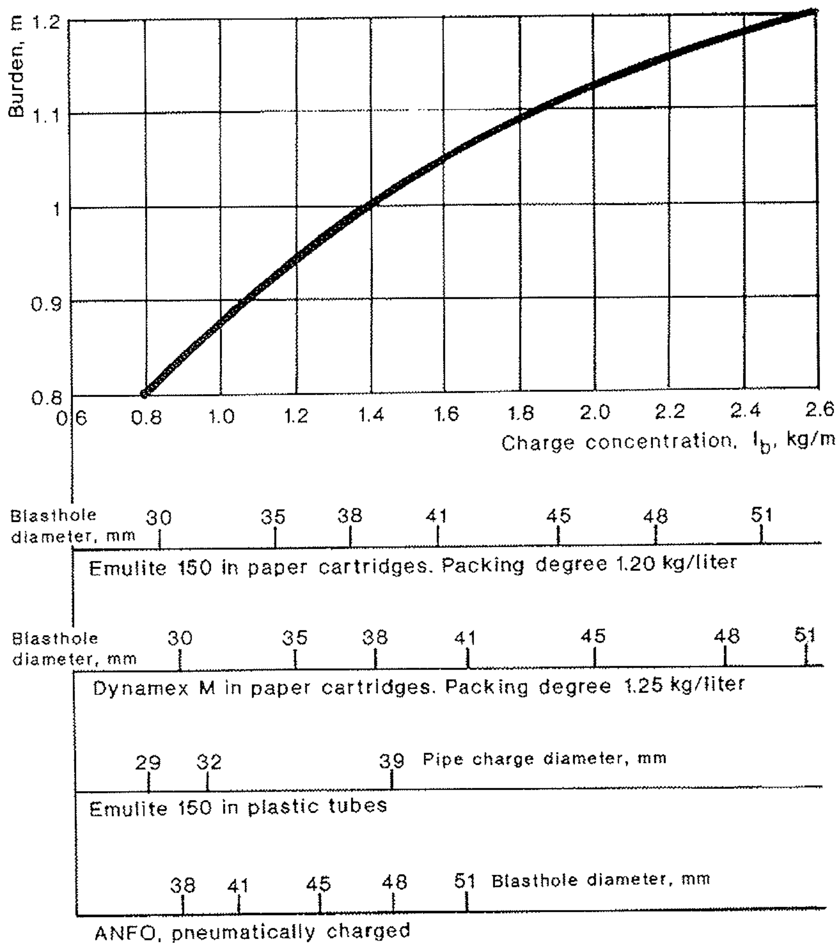



*Fig. 7.14 The burden B in relation to the concentration of the bottom charge for different hole diameters and different explosives.*

For Emulite 150 in paper cartridges, the uppermost blasthole diameter table is used as input data.
For Emulite 150 and Dynamex M in plastic pipe cartridges, the pipe diameter is used as input data and for ANFO the lowest blasthole diameter table is used as input data.

When the burden (B), the hole depth (H) and the concentration of the bottom charge ($l_b$) are known, the following table will give the drilling and charging geometry of the round.

| Part of the round: | Burden (m) | Spacing (m) | Height bottom charge (m) | Charge concentration Bottom (kg/m) | Charge concentration Column (kg/m) | Stemming (m) |
|---|---|---|---|---|---|---|
| Floor | 1×B | 1.1×B | 1/3×H | $l_b$ | 1.0×$l_b$ | 0.2×B |
| Wall | 0.9×B | 1.1×B | 1/6×H | $l_b$ | 0.4×$l_b$ | 0.5×B |
| Roof | 0.9×B | 1.1×B | 1/6×H | $l_b$ | 0.3×$l_b$ | 0.5×B |
| Stoping: | | | | | | |
| Upwards | 1×B | 1.1×B | 1/3×H | $l_b$ | 0.5×$l_b$ | 0.5×B |
| Horizontal | 1×B | 1.1×B | 1/3×H | $l_b$ | 0.5×$l_b$ | 0.5×B |
| Downwards | 1×B | 1.2×B | 1/3×H | $l_b$ | 0.5×$l_b$ | 0.5×B |

The design of the drilling pattern can now be carried out and the cut located in the cross section in a suitable way.

### 7.1.3 The contour.

The contour of the tunnel is divided into floor holes, wall holes and roof holes. The burden and spacing for the floor holes are the same as for the stoping holes. However, the floor holes are more heavily charged than the stoping holes to compensate for gravity and for the weight of the rock masses from the rest of the round which lay over them at the instant of detonation.
For the wall and roof holes two variants of contour blasting are used, **normal profile blasting** and **smooth blasting.**
With **normal profile blasting** no particular consideration is given to the appearance and condition of the blasted contour. The same explosives as in the rest of the round are utilized (but with a lesser charge concentration) and the contour holes are widely spaced. The contour of the tunnel becomes rough, irregular and cracked. The **smooth blasting** technique has been developed to obtain a smoother and stronger tunnel profile.
Smooth blasting is carried out by drilling the contour holes rather close to each other and using weaker explosives. (Gurit 17×500 mm and Gurit 11×460 mm have been specially developed for the requirements of smooth blasting.)
Smooth blasting is today a common technique in underground rock excavation as it produces tunnels with a regular profile, requiring substantially less reinforcement than if normal profile blasting is used.
Smooth blasting is dealt with in detail in Chapter 8.4 Smooth blasting, where charging tables for smooth blasting can be found.

### 7.1.4 The firing pattern.

The firing pattern must be designed so that each hole has free breakage. The angle of breakage is smallest in the cut area where it is around 50°. In the stoping area the firing pattern should be designed so that the angle of breakage does not fall below 90°.

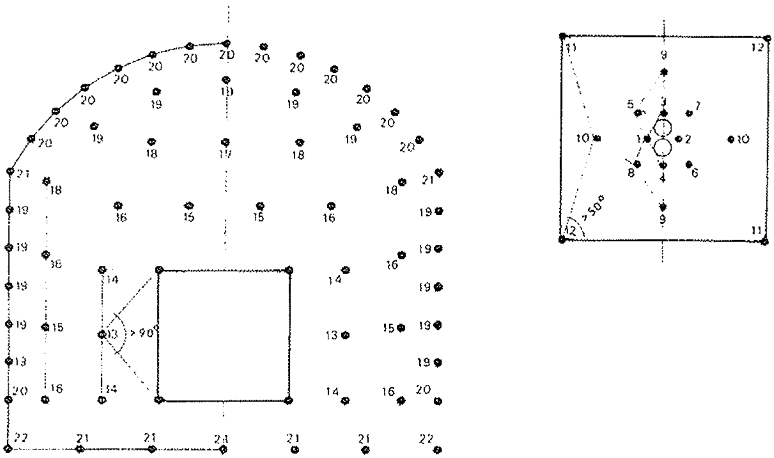



*Fig. 7.15 Firing sequence for tunnel in numerical order.*

It is important in tunnel blasting to have long enough time delay between the holes. In the cut area, the delay between the holes must be long enough to allow time for breakage and throw of rock through the narrow empty hole. It is proved that the rock moves with a velocity of 40 to 60 meters per second. A cut drilled to 4 m depth would thus require a delay time of 60 to 100 ms to be clean blasted. Normally delay times of 75 to 100 ms are used in the cut.

In the first two squares of the cut only one detonator of each delay should be used. In the following 2 squares two detonators of each delay may be used. In the stoping area, the delay time must be long enough for the movement of the rock. Normally the delay time is 100 to 500 milliseconds.

For the contour holes the scatter in delay between the holes should be as small as possible to obtain a good smooth blasting effect. Therefore, the roof should be blasted with the same interval number, normally the second highest of the series. The walls are also blasted with the same period number but with one delay lower than that of the roof.

Detonators for tunneling can be electric or non-electric.
The electric detonators are manufactured as MS (millisecond) and HS (half-second) delay detonators.

The non-electric detonators are manufactured as deci-second and half-second delay detonators.

Recommended detonators for tunneling:

Electric detonators:

| | Interval No. | Delay time |
|---|---|---|
| VA/MS | 1 | 25 ms |
| VA/MS | 4 | 100 ms |
| VA/MS | 7 | 175 ms |
| VA/MS | 10 | 250 ms |
| VA/MS | 13 | 325 ms |
| VA/MS | 16 | 400 ms |
| VA/MS | 18 | 450 ms |
| VA/MS | 20 | 500 ms |
| VA/HS | 2 | 1.0 sec |
| VA/HS | 3 | 1.5 sec |
| VA/HS | 4 | 2.0 sec |
| VA/HS | 5 | 2.5 sec |
| VA/HS | 6 | 3.0 sec |
| VA/HS | 7 | 3.5 sec |
| VA/HS | 8 | 4.0 sec |
| VA/HS | 9 | 4.5 sec |
| VA/HS | 10 | 5.0 sec |
| VA/HS | 11 | 5.5 sec |
| VA/HS | 12 | 6.0 sec |

The MS and HS series give 19 periods which is sufficient in most cases. The VA/MS and VA/HS detonators may be used in the same round, as the electric characteristics of the VA detonators are the same, independent of the delay times.

Recommended legwire lengths for a 4 m hole depth are 5.0 and 6.0 m.

Non-electric detonators:

| | Interval numbers | Delay time | Delay time between intervals |
|---|---|---|---|
| Nonel GT/T | 0 | 25 ms | |
| Nonel GT/T | 1–12 | 100–1200 ms | 100 ms |
| Nonel GT/T | 14, 16<br>18, 20 | 1400–2000 ms | 200 ms |
| Nonel GT/T | 25, 30, 35<br>40, 45, 50<br>55, 60 | 2500–6000 ms | 500 ms |

This tunnel series gives 25 different periods and is thus even more versatile than the electric tunnel series.

Recommended tube lengths for bunch blasting with Nonel are 6.0 to 7.8 m.

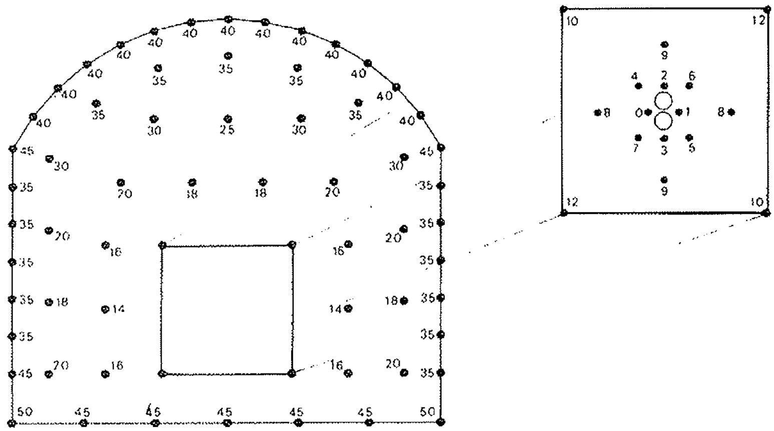


*Fig. 7.16 Typical firing pattern for NONEL GT/T.*

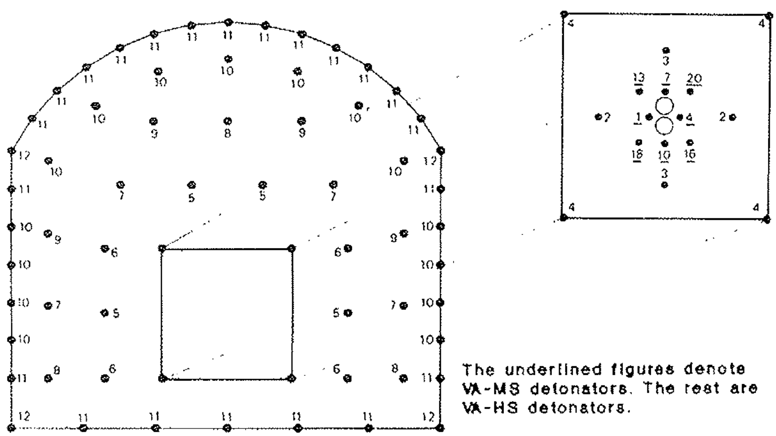



*Fig. 7.17 Typical firing pattern for VA/MS and VA/HS detonators.*

In the 4th square of the cut, four units of VA/HS interval No. 4 are used. This is made possible by wide range of scatter (±200 ms) within the interval for HS detonators.

### 7.1.5 Cuts with angled holes.

**The V-cut.**

The most common cut with angled holes is the **V-cut.**

A certain tunnel width is required in order to accommodate the drilling equipment. Furthermore, the advance per round increases with the width and an advance of 45 to 50 % of the tunnel width is achievable.
The angle of the cut must not be too acute and should not be less than 60°. More acute angles require higher charge concentration in the holes.
The cut normally consists of two V:s but in deeper rounds the cut may consist of triple or quadruple V:s.
Each V in the cut should be fired with the same interval number using MS detonators to ensure coordination between the blastholes with regard to breakage. As each V is blasted as an entity one after the other, the delay between the different V:s should be in the order of 50 ms to allow time for displacement and swelling.

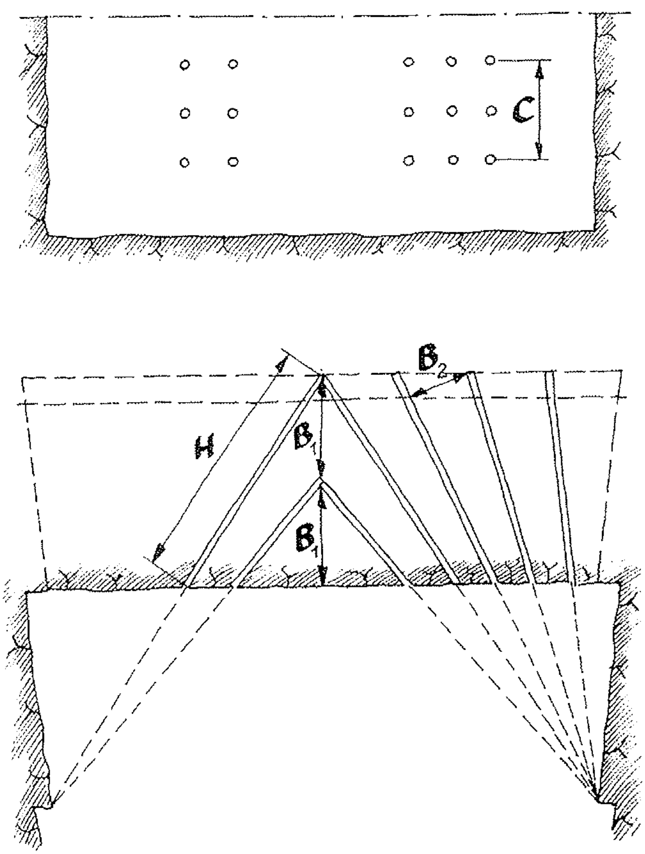



*Fig. 7.18 V-cut.*

**Calculation of the V-cut.**

The following graph (7.19) gives the height of the cut (C) and the burdens $B_1$ and $B_2$ for the cut.

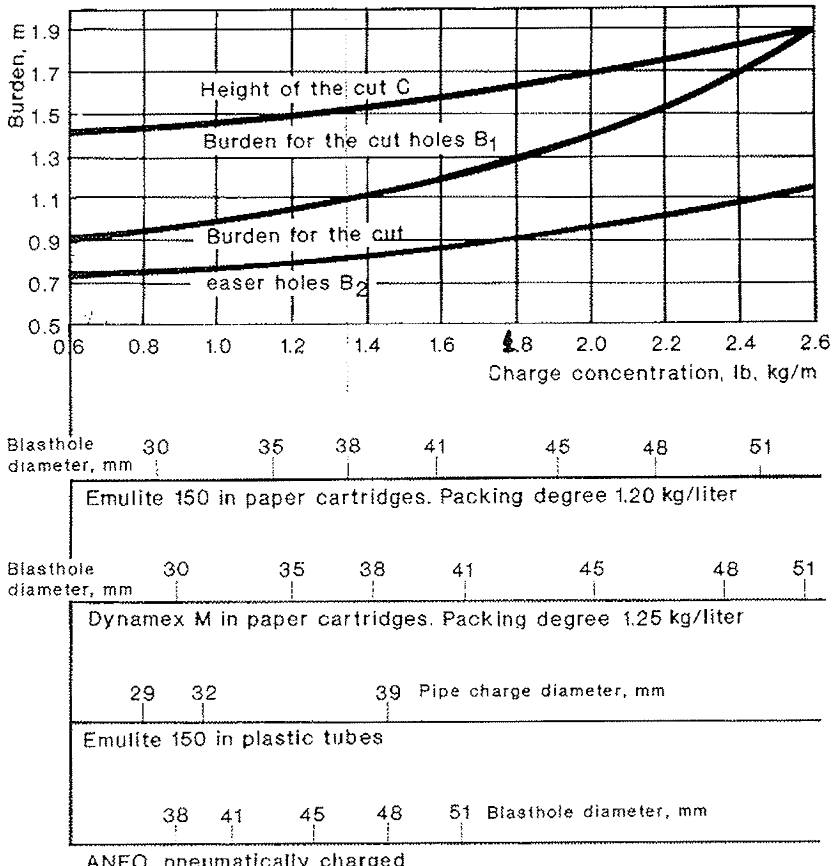



*Fig. 7.19 The burdens $B_1$, $B_2$ and the cut height C in relation to the bottom charge for different blasthole diameters and different explosives.*

**Charging the cut holes.**

The charge concentration in the bottom of the cut holes ($l_b$) can be found in graph 7.19.

The height of the bottom charge ($h_b$) for all cut holes is:

$$h_b = \frac{1}{3} \times H \quad \text{where } H = \text{hole depth (m)}$$

The concentration of the column charge ($l_c$) is:

$l_c$ = 30 to 50 % of $l_b$

The uncharged part (stemming) of the holes in the cut ($h_o$) is:

$h_o = 0.3 \times B_1$

The uncharged part for the rest of the cut is:

$h_o = 0.5 \times B_2$

For the rest of the round, the method of calculation is the same as that in Chapter 7.1.2 Stoping.

**The fan cut.**

The **fan cut** is an other example of angled cuts. Like the V-cut, a certain width of tunnel is required to accommodate the drilling equipment to attain acceptable advance per round.

The principle of the fan cut is to make a trench like opening across the tunnel and the charge calculations are similar to those in Chapter 5.6 Opening the bench. Due to the geometrical design of the cut the constriction of the holes is not large, making the cut easy to blast.

The drilling and charging of the holes are similar to that of the cut holes in the V-cut.

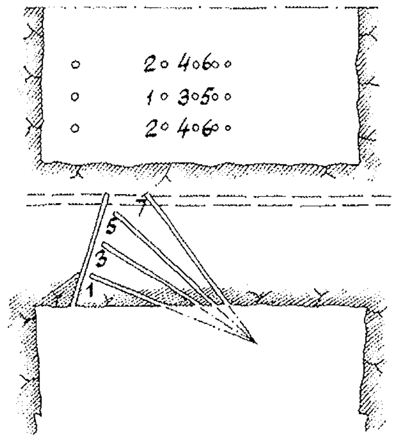
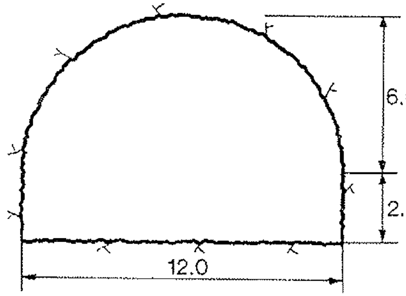


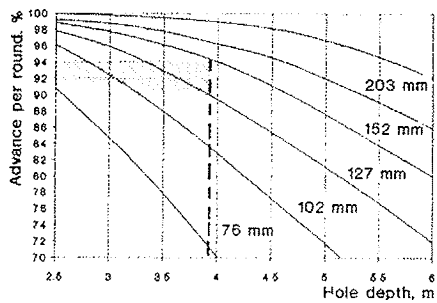

*Fig. 7.20 Fan cut.*

### 7.1.6 Example of calculation.

The project is a 1,500 m long road tunnel with a cross section area of 88 sq.m.
A blasthole diameter of 38 mm is chosen as the tunnel contour is to be smooth blasted. A larger blasthole diameter might cause overbreak from the stoping part of the round.
The drilling equipment is an electro hydraulic jumbo with 4.3 m steel length and feed travel of 3.9 m.
The expected advance is over 90 % of the blasthole depth.
The explosive is Emulite 150 in 29 and 25 mm cartridges for the cut, stoping and floor. Gurit 17×500 mm in plastic cartridges is used for the contour. Nonel GT/T is used for initiation.




To attain an advance of more than 90 % of the blasthole depth, 3.9 m, a large hole diameter of 127 mm should be chosen.
2×89 mm large holes can be an alternative.




**1st square.**

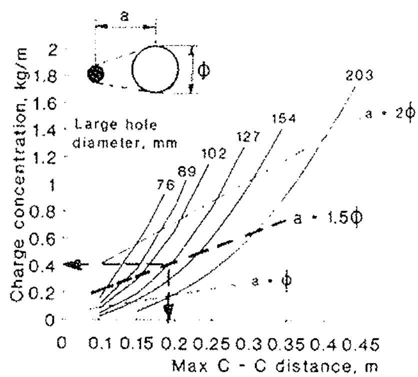
The distance from the center of the large hole to the center of the closest blasthole is:

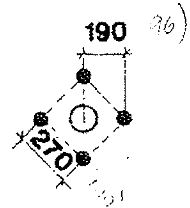
$a = 1.5\ \varnothing$
$a = 1.5 \times 127 = 190$ mm

The width of the 1st square is:

$W_1 = a\sqrt{2}$
$W_1 = 190\sqrt{2} = 270$ mm

The requisite charge concentration for the holes in the 1st square is 0.4 kg/m of Emulite 150. For practical reasons Emulite in 25×200 mm cartridges are used giving a charge concentration of 0.55 kg/m.



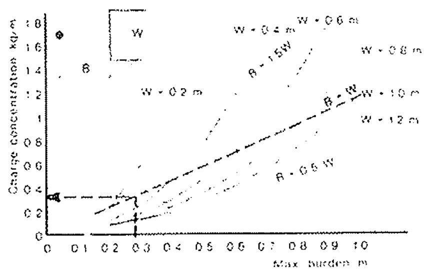
An overcharge of this magnitude does not cause any inconvenience. The uncharged part of the hole is equal to the C–C distance: $h_o = a$

The charge of the hole is the length of the charge $H - h_o$ times the actual charge concentration.

$Q = l_c(H-h_o)$
$Q = 0.55(3.9-0.2)$
$Q = 2.0$ kg

**Key data for the 1st square:**
**$a = 0.19$ m**
**$W_1 = 0.27$ m**
**$Q = 2.0$ kg.**

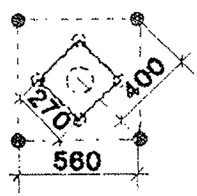
**2nd square.**




The blasting of the 1st square created an opening of 0.27×0.27 m. The burden in the 2nd square is equal to the width of the opening created.

$B_1 = W_1$
$B_1 = 0.27$ m
$C-C = 1.5W_1$
$C-C = 0.40$ m

$W_2 = 1.5W_1\sqrt{2}$
$W_2 = 0.56$ m



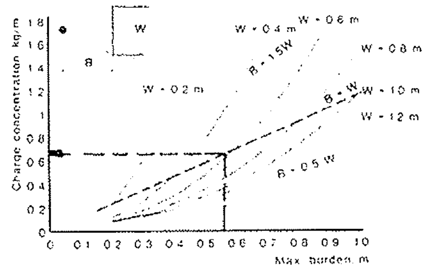

The requisite charge concentration for the holes in the 2nd square is approx. 0.37 kg/m.
Emulite 150 in 25×200 mm paper cartridges is used making the practical charge concentration 0.55 kg/m. The uncharged part of the hole is 0.5×B.

$Q = l_c(H-h_o)$
$Q = 0.55(3.9-0.15)$
$Q = 2.0$ kg.

**Key data for the 2nd square:**
**$B = 0.27$ m**
**$W_2 = 0.56$ m**
**$Q = 2.0$ kg**

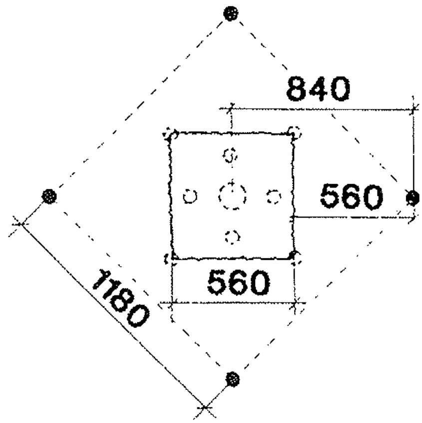
**3rd square.**

The opening has now a width W=0.56 m. The burden B is equal to $W_2$.




$B_2 = W_2$
$B_2 = 0.56$ m
$C-C = 1.5W_2$
$C-C = 0.84$ m

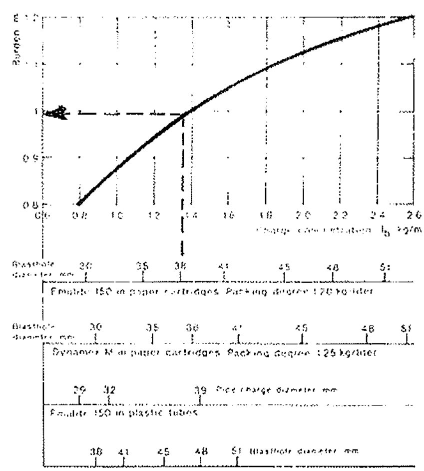
$W_3 = 1.5W_2\sqrt{2}$
$W_3 = 1.18$ m




The requisite charge concentration is approx. 0.65 kg/m. Now the 25×200 mm cartridges do not provide sufficient charge concentration to ensure breakage. A larger dimension of Emulite 150 must be used unless the cartridges are tamped.
Emulite 29×200 mm in paper cartridges give a charge concentration of 0.90 kg/m. The hole will thus be overcharged.
The uncharged part of the hole is 0.5×B.

$Q = l_c(H-h_o)$
$Q = 0.90(3.9-0.3)$
$Q = 3.2$ kg

**Key data for the 3rd square:**
**B = 0.56 m**
**$W_3$ = 1.18 m**
**Q = 3.2 kg.**




**4th square.**

The width of the opening is now 1.18 m. If B is chosen equal to W, the burden will be greater than that of the stoping part of the round. Therefore, the burden must be adjusted to that of the stoping part and the charge calculations are made as for stoping holes.

The burden is chosen from the graph 7.14 to 1.0 m.

The charge concentration of the bottom charge is found in the same graph to be 1.35 kg/m.
From the adjoining table the charge of the hole can be calculated.

$l_b$ = 1.35 kg/m
$h_b$ = 1/3H
$h_b$ = 0.33×3.9
$h_b$ = 1.3 m
$Q_b$ = $l_b$×$h_b$
$Q_b$ = 1.35×1.3
**$Q_b$ = 1.75 kg**

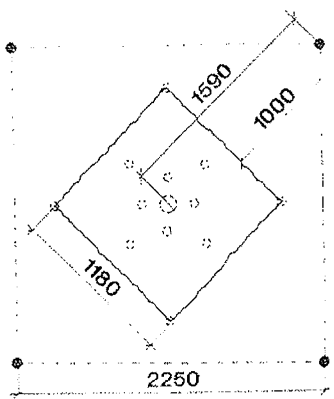
| Part of the round | Burden (m) | Spacing (m) | Height bottom charge (m) | Charge concentration Bottom (kg/m) | Column (kg/m) | Stemming (m) |
|---|---|---|---|---|---|---|
| Floor | 1 · B | 1.1 · B | 1/3 · H | $l_b$ | 1.0 · $l_b$ | 0.2 · B |
| Wall | 0.9 · B | 1.1 · B | 1/6 · H | $l_b$ | 0.4 · $l_b$ | 0.5 · B |
| Roof | 0.9 · B | 1.1 · B | 1/6 · H | $l_b$ | 0.3 · $l_b$ | 0.5 · B |
| ★ Stoping | | | | | | |
| Upwards | 1 · B | 1.1 · B | 1/3 · H | $l_b$ | 0.5 · $l_b$ | 0.5 · B |
| Horizontal | 1 · B | 1.1 · B | 1/3 · H | $l_b$ | 0.5 · $l_b$ | 0.5 · B |
| Downwards | 1 · B | 1.2 · B | 1/3 · H | $l_b$ | 0.5 · $l_b$ | 0.5 · B |

In the bottom charge Emulite in paper cartridges with 29 mm diameter is used and tamped well.

The column charge is:
$l_c$ = 0.5×$l_b$
$l_c$ = 0.5×1.35
$l_c$ = 0.67 kg/m

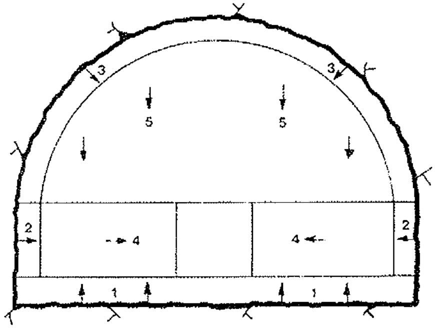
The product with dimensions closest to this is Emulite 150, 29×200 mm with an $l_c$=0.90 kg/m

Practical $l_c$ = 0.90 kg/m
$h_o$ = 0.5B
$h_o$ = 0.5×1.0=0.5 m
$h_c$ = H−$h_b$−$h_o$
$h_c$ = 3.9−1.3−0.5
$h_c$ = 2.1 m.
**$Q_c$ = $l_c$×$h_c$**
$Q_c$ = 0.90×2.1
$Q_c$ = 1.9 kg
$Q_{tot}$ = $Q_b$+$Q_c$
$Q_{tot}$ = 1.75+1.9
**$Q_{tot}$ = 3.65 kg**



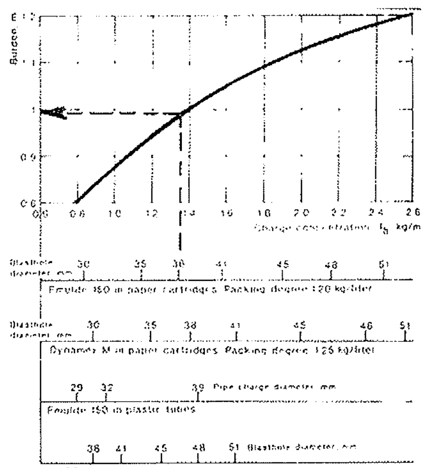

**Key data for the 4th square:**
**B = 1.0 m**
**$W_4$ = 2.2 m**
**Q = 3.65 kg.**

After the cut has been designed, the rest of the round is calculated.




This is most simply done in the following order:
1. Floor holes.
2. Wall holes.
3. Roof holes.
4. Stoping, upwards and horizontal.
5. Stoping downwards.

The reason for starting with the perimeter holes is to decide the burdens and spacings for the outer boundaries of the round.
When these calculations are completed the cut and the stoping holes may be located in accordance with the parameters which apply to them.



**1. The floor holes.**

In the calculation of all perimeter holes, the **"look-out"** has to be taken into account. As mentioned earlier, the "look-out" should not exceed 10 cm + 3 cm/m of hole depth. In this case the "look-out" should be limited to 20 cm.
The **burden** is 1.0 m according to the graph and the **spacing** is 1.1×B.
Due to "look-out", the holes above the floor holes are set out 0.8 m above the floor. The spacing is 1.1 m.

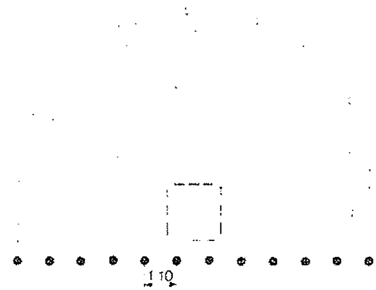
Bottom charge:
$l_b = 1.35$ kg/m
$h_b = 1/3 \times 3.90 = 1.30$ m
$Q_b = 1.35 \times 1.3 = 1.75$ kg

Column charge:
$l_c = l_b = 1.35$ kg/m
$h_o = 0.2 \times B = 0.2$ m
$h_c = H - h_b - h_o = 2.4$ m
$Q_c = 1.35 \times 2.4 = 3.25$ kg

Total charge:
$Q = 1.75 + 3.25 = 5.0$ kg

| Part of the round | Burden (m) | Spacing (m) | Height bottom charge (m) | Charge concentration Bottom (kg/m) | Charge concentration Column (kg/m) | Stemming (m) |
|---|---|---|---|---|---|---|
| Floor | $1 \times B$ | $1.1 \times B$ | $1/3 \times H$ | $l_b$ | $1.0 \times l_b$ | $0.2 \times B$ |
| Wall | $0.9 \times B$ | $1.1 \times B$ | $1/6 \times H$ | $l_b$ | $0.4 \times l_b$ | $0.5 \times B$ |
| Roof | $0.9 \times B$ | $1.1 \times B$ | $1/6 \times H$ | $l_b$ | $0.3 \times l_b$ | $0.5 \times B$ |
| Stoping | | | | | | |
| Upwards | $1 \times B$ | $1.1 \times B$ | $1/3 \times H$ | $l_b$ | $0.5 \times l_b$ | $0.5 \times B$ |
| Horizontal | $1 \times B$ | $1.1 \times B$ | $1/3 \times H$ | $l_b$ | $0.5 \times l_b$ | $0.5 \times B$ |
| Downwards | $1 \times B$ | $1.2 \times B$ | $1/3 \times H$ | $l_b$ | $0.5 \times l_b$ | $0.5 \times B$ |

**Key data for floor holes:**
**B = 1.0 m**
**S = 1.1 m**
**Q = 5.0 kg.**




**2. The wall holes.**

In this particular case the walls are very low and do not make a good example for the design of the drilling and charging pattern.
The drilling pattern is taken from the **smooth blasting** table and the burden is chosen to 0.8 m and the spacing to 0.6 m.
The uncharged part of the hole is 0.2 m.
The charge concentration for Gurit 17×500 mm is 0.23 kg/m. The holes will be charged with 7 tube charges and 1 stick of Emulite 150, 25×200 mm in the bottom.

Bottom charge:
$Q_b = 0.11$ kg

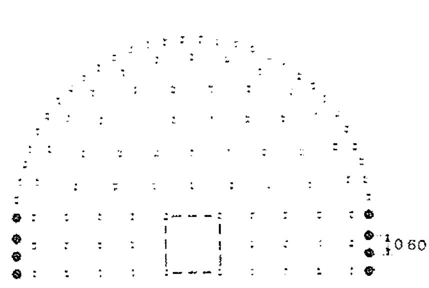
Column charge:
$Q_c = 7\times0.115=0.81$ kg

Total charge:
$Q = 0.11+0.81=0.92$ kg

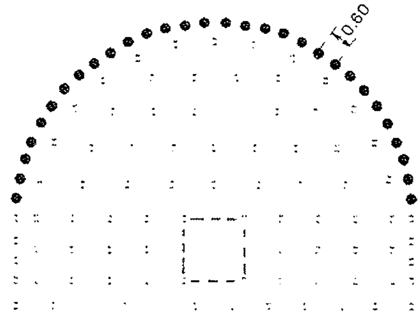
The "look-out" has to be considered, so the burden to be set out on the face is $0.8-0.2=0.6$ m.

**Key data for the wall holes:**
**B = 0.8 m**
**S = 0.6 m**
**Q = 0.92 kg**

| Perimeter hole diam mm | Charge concentration kg/m | Charge type | Burden m | Spacing m |
|---|---|---|---|---|
| 25–32 | 0.11 | 11 mm Gurit | 0.3–0.5 | 0.25–0.35 |
| 25–48 | 0.23 | 17 mm Gurit | 0.7–0.9 | 0.50–0.70 |
| 51–64 | 0.42 | 22 mm Gurit | 1.0–1.1 | 0.80–0.90 |
| 51–64 | 0.45 | 22 mm Emulite | 1.1–1.2 | 0.80–0.90 |



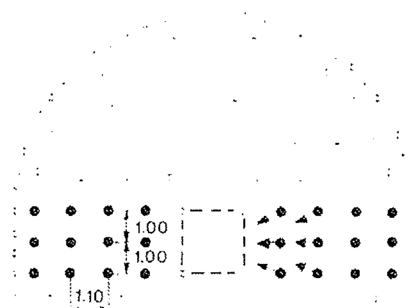
### 3. The roof holes.




The conditions for the roof holes are equal to those of the wall holes. The burden is chosen to 0.8 m and the spacing to 0.6 m.
The charge concentration is the same as for the wall holes.
The "look-out" must be considered in this case as well.

**Key data for the roof holes:**
**B = 0.8 m**
**S = 0.6 m**
**Q = 0.92 kg.**

### 4. Stoping upwards and horizontally.




The stoping holes are calculated in a similar way to the floor holes, but less explosives are needed. While the floor holes must be charged to compensate for gravity and heavage of broken rock, the stoping holes can normally contain less explosives as the direction of breakage is horizontal or close to horizontal.
Charge: Bottom, tamped Emulite 29 mm, $l_b$=1.35 kg/m.
Charge: Column, Emulite 29 mm in paper cartridges with $l_c$=0.90 kg/m.
The burden B is 1.0 m, according to the graph 7.14.

The spacing S will be 1.1 m according to adjoining table.

| Part of the round | Burden (m) | Spacing (m) | Height bottom charge (m) | Charge concentration Bottom (kg/m) | Charge concentration Column (kg/m) | Stemming (m) |
|---|---|---|---|---|---|---|
| Floor | 1·B | 1.1·B | 1/3·H | $l_b$ | 1.0·$l_b$ | 0.2·B |
| Wall | 0.9·B | 1.1·B | 1/6·H | $l_b$ | 0.4·$l_b$ | 0.5·B |
| Roof | 0.9·B | 1.1·B | 1/6·H | $l_b$ | 0.3·$l_b$ | 0.5·B |
| Stoping | | | | | | |
| ★ Upwards | 1·B | 1.1·B | 1/3·H | $l_b$ | 0.5·$l_b$ | 0.5·B |
| ★ Horizontal | 1·B | 1.1·B | 1/3·H | $l_b$ | 0.5·$l_b$ | 0.5·B |
| Downwards | 1·B | 1.2·B | 1/3·H | $l_b$ | 0.5·$l_b$ | 0.5·B |

Bottom charge:
$l_b$ = 1.35 kg/m
$h_b$ = 1/3×3.90=1.30 m
$Q_b$ = 1.35×1.3=1.75 kg

Column charge:
$l_c$ = 0.90 kg/m
$h_o$ = 0.5×B=0.5 m
$h_c$ = $H-h_b-h_o$=2.1 m
$Q_c$ = 0.90×2.1=1.9 kg

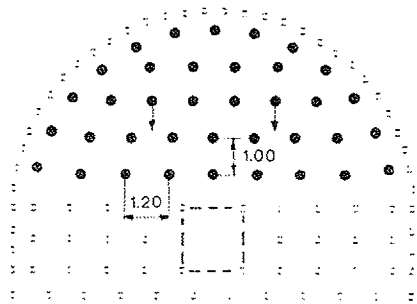
Total charge:
Q = 1.75+1.9=3.65 kg

**Key data for stoping holes upwards and horizontal:**
**B = 1.0 m**
**S = 1.1 m**
**Q = 3.65 kg**

| Part of the round | Burden (m) | Spacing (m) | Height bottom charge (m) | Charge concentration Bottom (kg/m) | Charge concentration Column (kg/m) | Stemming (m) |
|---|---|---|---|---|---|---|
| Floor | 1 · B | 1.1 · B | 1/3 · H | $l_b$ | 1.0 · $l_b$ | 0.2 · B |
| Wall | 0.9 · B | 1.1 · B | 1/6 · H | $l_b$ | 0.4 · $l_b$ | 0.5 · B |
| Roof | 0.9 · B | 1.1 · B | 1/6 · H | $l_b$ | 0.3 · $l_b$ | 0.5 · B |
| Stoping | | | | | | |
| Upwards | 1 · B | 1.1 · B | 1/3 · H | $l_b$ | 0.5 · $l_b$ | 0.5 · B |
| Horizontal | 1 · B | 1.1 · B | 1/3 · H | $l_b$ | 0.5 · $l_b$ | 0.5 · B |
| ★ Downwards | 1 · B | 1.2 · B | 1/3 · H | $l_b$ | 0.5 · $l_b$ | 0.5 · B |

**5. Stoping downwards.**

The design of the drilling pattern for stoping downwards is similar to stoping in other directions with the difference that larger spacing may be permitted. The charge of the holes is the same in all stoping.




**Key data for stoping holes downwards:**
**B = 1.0 m**
**S = 1.2 m**
**Q = 3.65 kg**

SUMMARY

The round consists of 127 blastholes with 38 mm diameter and 1 large hole with 127 mm diameter.

The round is charged as follows:

| Part of the round | No. of holes | Kind of explosive | Weight per hole (kg) | Total (kg) |
|---|---|---|---|---|
| Cut | | | | |
| 1st square | 4 | Emulite 150, 25 mm | 2.0 | 8.0 |
| 2nd square | 4 | Emulite 150, 25 mm | 2.0 | 8.0 |
| 3rd square | 4 | Emulite 150, 29 mm | 3.2 | 12.8 |
| 4th square | 4 | Emulite 150, 29 mm | 3.65 | 14.6 |
| Floor holes | 12 | Emulite 150, 29 mm | 5.0 | 60.0 |
| Wall holes | 8 | Emulite 150, 25 mm | 0.11 | 0.9 |
| | | Gurit 17 mm | 0.81 | 6.5 |
| Roof holes | 30 | Emulite 150, 25 mm | 0.11 | 3.3 |
| | | Gurit 17 mm | 0.81 | 24.3 |
| Stoping: | | | | |
| Upwards | 8 | Emulite 150, 29 mm | 3.65 | 29.2 |
| Horizontal | 16 | Emulite 150, 29 mm | 3.65 | 58.4 |
| Downwards | 37 | Emulite 150, 29 mm | 3.65 | 135.1 |

| | | |
|---|---|---|
| Consumption per round: | Emulite 150, 25×200 mm | 20.1 kg |
| | Emulite 150, 29×200 mm | 310.1 kg |
| | Gurit | 30.8 kg |
| | Nonel GT/T | 127 units |

The expected advance per round is over 90 %. It is assumed to be 3.55 m.

$$\text{Specific charge: } \frac{361.1}{3.55 \times 88.0} = 1.16 \text{ kg/cu.m.}$$

**Explosives consumption for the whole project:**
**Number of rounds: 1500/3.55=425**
**Consumption of**

| | |
|---|---|
| **Emulite 150, 25×200 mm** | **20.2×425 = approx. 9 tons** |
| **Emulite 150, 29×200 mm** | **310.1×425 = approx. 132 tons** |
| **Gurit** | **30.8×425 = approx. 13 tons** |
| **Nonel GT/T** | **127×425 = approx. 54000 units.** |

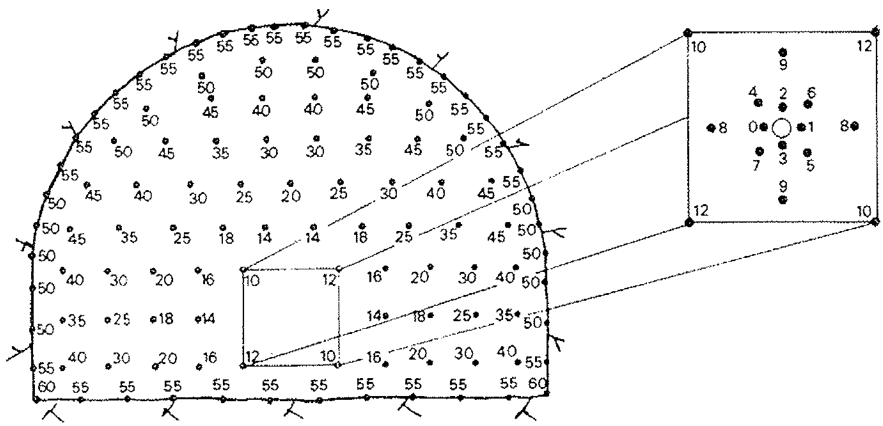


*Fig. 7.21 Drilling and firing pattern.*

## 7.2 Shafts.

In mining, shafts form a system of vertically or inclined passageways which are used for transportation of ore, refill, personnel, equipment, air, electricity, ventilation etc.
In underground construction, shafts are driven for the building of penstocks, cable shafts, ventilation and elevator shafts, surge chambers etc. In addition, shafts are driven as "glory holes" for transportation of material which is not accessible by other means than vertical or close to vertical tunnels.
Shafts are either driven downwards, sink shafts, or upwards, raise shafts.

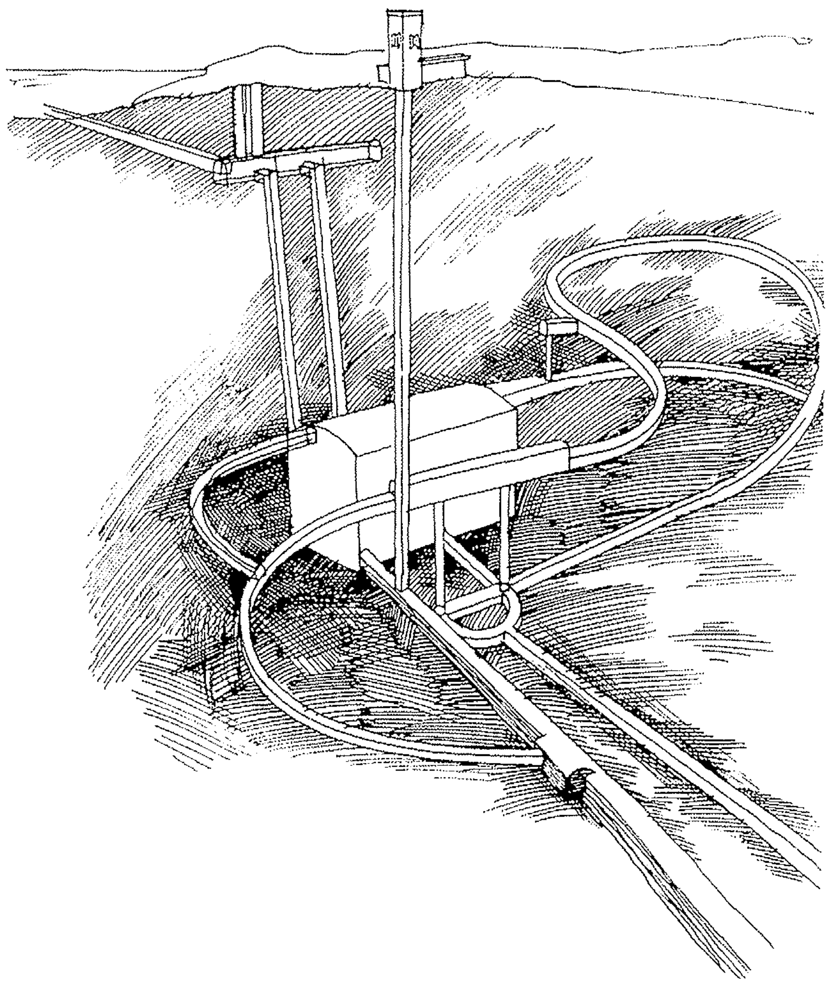


*Fig. 7.22 Typical tunnel system in a hydroelectric power plant.*

### 7.2.1 Sink shafts.

Sink shafts are passageways sunk from the surface downwards or underground from one level to a lower one. The majority of the sink shafts are driven vertically.
Shaft sinking is one of the most difficult and risky blasting jobs as the work area is normally wet, narrow and noisy. Furthermore, the drilling and blasting crews are exposed to falling objects.
The advance is slow as the rock has to be removed between each blast with special equipment which has limited digging capacity. The blasted rock must be well fragmented to suit the excavation equipment.

The design of the cross section of the shaft principally depends on the quality of the rock. Nowadays most of the shafts are made with a circular cross section which gives better distribution of the rock pressure, thus decreasing the need for reinforcement, especially in deep shafts.

The most common drilling and blasting methods are benching and blasting with pyramid cut.

The **benching** method is a fast and efficient method as the time-consuming cleaning of the floor between the blasts can be minimized. It is also easy to keep the shaft free from water as a pump can always be placed in the lower blasted part of the shaft. The drilling and charging pattern is similar to that of smaller surface blastings.
The burden and spacing vary with the hole diameter but the drilling pattern is more closely spaced than for surface blasting due to higher constriction.

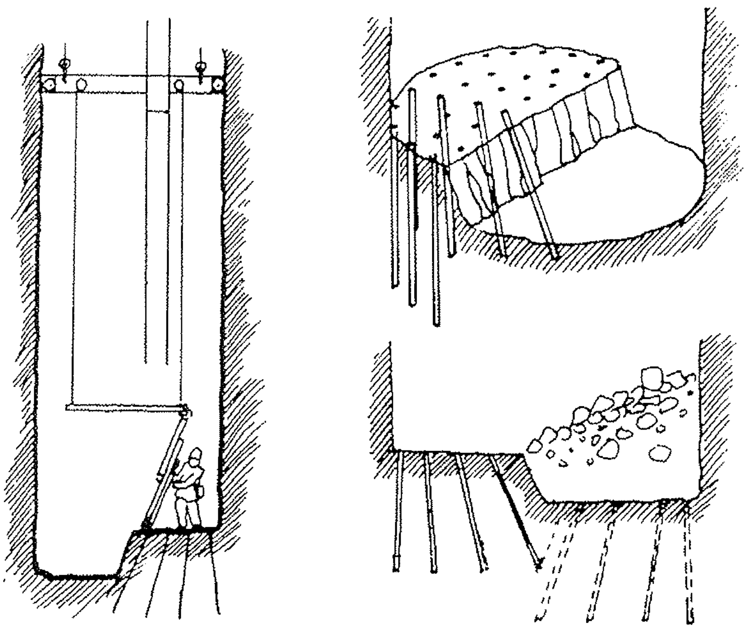


*Fig. 7.23 Shaftsinking by benching.*

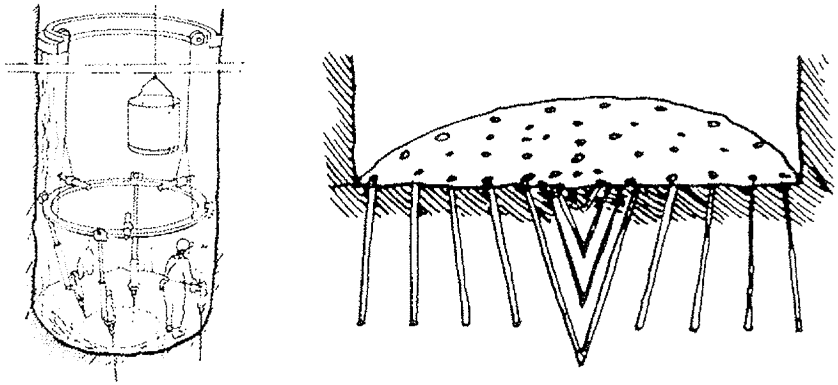


*Fig. 7.24 Shaft sinking with pyramid cut.*

Shaft sinking with **pyramid cuts** is similar to tunnel blasting with V-cuts. The drilling is done with a "drill-ring" which is composed of a circular I-beam to which the drilling machines are fixed. The "drill-ring" may be fixed to the shaft walls with bolts. Due to the construction of the "drill-ring", the cut will be conical.

The explosives used in shaft sinking must always be water resistant. Even if the ground is dry, the flushing water from the drilling will always stay in the blastholes.
For this reason explosives with excellent water resistance properties are preferred. Emulite 150 and Dynamex M are easily tamped to utilize the hole volume well, thus decreasing the number of holes and the drilling and charging time. The specific charge in shaft sinking is rather high, ranging from 2.0 kg/cu.m. to 4.0 kg/cu.m.

The initiation of the blast may be done with electric detonators or non-electric detonators. As a sink shaft is a small confined area, thunderstorms are a particular hazard as stray currents tend to be transmitted down the shaft on pipes and cables. To avoid problems with evacuation of the blasting crew during a thunderstorm, NONEL detonators should be used.

### 7.2.2 Raise shafts.

The drifting of raise shafts – shafts which are driven from blasted underground chambers or tunnels, vertically or inclined upwards – is one of the most difficult, most costly and most dangerous undertakings in mining and construction.
As the drifting of raise shafts has increased in the world, new methods have been developed to make the work more mechanized, cheaper and safer.
Raise shafts were drifted in more or less the same way for decades until the 1950's when new types of raise shaft elevators were taken into use.

**Various raise shaft drifting methods where blasting is part of the method.**

Older methods:

* Timbered shafts
* Open shafts

Modern methods:

* Boliden elevator type Jora
* Alimak Raise Climber
* Longhole drilling

To start with the older methods, the timbered shaft method was the most common method in Sweden until some 40 years ago and is still occasionally used for shorter shafts. The raise shaft is driven vertically and divided into two sections by a timber wall which is extended before each blast. When the round is fired, one section is filled with rock. The blasted rock will then act as a working platform for the next round. In order to maintain the working height at the face some rock has to be excavated after each blast. The second section is used as a ladderway and for transportation of equipment, drill steel, explosives and timber. The ventilation is also placed in this section which is covered during blasting.

Timbered raise shafts have been driven up close to 100 m, but normally the maximum height should not exceed 60 m. The cross section area is usually 4 sq.m. and the advance per round approx. 2.2 m.

*Fig. 7.25 Timbered raise shaft.*

The timbered shaft method was replaced by open shaft methods when the cost of timber became too high. In one of these methods a working platform of planks is laid on timber which is supported by bolts in the shaft walls. New bolt holes are drilled in the shaft walls when the round is drilled so the platform can be moved upwards as the work proceeds.

Another open shaft method is to use steel tubes instead of timber. The steel tubes are bolted to the shaft walls and the tubes support the platform.

The open shaft methods are rarely used and when used, only for short raises, up to 25 m. From a safety point of view none of the open shaft methods is to be recommended.

The cross section is normally 4 sq.m. and the advance approx. 2.2 m.

**The JORA lift method.**

Raise shafting using a lift cage hanging on a wire which runs through a large drillhole has been used in Sweden and other countries since the 1940's, but it was not until the 1950's when Boliden AB developed the JORA lift, that the method came into wider use.

A large hole, diameter 110 to 150 mm, is drilled from an upper level in the center of the intended shaft. Through the hole a wire is sunk down to the lower level and a working platform with a lift cage is fastened to it. By a lifting gear the platform is elevated up to the shaft face by remote control from the lift cage. The drilling and charging are carried out from the platform on the top of the lift cage and some scaling can be done from the cage with the protection of the platform. During the scaling, drilling and charging operations the platform is fixed with bolts to the shaft walls. Before blasting the platform is lowered down and placed on a sledge like vehicle and towed aside. The wire is lifted up through the large hole before blasting. The large hole is used as cut hole in the blasting of the round. Due to the large size of the cut hole, advances of up to 4 m are obtained. The area is approx. 4 sq.m. and the maximum height is 100 m. In this method it is necessary to have free space above the shaft for the drilling of the large hole and for the placing of the lifting gear.

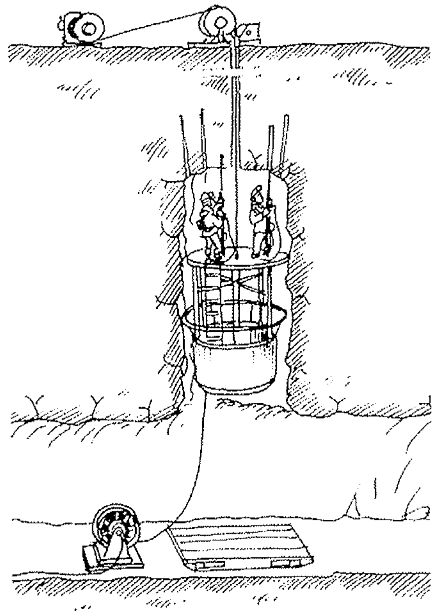


*Fig. 7.26 The JORA lift.*

**The ALIMAK Raise Climber.**

The Alimak raise shaft driving method was introduced in 1957 and became the most utilized system in the world because of its flexibility, safety, economy and speed.

The equipment consists of a raise climber with a working platform, which covers practically the entire area of the shaft. Under the platform there is a cage for the transport of personnel, material and equipment. The raise climber is propelled by a rack and pinion system along a special guide rail. The rail system incorporates a tube system for the air and water supply to the drilling equipment. The system also provides air for the blasting with NONEL and to ventilate the raise after the blasting.

The platform is equipped with a protective roof under which the blaster stands during scaling and drilling operations. If the inclination of the raise shaft is 60° or less the scaling may be done gradually during the ascent under the protection of the previously scaled hanging wall.

The Alimak method can be used for vertical as well as inclined shafts. The lower limit of the inclination depends on the angle of repose of the rock.

Unlike other modern methods for raise shafting, the Alimak needs only one point of attack, the lower one. The upper break-through point may be prepared while the raise is driven.

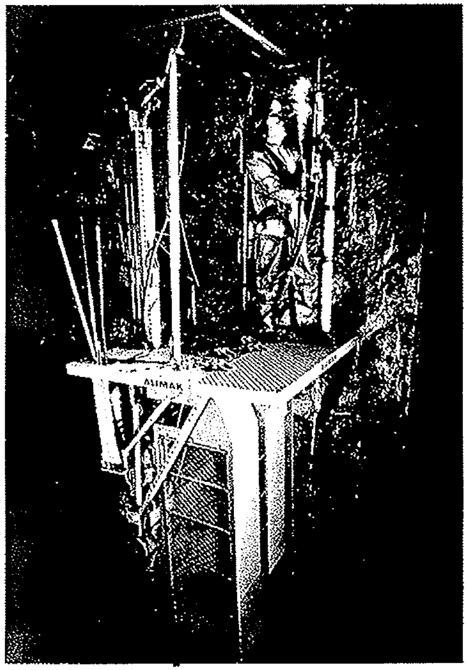


*Fig. 7.27 The ALIMAK Raise Climber.*

The lengths which may be driven are only limited by the time which is at the blasting crews' disposal for ascent, scaling, drilling, charging, descent and blasting. For an 8 hour shift, the upper limit should be around 2,000 m. The lengths are also limited by the type of drive. The air-driven raise climber may be used for up to 150 m shaft length, electric drive up to 900 m. For longer shafts diesel-hydraulic driven climbers are used.

The area is normally 4 sq.m., but inclined shafts have been driven full face upp to 36 sq.m.

Drilling and charging patterns are the same for all above mentioned raise shafting methods. Normally a raise shaft of 4 sq.m. is driven upwards and then the shaft is stoped to its final area. However, sometimes the shaft is driven "fullface" and as mentioned earlier areas up to 36 sq.m. have been successfully blasted.

The drilling and firing pattern for a raise shaft does not differ from that of a horizontal tunnel of the same size.

The Alimak work cycle:

**Drilling:**

The drilling and charging is carried out from the raise climber's platform under a specially designed protective roof. Both air and water to the drilling machines are supplied through tubes in the guide rail sections.

**Blasting:**

After drilling and charging the round, the raise climber is driven to the bottom and under the roof of the drift. During the blast, the climber is therefore well protected from falling rock.

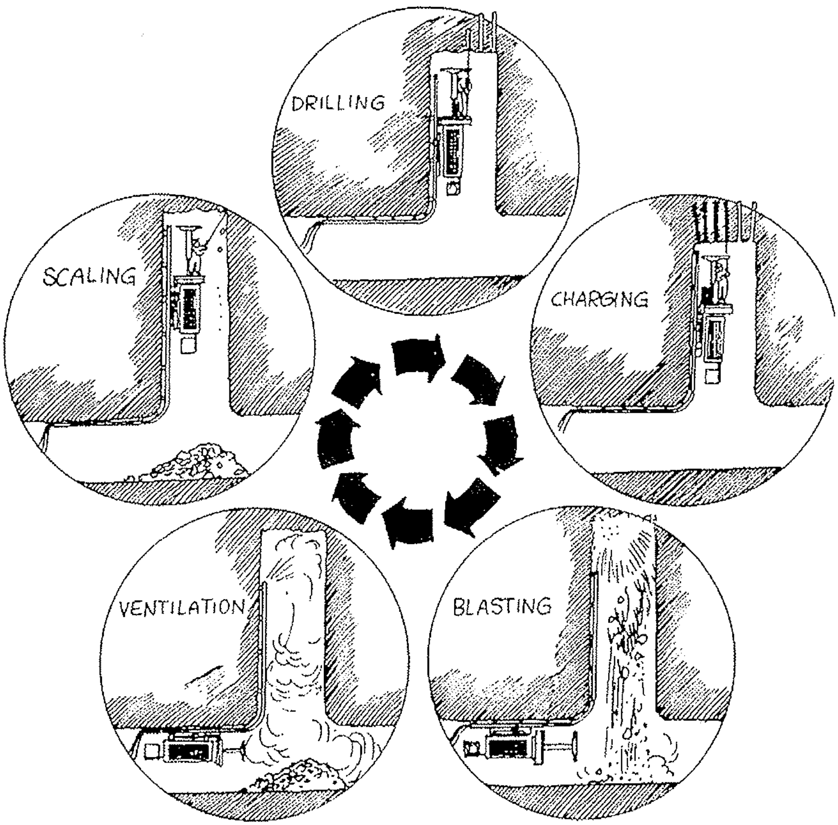



*Fig. 7.28 The ALIMAK work cycle.*

**Ventilation:**

After blasting the raise is ventilated and sprayed with water. The top of the guide rail is protected by a header plate which also acts as a water diffuser during the ventilation phase.

**Scaling:**

Scaling of the roof and walls of the raise is done from under the protective roof which gives the workmen good protection.

Generally large hole cuts are used and the design of the cut varies with the diameter of the large hole. (See 7.1.1 The cut, in Chapter Tunneling.)

The normal hole depth is 2.4 m and the expected advance 2.1 to 2.2 m.
The drilling is done with stopers, which are designed for raise driving, overhead drilling and roof bolting or drilling machines with jack legs.
For the blastholes drill series 11 (34 to 32 mm) is used and the large hole diameter is normally 75 mm.
For the stability of the walls and to avoid overbreak, the walls of the raise are normally smoothblasted. The smooth blasting method is also used if the shaft is to be widened at a later stage in order to avoid excessive scaling and to decrease the risk of rockfall.

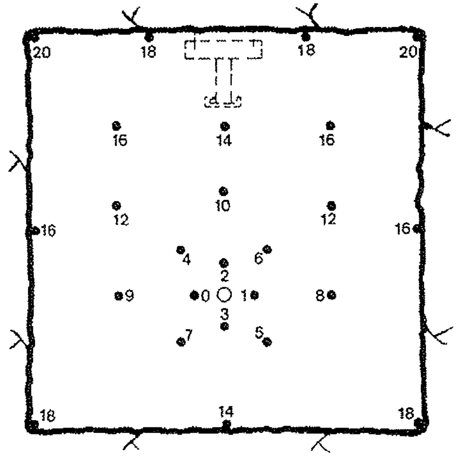


*Fig. 7.29 Drilling and firing pattern for 4 sq.m. raise shaft.*

A normal pilot shaft has an area of 4 sq.m. Normally one round is drilled and blasted per shift with an advance of 2.2 m. Working 2 shifts per day, the advance should be 4.4 m but taking disturbances in the work cycle into account, the long term advance is approx. 3.5 m/day or 70 to 90 m per month.

**Shaft raising by long hole drilling.**

In this method, all drilling is done downwards with parallel holes and the whole area is drilled at the same time.
Great precision in drilling and charging is a must and the lack of precision has earlier limited the practical height to 25 to 30 m. Now, with new drillrigs e.g. Atlas Copco Simba, the drilling can be carried out with great precision in any direction from vertical to 50°. With the Simba the deviation can be kept under 0.5 % for holes up to a length of 50 m.

The long hole drilling method is also advantageous from a safety point of view as all drilling and charging work is carried out from a safe location.

Two different cuts are used:

- large hole cut (blasting towards a large hole).
- crater cut (blasting towards the lower free face of the raise).

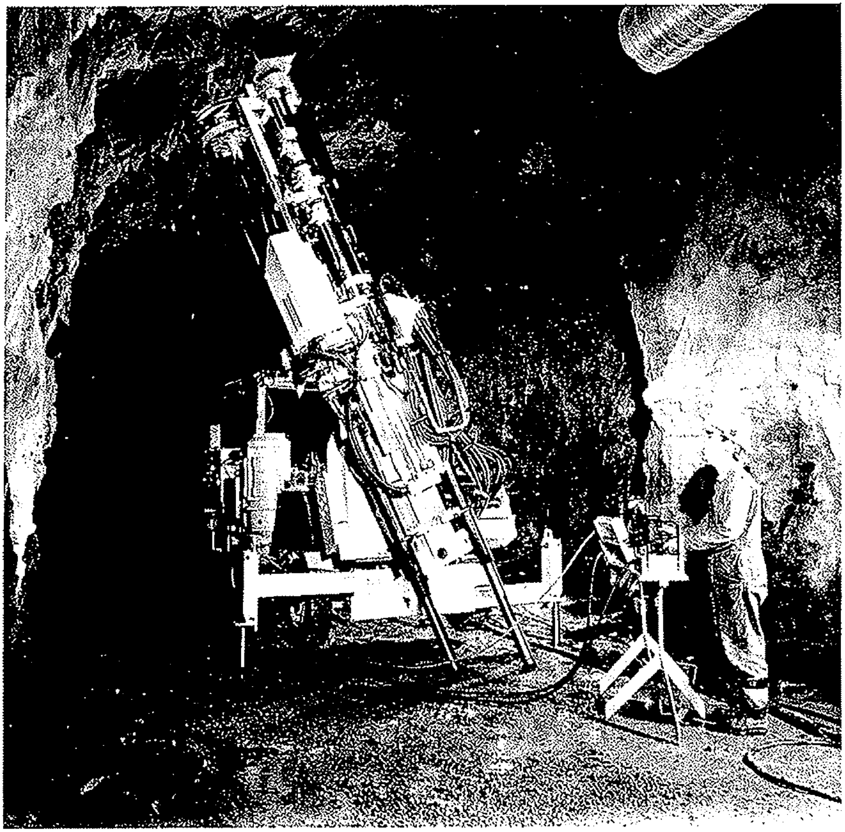


*Fig. 7.30 Simba.*

The large hole cut came first and is still the most common one.
The drill holes in the round have a diameter of 50 to 75 mm and the central large hole is reamed to a diameter of 102 to 203 mm.

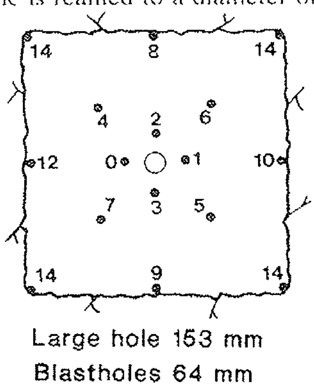



*Fig. 7.31 Firing sequence for 4 sq.m. raise.*

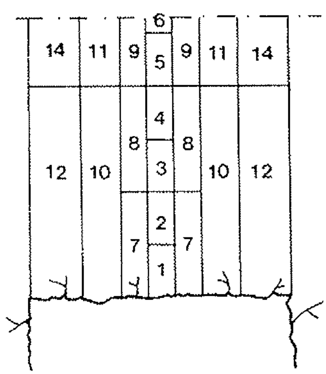



*Fig. 7.32 Round sequence for raises with larger cross section.*

The design and charging of the cut follow the same principles as described in Chapter 7.11 Tunneling, The cut. The firing sequence depends on the faulty drilling so the hole with the smallest real burden is fired with the lowest period number. It is therefore necessary to map every hole with regard to the faulty drilling.

The charging is done from the upper level. A piece of wood is lowered down on a rope and when the wood passes the lower mouth of the hole the rope is tightened and the piece of wood forms a plug for the lower part of the hole. The charges are lowered to the bottom of the hole. The hole should not be stemmed as the stemming may sinter and block the hole for the subsequent blast. The holes may be relatively overcharged compared with a tunnel cut as the charges are not confined at either end. Furthermore, the blastholes are normally of larger diameter than those used in tunnels. The risk of recompaction of the rock in the cut section can be considered as low even if the holes are considerably overcharged.

**Crater blasting.**

The blasting of a long hole drilled raise can also be carried out towards the free lower surface of the raise with a crater cut. No large diameter center hole is needed but the blastholes normally have a larger diameter than in the previous method. The crater blasting method is used only for the cut section to open a hole of approx. 1 sq.m., then normal stoping will follow.

The crater cut consists of five holes, one center hole and four edge holes. The center hole is blasted first whereupon the edge holes are blasted one by one with different delays.

Before charging, the holes are plugged with a piece of wood which is lowered down from the upper surface on a rope and secured to the lower rock surface. The hole is then filled with sand to the calculated level of the explosives charge. The charge should have a diameter close to that of the hole.

The charge is then stemmed with water. (Any other stemming may sinter and block the hole, making subsequent blasting operations impossible.)

The requisite charge weight and depth of the charge are calculated from Livingstone's theories as follows:

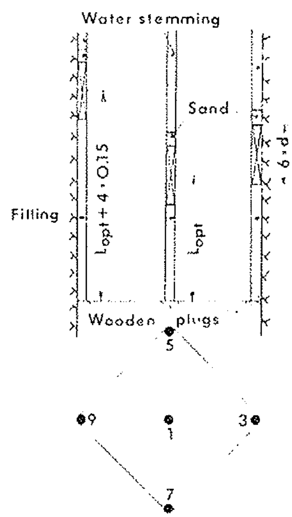



*Fig. 7.33 Drilling, charging and firing pattern for crater cut.*

1. The length of the charge shall be 6 times the blasthole diameter.

$$l = 6 \times d \qquad \text{(mm)}$$

2. The optimum depth of the charge is 50 % of the critical depth.

$$L_{opt} = 0.5 \times L_{crit} \qquad \text{(mm)}$$

3. The critical depth depends on the charge weight.

$$L_{crit} = S \times Q^{1/3} \qquad \text{(mm)}$$

where S = the strain energy factor approx. 1.5 (depending on the explosive used and the type of rock)
Q = charge weight in kg.

4. The charge weight is then

$$Q = \frac{3 \times d^3 \times \pi \times p}{2} \qquad \text{(kg)}$$

where p = charging density (1.2 kg/liter for Emulite 150 and 1.35 kg/liter for Dynamex M)

5. The optimum charge depth is then related to charge weight, explosives density, blasthole diameter and strain energy factor as follows:

$$L_{opt} = 0.5 \times S \times \sqrt[3]{\frac{3 \times \pi \times d}{2}} \times d \times 10 \qquad \text{(mm)}$$

The crater theory is valid only for the center hole. The charge of the edge holes is placed so that the burden is less than the charge depth of the crater hole. The charge depth increases with 10 to 20 cm between each hole.

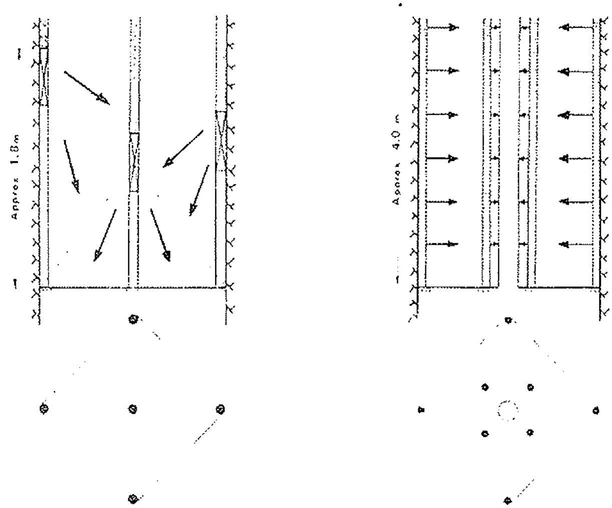



*Fig. 7.34 Comparison of crater cut and standard large hole cut.*

The advantages with crater cut compared to large hole cut are:

1. Lower cost for drilling and explosives as less holes are drilled in the cut. The same hole diameter is used in all holes.
2. Drilling precision is not as essential as for large hole cuts.
3. Simpler blasting practice with less need for well trained personnel.

The disadvantage with the crater cut method is the relatively short rounds that may be shot each time.

## 7.3 Underground chambers.

The military defense forces started early to utilize solid rock for construction of fortifications which gave many advantages over surface construction. Solid rock is difficult to penetrate and underground chambers are difficult to discover and easy to guard.
The field of application is huge: Protection for guns, ammunition and soldiers, protection for submarines and smaller warships, storage for material, fuels and foodstuffs and not least as air-raid shelters for civilians.
Oil was initially stored in surface tanks, but after WWII storage in unlined storage chambers has become the most common method. The increased exploitation of sub-surface storage has to a great extent been due to the rapid development of rock blasting techniques. The increased mechanization of the operations has resulted in relatively unchanged construction costs over a number of years, while at the same time the price of land has increased considerably.
Common to all types of underground chambers is that they are well protected from a military point of view. They are well camouflaged and more difficult to damage than surface storage facilities if attacked from the air or overland. They require little land: surface space is only needed for access roads, ventilation etc. From an environmental point of view sub-surface storage is safer, as leakage does not often occur from underground chambers. It is safer than surface storage in case of fire, as the supply of oxygen is often insufficient to allow a bigger fire to develop.
Underground chambers have many fields of application:

- storage for different products
  cold storage for food, wines, water, oil etc.
- garages, telephone exchanges, swimming pools
- military and civil stores and workshops
- air-raid shelters for people
  aircrafts
  warships
  archives
- storage for lightly contaminated nuclear waste
- storage of nuclear residue
- hydro-electric powerstations

Some of the applications may be combined. In wartime, the space which is normally used for garages, workshops or swimming pools can be utilized as air-raid shelters.

The basis for underground chambers is a qualitative sound rock to build in. Some economic aspects have to be considered. If the chamber is located at too shallow a level, the cost of reinforcing the rock may be high as the quality of the surface rock is normally poorer than rock at deeper levels. However, deep location results in long access roads, which may cause problem both during construction and when the chambers come into use.

From the point of view of rock blasting techniques, the construction of underground chambers does not differ from that of tunnels of the same magnitude. The width of underground chambers cannot be too great due to the inability of the rock to support the roof by its own strength. For oil storage chambers and machine halls for hydro-electric power-plants, widths of 20 to 24 m have been constructed without need for heavy reinforcement. The height of the chambers may be up to 40 m.
Small underground chambers, with a height of less than 8 m are blasted as tunnels. In larger chambers, the operation has to be divided into several stages of drilling and blasting in which different methods are used:

* pilot tunnel with side stoping
* horizontal benching
* vertical benching.

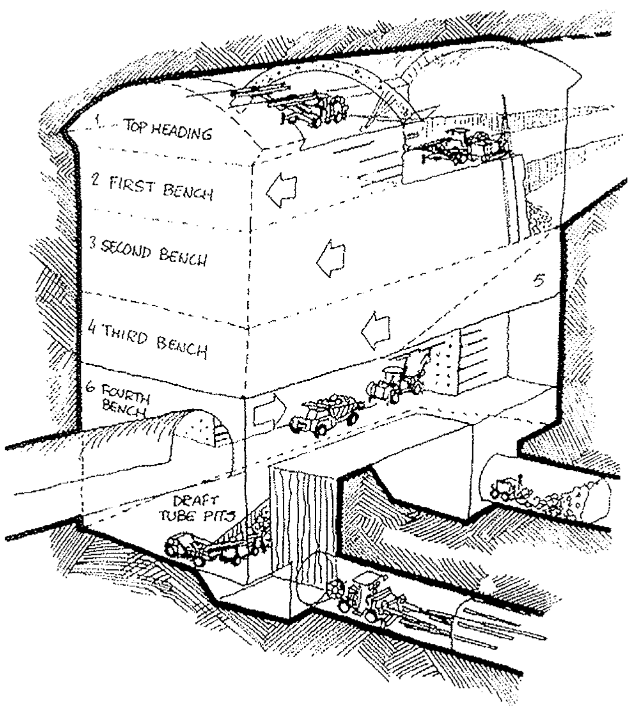



*Fig. 7.35 Drifting stages in underground chamber.*

The pilot tunnel is drifted at the roof of the chamber to facilitate scaling and reinforcement. The side stoping to full width is then carried out. Scaling and, if necessary, bolting and shotcreting of the roof are done simultaneously to avoid future expensive reinforcement work.
Then blasting is carried out in one or several benches. It is common for the first bench to be a horizontal bench utilizing the drilling equipment for the tunnel. Some rock chambers are also designed in such way that no space is available close to the wall for the boom of the vertical drilling equipment. The disadvantage with horizontal benching is that the height and depth of the round depends on the drilling equipment. The height is normally limited to 8 m and the depth of the round to 4 m. Other limitation on the blast design is that the blasthole diameter can rarely exceed 51 mm.
Excavation of the blasted material must be carried out between each blast.
Vertical benching is the dominant method for benching in rock chambers. The advantages with vertical benching is that drilling and excavation may be carried out simultaneously. The bench height may be varied within a wide range and larger blastholes may be used, often with better economy as a consequence. It is also easier to obtain a smoother contour with vertical benches than with horizontal.
The charge calculations for the pilot tunnel, side stoping and horizontal benching are the same as presented in Chapter 7 Tunneling, where the side stoping is calculated as stoping holes with horizontal breakage and the vertical bench as stoping holes with upwards breakage.
The vertical benching is calculated in accordance with Chapter 5 Bench blasting. If excavation is not carried out between the blasts, the specific charge has to be increased in order to compensate for movement of rock from previous rounds. See 5.8 Swelling.
Access tunnels are required for each bench for the transport of rock and equipment.

In certain cases, restrictions due to geological reasons, ground vibrations etc., may affect the execution of the work.
In Fig. 7.36 the roof must be bolted with 8 m long bolts and sprayed with concrete before any side stoping can be done.
The vertical bench is limited to a height of 4 m which makes it feasible to make a raise shaft, "glory hole", for the transport of the blasted rock. The raise shaft is a long hole drilled one, from the upper level and the blasting starts at the lower level. See Chapter 7.2.2.

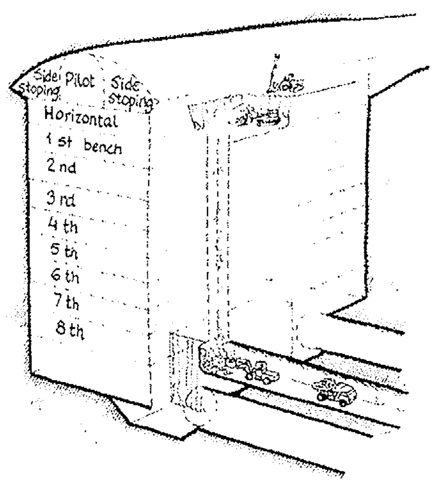



*Fig. 7.36 Drifting stages for machine hall in hydro-electric power plant.*

# 8. CONTOUR BLASTING

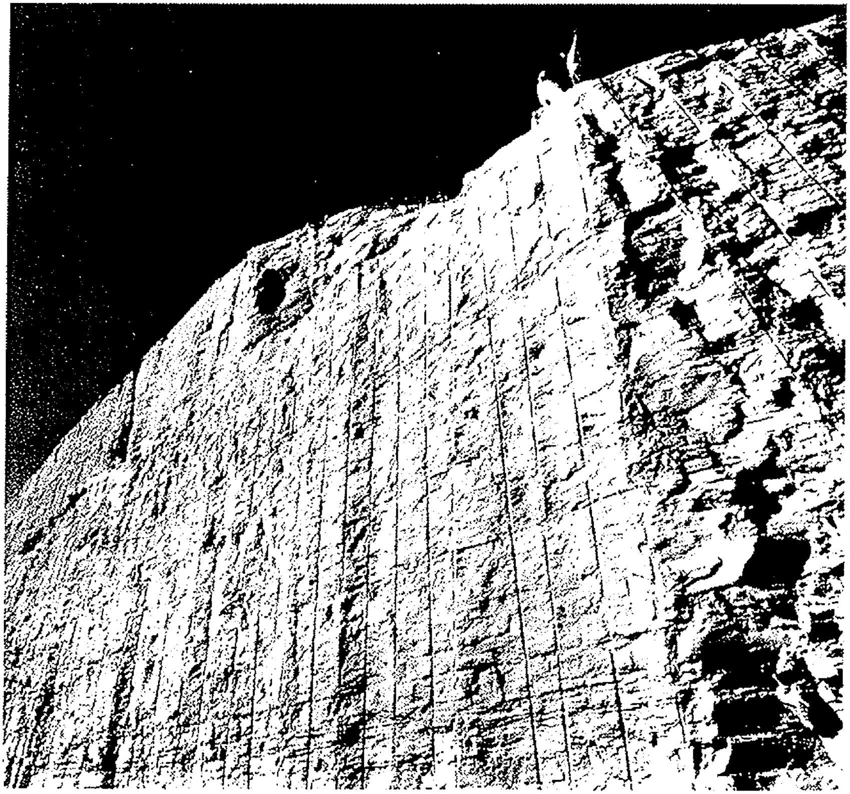


*Fig. 8.1.*

## 8.1 General.

From the beginning of the explosives era in the mining and construction industry, many attempts have been made to find methods to control overbreak and damage to the remaining rock.

In tunneling, road and railroad cuts, it is of the utmost importance that the remaining rock is of high quality in order to avoid rockfall, rockslides and excessive maintenance work.

Accurate blasting is needed especially in those tunnels where the overbreak has to be replaced with expensive concrete.

Numerous blasting techniques have been used to control overbreak. They all have one objective in common: To minimize the stress and fracturing of the rock beyond the theoretical excavation line by reduction and better distribution of the explosives charges.

The approach was initially mostly by trial and error, but lately more sophisticated and scientific methods have been worked out in both Europe and U.S.A.

It is often claimed that good overbreak control cannot be expected in all geological formations. That is true, but carefully executed blasting will minimize the overbreak even in severe geological conditions.

The first approach to control overbreak was by Line Drilling, which simply involved a single row of uncharged holes closely spaced along the perimeter of the excavation, providing a weak plane to which the blast could break. Line Drilling was modified over the years, all or just some of the holes were charged with light charges. The spacing between the holes was then modified and was made larger.

New methods like Cushion Blasting and Smooth Blasting were created and the perimeter holes were blasted after the main blast.
The idea of cutting the blasting area from the remaining rock by forming a crack along the theoretical excavation plane created the development of Pre-splitting Blasting, where the perimeter holes are blasted before the rest of the round.

The above mentioned methods have applications in both underground and surface blasting work.

Common to all four methods is that in the charge calculations, not only the perimeter holes have to be considered, but also the holes closest to the perimeter line have to be charged in such a way that they do not create cracks which reach beyond the perimeter of the blast.

The methods which will be described are:

* Line drilling
* Cushion blasting
* Smooth blasting
* Presplitting

Line drilling and cushion blasting will be described briefly as the use of these methods has declined as they are too time-consuming and costly. The development of special charges for controlled contour blasting has turned the blasters to smooth blasting and presplitting methods.

## 8.2 Line drilling.

Application: Mostly surface excavation.

The idea with line drilling is to create a plane of weakness by drilling closely spaced, small diameter holes along the perimeter of the excavation to which the blast can break.

Line drillholes are usually not over 75 mm (3") in diameter and the spacing is 2 to 4 times the diameter of the hole. To use larger hole diameters is often too costly.

Precision in drilling is very important for good result. Any deviation from the plane will have an adverse effect on the result.

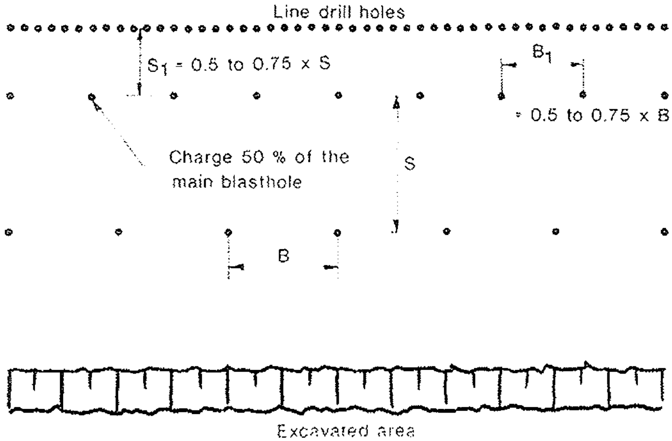



*Fig. 8.2 Line drilling.*

The blastholes directly adjacent to the line drillholes are usually more closely spaced than the holes in the rest of the round. They are also more lightly charged. The common practice is to reduce the burden and spacing by 25 to 50 % and to reduce the charge by approx. 50 %. The charges should be well distributed in the hole i.e. by using Emulite 150 or Dynamex M, 25×200 mm cartridges taped to a detonating cord downline.

The best results are obtained in homogeneous rock formations with a minimum of joints, planes and dirt seams. In fractured rock formations, smooth blasting and presplitting will give a better result.

Advantages:
Applicable where even light charges may cause damage beyond the excavation line.

Disadvantages:
Unpredictable result except in very homogeneous rocks.
High drilling costs due to close spacing.
Time consuming due to the extensive drilling.
The slightest deviation in drilling causes poor result.

## 8.3 Cushion blasting

Application: Mainly surface excavation.

Cushion blasting was first introduced in Canada.
A single row of holes is drilled along the perimeter of the excavation. The size of the drillholes varies between 50 mm (2") and 164 mm (6 1/2").

Cushion blastholes are charged with small, well distributed charges in completely stemmed holes, which are fired after the main blast is excavated. The stemming "cushions" the shock from the explosive to the rock to remain, minimizing cracking and tension. The charges in cushion blasting should be fired with no delay, or minimum delay, between the holes. Detonating cord is the best means of initiation where noise or airblast do not cause any problem. The united force of the charges cuts the web between the holes forming a smoth rock surface.

The burden and spacing will vary with the hole diameter in the perimeter drilling.
The holes are charged with explosives cartridges taped to a detonating cord downline. Cap sensitive explosives cartridges with a diameter of 25 to 32 mm and a length of 200 mm are taped to the downline with a relative distance of 30 to 50 cm depending on the hole diameter.
To avoid stumps in the bottom part of the excavation and to promote shearing between the holes, the charge concentration must be increased in the bottom of the hole.

Precision in drilling is important and deviation of more than 15 cm from the theoretical plane tends to give poor result.

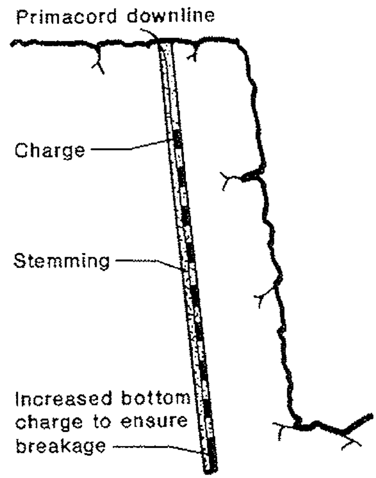



*Fig. 8.3 Cushion blasting.*

When cushion blasting is used in 90° corners it may be combined with presplitting for better result.

Proposed charging and drilling pattern for cushion blasting:

| Perimeter hole diam. mm | Charge concentration Emulite or Dynamex kg/m | Recommended charge type | Burden m | Spacing m |
|---|---|---|---|---|
| 50–64 | 0.12–0.35 | Em* 150, DxM*, 25 mm | 1.20 | 0.90 |
| 75–89 | 0.20–0.70 | Em 150, DxM, 32 mm | 1.50 | 1.20 |
| 102–114 | 0.35–1.10 | Em 150, DxM, 32 mm denser charged | 1.80 | 1.50 |
| 127–140 | 1.10–1.50 | Em 150, DxM, 55 mm | 2.10 | 1.80 |
| 152–165 | 1.50–2.20 | Em 150, DxM, 65 mm | 2.70 | 2.10 |

* Em 150 = Emulite 150 * DxM = Dynamex M

Advantages:
Increased spacing between the holes, less drilling.
Functions reasonably well in incompetent rock formations.

Disadvantages:
It is necessary to excavate the main blast before firing the cushion blast.
Difficult to cut 90° corners without combining with another method i.e. presplitting.

## 8.4 Smooth blasting.

Application: Surface and underground excavation.

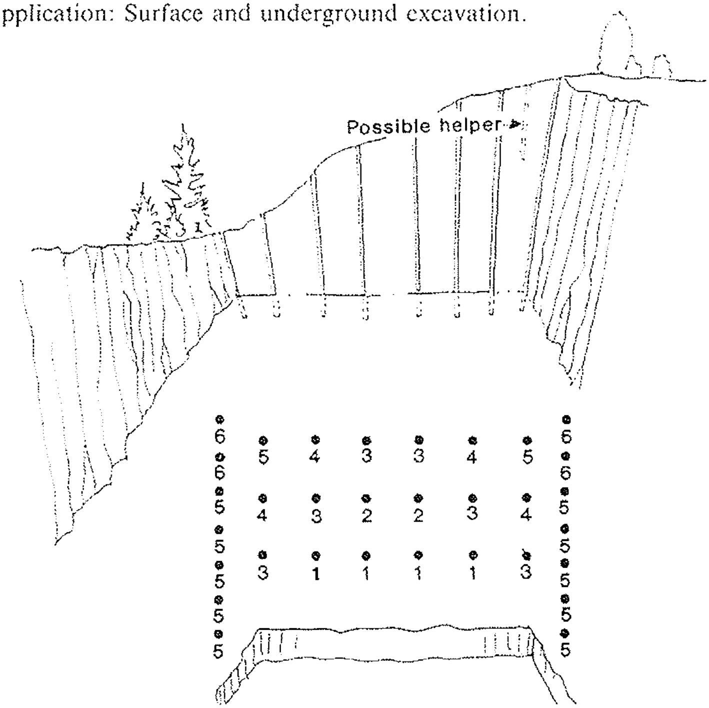



*Fig. 8.4 Principle of smooth blasting.*

Smooth blasting was developed and refined in Sweden during the 1950s and 60s. The principle is mainly the same as for cushion blasting, but the smooth blasting holes may be fired together with the rest of the round. There is no need for excavation of the main blast beforehand.

New explosives were developed for smooth blasting, small diameter light explosives with low VOD and relatively low gas content were tried, with good results. The trials led to the development of Gurit, a nitroglycerin based explosive containing kieselguhr (the original material that Alfred Nobel used to tame the nitroglycerin) and other components to obtain a suitable explosive.

Gurit is available in 11, 17 and 22 mm diameter cartridges to suit all possible applications.

Smooth blasting is carried out in connection with the rest of the round and the smooth blasting holes are fired with higher period numbers than the rest of the round.

Not only the perimeter holes should have light charges. The holes directly adjacent to the perimeter holes must be charged with well balanced charges, as excessive charge may break beyond the intended rock excavation line before the perimeter holes are blasted, spoiling the final contour.

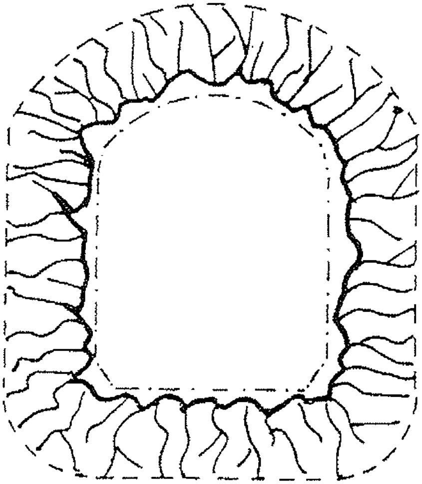


*Fig. 8.5 Crack zone from blasting with conventional explosives.*

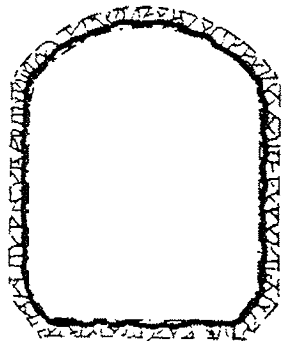


*Fig. 8.6 Crack zone from smooth blasting with Gurit 17×500 mm.*

The perimeter holes must be carefully charged with joined-up charges, which are locked in the hole by stemming. To prevent the sand from running down the hole, a paper plug can be placed on top of the last cartridge.

The quality of the remaining rock depends to a large extent on the relation between the spacing of the holes (S) and the burden (B). For a good result, the ratio S/B should be ≤0.8. The burden should be greater than the spacing.

It is in underground excavation that smooth blasting has become an undisputed success. The increased demand for stable rock surfaces in underground chambers has resulted in the smooth blasting method being prescribed as the standard method for controlled contour blasting in permanent underground installations.

Not only is a smooth surface called for but also less fissures in the remaining rock, decreasing the need for subsequent reinforcing works.

In underground blasting it is even more important than in surface blasting to charge the holes adjacent to the perimeter holes carefully. The holes in a tunnel blast are normally more closely spaced than in surface blasting and are also more constricted. If the stoping holes in the blast are heavily charged, the crack formation from these holes may extend beyond the final contour.

SVEDEFO* has worked out an empirical formula to predict the vibration velocity which can be expected from different linear charge densities at different distances.

$v = 700 \times Q^{0.7}/R^{1.5}$ (mm/sec.)

where $v$ = vibration velocity (mm/sec.)

$Q$ = charge (kg)

$R$ = distance (m)

* Swedish Detonic Research Foundation.

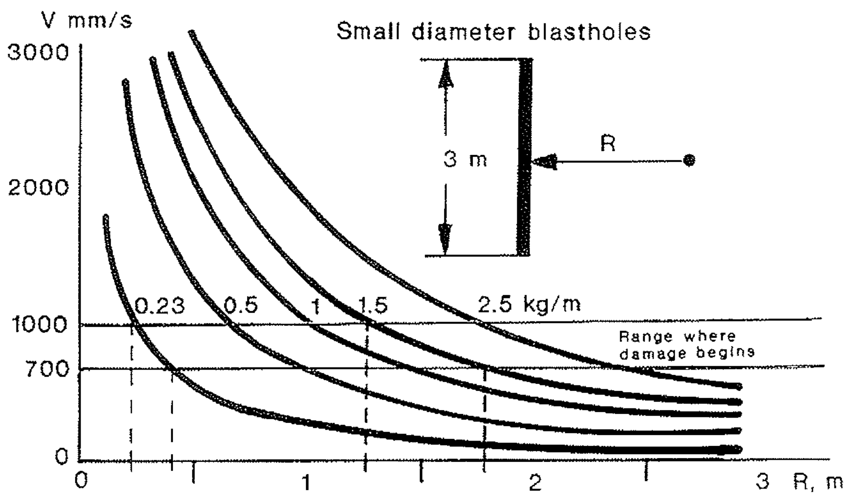



*Fig. 8.7 Vibration velocity (v) as a function of the distance (R) with different charge concentrations.*

The graph shows that a 45 mm hole fully charged with ANFO, charge concentration 1.5 kg/m, forms a crack zone extending 1.2 to 1.8 m, while 17 mm Gurit (0.23 kg/m) has a crack extension of 0.2 to 0.4 m.

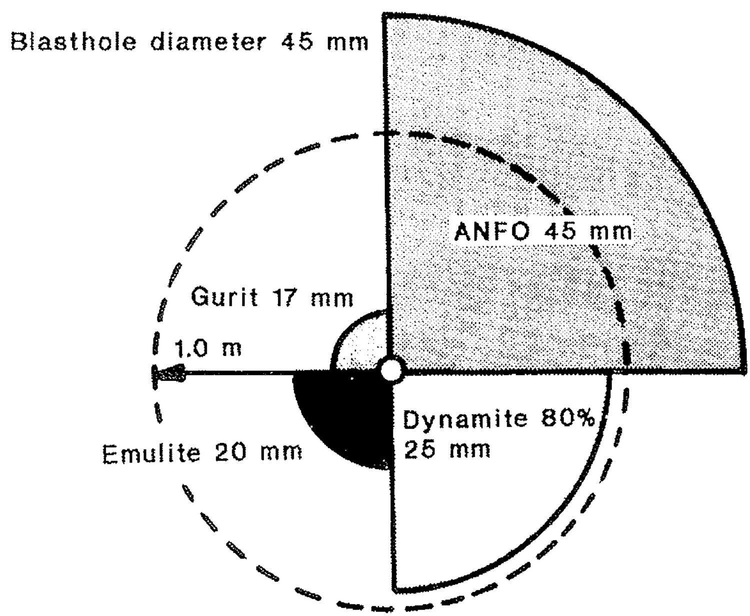



*Fig. 8.8 Crack extension with different explosives.*

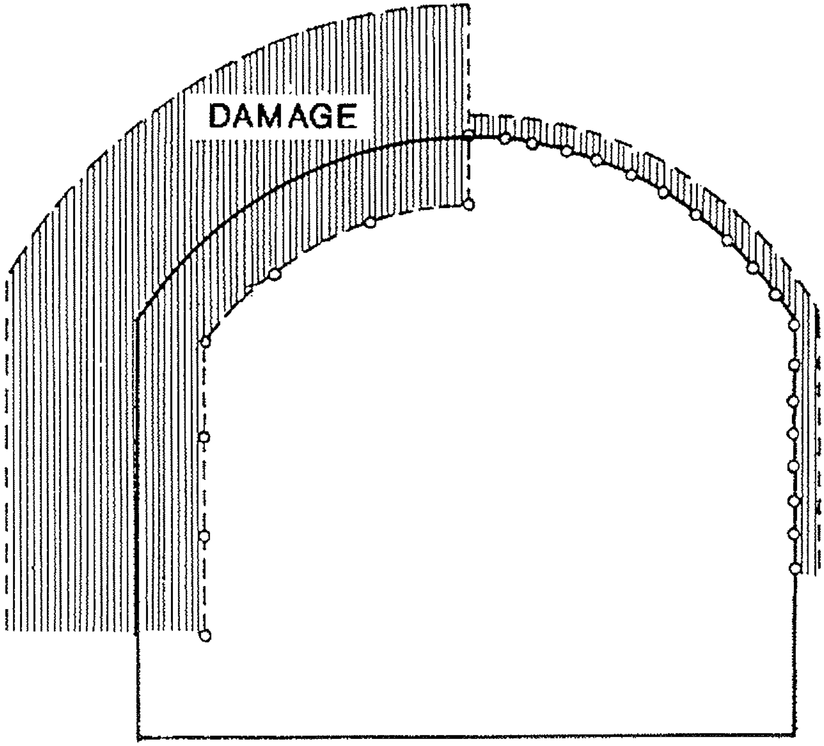



*Fig. 8.9 The holes adjacent to the perimeter holes may cause more damage to the remaining rock than the perimeter holes.*

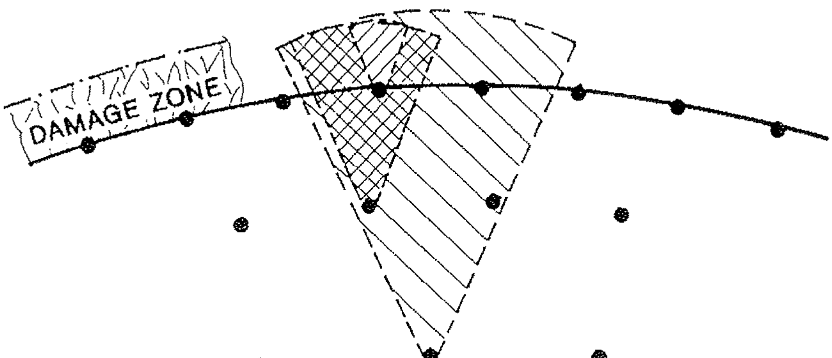



*Fig. 8.10 The adjacent holes must also be carefully blasted.*

Well-balanced charges in the holes next to the perimeter are a must for the best result.
As mentioned before, smooth blasting is carried out with special explosives in closely spaced blastholes. The following table gives the recommended charge and drilling patterns for different diameters of the perimeter holes.

| Perimeter hole diam. mm | Charge concentration kg/m | Charge type | Burden m | Spacing m |
|---|---|---|---|---|
| 25–32 | 0.11 | 11 mm Gurit | 0.3–0.5 | 0.25–0.35 |
| 25–48 | 0.23 | 17 mm Gurit | 0.7÷0.9 | 0.50–0.70 |
| 51–64 | 0.42 | 22 mm Gurit | 1.0–1.1 | 0.80–0.90 |
| 51–64 | 0.45 | 22 mm Emulite | 1.1–1.2 | 0.80–0.90 |

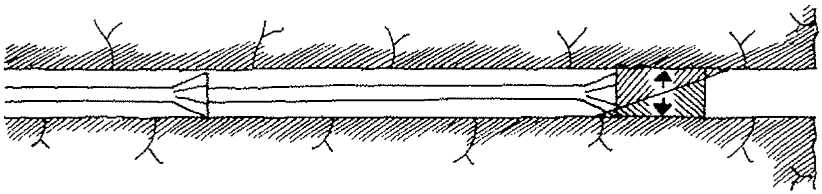


*Fig. 8.11 The gas pressure locks the plug more tightly to the walls of the blasthole.*

The charges should be connected together, string charged, and the hole plugged, otherwise the charges will be sucked out of the hole by the explosions from the previous blasted holes in the round.

A special blasthole plug has been developed for this purpose. The plug locks the charge in the hole efficiently.

The firing of the perimeter holes should be made with the same period number for the best result.

Advantages:
Increased spacing reduces drilling cost.
Better result in incompetent rock formations.
No excavation needed before the smooth blasting is executed.
Special charges give light and well distributed charging of the perimeter holes.

Disadvantages:
No real disadvantages compared with previously presented contour blasting methods.

## 8.5 Presplitting.

Applications: Mostly surface blasting.

The idea of presplitting is to isolate the blasting area from the remaining rock formation by forming an artificial crack along the theoretical excavation plane.

This is done by drilling a single row of relatively closely spaced holes along the perimeter of the blast. The hole diameter is usually 30 to 64 mm and in most cases all holes are charged.

Presplitting differs from smooth and cushion blasting in that way that the presplitting holes are fired before the main blast. The presplit may be fired together with the main blast but with the lowest interval number.

Sometimes the presplitting of the perimeter is done before the drilling of the main blast.

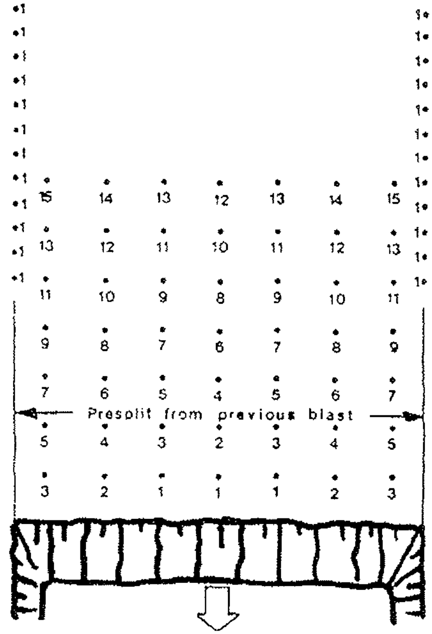



*Fig. 8.12 Presplitting.*

The theory of presplitting is that when shock waves from simultaneously detonating charges in adjoining blastholes collide, tension occurs in the rock, forming a crack in the web between the holes. For that reason it is important that the charges are detonated simultaneously or as close to that as possible. For the best result, detonating cord or instantaneous detonators should be used for initiation. If noise or ground vibrations make it necessary to use other means of initiation a reduction of the distance between the holes is necessary.

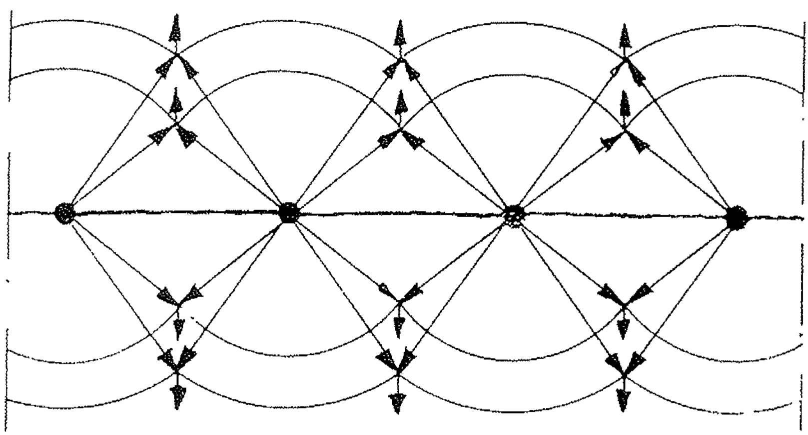


*Fig. 8.13 The theory of presplitting.*

Presplitting creates an artificial plane along the limits of the excavation against which the subsequent main blast may break, resulting in a smooth wall with little or no overbreak.

Some of the shock waves from the subsequent main blast are reflected against the presplit plan, preventing them from being transmitted into the remaining rock formation. This tends to reduce ground vibrations.

Precision in drilling is of the utmost importance for the final result, even small deviations may adversely effect the presplit.

As mentioned before, the best result is obtained by firing the holes simultaneously using detonating cord. If long presplit lines are fired, the lines may be parted and delayed with relays.

In incompetent rocks the result may be improved by drilling guide holes between the charged holes to promote the cut along the intended plane. Unloaded guide holes between the charged holes give better final result in all rock formations, but are rarely used due to the increased drilling costs.

The presplit holes are normally more closely spaced than they are in smooth blasting. The holes are charged with light special charges, Gurit, and usually initiated with detonating cord. A detonating cord downline will also secure initiation of the charges, especially in incompetent rock.

Proposed charging and spacing for presplitting.

| Perimeter hole diam. mm | Charge concentration | Recommended charge type | Hole spacing m |
|---|---|---|---|
| 25–32 | 0.11 | 11 mm Gurit | 0.2–0.3 |
| 25–41 | 0.23 | 17 mm Gurit | 0.4–0.6 |
| 41–51 | 0.46 | 2×17 mm Gurit | 0.5–0.7 |
| 41–51 | 0.42 | 22 mm Gurit | 0.5–0.7 |
| 51–64 | 0.45 | 22 mm EMULITE* | 0.6–0.8 |

* Paper cartridges taped to a detonating cord downline.

The bottom is charged with a higher charge concentration to promote shear in the bottom part. Emulite 150 or Dynamex M are suitable for this purpose.
If the depth of the hole is less than 1.5 m it may be necessary to reduce the distance between the holes.

Recommended bottom charge for different hole depths.

| | | | | |
|---|---|---|---|---|
| Hole depth (m) | 2.0 | 2.0–4.0 | 4.0–6.0 | 6.0–10.0 |
| Bottom charge (kg) | 0.05 | 0.10 | 0.20 | 0.30 |

The holes should be charged up to 3/4 of the hole depth. In rock with horizontal planes it may be necessary to charge higher in the hole for better shear of the rock in the upper part of the round.

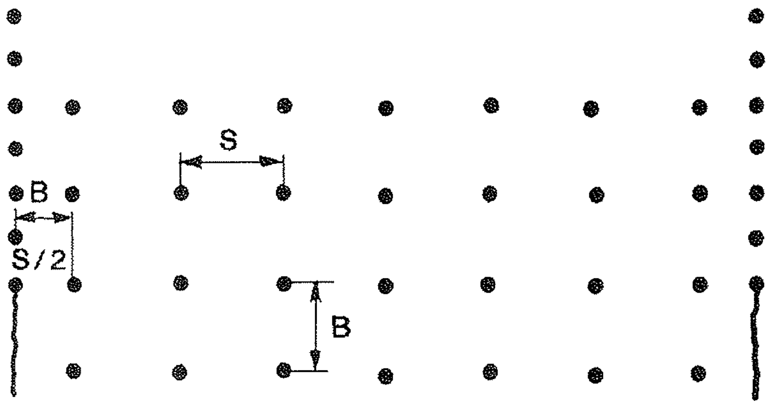



*Fig. 8.14 Drilling pattern for presplitting.*

The holes should not be stemmed. When long lines are presplit and the lines are parted with delays it may be necessary to prevent the charges from being blown or sucked out of the holes by using blasthole locks.

The distance from the presplit plane to the adjacent holes in the round should be half the spacing in the round. B=S/2.

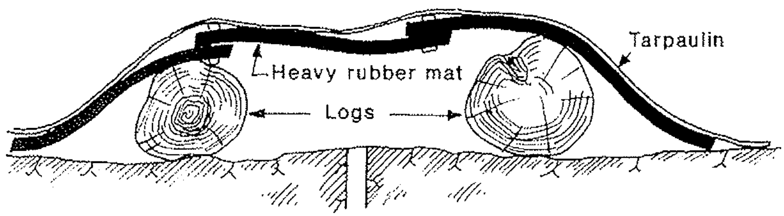



*Fig. 8.15 Covering of presplitting blast.*

The risk of flyrock from presplitting blasts is great. Good covering is a necessity. The covering material must not be placed too close to the rock surface. Space for the gas expansion has to be provided for. See fig. above.

Presplitting of trenches is often done to reduce overbreak outside the theoretical plane. Presplitting of the two parallel lines cannot be done simultaneously if the distance between the lines is less than 4.0 m. The shock waves from the two presplit blasts disturb each other and no real tension occurs in the web between the holes in the line. If parallel presplit lines are fired in the same blast, one line should be delayed at least 50 ms.

Ground vibration, airblast and noise are the three problems which restrict the use of presplitting in contour blasting.

Ground vibrations are normally greater in presplitting operations than in other methods of contour blasting because presplit holes have no free breakage, making constriction of the hole complete.

The airblast and noise problems are greater than in other methods of contour blasting. As the presplitting holes are not stemmed and the normal initiation method is detonating cord, care has to be taken when working close to populated areas. Even if instantaneous electric initiation is used, the airblast problem has to be considered.

Advantages:
Gives excellent results in homogeneous rocks and better results than other methods in incompetent rock, especially if guide holes are drilled.

Disadvantages:
More drilling than in smooth blasting.
Noise and ground vibration problems.

## 8.6 Combined methods.

As mentioned previously, it may be necessary to combine different methods for contour blasting. In unconsolidated rock a combination of line drilling and smooth blasting or presplitting gives a better result than if only the later methods are used. The line drill between the smooth blasting or presplitting holes acts as a guide for the shear between the holes.

When smooth blasting or presplitting in curved lines or corners, the use of guide holes gives better result. It must also be considered that closer spacing is required than in straight line blasting. 90° corners may be presplit or line drilled. The presplit is preferable if blastings are not restricted by ground vibration, noise etc.

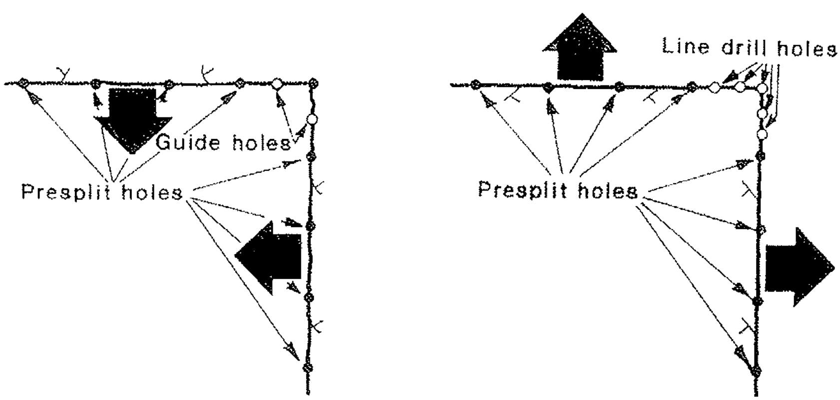



*Fig. 8.16 Combinations of contour blasting methods on 90° corners.*

## 8.7 Economic aspects of contour blasting.

The economic advantage of contour blasting appears mainly in tunnel blasting. In solid rock the overbreak has to be transported out of the tunnel and an overbreak outside the theoretical payline of 1 cu.m. per meter of the tunnel gives extra handling of 1.000 cu.m. per kilometer.

The situation is more serious in incompetent rocks where concrete lining is needed.

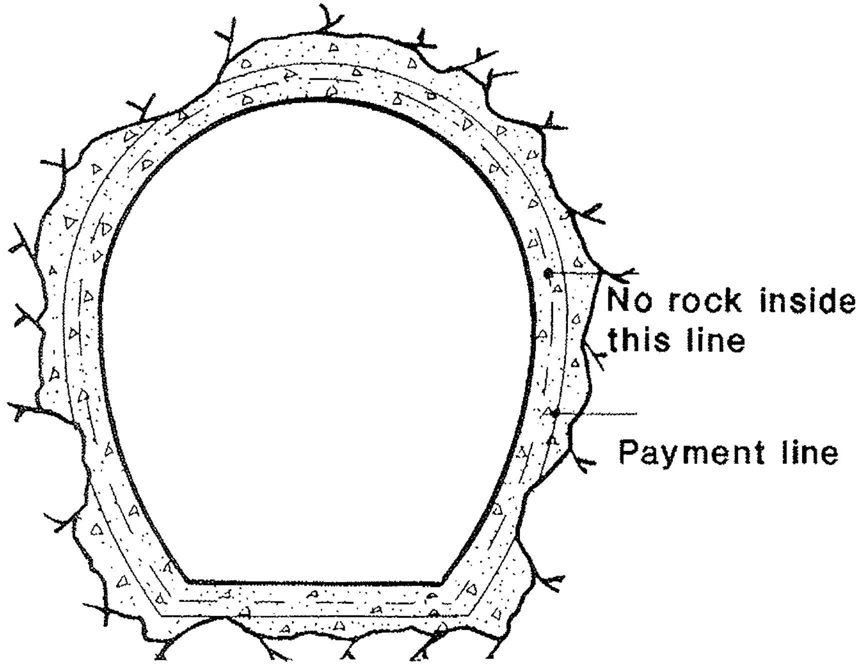



*Fig. 8.17 Water tunnel.*

Example:

A 5,000 m long water tunnel is going to be blasted. Its cross section is 55 sq.m. and the circumference of the roof and walls is 20 m.

The extra cost for contour blasting is 19 USD/sq.m. and the cost of concrete is 130 USD/cu.m. (Prices 1984.)

If conventional blasting is applied, without doubt an overbreak of 30 cm more than if smooth blasting techniques are used must be expected. (See fig. 8.5, 8.6, 8.7 and 8.8.)

The overbreak will require refill with concrete, in this case 6 cu.m. extra per meter of the tunnel. 5,000 m will then require 30,000 extra cubic meters of concrete at a cost of 130 USD/cu.m. Totally 3,900,000 USD.

If smooth blasting techniques had been applied, the cost would have been $20 \times 5{,}000 \times 19$USD$=1{,}900{,}000$ USD.

In this example no costs for extra scaling, bolting and shotcreting during the operation have been taken into account, but may reach substantial sums.

# 9. CHARGING THE BLASTHOLE

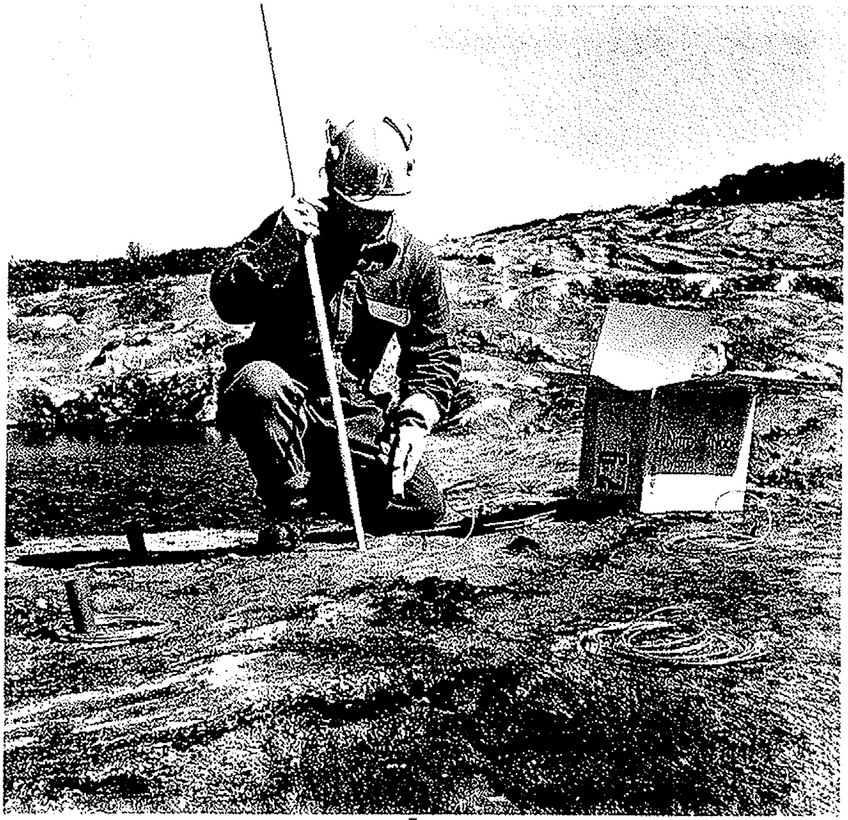


*Fig. 9.1 Charging.*

## 9.1 General.

The charging methods are different for different blasthole diameters. For that reason, the blastholes have been classified according to the diameter as follows:

| | | |
|---|---|---|
| small size | <50 mm | (2″) |
| medium size | 50–100 mm | (2″–4″) |
| large size | >100 mm | (4″) |

Small diameter blastholes often have a limited depth. They are mainly used in smaller bench blasting operations, trench blasting, tunnel blasting and mining. The inclination can range from vertically down to vertically up. The holes are normally charged with high explosives and a tamping rod is used to introduce and compact the explosive.

The medium sized blastholes are used in construction and production blasting. The inclination is usually vertical or close to vertical downwards (an inclination of 3:1 is recommended and gives a good breakage). The holes are normally primed with a high explosive and the main charge in the hole may either be a high explosive or a blasting agent. The holes may be charged with a tamping rod (if they are not too deep). In deep holes, the charges may be compacted with a loading weight. Liquid or dry free flowing blasting agents may be poured into the holes.

Large size blastholes are used in large scale operations like quarrying and mining. The inclination is usually vertical and the explosives used are normally blasting agents which are primed with a high explosive. The blasting agent is poured or pumped into the blasthole.

Before charging is started the blasthole should be checked for obstructions, hole depth and water/soil content. In shallow holes a tamping rod is suitable, while in deep holes a weighted measuring tape should be used. If the hole is too deep it should be filled up to the intended level with drill cuttings or similar. (Holes which are too deep in a blast are more constricted and cause more ground vibration.) Holes that are too short should be flushed with compressed air to be cleared for soil. If they still are too short they should be deepened by drilling or a new hole drilled. Sometimes too short holes have to be blasted because of equipment shortage or other reasons. In these cases sceondary blasting of the toe has to be counted on.

Obstructions in small holes may be removed with a tamping rod. In large downward holes, a heavy weight tied to a rope and dropped repeatedly on the obstruction may clear the hole. If the obstruction cannot be cleared with a tamping rod or a weight it may be necessary to redrill the hole, either in the blocked hole or a new hole close to the obstructed one.

**No new hole should be drilled in the round where there is a risk of drilling into a charged blasthole.**

The blastholes should be cleared from water before the charging of the holes begins. This is important especially when dry blasting agents like ANFO are used. ANFO deteriorates fast when it comes in contact with water. As the water is in the bottom of the hole, the most constricted part of the blast has to be broken by a water contaminated explosive, resulting in bad toe breakage.
Where water is present, a water resistant explosive should be used in the bottom of the hole and up above the water level.

## 9.2 Priming.

The terms "**primer**" and "**booster**" are often confused. MSHA (The Mine Safety and Health Administration of U.S.A.) defines the **primer** as a unit of cap-sensitive explosive used to initiate other explosives or blasting agents. A **primer** contains a detonator or other initiating device i.e. detonating cord. A **booster** is usually cap-sensitive but does not contain a detonator. A **booster** is used to maintain or intensify the explosive reaction.

The primer cartridge should be assembled at the work-site. The transport of capped primers adds further risk to the blasting operation and is against the regulations in most countries.

The primer cartridge must be made of a cap-sensitive explosive with high VOD and high detonation pressure.
Explosives like Emulite and Dynamex have the properties needed to be used as primers.

The detonator may be introduced into soft plastic explosives by gently pushing the detonator into the explosives cartridge.
In harder explosives, a wooden stick should be used to make a hole in the cartridge before the detonator is introduced.

Factory made primers are delivered with cap wells suitable for most detonators.

The detonator should be introduced into the primer cartridge from the end and not from the side. Introduction from the side may cause misfire if there is insufficient explosive around the end of the detonator.

The primer cartridge should be placed in the bottom of the blasthole with the base of the detonator facing the explosives column. Bottom priming gives the best confinement at the initiation point and makes sure that no explosive is left undetonated in the bottom of the hole if it should be blocked during the charging operation. **The primer cartridge must not be tamped nor dropped into the blasthole.**
When priming blasting agents, the primer should have a diameter which is close to the diameter of the blasthole.
This is especially important when priming ANFO.

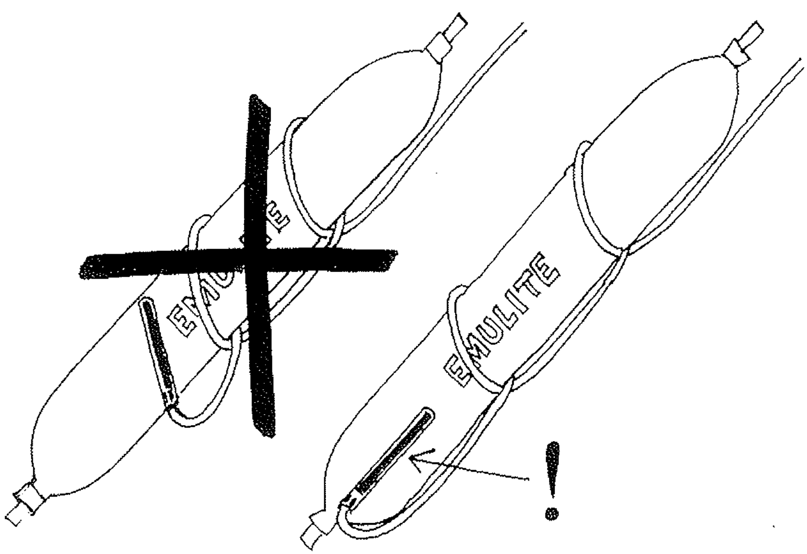



*Fig. 9.2 Correctly assembled primer cartridge to the right.*

### 9.2.1 Priming of ANFO.

When ANFO is efficiently primed it rapidly reaches its steady state velocity of detonation and maintains it.

The steady state velocity depends on the density, the confinement and particle size of the ANFO as well as the blasthole diameter. If none of the above conditions changes, the ANFO will detonate at one, and only one velocity. If one of the conditions is changed the steady state velocity changes.

**STEADY STATE VELOCITY FOR DIFFERENT BLASTHOLE DIAMETERS.**

| Blasthole diameter mm | VOD m/sec |
|---|---|
| 89 | 3.700 |
| 102 | 3.800 |
| 152 | 4.200 |
| 270 | 4.400 |

The VOD increases as the blasthole diameter increases and reaches its highest value at a blasthole diameter of 300 mm.

As mentioned above, the purpose of the primer is to initiate the ANFO so that it rapidly reaches its steady state velocity. The primer may initiate the ANFO with low order velocity (VOD lower than steady state VOD) or overdrive velocity (VOD higher than steady state VOD).

Low order initiation is caused by too small primer or too low detonation pressure.

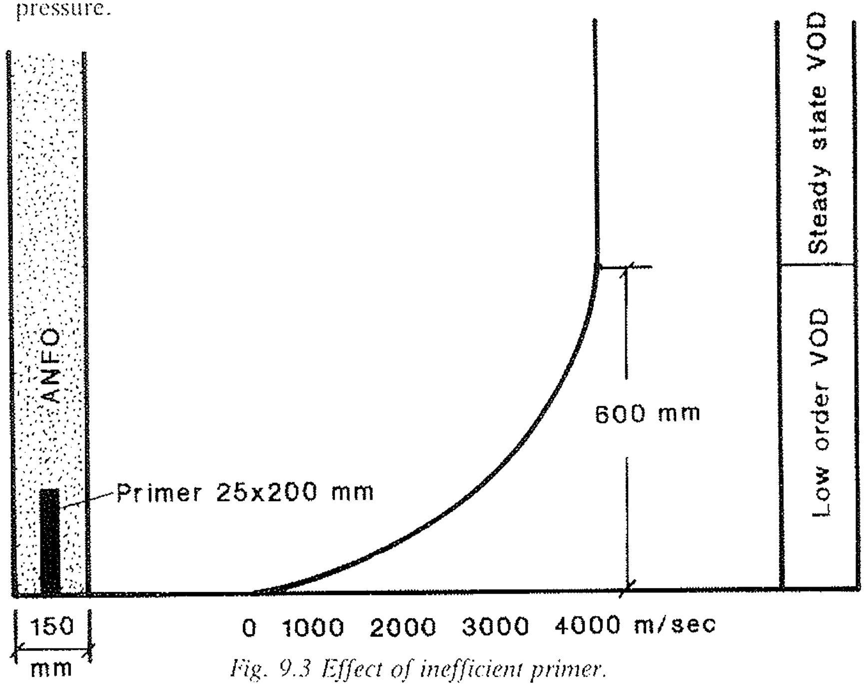



*Fig. 9.3 Effect of inefficient primer.*

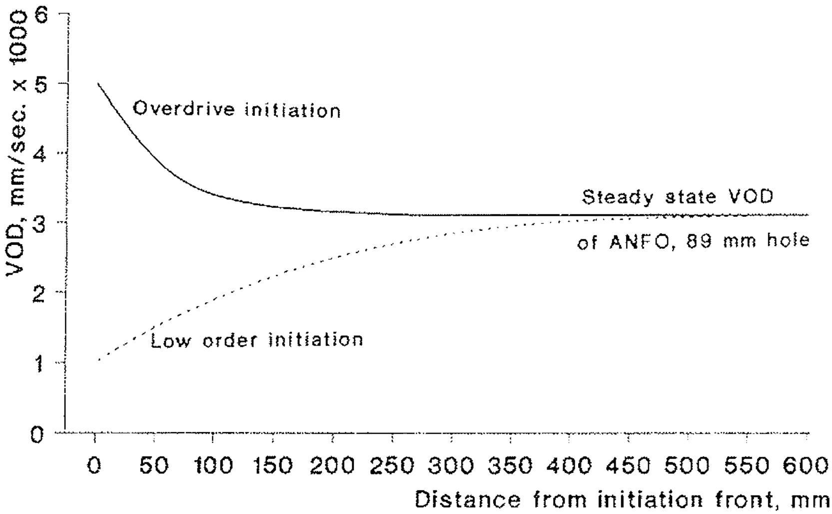



*Fig. 9.4 Effect of primer.*

The velocity distance curve shows that it takes approximately the length of four blasthole diameters to build up the VOD to steady state. The low energy initiation in the bottom of the blasthole may have a serious effect on the blasting result.

Small inefficient primers initiate ANFO with low order velocity while powerful primers with a diameter close to that of the blasthole initiate with overdrive velocity.

The two most important properties of a primer are:

* its detonation pressure
* its diameter

**The effect of the primer detonation pressure.**

The detonation pressure is the pressure generated by the explosive in its detonation wave. The pressure is a function of the velocity and the density of the explosive.

Detonation pressure for different explosives:

| Explosive | Detonation pressure kbars |
|---|---|
| Composition B (military) | 225 |
| Dynamex M | 110 |
| Emulite 150 | 95 |
| 40 % dynamite | 40 |
| ANFO | 20 |

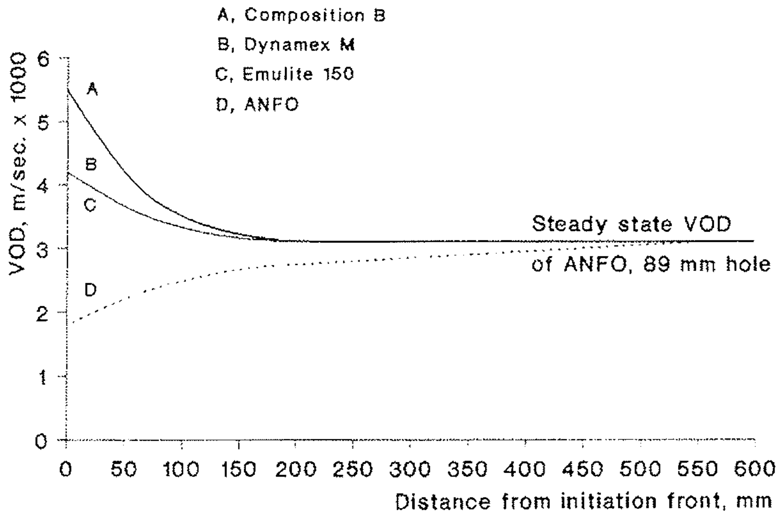



*Fig. 9.5 Effect of detonation pressure on initial VOD of ANFO.*

Composition B gives the highest detonation pressure initiating the ANFO with the greatest degree of overdrive. Dynamex M and Emulite also give a good degree of overdrive, thus initiating the ANFO efficiently.

Primers with less than 50 kbars detonation pressure do not initiate ANFO efficiently and should not be used.

**The effect of the primer diameter.**

The second important property of a primer affecting the initial velocity of ANFO is the diameter of the primer.

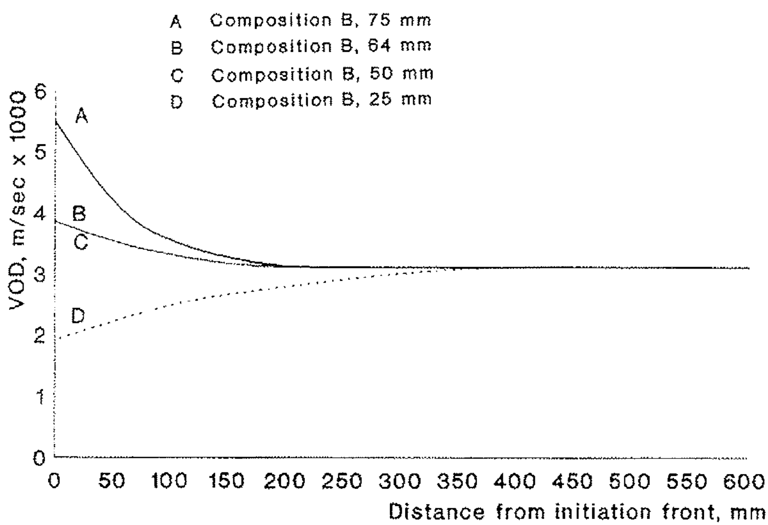



*Fig. 9.6 Effect of primer diameter on initial VOD of ANFO.*

As may be seen from the graph the initial velocity of ANFO is reduced when the diameter of the primer is reduced. The drastic reduction of initial velocity when 25 mm primer cartridges are used is worth noting.

Atlas Powder Company of U.S.A. has explained the effect of the primer diameter as follows in their brochure, "The Basic Principles of priming ANFO": "Initiation of ANFO is effected by transferring the detonation pressure from the primer onto the surface of the ANFO. In the case of matching diameters between the primer and ANFO, the pressure wave is transferred uniformly over the entire surface of the ANFO. If detonation pressure is high enough, the result will be efficient initiation of the ANFO, perhaps with overdrive. However, if the primer is small in relation to the ANFO column, the area of pressure transfer is greatly reduced. Consequently, only an equivalent area of ANFO (equal to the diameter of primer) will receive the pressure wave. For example, when the 75 mm column of ANFO was initiated with a 25 mm diameter primer, the initial velocity of the ANFO will be equal to a 25 mm diameter ANFO column. As the detonation progresses through the column, it will gradually expand to the 75 mm diameter of ANFO and will then finally achieve steady state velocity.

The adverse combination of low detonation pressure and small diameter in a primer has given initial velocities as low as 600 m/sec.

The length of the primer should for geometric reasons, always be equal to or longer than its diameter. Preferably at least two diameters in length to assure that a stable flat pressure wave is formed in the primer."

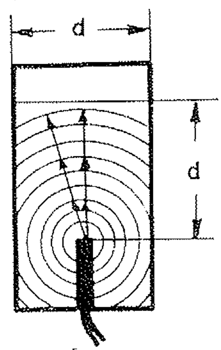



*Fig. 9.7 Detonation wave from a detonator.*

The use of standard (10 gr/m) detonating cord as downline in blastholes charged with ANFO is not recommended. The cord initiates the ANFO column along its entire length at very low order, with substantially reduced energy output from the blasting agent. Lately Ensign Bickford of U.S.A. has introduced a downline cord with 3 gr/m core load which neither initiates nor dead presses the ANFO. It may be used in blastholes with 64 mm (2 1/2") diameter and above.

To summarize the Chapter "Priming of ANFO":

Efficient initiation of ANFO requires a primer of sufficient diameter which will nearly fill the blasthole and have a minimum length of at least one blasthole diameter.

The detonation pressure should not be less than 80 kilobars.

Reinforced priming of ANFO as mentioned in Chapter 5.2 Charge calculations, is an economic alternative, especially if the blastholes contain water. The constricted bottom part of the hole is charged with a water resistant explosive of high and stable velocity and high detonation pressure. The burden and spacing can be widened considerably.

## 9.3 Different charging methods.

### Charging with tamping rod.

Tamping rod is used to tamp explosives cartridges in holes of small to medium diameters. The tamping rod should be made of wood or plastic. **IRON RODS ARE ABSOLUTELY PROHIBITED.** Any metallic fitting or pike should be of copper or brass. The diameter of the rod should be approx. 10 mm smaller than that of the blasthole thus giving space for legwires, Nonel tube, safety fuse or detonating cord. To obtain a good standard of packing during the charging, only one cartridge should be tamped at a time. **DO NOT TAMP THE PRIMER CARTRIDGE.** For the best results, cartridges with a diameter close to that of the blasthole should be used. If more than one cartridge is introduced in the blasthole between tamping operations, it will result in inadequate packing of the explosive.

### Charging with tamping weight.

A tamping weight is used in medium and large size blastholes which are charged with cartridged explosives, normally in plastic hoses.
The tamping weight could consist of wooden casing, normally oak, with cast-in lead or a copper casing containing lead. The wooden weight is an oak cylinder which is soaked in water for a couple of days before the lead is cast in. A hook or a loop should be included for a rope. The copper weight is made the same way as the wooden weight.
The tamping weight should be used between the introduction of each cartridge, especially in the bottom part of the hole. A tamping weight with a measuring tape is also useful for the preblast investigation of the blasthole.

### Charging with pneumatic machines.

Principally two types of pneumatic charging machines are available:

* semi-automatic charging machines for cartridged explosives
* pressure-ejector vessels for ANFO.

Semi-automatic charging machines are useful for upward holes, underwater blasting and in fissured rocks where cartridges tend to jam but where a semi-riged plastic hose could be introduced to the bottom of the hole.

### Charging of cartridged explosives.

The principle of the charging machine is that the explosive is transported through a plastic hose by pneumatic pressure. The pressure is reduced by a reduction valve to 3 kg/sq.cm. to eliminate the risk of accidental initiation. Only high safety explosives like Emulite or Dynamex should be used in charging machines for cartridged explosives.

The semi-automatic charger permits continuous insertion of explosive cartridges at the same rate as they are charged in the hole by the hose. The cartridges pass through an airlock between two valves. The air pressure, 3 kg/sq.cm., is always maintained in the hose while the cartridges are being inserted. The semi-

*Fig. 9.8 Semi-automatic charger with ROBOT tamper.*

automatic charger permits a considerably higher charging capacity than when charging with a tamping rod.
Charging can be done with 25, 32 or 40 mm explosives cartridges.

The charging hose is made from an anti-static plastic material, with a nozzle fitted at the end. The nozzle contains knives which cut the paper/plastic around the cartridge when it is forced through the nozzle, to make compaction of the explosive easier.
During the charging, the charging hose is moved backwards and forwards in the blasthole with short movements to ensure good tamping of the explosive.
When charging under water, the charging work is more difficult because of the counter pressure exerted by the water. If charging is carried out at considerable depths, the reduction valve may be immersed in the water to compensate for the counter pressure. The compensation will be 1 kg/sq.cm. per 10 m water depth. It is recommended that the pressure at the nozzle should be at least 1.5 kg/sq.cm., meaning that the valve should be lowered 15 m below the surface when charging at a waterdepth of 30 m.

**WARNING!**
**It is under NO circumstances permitted to change the pressure level in the reduction valve. (3 kg/sq.cm.)**

In underground operations with upward holes, the semi-automatic charger may be combined with a ROBOT tamper, which carries out the tamping work with the charging hose.
The principle of the ROBOT is that the charging hose is firmly held by a pneumatic gripper which moves the hose backwards and forwards by a double acting cylinder. If the air pressure in the gripper is varied, the friction between the hose and the gripper varies, making it possible to change the tamping degree. Charging and tamping with the semi-automatic charger and the ROBOT tamper may be carried out by one man making it an economic and technically suitable method for charging upward blastholes.

**Charging machines for ANFO.**

Free flowing ANFO is normally poured into blastholes which are vertical or close to vertical downwards.
For horizontal and upward blastholes, the principal method of charging is by means of pneumatic charging devices. Such devices are also used for the charging of downward blastholes where a higher charging density is required.
The principle of the charging machines is that the ANFO is transported from a container, through a plastic hose, into the blasthole by pneumatic pressure.

Two main types of pneumatic charging machines for the charging of ANFO are available:

* pressure vessel machines which work with high pressure in the container. The ANFO is pressed through the hose into the blasthole.
* ejector units where the ANFO is sucked from the container and blown through the hose into the blasthole.

Combined pressure/ejector machines are also available.

**ANOL** is a pressure vessel device for the charging of ANFO in all kinds of applications. Prilled ANFO can be charged in upward blastholes with an inclination of up to 35° without running out.

The flow of ANFO is remotely controlled by the charger.

As ANFO is highly corrosive, all machine parts that come in contact with ANFO are made of stainless

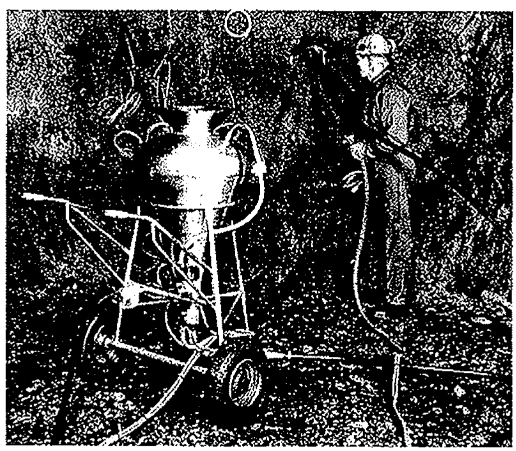


*Fig. 9.9 ANOL charging machine.*

steel.
ANOL is manufactured in sizes of 100, 150, 300 and 500 liters.

The **JET-ANOL** charging machine is a combined pressure/ejector unit for the charging of prilled ANFO in upward blastholes with diameters between 32 and 51 mm and a depth of up to 45 m.
The ANFO is transported by the ejector with such a high velocity into the blasthole that the prills are crushed and stay in the blasthole. The flow of ANFO as well as the velocity of the ANFO through the hose are remotely controlled by the charger.


*Fig. 9.10 JET-ANOL charging machine.*

As for the ANOL, all parts in contact with ANFO are made of stainless steel. JET-ANOL is manufactured in sizes of 100, 150, 300, 550 and 750 liters.
The charging hose has to be anti-static as the ANFO is transported through the hose with high velocity causing a certain risk of accumulation of static electricity. Because of the risk of static electricity, all ANFO charging units have to be earthed during charging operations.

ANOL and JET-ANOL are registered trademarks of Nitro Nobel AB Sweden.

**Charging with pump trucks.**

In large scale blasting operations with large diameter blastholes (over 100 mm), the explosive or blasting agent may be charged into the hole by a pump truck. The explosive or blasting agent (Emulite or EMULAN) can be manufactured in a plant on the site and pumped directly from the plant into the pump truck. The explosive is then transported to the blasting site in the truck and pumped into the blasthole with a capacity of approx. 200 kg/minute.
Care must be taken when charging holes containing water. The charging hose must be introduced below the water level to the bottom and lifted with the same pace as the hole is filled to avoid separation in the explosives column by waterpockets.

*Fig. 9.11 Pumping Emulite with a pump truck.*

# 10. CAUTIOUS BLASTING

*Fig. 10.1 Cautious blasting.*

## 10.1 General.

The development of blasting techniques has made it possible to carry out advanced blasting operations close to and under existing structures.
In the last decades, blasting activities in populated areas have increased due to the need for better communications like metros, tunnels for telephone cables as well as tunnels for water supply, sewerage, electric cables etc.
Another area of cautious blasting is the expansion of existing hydroelectric and nuclear power plants, where it is of the utmost importance that power production is not disturbed during the construction period.

The increased prices of land in urban areas have also made it feasible to utilize the "below street level" space for various purposes such as garages, offices, bomb shelters etc.
In all these blasting operations, ground vibrations and to a certain extent air shock waves and flyrock constitute a threat to property and life and it is therefore necessary to control these hazards to avoid damage.
It is primarily the ground vibrations which affect neighboring structures but special attention has to be given to the possible occurrence of flyrock, which is the main cause of on-site fatalities and damage to equipment.

## 10.2 Ground vibrations.

### 10.2.1 The theory of ground vibrations.

Ground vibrations are seismic movements in the ground caused by rock blasting, piling, traffic, excavation, vibration compaction etc.
Ground vibrations, which are a form of energy transport through the ground, may damage adjacent structures when they reach a certain level.
Some of the energy released from a blast propagates in all directions from the hole as seismic waves with different frequencies. The energy from these seismic waves is damped by distance and the waves with the highest frequency are damped fastest. This means that the dominant frequencies from the blast are high at short distances and lower at longer distances.

The size of the ground vibrations depends on:

* quantity of co-operating charges
* constriction
* characteristics of the rock
* distance from the blasting site
* geology of covering earth deposits

By selecting the right blasting method and correct drilling and firing patterns the size of the ground vibrations can be controlled.

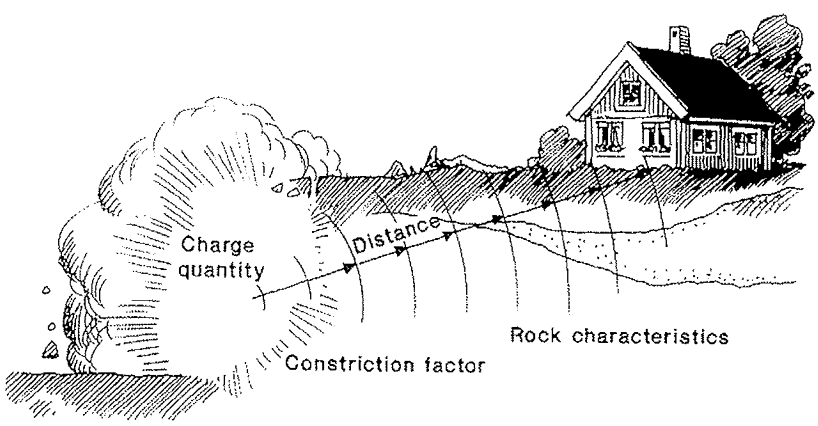



*Fig. 10.2*

Ground vibrations are a complicated type of seismic waves and consist of different kinds of waves:

* **P-wave.** The P-wave is also called the primary or compressional wave. It is the fastest wave through the ground. The particles in the wave move in the same direction as the propagation of the wave. the density of the material will change when the wave passes.

* **S-wave.** The S-wave is also called the secondary or shear wave. It moves through the medium at right angle to the wave propagation but slower than the P-wave. The S-wave changes the shape of the material but not the density.

Abbreviations:
SH = Shear wave, horizontal
SV = Shear wave, vertical
R = Rayleigh wave
P = Compressional wave

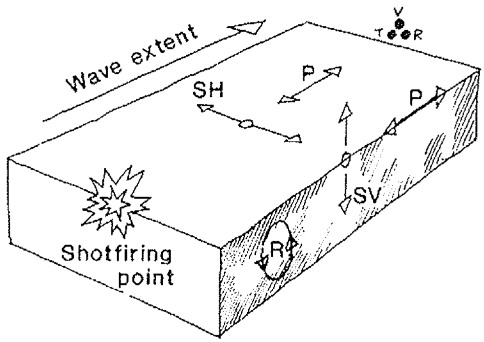



*Fig. 10.3 Seismic waves.*

The common denomination for P-waves and S-waves is body waves.

* **R-wave.** The R-wave (Rayleigh wave) is a surface wave which fades fast with depth. It propagates more slowly than the P and S waves and the particles move elliptically in the vertical plane and in the same direction as the propagation. At the surface the movement is retrograde to the movement of the wave.

The measuring of the ground vibrations is usually done at one or several points at ground level. For a total analysis, the practice is to measure in three directions: vertical, longitudinal and transverse. Normally the vertical component is dominant at shorter distances. It is therefore, usually sufficient to measure in the vertical direction. For vibration analysis of the measured values, the vibration phenomenon may be recorded as a function of time – time history. Then the displacement, particle velocity and acceleration can be recorded.
The basic rule is that the vibration velocity is measured on structures (buildings etc.) with a geophone and the acceleration on installations (computers etc.) with an accelerometer.
If the vibration velocity is measured, the acceleration can be calculated and vice versa. Which of these parameters that is the most interesting depends on the damage criterion for the structure to be protected. If this is known, it is normally sufficient to measure the peak value of the desired parameter.

### 10.2.2 Damage criteria and recommendations.

Experience over many years of measuring has shown that the particle velocity of the ground vibrations affecting a foundation constitutes the best parameter for the risk criterion for damage. As ground vibrations is approximately a sine formed vibration, the particle velocity can be calculated in accordance with the following formula:

$$v = 2\pi fA$$

where v = particle velocity (mm/sec)
f = frequency (periods/sec)
A = displacement in mm

From the above formula, the acceleration of the vibration can be calculated:

$$a = 4\pi^2 f^2 A$$

where a = acceleration in g (9.81 m/sec²)
A = displacement in mm

Control of the particle velocity is important, as it has been shown to be directly proportional to the stress to which the building material is exposed.
The relationship between particle velocity and stress in an ideal case, when a plane shock wave passes through an infinite elastic medium can be expressed as follows:

$$y = \frac{v}{c}$$

where y = shearing angle (mm/m)
v = particle velocity (mm/sec)
c = propagation velocity (m/sec)

To recommend realistic permitted levels of ground vibrations for buildings, engineers with extensive experience of rock blasting and vibration measurement evaluation should be consulted. Any restriction in the form of reduced vibration levels will increase the cost of drilling and blasting considerably.
For that reason it is important to start all blasting operations in populated areas with an inspection of surrounding buildings. This will be followed by a risk analysis in order to assess the sensitivity of the buildings and foundations to ground vibrations.

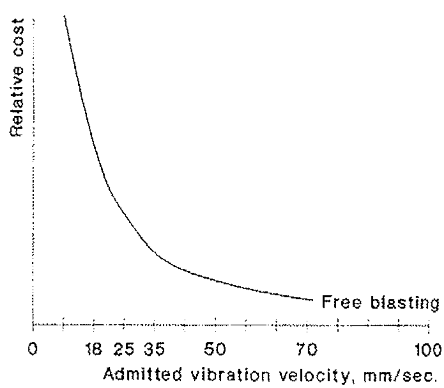



*Fig. 10.4 The effect on cost of different levels of vibration velocity.*

The most important parameters are:

* Vibration resistance of the building materials.
* The general condition of the building.
* Duration and character of the ground vibration.
* Presence of equipment sensitive to ground vibrations within the building.
* How the foundation is constructed.
* The quality of the foundation.
* The velocity of the wave propagation in rock, soil and construction material.

The following table shows the values that are normally permitted and which are used to evaluate the potential damage risk through ground vibration to standard residential housing.

Although the vibration velocity is stated as the permitted value it is the shearing angle which determines the dimensions. The accuracy of the values in the table has been confirmed by hundreds of thousands of readings over more than 40 years.

**Vibration velocities normally recommended in appraising ground vibration damage risk to residential buildings with respect to the foundation of the building.**

| Wave velocity c m/sec | 1000–1500 Sand, gravel clay under ground water | 2000–3000 Moraine slate, soft limestone | 4500–6000 Granite, gneiss, hard limestone, diabase quartzite, sandstone | Result in typical housing structures | Level at c=4500 to 6000 m/sec. |
|---|---|---|---|---|---|
| Vibration velocity v mm/sec | 9<br>13<br>18 | 18<br>25<br>35 | 35<br>50<br>70 | No visible cracking | 0.008<br>0.015<br>0.03 |
| | 30 | 55 | 100 | Fine cracks, falling plaster | 0.06 |
| | 40 | 80 | 150 | Notice-able cracking | 0.12 |
| | 60 | 115 | 225 | Severe cracking | 0.25 |

In the case of older buildings of poorer quality, it is customary to decrease the permissible vibration velocity from 70 mm/sec. to 50 mm/sec., in buildings of light concrete it should be decreased to 35 mm/sec. Conversely, there have been occasions where velocity values of more than 100 mm/sec. were attained without damage to buildings. In the case of individual blasting operations, sturdy concrete structures can stand values exceeding 150 mm/sec.

If the limit values in the table above regarding "no visible cracking" are transferred into a three-party graph, the curve will look like in Fig. 10.5 curve 3. However, in the curve the limit value for buildings founded on rock has been reduced from 70 mm/sec. to 50 mm/sec. Curve 3 can be said to represent the recommended limit values for normal residential areas. For frequencies exceeding 40 Hz the particle velocity is the criterion for damage but at lower frequencies the displacement represents the criterion.

The dominant frequencies for vibrations passing through soft kinds of rock, moraine, sand, gravel, clay etc. are lower than for example granite. This is shown in the above table and curve 3 reflects this for lower frequencies where displacement is used as the criterion. Curve 2 in Fig. 10.5 represents values at which buildings receive direct damage. (Langefors and Kihlström, 1967.)

It must be pointed out that curve 3 only indicates the recommended limiting value and expert judgment is needed to determine more accurately which upper limit values should be set for structures adjacent to blasting operations.

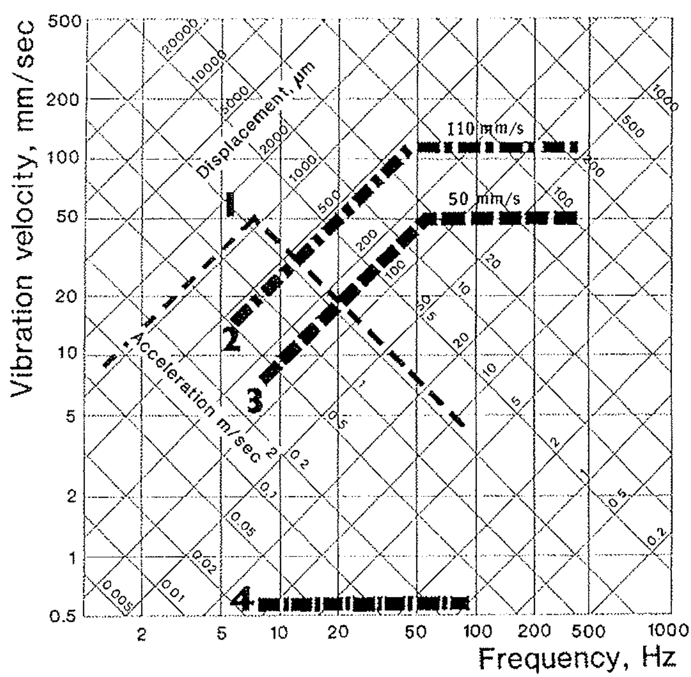



*Fig. 10.5*

Criteria for damage and recommendations:

Curve 1: Recommended upper limit for IBM computers with a duration of vibration less than 5 sec.
Curve 2: Direct damage from vibrations to buildings during blasting.
Curve 3: Recommended upper limit for blasting.
Curve 4: Vibrations disturbing to human beings.

In connection with blasting operations close to telephone and relay stations or buildings containing other sensitive equipment such as computers, electron microscopes, turbines etc. consideration must be given to the acceleration in order to avoid disturbances.
The recommended permissible values for ground vibration close to this type of equipment are:

* Telephone – relay stations
  v=50 mm/sec. and a=0.1–3.0 g depending on type of station.
* TV-stations
  v=35 mm/sec. and a=3.0 g.
* Axe-electronic switch boards
  v=20 mm/sec.
* Computers
  a=0.25 g. (For certain parts of the computer.)

Blasting close to computer installations (not micro computers or PC:s), where the manufacturer prescribes a maximum acceleration of 0.25 g, is difficult and under certain circumstances impossible, if special arrangements are not made. Nitro Consult AB, a subsidiary of Dyno Industries, Norway, has therefore developed a special method to dampen these installations, thus reducing the vibrations coming into the equipment. Dampening should always be followed up with vibration measurement.

The size of the ground vibrations depends on:

* number of co-operating charges
* the constriction of the blast
* the characteristics of the rock
* the distance from the blasting site
* the geology of the surrounding ground

For the planning of blasting operations where ground vibration problems occur, it is important to be aware of the relationship between distance, charge and ground vibration.

Using Langefors' formula for determining the charge level the vibration velocity can be calculated:

$$\text{Charge level} = \frac{Q}{R^{3/2}}$$

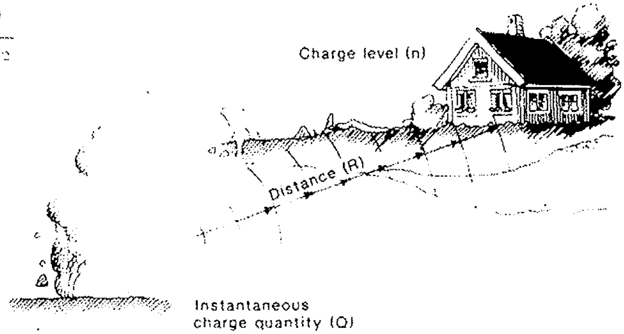



*Fig. 10.6.*

where Q indicates the charge in one hole in kg or several instantaneously fired charges at the same distance R in meters.

Vibration velocity:

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

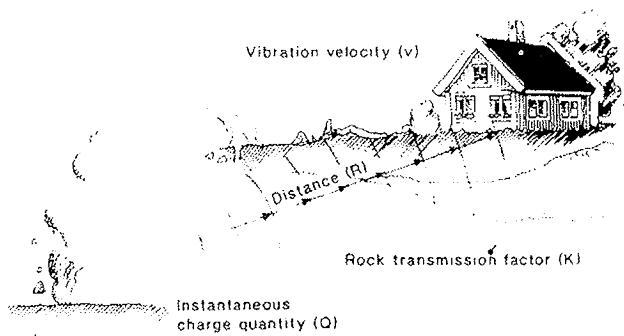



*Fig. 10.7.*

| | | | |
|---|---|---|---|
| where Q | = | instantaneously detonating charge | (kg) |
| R | = | distance | (m) |
| v | = | vibration (particle) velocity | (mm/sec) |
| K | = | transmission factor, constant depending on the homogeneity of the rock and the presence of faults and cracks. For hard Swedish granite it is approx. 400 but it is normally lower. | |

The relationship between charge/distance and ground vibration can be used to make a simple table which may serve as ready-reckoner for the planning of blasting operations.

| Distance | Charge in kg (instantaneous detonation) | | | | | | |
|---|---|---|---|---|---|---|---|
| m | Level: 0.008 | 0.015 | 0.03 | 0.06 | 0.12 | 0.25 | 0.50 |
| 0.5 | | | | 0.02 | 0.04 | 0.08 | 0.16 |
| 1 | 0.008 | 0.015 | 0.03 | 0.06 | 0.12 | 0.25 | 0.50 |
| 2 | 0.023 | 0.04 | 0.08 | 0.17 | 0.34 | 0.68 | 1.35 |
| 3 | 0.04 | 0.08 | 0.16 | 0.32 | 0.65 | 1.30 | 2.60 |
| 4 | 0.06 | 0.12 | 0.24 | 0.48 | 1.0 | 2.0 | 4.0 |
| 5 | 0.09 | 0.17 | 0.35 | 0.70 | 1.4 | 2.8 | 5.6 |
| 6 | 0.12 | 0.22 | 0.44 | 0.88 | 1.8 | 3.7 | 7.3 |
| 7 | 0.15 | 0.28 | 0.56 | 1.1 | 2.2 | 4.6 | 9.2 |
| 8 | 0.18 | 0.34 | 0.68 | 1.35 | 2.7 | 5.7 | 11.3 |
| 9 | 0.22 | 0.4 | 0.8 | 1.6 | 3.2 | 6.7 | 13.5 |
| 10 | 0.25 | 0.5 | 1.0 | 2.0 | 4.0 | 8.0 | 16.0 |
| 12 | 0.3 | 0.6 | 1.2 | 2.5 | 5.0 | 10.4 | 20.8 |
| 14 | 0.4 | 0.8 | 1.6 | 3.1 | 6.3 | 13.0 | 26.0 |
| 16 | 0.5 | 1.0 | 1.9 | 3.8 | 7.7 | 16 | 32 |
| 18 | 0.6 | 1.2 | 2.3 | 4.6 | 9.2 | 19 | 38 |
| 20 | 0.7 | 1.3 | 2.7 | 5.4 | 10.7 | 22 | 44 |
| 25 | 1.0 | 1.9 | 3.8 | 7.5 | 15 | 31 | 62 |
| 30 | 1.3 | 2.5 | 4.9 | 9.8 | 20 | 41 | 82 |
| 40 | 2.0 | 3.8 | 7.6 | 15 | 30 | 63 | 126 |
| 50 | 2.8 | 5.3 | 10.6 | 21 | 42 | 88 | 176 |
| 60 | 3.7 | 7.0 | 14 | 28 | 56 | 116 | 232 |
| 70 | 4.7 | 8.8 | 18 | 35 | 70 | 146 | 292 |
| 80 | 5.7 | 10.7 | 21 | 43 | 86 | 178 | 358 |
| 90 | 6.8 | 12.8 | 25 | 51 | 102 | 213 | 427 |
| 100 | 8.0 | 15.0 | 30 | 60 | 120 | 250 | 500 |
| 120 | 10.5 | 19.7 | 39 | 79 | 158 | 328 | 657 |
| 140 | 13.2 | 24.8 | 50 | 100 | 200 | 410 | 820 |
| 160 | 16.2 | 30 | 60 | 120 | 240 | 500 | 1000 |
| 180 | 19.3 | 36 | 72 | 145 | 290 | 600 | 1200 |
| 200 | 22.6 | 42 | 85 | 170 | 340 | 700 | 1400 |

The charge levels in the previous table correspond to the following vibration velocities if the rock transmission factor K=400.

| Level | Vibration velocity | |
|---|---|---|
| $Q/R^{3/2}$ | mm/sec. | |
| 0.008 | 35 | |
| 0.015 | 50 | |
| 0.03 | 70 | |
| 0.06 | 100 | (threshold value granite) |
| 0.12 | 150 | |
| 0.25 | 225 | |
| 0.50 | 300 | |

The relationship charge/distance and vibration velocity can also be expressed graphically:

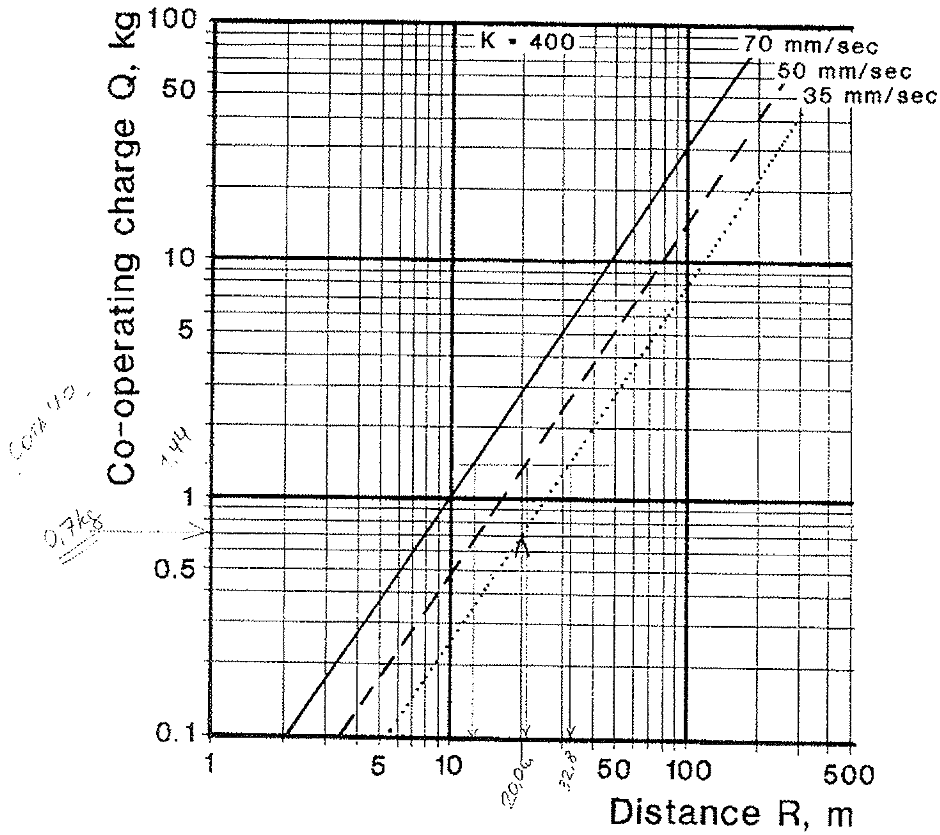



*Fig. 10.8 Charge (Q) as a function of distance (R) for different levels of vibration velocity. Rock transmission factor K=400. At a distance of 20 m, the charge must not exceed 1.3 kg to ensure a vibration velocity of less than 50 mm/sec.*

The distance and charge tables which are based on the determined rock transmission factor K should be used with care close to buildings where the foundation is unknown e.g. buildings built partly on rock and partly on soil and buildings founded on wooden piles in clay etc. The value of the rock transmission factor K will also change depending on the characteristics of the ground and the distance. Looser materials such as moraine and clay have lower K values than homogeneous hard rock. The rock transmission factor K is also lower in weathered and fissured rocks.

The actual value of the factor K is best determined by test blastings at the actual site, followed up by scrupulous vibration measurement.

To evaluate the test blasts, the constriction of the blast must be considered e.g. if the test hole has free breakage, if it just cracks the rock or if it does not affect the rock at all. To evaluate a test blast correctly, experience of test blastings and knowledge of the field of ground vibrations is necessary.

When the rock transmission factor K is determined, the graph in fig. 10.8 may be adjusted accordingly and the realistic relationship between charge/distance and vibration velocity adapted to the local conditions.

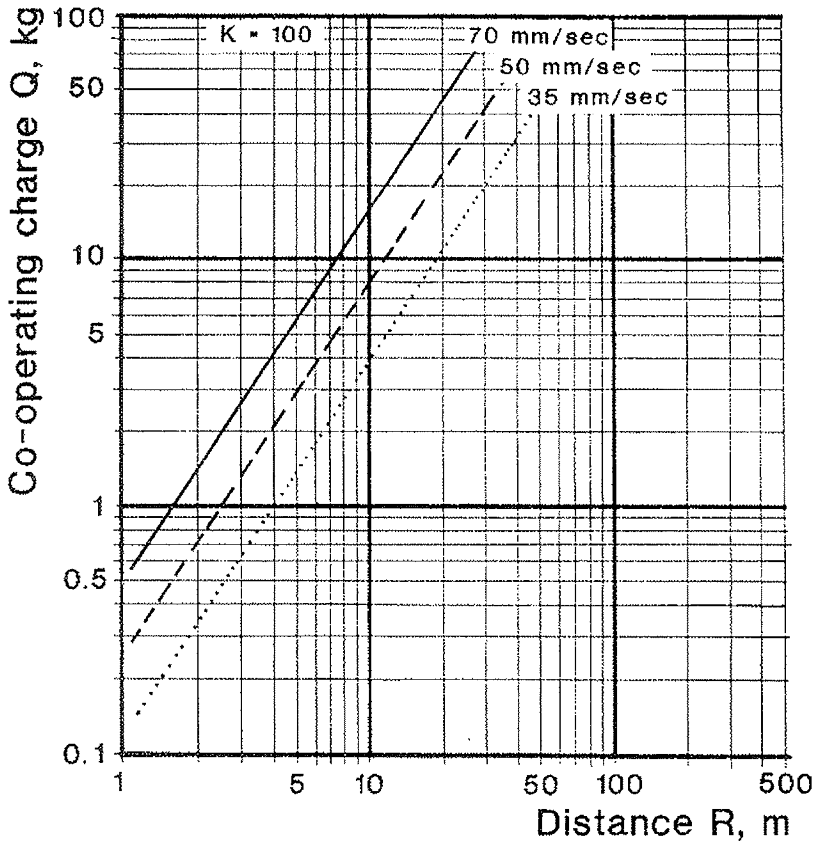



*Fig. 10.9 Charge (Q) as a function of distance (R) for different levels of vibration velocity. Rock transmission factor K=100. At a distance of 20 m, the charge must not exceed 20 kg to ensure a vibration velocity of less than 50 mm/sec.*

The comparison of the two graphs with rock transmission factors K=400 and K=100 respectively shows that the dampening effect is higher in the softer rock (K=100) and the vibration velocity is lower if the relationship charge/distance is maintained.

### 10.2.3 Geological factors influencing ground vibrations.

Soils and rocks are porous materials with a relatively rigid skeleton of particles. The pores are filled with water or air. In soil, the soil skeleton consists of mineral grains which are held together by frictional and cohesive forces. In sedimentary rocks the mineral grains are cemented together and in magma rocks and metamorphous rocks the minerals have crystallized to a rockmass which usually contains water-bearing fissures and joints. In practice it may be difficult to state a precise propagation velocity of the seismic wave in different soils and rocks.

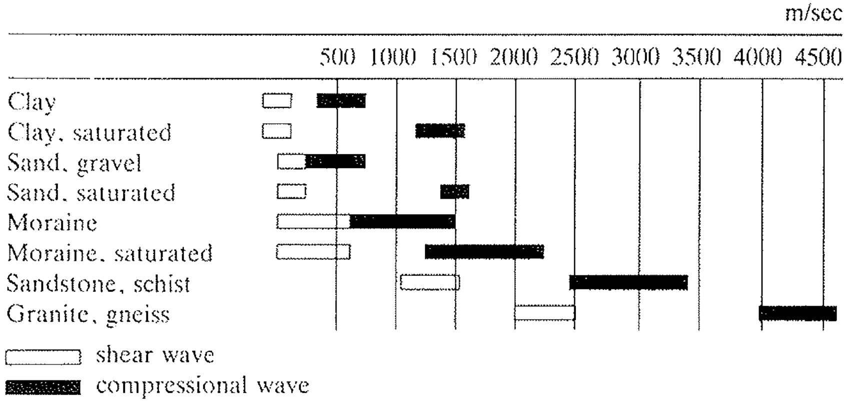



*Fig. 10.10 The propagation velocity of compressional and shear waves through different soils and rocks.*

The propagation velocities of the Rayleigh wave depend on the frequency and are lower than those of the shear wave.

Every geological environment has its own ground vibration characteristics which affect the propagation of the vibration wave. The ground vibration characteristics depend on the following properties of the ground:

* the elastic constants of the ground (elastic and shearing moduli) which determine the propagation velocity of the waves.
* the type of soil and its depth which determine the predominant range of frequency and type of waves.
* the moistness of the soil and the ground water level.
* the topography and morphology, which may lead to focusing of seismic waves.
* the damping characteristics of the ground.

An example of geological factors influencing the rock blasting operation is the difference in permitted charges at different distances in Sweden and the U.S.A.

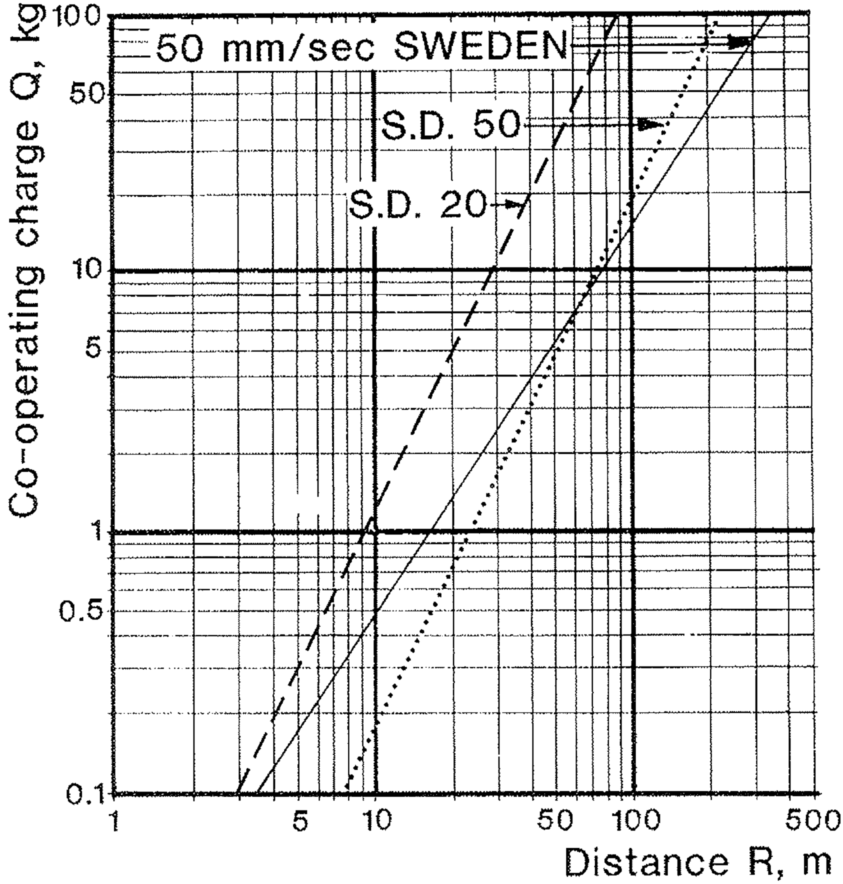



*Fig. 10.11 Comparison of quantity-distance relationships for specified values of vibration velocity in Sweden and U.S.A.*

In U.S.A. the highest permitted vibration velocity is 50 mm/sec. and safe scaled distances have been established for use in the field. The equations for blast design are:

$D/\sqrt{W} \geqslant 50\ ft/lb^{1/2} = S.D.$ (1)
$D/\sqrt{W} \geqslant 20\ ft/lb^{1/2} = S.D.$ (2)

where D is the distance in feet from blasting site to structure in question
W is maximum charge weight in pounds per delay
S.D. is scaled distance

Equation #1 is recommended for sites where no instrument measurements are made as the equation contains a rather high safety factor.
Equation #2 is only recommended for sites where instrument recordings of the blasts are made.

A scaled distance of 50 or more will protect against vibrations greater than 50 mm/sec. As may be seen in Graph 10.11. the scaled distance equstion #1 gives rather conservative values, especially on shorter distances, e.g. at a distance of 10 m, 0.2 kg per delay is permitted according to scaled distance equation in U.S.A. while 0.5 kg is permitted instantaneously in Sweden. (Which can be doubled or trebled per delay if MS detonators are used, see page 219.)
On the other hand, the Graph 10.11 also shows that, when using equation #2, the charge level may be increased considerably, e.g. at a distance of 10 m 1.0 kg can be fired simultaneously. However, equation #2 is only recommended for sites where vibrations are measured and peak particle velocities of 50 mm/sec or less are obtained. If the blaster wants to use a smaller scaled distance, that is, he wants to use more explosives per delay, then a consultant and/or permission from regulatory authorities is/are required.
At larger distances S.D. 50 is not so conservative, at 100 m 20 kg per delay may be charged compared to 15 kg in Sweden, but S.D. 20 permits 100 kg per delay. The charge levels are considerably higher in U.S.A. especially at larger distances. The reason for that is that the rock characteristics in U.S.A. are different from those in Sweden. The rock is generally softer and more weathered with lower propagation velocity of the vibration wave. Furthermore, and equally important, the vibrations are dampened faster and the vibration velocity is thus lowered.

### 10.2.4 Planning of blasting operations.

At the planning stage of the blasting operation, attention must be paid to the geological characteristics of the rock. If there are zones of weathered and fissured rock between the blasting site and objects sensitive to vibrations with a damping effect on the ground vibrations, the geological characteristics of the rock may change to more homogeneous rock as the work proceeds, increasing the ground vibrations. It may than be necessary to decrease the charge to avoid damage.
Therefore the test blastings should be measured to make a seismic profile where the seismic waves are measured at various points giving information on how the characteristics of the rock varies.

When planning and executing the blasting operation, it is important that the constriction of the round is minimized by correct drilling and firing patterns.

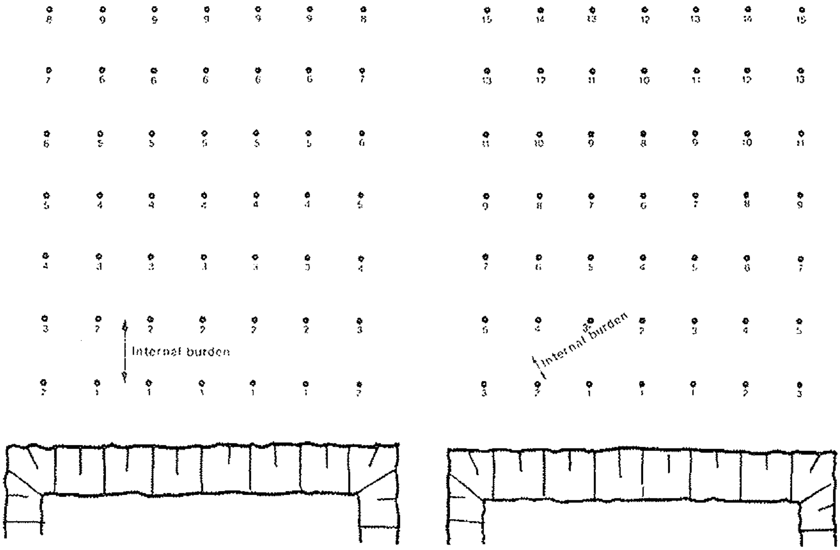



*Fig. 10.12 By changing the firing pattern, the internal burden may be minimized and the constriction lowered.*

The vibration velocity also depends on the inclination of the hole. Steeper hole inclinations or other conditions increasing the constriction of the blast (misfires etc.) may cause considerable increase of the vibration velocity.

The ground vibrations will also increase if the blast fails to break the rock down to the intended level.

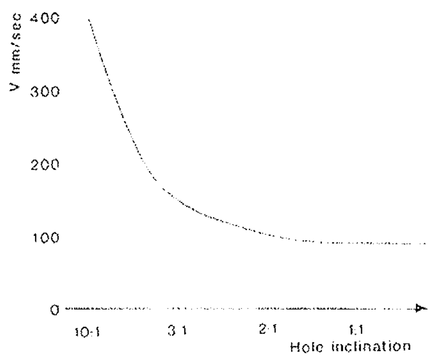



*Fig. 10.13 Vibration velocity in relation to hole inclination with the same burden and explosive charge (trench blasting).*

The first rounds blasted at a work site must be considered as test blastings and the vibration measurements should be used as a guidance for the planning of an optimum blasting operation. The results from the vibration measurements should be utilized during all blasting operations to find the most economic drilling and firing pattern. However, a certain margin to the permitted vibration velocity should always be maintained as the ground vibrations may increase sharply if the blast does not go according to plan. This can be difficult in cases when the drilling is far ahead of the blasting operation, but using the result of the initial risk analysis and a thorough follow-up during the blasting operation, the drilling pattern may be selected in such a way that several charges may be used in each hole if the vibration velocity values become too high.

Investigations show that people in general react to vibration values far below the limit for damage on buildings. It has also been demonstrated that blasting operations which are executed in a short time are better accepted by people in the area than operations lasting for a long time, even if there are long gaps between the blasts.
The best way to forestall complaints is if those responsible for the blasting operations give comprehensive information to the people affected.

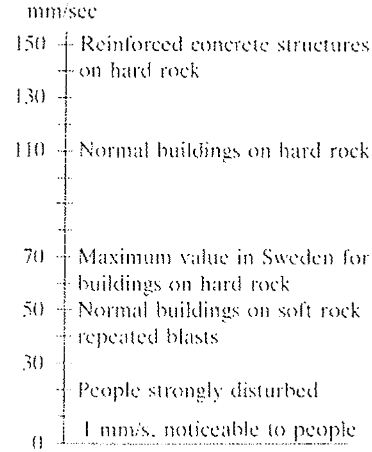



*Fig. 10.14 Maximum permitted vibration velocities for residential buildings.*

More often than not it is a good investment to employ a consultant at the beginning of a blasting operation. The consultant will take care of the primary risk analysis and before blasting starts point out the problems which are likely to occur. Using the knowledge gained from the risk analysis the blasting operation can be better planned both technically and economically. It is normally cheaper to prevent problems than take measures when they arise.

### 10.2.5 Instruments to measure ground vibrations.

Different types of instruments have been developed for ground vibration measurements.
The early instruments were mechanical. They were fixed to the object which was subjected to ground vibrations. The principle was that the instrument contained a heavy weight which was suspended in a spring, acting as an inert mass. During the vibration the instrument moved but the weight did not. The movement of the instrument was recorded on paper and the level of ground vibration could be evaluated. The mechanical instruments have nowadays been replaced by electronic ones.
In the electronic instrument, the mechanical vibration is sensed and converted to an electric signal by an electro dynamic transducer called a geophone. This transducer gives an electric signal which is direct proportional to the particle velocity of the vibration, which is the parameter being recorded.

The geophone consists of a spring (1) loaded – moving mass (2) system. A coil (3) is wound around the moving mass. The system moves freely in a magnetic field created by a permanent magnet (4). When the coil moves in the magnetic field an electric current is induced with a magnitude proportional to the velocity of the coil.
In the geophone in use, the coil is steady while the magnet mounted in the outer case moves in relation to the received mechanical vibration.

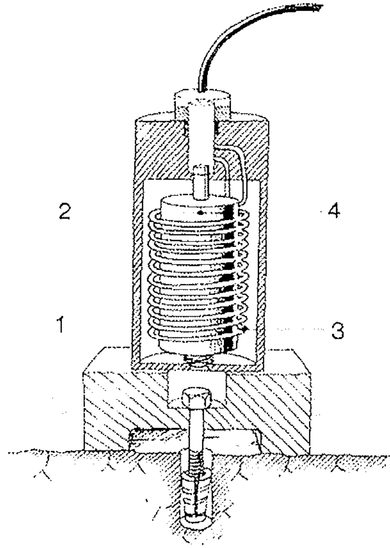



*Fig. 10.15 Basic design of geophone.*

There are instruments available for the measurement of vertical and horizontal components at one or several measuring points. As regards blasting it is the magnitude of the vertical component that is important. The instruments may also be supplemented with an accelerometer to measure the acceleration.

The two main types of available vibration measurement instruments are:
- Peak value monitors for continuous registration
- Time-History recorders for registration of events.

The peak value monitor UVS 1201 measures the peak particle velocity (PPV) of a vibration wave at one single point using a geophone. The velocity signal is also processed into corresponding peak values of displacement and acceleration. The values are updated continuously and stored in an electronic memory at 2-minute intervals.
The UVS 1201 has one month's memory capacity.

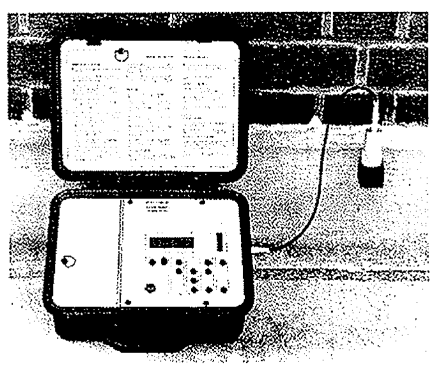


*Fig. 10.16 UVS 1201.*

Measurement values from any part of the memory can be displayed on an LCD (Liquid Crystal Display), and hard copy reports may be obtained by connecting a printer to the UVS 1201. By selecting start and stop times and setting suitable threshold values, the printout can be edited according to preference.
Furthermore, the content of the memory of the UVS 1201 can be copied to a portable disc drive for further processing.

The UVS 1404 is a four-channel instrument for measurement and recording of peak particle velocity, acceleration, water shock waves and air shock waves, depending on the type of sensor used for each channel.
The recorded data are presented on a dual LCD system. The primary LCD shows a continuous graphical diagram along a horizontal time-axis while alpha-numeric information (including date, time, graphical scale and channel number) is given on the second LCD.

The one-month memory of the UVS 1404 is updated continuously in the same manner as for the UVS 1201. Printed copies can be produced by connecting a printer to the machine. Stored data can also be duplicated to a portable disc drive for later editing and printout.

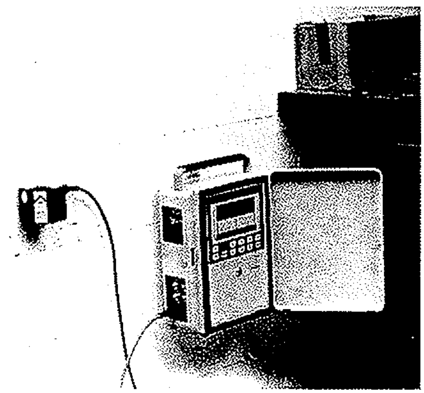


*Fig. 10.17 UVS 1404.*

The Time History recorder UVS 1608 records complete Time History sequences of vibration or shockwave transients. The incoming signals are electronically stored and readily available for presentation on an illuminated LCD screen. The built-in printer provides on demand an exact hard copy of what is shown on the display at any particular moment.
Furthermore, the entire content of the UVS 1608 memory can be copied to a 3.5″ disc, which can be reread, processed and printed by the UVS 1608 at any time.

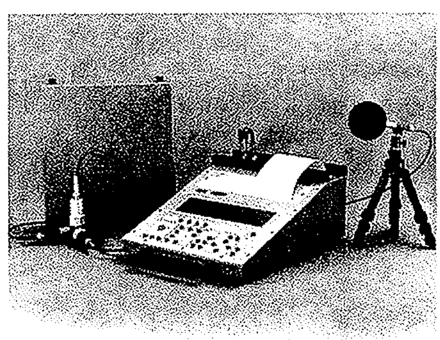


*Fig. 10.18 UVS 1608.*



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The UVS 1608 is available in 4-channel and 8-channel versions. Each channel is compatible with geophones as well as accelerometers and microphones.
The UVS 1608 can also be interconnected with a UVS 1404 Vibration Monitor when Time History recordings are required in parallel with long-term monitoring of peak values.

All the instruments described above are designed to stand harsh working and anvironmental conditions. The electronic components which are placed in heavy-duty casings are specially designed to resist extreme temperature variations, from the tropics to the arctic.
The UVS 1201, UVS 1404 and UVS 1608 instruments are manufactured by Nitro Consult AB Sweden.

## 10.3 Charge calculations.

### 10.3.1 General.

When blasting close to buildings and other installations sensitive to vibrations it is not always possible to utilize the blastholes in the same way as in normal blasting operations. The ground vibrations which always occur in blasting operations depend on the maximum co-operating charge weight. Thus, the charge weight for each delay must be kept within certain limits for different distances. However, for big blasts and long distances the total amount of explosives may be a determining factor for the size of the vibrations.
The constriction of the blast is another factor which affects the size of the ground vibrations. A constricted charge gives higher vibrations than one with free breakage.

The maximum co-operating charge can be determined from the charge/distance graph. Knowing the distance to the sensitive object it is easy to determine the correct charge weight.
The charge weight depends on the permitted vibration velocity and the rock transmission factor.

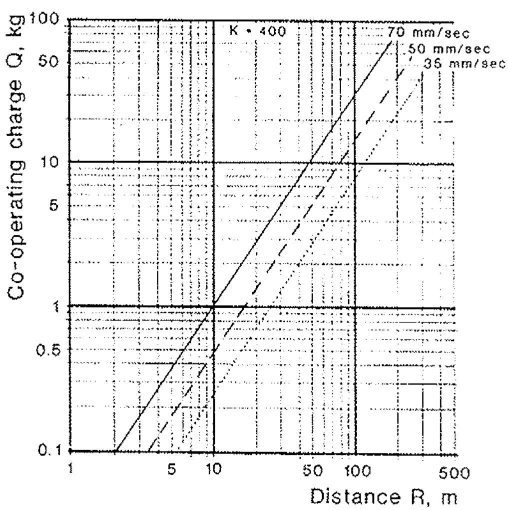



*Fig. 10.19 Charge/distance graph for different vibration velocities.*

The maximum co-operating charge can be reduced in the following way:

- The firing pattern.
  The number of holes with the same period number is reduced so the maximum co-operating charge is not exceeded.
- Reduced drilling pattern.
  The blasthole volume is not utilized to the maximum for the explosives charge as in normal blasting. The drilling pattern is more closely spaced with less explosives in each hole.
- Divided charges.
  The requisite charge amount for the hole is divided into several partial charges fired with different delays. The charges are separated by sand stemming.
- Divided benches.
  The bench is not blasted to its full depth in one go but divided into several lower benches.

**10.3.2 The firing pattern.**

In cautious blasting it may be necessary to decrease the co-operating charge. This can be done by decreasing the number of detonators with the same period number.
Detonators with the same period number always have a certain scatter. In other words, the delay time of the delay element is not exactly the same for detonators with the same period number. This means that only some of the detonators within the period will co-operate.
The co-operation within the period or between various periods depends on the frequency of the ground vibrations. For hard homogeneous bedrock the frequency is normally over 60 Hz and here the following practical rule for co-operation will apply:

| Detonator type | Period number | Co-operation within period (reduction factor) |
|---|---|---|
| VA/MS | 1–10 | 1/2 |
| Nonel GT | 3–10 | 1/2 |
| VA/MS | 11–20 | 1/3 |
| Nonel GT | 11–20 | 1/3 |
| Nonel GT/T | 1–20 | 1/4 |
| | 25–60 | 1/6 |
| VA/HS | 1–12 | 1/6 |

The table is based on the scatter within the period which is lowest for MS detonators but may be as high as 200 ms for HS detonators.

According to Langefors, the risk of co-operation is greater at low frequencies.

Less than 60 Hz:

| | | |
|---|---|---|
| VA/MS | 1–10 | 1 |
| VA/MS | 11–20 | 1/2 |
| VA/HS | 1–12 | 1/6 |

Less than 20 Hz:

| | | |
|---|---|---|
| VA/MS | 1–20 | 1 |
| VA/HS | 1–12 | 1/3 |

The low frequencies occur in soft rocks and when blasting at relatively great distances.
At the lowest frequencies it may theoretically be co-operation between different period numbers.
In the U.S.A., with its softer rocks, the charges are supposed to co-operate if the delay between them is shorter than 9 ms.

The following example shows the effect of the reduction factor in cautious blasting:

Conditions:

- Rock, granite
- Bench blasting
- Blasthole diameter, drill series 11 (34–29 mm)
- Bench height 4.0 m
- Charge per hole 1.95 kg
- Maximum permitted co-operating charge 5.0 kg

A. Blasting without considering the reduction factor.

$2 \times 1.95$ kg = 3.9 kg

Conclusion: Maximum 2 blastholes may co-operate, which in this case implies 2 detonators per period number.

B. Blasting considering the reduction factor.

- MS detonators with period numbers 1 to 10 have a reduction factor of 1/2, which means that only half the detonators within the same period are likely to co-operate.

  If 4 detonators with the same period number are used only 2 will co-operate. The maximum charge which will detonate instantaneously is then $(4 \times 1.95) \times 1/2 = 3.9$ kg. If for example 5 detonators are used in the same period there is a risk of over-charging, as 3 of the 5 detonators are likely to co-operate ($3 \times 1.95$ kg$=5.85$ kg).

  Conclusion: 4 detonators MS 1 to 10 may be used in the same period without risk of excessive charge.

- MS detonators with period numbers 11 to 20 have a reduction factor of 1/3, meaning that one third of the detonators within the period are likely to co-operate.

If 6 detonators are used with the same period number only 2 will co-operate. The maximum charge which will detonate instantaneously is $(6\times1.95)\times 1/3=3.9$ kg.

Conclusion: 6 detonators MS 11 to 20 may be used in the same period without risk of excessive charge.

If a firing pattern starting with 4 pcs MS No. 1 is changed to one starting with 6 pcs MS No. 11, it will not increase the co-operating charge.
In the case of different amounts of explosives in the blastholes, the least favorable case has to be reckoned on, that is, the holes with the biggest amounts of explosives will co-operate.
When the number of detonators within each period is limited because of restricted ground vibrations, it may be a problem to obtain enough periods for the blast. In cases like this it is practical to use NONEL UNIDET with its unlimited number of delays (See Chapter 3b.2.4 NONEL).
In blasting operations very close to objects which are sensitive to ground vibrations, where vibrations over the permitted limit may result in severe damage, no reduction factor should be used and only the real number of detonators per period taken into account.

### 10.3.3 Bench blasting with reduced drilling pattern.

When the maximum permitted co-operating charge is smaller than the requisite charge for the blasthole, it is no longer possible to reduce the co-operating charge with the firing pattern. One possibility to reduce the charge is to reduce the blasthole diameter, which gives less explosives in each blasthole. Then normal drilling and charging tables may be used for the actual blasthole diameter.
However, it is often practically impossible to change the drilling equipment. In these cases a reduced drilling pattern is used, drilled with the existing equipment, where the permitted charge determines the drilling pattern.
The reduced drilling pattern increases the specific drilling which naturally increases the cost. To what degree the specific drilling may increase from an economic point of view must be decided upon in each case.
One basis for forming a judgment is to compare with the specific drilling when the blasthole is fully utilized:

| Hole diameter mm | Specific drilling m/cu.m. |
|---|---|
| Drill series 11 | |
| 34–27 | 0.8–1.3 |
| Drill series 12 | |
| 40–29 | 0.6–0.9 |
| 51 | 0.3 |
| 64 | 0.2 |
| 76 | 0.15 |

To calculate the reduced drilling pattern, the permitted co-operating charge must be known. The permitted co-operating charge may be found in the charge/distance table or in the graph 10.8. Knowing the vibration velocity which is permitted for the object in question and the distance to the blasting site, the actual permitted co-operating charge will be found.

The correct permitted vibration velocity is found in the table on page 204, which shows the vibration velocity that is normally permissible for residential buildings for the kind of material on which the buildings are built.

The basis for the calculations is that the specific charge should be 0.40 kg/cu.m. This value is the normal value and changes may be needed due to the blastability of the rock. The change of specific charge does not change the calculation procedure.

By carrying out test blasts followed by analysis of the vibration measurement results, it is often possible to use more explosives than indicated in the graph thus lowering the blasting cost.

**Charge calculation procedure.**

Specific charge q (kg/cu.m.)

Permitted co-operating charge $Q_{per}$ (kg)

**Drilling pattern.**

1. The volume of rock which is blasted by each hole:

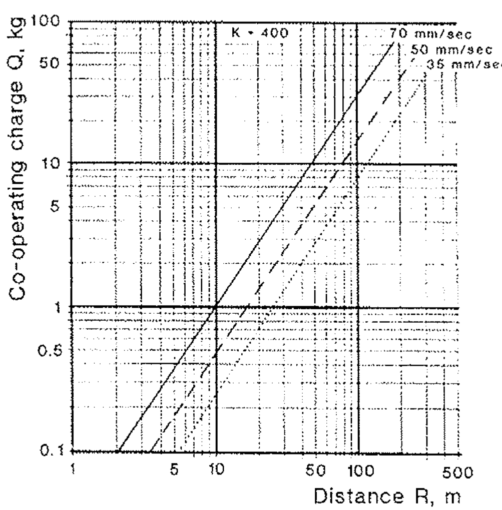
$$\text{Volume} = \frac{Q_{per}}{q} \quad \text{(cu.m.)}$$

2. When the volume is known, the drilling pattern can be calculated.

The surface area each blasthole can cover:

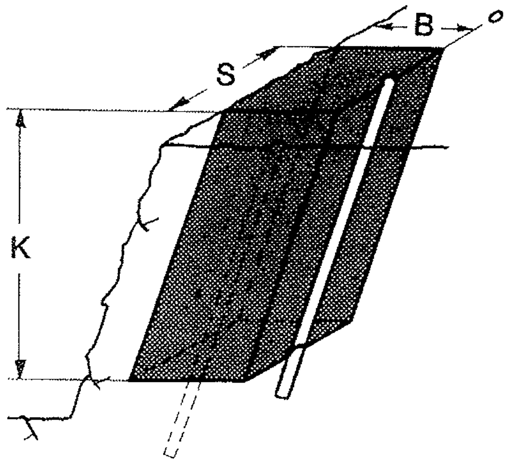
$$\text{Area} = \frac{\text{volume}}{K} \quad \text{(sq.m.)}$$

(K, in this case is bench height)



3. Practical drilling pattern. When the surface area for the blasthole is known the practical burden is:

$$B = \sqrt{\frac{Area}{1.25}} \qquad (m)$$




Practical spacing:

$$S = 1.25 \times B \qquad (m)$$

The practical spacing should be adjusted to the width of the bench if necessary.

4. Hole depth
The hole depth H may be estimated from tables in Chapter 5.2 Charge calculations.

5. Specific drilling

$$b = \frac{H_{est}}{B \times S \times K} \qquad (m/cu.m.)$$

An estimate should be carried out to see if the specific drilling is acceptable. If not, the use of divided charges in the holes or several lower benches should be considered.

If the specific drilling is acceptable, the calculations continue as follows:

6. Drilling error.

$$E = \frac{d}{1000} + 0.03K \qquad (m)$$

7. Subdrilling

$$U = 0.3(B+E) \qquad (m)$$

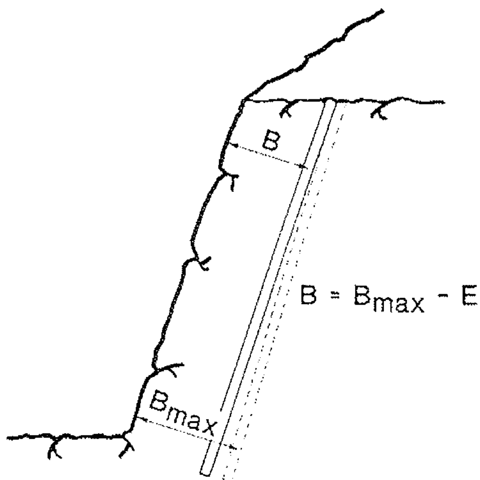
8. Hole depth

$$H = a(K+U) \qquad (m)$$

a = 1.05 for inclination 3:1 and 1.0 for vertical holes.

9. Maximum burden

$$B_{max} = B+E \qquad (m)$$



**Bottom charge.**

10. Charge concentration

$$l_b = \frac{B_{max}^2}{2} \text{ for Dynamex M (kg/m)}$$

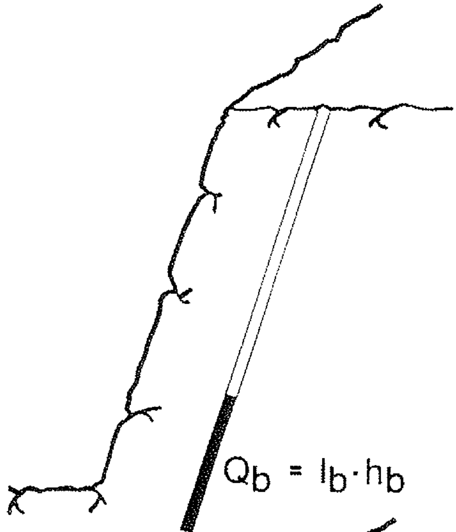
$$l_b = \frac{B_{max}^2}{2} \times \frac{1.15}{1.25} \text{ for Emulite 150}$$

11. Height

$$h_b = 1.3 \times B_{max} \qquad \text{(m)}$$

12. Weight

$$Q_b = l_b \times h_b \qquad \text{(kg)}$$




**Column charge.**

13. Weight

$$Q_c = Q_{per} - Q_b \qquad \text{(kg)}$$

14. Stemming

$$h_o = B \qquad \text{(m)}$$

15. Height

$$h_c = H - h_o - h_b \qquad \text{(m)}$$

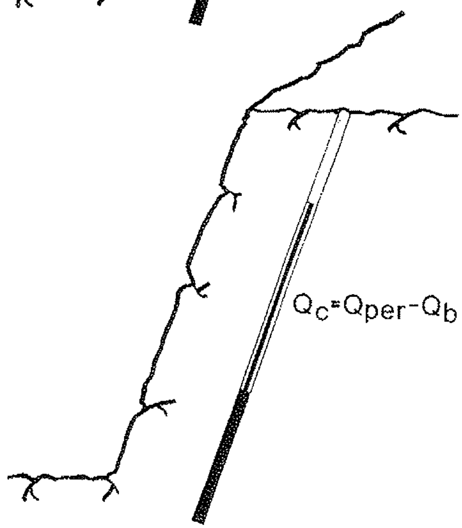
16. Charge concentration

$$l_c = \frac{Q_c}{h_c} \qquad \text{(kg/m)}$$

$l_c$ should be at least 40 % of $l_b$. Explosives of suitable dimensions should be selected for the charge.

17. Total charge weight

$$Q_{tot} = Q_b + Q_c \qquad \text{(kg)}$$




Check that $Q_{tot} \leq Q_{per}$. If this is not the case, the column charge should be reduced. If that is not possible the drilling pattern must be reduced further.

18. Specific drilling

$$b = \frac{\text{number of drilled meters per row}}{\text{volume per row}} = \frac{n \times H}{w \times B \times K} \qquad \text{(m/cu.m.)}$$

19. Specific charge

$$q = \frac{\text{Total charge per row}}{\text{Volume per row}} = \frac{n \times Q_{tot}}{w \times B \times K} \qquad \text{(kg/cu.m.)}$$

**Calculation example.**

Conditions: Blasting to be carried out close to a TV transmitting station. The permitted vibration velocity is 35 mm/sec. and the distance to the blasting site is 20 m.

| | |
|---|---|
| Blasthole diameter: | Drill series 11, 34–27 mm in this case 31 mm. |
| Bench height: | 2.5 m |
| Hole inclination: | 3:1 |
| Width of the round: | 12 m |
| Rock transmission factor K: | 400 |
| Explosive: | Dynamex M |
| | Gurit |
| Specific charge q: | 0.4 kg/cu.m. |
| Permitted co-operating charge: | 0.65 kg |

(in accordance with graph Fig. 10.8)

1. Rock volume per hole.

$$\text{Volume} = \frac{Q_{per}}{q} = \frac{0.65}{0.4} = 1.63 \text{ cu.m.}$$

2. Surface area per hole.

$$\text{Area} = \frac{\text{Volume}}{K} = \frac{1.63}{2.5} = 0.65 \text{ sq.m.}$$

3. Practical drilling pattern.

Burden:

$$B = \sqrt{\frac{\text{Area}}{1.25}} = \sqrt{\frac{0.65}{1.25}} = 0.72 \text{ m}$$

Spacing:
$S = 1.25 \times 0.72 = 0.90$ m
Adjustment of the spacing to the width of the bench.

$$\text{Number of hole spaces } \frac{12.0}{0.90} = 13.33, \text{ that is } 14.$$

$S_{adj} = 12/14 = 0.86$ m
Number of holes per row $14+1=15$.

4. Estimated hole depth
From table in Chapter 5.2 Charge calculations

$H_{est} = 3.05$ m

5. Specific drilling

$$b = \frac{H_{est}}{B \times S \times K} = \frac{3.05}{0.72 \times 0.86 \times 2.50} = 1.97 \text{ m/cu.m.}$$

The specific drilling for drill series 11 is 0.8 to 1.3 m/cu.m. when the blasthole is fully utilized. The specific drilling is somewhat high in this case, but may be accepted.

6. Drilling error.

$$E = \frac{d}{1000} + 0.03K = \frac{31}{1000} + 0.03 \times 2.5 = 0.11 \text{ m}$$

7. Subdrilling

$U = 0.3(B+E) = 0.3(0.72+0.11) = 0.25$ m

8. Hole depth

$H = a(K+U) = 1.05(2.5+0.25) = 2.89$ approx. 2.90 m

9. Maximum burden

$B_{max} = B+E = 0.72+0.11 = 0.83$ m

Bottom charge.

10. Charge concentration

$$l_b = \frac{B_{max}^2}{2} = \frac{0.83^2}{2} = 0.35 \text{ kg/m}$$

11. Height

$h_b = 1.3 \times B_{max} = 1.3 \times 0.83 = 1.08$ approx. 1.10 m

12. Weight

$Q_b = l_b \times h_b = 0.35 \times 1.10 = 0.39$ kg

The bottom charge may consist of 4 cartridges of Dynamex M, 22×200 mm with a weight of 0.1 kg each = 0.4 kg.

The practical height of the bottom charge, $h_b$, will be 0.8 m.

Column charge.

13. Weight

$Q_c = Q_{per} - Q_b = 0.65 - 0.40 = 0.25$ kg

14. Stemming

$h_o = B = 0.72$ m

15. Height

$h_c = H - h_b - h_o = 2.90 - 0.80 - 0.72 = 1.38$ m

16. Charge concentration

$$l_c = \frac{Q_c}{h_c} = \frac{0.25}{1.38} = 0.18 \text{ kg/m}$$

The concentration of the column charge should be at least 40 % of the concentration of the bottom charge, which is found to be the case.

The column may be charged with 2 cartridges of Gurit 17×500 mm with a cartridge weight of 0.115 kg each. Total weight 0.23 kg.

The total length of the bottom charge, 0.8 m, and the column charge, 1.0 m, will leave a stemming length of 1.1 m which may cause some boulders from the upper part of the round.

17. Total charge

$Q_{tot} = Q_b + Q_c = 0.41 + 0.23 = 0.64$ kg

18. Specific drilling

$$b = \frac{n \times H}{w \times B \times K} = \frac{15 \times 2.90}{12.0 \times 0.72 \times 2.5} = 2.01 \text{ m/cu.m.}$$

19. Specific charge

$$q = \frac{n \times Q_{tot}}{w \times B \times K} = \frac{15 \times 0.64}{12.0 \times 0.72 \times 2.5} = 0.44 \text{ kg/cu.m}$$

Summary of important data:

| Bench height $K$ (m) | Hole depth $H$ (m) | Burden $B$ (m) | Spacing $S$ (m) | Bottom charge $Q_b$ (kg) | Column charge $Q_c$ (kg) | Specific drilling $b$ (m/cu.m) | Specific charge $q$ (kg/cm.m) |
|---|---|---|---|---|---|---|---|
| 2.5 | 2.9 | 0.72 | 0.86 | 0.40 | 0.23 | 2.01 | 0.44 |

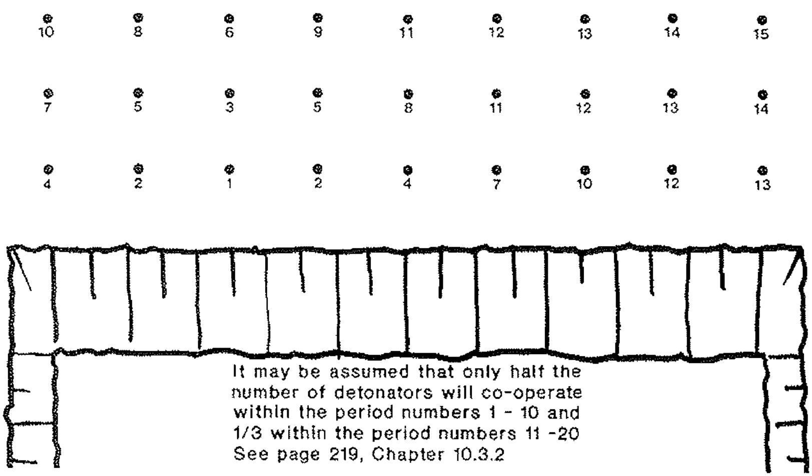



*Fig. 10.20 Firing pattern.*

### 10.3.4 Bench blasting with divided charges.

In blasting operations, it is common that the drilling is carried out well ahead of the blasting operation, with the result that the drilling pattern is fixed and cannot be changed if it is found that the requisite charge for the blasthole is higher than the permitted one.

In cases like this, the charge may be divided into two or more smaller charges in the hole which are shot with different period numbers. The upper charge must then always be initiated with the lower period number.

An intermediate sand stemming divides the charges from each other to avoid flash-over between the charges. An explosive's susceptibility to flash-over depends on parameters like age of the explosive, temperature, charge diameter, quality and length of the stemming. The length of the stemming needed between charges varies from 0.4 m for drill series 11 (34–26 mm) to 2.0 m for a blasthole diameter of 150 mm. Too long intermediate stemming could result in more difficult breakage for the lower bottom charge resulting in higher vibration values. The best stemming material has a particle size of 1/10 of the blasthole diameter (for diameters up to 100 mm).

**Charge calculation procedure.**

**Drilling pattern.**

1. Maximum burden.
$B_{max}$ depends on $Q_{per}$ and is found in table $P_1$.

2. Charge concentration.
$l_b$ depends on $B_{max}$ and is found in table $P_1$. Chose suitable exploisives units considering the $l_b$.

3. Subdrilling.

$$U = 0.3 \times B_{max} \qquad (m)$$

4. Hole depth.

$$H = a(K+U) \qquad (m)$$

a = 1.05 for hole inclination 3:1 and 1.0 for vertical holes.

5. Error in drilling.

$$E = \frac{d}{1000} + 0.03H \qquad (m)$$

TABLE $P_1$

Determination of $B_{max}$ and $l_b$ with regards to $Q_{per}$.

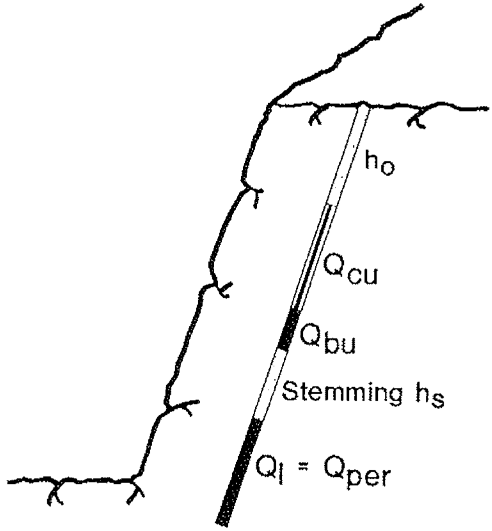
| Permitted charge kg | Maximum burden m | Charge concentration kg/m |
|---|---|---|
| $Q_{per}$ | $B_{max}$ | $l_b$ |
| 0.25 | 0.7 | 0.25 |
| 0.5 | 0.9 | 0.4 |
| 1.0 | 1.2 | 0.7 |
| 1.5 | 1.35 | 0.9 |
| 2.0 | 1.5 | 1.1 |
| 2.5 | 1.6 | 1.25 |
| 3.0 | 1.7 | 1.4 |
| 4.0 | 1.85 | 1.7 |
| 5.0 | 2.0 | 2.0 |
| 6.0 | 2.1 | 2.2 |
| 7.0 | 2.2 | 2.4 |
| 8.0 | 2.3 | 2.7 |
| 9.0 | 2.4 | 2.9 |
| 10.0 | 2.5 | 3.1 |
| 12.0 | 2.65 | 3.5 |
| 14.0 | 2.8 | 3.9 |
| 16.0 | 2.9 | 4.2 |
| 18.0 | 3.0 | 4.5 |
| 20.0 | 3.15 | 5.0 |
| 25.0 | 3.4 | 5.8 |

6. Practical drilling pattern.

Practical burden:
$B = B_{max} - E$ (m)

Practical spacing:
$S = 1.25 \times B$ (m)

The hole spacing is adjusted to the width of the round.




**Charging.**

Lower partial charge.

7. Weight

$Q_l = Q_{per}$ (kg)

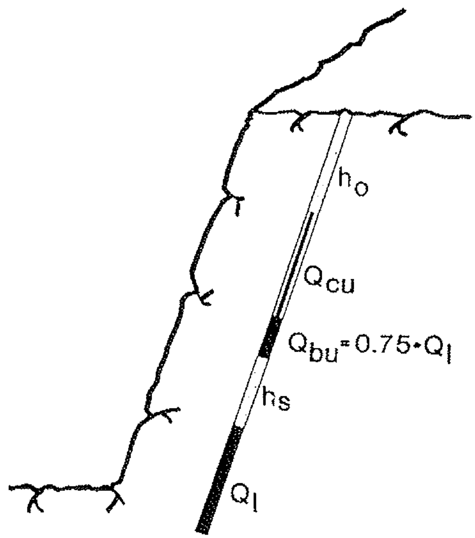
8. Height
$h_l = 1.3 \times B_{max}$ (m)

9. Length of the intermediate stemming, $h_s$, is between 0.4 and 2.0 m depending on the hole diameter.
Approx. 15 times the charge diameter.

Upper partial charge.

10. Residual chargeable length of blasthole.

$h_r = H - h_l - h_s$ (m)




Upper bottom charge.

11. Weight.

$Q_{bu} = 0.75 \times Q_l$ (kg)

Less bottom charge is required as the charge has free breakage which is not the case in the lower charge.

12. Height.

$h_{bu} = 1.3 \times B_{max}$ (m)

Upper column charge.

13. Stemming part.

$h_o = B$ (m)

The length of the stemming may be adjusted depending on the charge concentration in the column.

14. Height.

$h_{cu} = h_r - h_{bu} - h_o$ (m)

15. Concentration of column charge.

$$l_{cu} = \frac{Q_{per} - Q_{bu}}{h_{cu}} \quad \text{(kg/m)}$$

Judge if the calculated charge concentration in the column is sufficient in relation to the bottom charge. It should be at least 40 % of the concentration of the bottom charge. If the remaining charge weight is not large enough to obtain an acceptable charge concentration in the column, a third charge must be used in the hole.

16. Total charge weight – upper partial charge.

$Q_u = Q_{bu} + Q_{cu}$ (kg)

Check that $Q_u \leq Q_{per}$.

### 10.3.5 Bench blasting with divided benches.

The blasting of the area closest to a building often means that the methods of reduced drilling pattern or divided charges will not suffice, but the bench heights have to be reduced.

Over short distances, under 5 m, the ground vibrations are only slightly damped so the values in the charge/distance graph should be followed. The blastings must also be continuously followed up with ground vibration measurement. Any change of the vibration velocity must be taken into account for the planning of subsequent blasts.

Faults and incompetent zones may cause unexpected problems in the immediate vicinity of buildings by displacement of surface rock or gas expansion under the building.

The rounds close to the building should have free breakage to avoid upward movement of the surface rock. The blastholes closest to the building should have weak column charges which cut off the rock thus preventing backbreak.

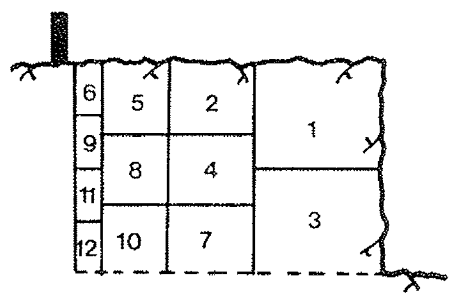



*Fig. 10.21 Blasting order close to buildings.*

### 10.3.6 The slot drilling method.

Lately SKANSKA (Major Swedish contractor) has patented a method of reduc-

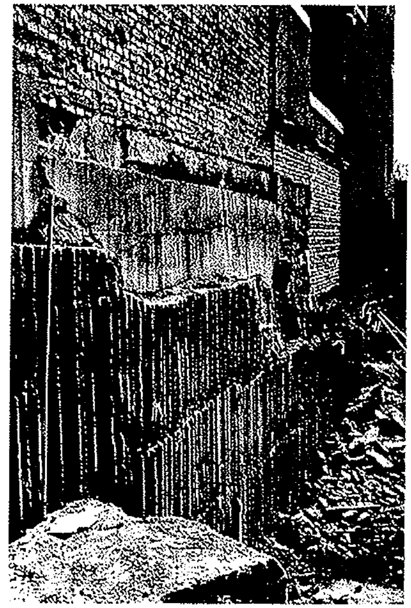


*Fig. 10.22 The SKANSKA slot drilling method.*

ing ground vibrations from blasting close to existing buildings.

A slot is drilled which separates the building from the blasting site. The slot consists of holes drilled in parallel to form a fully open slot. The slot must be free from drill cuttings and water and extend a certain distance outside the object to be protected to give the best result.

Ground vibration measurements have shown that the slot acts as an effective damper of ground vibrations. This implies that the drilling and blasting costs can be reduced, as more effective drilling and charging patterns can be used.

Furthermore, the method may imply further savings as the need to reinforce the rock is reduced or in certain cases eliminated.

**10.3.7 Trench blasting with reduced burden.**

A lot of today's trench blasting is done in populated areas and consequently in the immediate proximity of buildings.

Due to the increased constriction of the rock in trench blasting, the ground vibrations increase and consequently the risk of damage. Therefore, it is of the utmost importance that the blasting operation is planned and executed in a scrupulous way.

The charge calculations for cautious trench blasting will generally follow the same pattern which is used for bench blasting. As in bench blasting, the basis for the calculation is the required specific charge in kg/cu.m.

**Charge calculations procedure.**

**Drilling pattern.**

1. Number of holes in each row, n, is found in the drilling and charging tables for trench blasting, Chapter 6.

2. Permitted charge is found in charge/distance graph and the charge per row is:

$$Q_{row} = n \times Q_{per} \qquad \text{(kg)}$$

3. Hole depth.
H (m) is found in the drilling and charging table for trench blasting, Chapter 6.

4. Specific charge.
q (kg/cu.m.) is found in graph $R_1$.

5. Practical burden.

$$B = \frac{Q_{row}}{q \times H \times w} \qquad \text{(m)}$$

**Charging.**

6. Bottom charge.
$Q_b$ in accordance with graph $R_2$

7. Height of bottom charge

$$h_b = \frac{Q_b \times 1000}{d^2} \qquad \text{(m)}$$

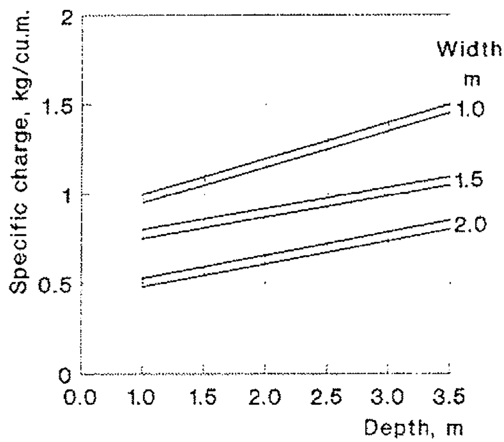
8. Height of stemming

$$h_o = B \qquad \text{(m)}$$

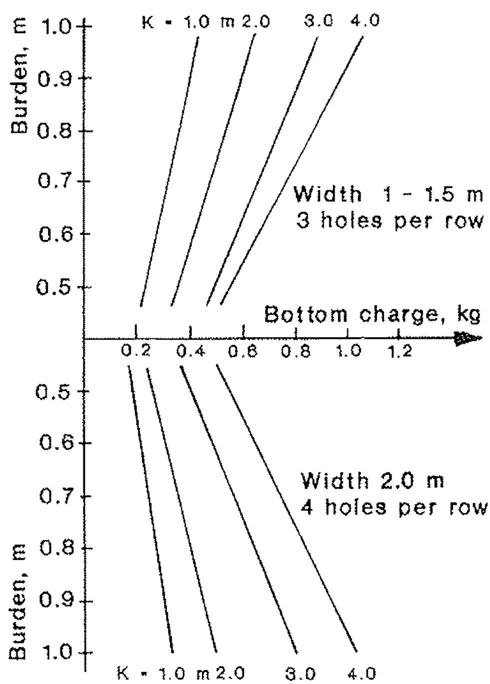
$h_o$ should be adapted to the burden B and the concentration of the column charge. From a ground vibration point of view it is favorable to have short stemming (higher column charge) on condition that throw can be controlled by covering the blast and that the required charge concentration is obtained.

GRAPH R1

Specific charge as function of depth and width.




GRAPH R2



9. Weight of column charge

$Q_c = Q_{per} - Q_b$ (kg)

10. Height of column charge

$h_c = H - h_b - h_o$ (m)

11. Concentration of column charge

$$l_c = \frac{Q_c}{h_c} \quad \text{(kg/m)}$$

12. Weight of column charge

$Q_c = l_c \times h_c$ (kg)

13. Total charge weight

$Q_{tot} = Q_b + Q_c$ (kg)

Check that $Q_{tot} \leq Q_{per}$.

TABLE $R_3$

Recommended charge concentration of the column charge in relation to the practical burden in trench blasting.

| Practical burden B (m) | Minimum required charge concentration $l_c$ (kg/m) | Suitable explosive | Real charge concentration $l_c$ (kg/m) |
|---|---|---|---|
| 0.3 | 0.05 | Gurit, 11×460 mm | 0.08 |
| 0.4 | 0.07 | Gurit, 11×460 mm | 0.08 |
| 0.5 | 0.1 | Gu 11* + 1/4 Em 150* | 0.12 |
| 0.6 | 0.15 | Gu 11* + 1/2 Em 150* | 0.15 |
| 0.7 | 0.18 | 1/4 cartr. Em 150* + 10 cm wooden stick | 0.20 |
| 0.8<br>0.9 | 0.22<br>0.25 | 1/2 cartr. Em 150* + 10 cm wooden stick | 0.25<br>0.25 |

Gu 11* = Gurit, 11×460 mm

Em 150* = Emulite 150, 25×200 mm.

### 10.3.8 Trench blasting with divided charges.

The method of using divided charges in trench blasting can be used when the hole depth exceeds 2.0 m. This is because the intermediate stemming and the normal stemming occupy a minimum length of 1.0 m.

**Charge calculation procedure.**

**Drilling pattern.**

1. The number of holes in each row is found in the drilling and charging tables for trench blasting.

2. Hole depth.
H (m) is found in the drilling and charging tables for trench blasting.

3. Practical burden.
B (m) is found in graph $R_2$ if the lower bottom charge is equal to $Q_{per}$.

**Charging.**

Lower bottom charge.

4. Weight.

$Q_{bl} = Q_{per}$ (kg)

5. Height.

$$h_{bl} = \frac{Q_{bl} \times 1000}{d^2} \quad \text{(m)}$$

6. Height of intermediate stemming.
$h_s$ = 0.4 to 1.0 m. The value depends on the circumstances. e.g. the blasthole diameter. Approx. 15 times the charge diameter.

Upper partial charge.

7. Residual chargeable height of the blasthole.

$h_r = H - h_{bl} - h_s$ (m)

Upper bottom charge.

8. Only 60 % of charge $Q_{bl}$ is needed for the breakage of the upper part as the hole has free breakage, i.e. no constriction.

$Q_{bu} = 0.6 \times Q_{bl}$ (kg)

9. Height.
$h_{bu}$ is estimated from the chosen charge unit.

Upper column charge.

10. Charge concentration.

$l_{cu}$ from table $R_3$ (kg/m)

11. Stemming.

$h_o \geq B$ (m)

Adjusted to the charge concentration in the column of the hole.

12. Height.

$h_c = h_t - h_{bu} - h_o$ (m)

13. Weight.

$Q_{cu} = l_{cu} \times h_{cu}$ (kg)

14. Total charge weight.
Upper partial charge.

$Q_u = Q_{bu} + Q_{cu}$ (kg)

Check that $Q_u \leqslant Q_{per}$.

**10.3.9 Cautious tunnel blasting.**

An increasing number of tunnels are being constructed under built-up areas where they pass under inhabited buildings as well as buildings with equipment sensitive to ground vibrations.
Cautious blasting followed up with ground vibration measurement has subsequently become more common.
The blastholes in a tunnel round are very constricted. To decrease ground vibrations, it is necessary not only to lower the co-operating charge, but also to endeavor to reduce constriction of the rock. This means that drilling pattern, hole depth, charge per hole and firing pattern have to be adjusted so that the permitted co-operating charge is not exceeded and that all holes have free breakage.
Some of the most important points to consider are:

- Choice of blasthole diameter and explosive.
- Choice of large hole diameter, one or several large empty holes in the cut to decrease constriction and the risk for flash-over.
- Accuracy in drilling.
- Suitable firing pattern which minimizes the co-operating charge and guarantees most favorable angle of breakage.
- Dividing the round into partial rounds.
- Reduction of the distance between the holes so the charge in each hole can be reduced.
- Reduction of the hole depth.

The firing pattern is of the utmost importance in cautious tunnel blasting. The constriction of the blastholes can be decreased by using the right period number, so that each hole has an angle of breakage of at least 90° in the stoping part of the round (See Fig. 7.15 in Chapter 7 Underground blasting). If the tunnel round is of such a size that the number of periods does not suffice without exceeding the permitted co-operating charge, the blast must be divided into two or more partial blasts e.g. blast the constricted cut holes as a separate blast, then the stoping holes and finally the contour holes. In order to obtain the best result in the

contour, the perimeter holes (except the floor holes) should be blasted with the same period number. This is normally no problem as the contour holes usually have very low charge concentration.
In cautious blasting, certain types of cuts like V-cuts are not suitable because of the risk of co-operation and flash-over between the large number of holes in the cut. Large hole cuts have earlier been considered to give rise to large ground vibrations but measurements of ground vibrations analyzed over a long period contradicts this. A parallel hole cut with two or more large holes is a very practical way of reducing constriction and unsuccessful breakage and is thus to be recommended.
It is often more advantageous to drill more holes in a round with reduced charges than to shorten the hole depth, thus maintaining normal advance of the round. However, sometimes the hole depth has to be reduced in order to keep the co-operating charge within the permitted limits.
A continuous follow-up of the blasting activities by ground vibration measurement may be beneficial by disclosing more favorable practical ground vibration values than those determined by theoretical calculations.
The adaptation of the blasting operation to the measured results means an optimum rate of drifting on the basis of vibration measurement.
Problems with flyrock and air shock waves do occur in the initial stage of the tunneling operation and constitute a risk when the work is started in populated areas, which is often the case nowadays. Therefore, it is important to investigate the rock with regard to fissures and incompetent zones and than cover the blast well. The air shock wave is troublesome not only in the initial stage of the work but also when the drifting has advanced further into the rock, especially in the direction of the tunnel.

## 10.4 Blasting close to hardening concrete.

Blasting works are often carried out simultaneously with construction work which give rise to problems with blasting close to hardening concrete.
The problem has been studied by the Ontario Hydro, Concrete and Masonry Research Section and the following recommendations are given for concrete with STD cement, without entering too deeply into theories and research results.

If it is presumed that concrete which hardens in a temperature of +5° C can stand a peak particle velocity of 100 mm/sec. after 90 days, the following vibration velocity values are recommended:

| Hardening time days | Maximum permitted vibration velocity mm/sec |
|---|---|
| 2 | 8 |
| 3 | 11 |
| 7 | 35 |
| 28 | 80 |
| 90 | 100 |

Note: Up to 10 hours after casting, the concrete can stand ground vibrations of up to 100 mm/sec. Between 10 and 70 hours after casting no blastings should be undertaken closer than 30 meters.

On the other hand, if it is presumed that the concrete is hardening in a temperature of +21° C and that the concrete stands 100 mm/sec of ground vibration after 90 days, the following is recommended:

| Hardening time days | Maximum permitted vibration velocity mm/sec |
|---|---|
| 1 | 14 |
| 2 | 30 |
| 3 | 40 |
| 7 | 60 |
| 28 | 85 |
| 90 | 100 |

Note: The concrete can stand ground vibrations of up to 100 mm/sec up to 5 hours after casting. No blastings should be undertaken closer than 30 meters between 5 and 24 hours after casting.

## 10.5 Flyrock.

Cautious blasting does not only mean the control of ground vibration but also the control of flyrock.
The control of flyrock and its prevention has been dealt with thoroughly in Chapter 5.8 Throw, flyrock.

## 10.6 Air shock waves.

The immediate effect of blasting is not only to cause ground vibrations and throw, but also an air shock wave.
In most routine blastings, in which the explosives are enclosed in blastholes, and which are designed for ground vibration velocities of 70 mm/sec or less, the blasting does not cause air shock waves of the magnitude that may cause damage to buildings.
However, a low level of air shock wave overpressure does play an important role in distressing neighboring residents by rattling windows etc. Therefore, complaints may be reduced by taking actions to reduce overpressure from air shock waves.

Air shock waves are pressure waves which radiate in the air from a detonating charge. The intensity of the pressure depends on the size of the charge and on its degree of confinement. When a pressure wave passes a given position, the pressure of the air rises very rapidly to a value over the ambient atmospheric pressure. It then falls relatively slowly to a pressure below the atmospheric value before returning to the atmospheric pressure after a series of oscillations. The maximum pressure is known as the peak air overpressure. The air shock waves are within a wide range of frequencies, typically between 0.1 Hz and 200 Hz. In the portion of the spectrum lying over 20 Hz the air shock waves are audible and known as noise, while concussion is the portion under 20 Hz and inaudible.

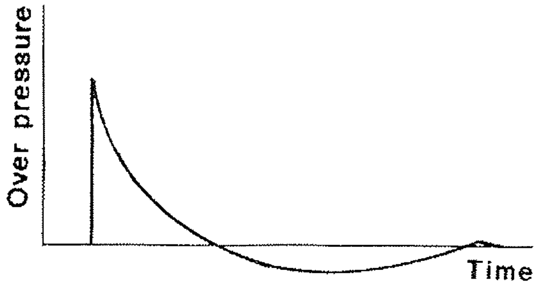



*Fig. 10.23 Pressure/time curve for air shock wave.*

The lower, inaudible, frequencies are damped more slowly than the higher, audible, frequencies and cause overpressure over greater distances. These low frequencies can occasionally cause direct damage onto structures, but can more commonly induce higher frequency vibrations which are noticed as noise in windows, doors, crockery etc. Under such circumstances it is impossible to determine whether the ground vibration or air shock wave is being perceived without monitoring the blast.

The air overpressure is measured as units of pressure and usually pressure unit millibar (mbar) is used. The units decibel (dB) and kilopascal (kPa) are also used.

The decibel unit is expressed as:

$$dB = 20 \log \frac{P}{Po}$$

where P is the measured pressure and Po the reference pressure of 0.00002 Pa.

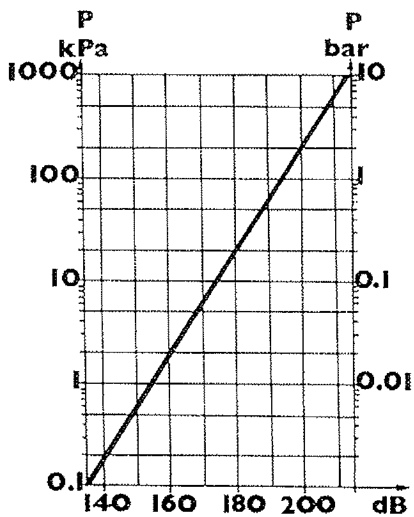



*Fig. 10.24 Relation kPa/dB.*

Knowing the charge weight Q (kg) and the distance R (m) to the charge, the overpressure can be calculated from the formula:

$$P = 700 \frac{Q^{1/3}}{R} \text{ (mbar)}$$

The relationship applies to TNT, which means that for civil explosives type Emulite 150 and Dynamex M the charge weight should be reduced by 25 % when used in the formula.

The relationship applies to unconfined charges.

The unconfined charges which cause problems in populated areas are concussion charges (mudcapping), trunklines of detonating cord, welding of powerlines with explosives, presplitting with unstemmed holes etc. As can be seen in Fig. 10.25, a trunkline consisting of 100 m 10 gr detonating cord can cause broken windows at a distance of up to 100 m.

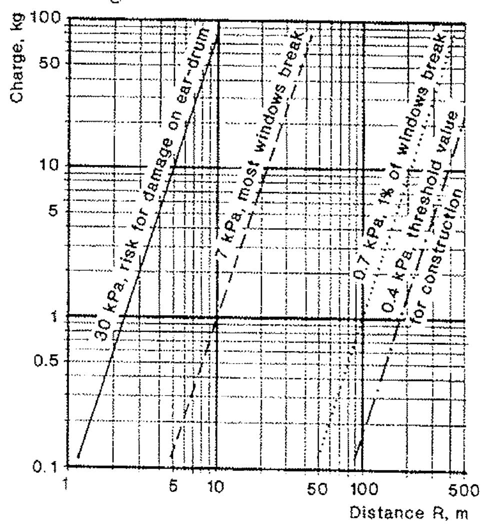



*Fig. 10.25 Charge as a function of distance for different levels of air overpressure.*

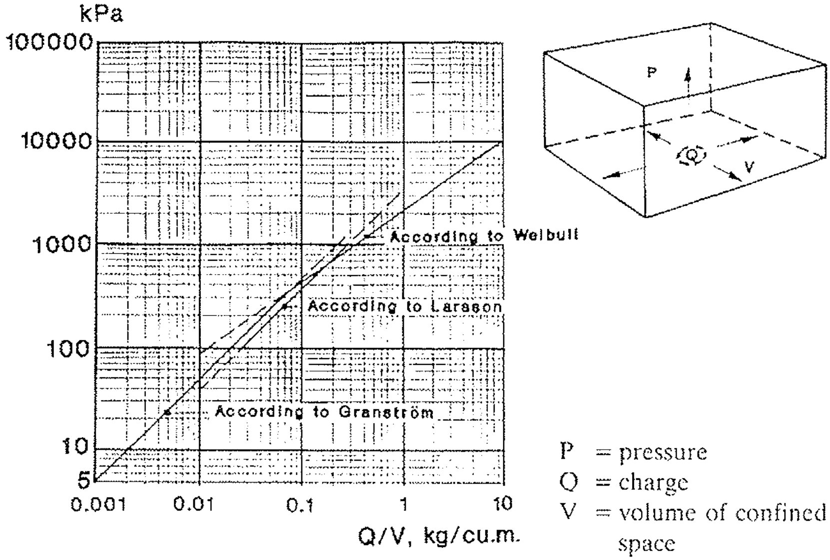
The propagation of the air shock waves is influenced by atmospheric conditions where the wind direction, wind velocity, temperature and air pressure have a great effect.

Reflexions in the atmosphere may be caused by temperature inversion, where the air shock wave is reflected against the boundary layer of air strata with different temperatures. Temperature inversion frequently occurs on cloudless evenings, nights and mornings. The phenomenon can cause local amplification of the air overpressure, which is greater than that which would normally have been expected at a certain distance.

Even if the air overpressure is kept under the threshold value for buildings (0.4 kPa), it is not always sufficient to safeguard against complaints. The blasts should therefore be designed for the minimum practical level.

**Air overpressure in confined spaces.**

In the case of blasting in underground chambers and tunnels, different conditions prevail as the pressure wave is confined and in the case of tunnels concentrated in one direction. This means that the pressure is amplified compared to blasts in an open space.




P = pressure
Q = charge
V = volume of confined space

*Fig. 10.26 Pressure as a function of charge and volume from a detonation in a confined space.*

The principal sources of air overpressure are:

- Detonation of unconfined charges. The most common are concussion charges (plaster charges), trunklines of unconfined detonating cord, blastwelding of powerlines and presplitting with unstemmed holes.
- Too short stemming and/or wrong stemming material. Inadequate stemming might not confine the explosive on detonation.
- Venting of high velocity gases may occur in poorly designed blasts where no consideration has been given to incompetent zones, principally in the burden area. Overcharging of a blasthole could cause the same effect.
- The sudden movement of the blasted rock mass towards the free face or faces will raise the air pressure.

In order to control the air shock waves, the following steps should be considered:

- Design the blast in such a way that the amount of explosives is in accordance with blasting requirements and minimum air overpressure.
- Pay particular attention to incompetent zones, overbreak from previous round, mudseams etc. through which gases may vent and cause overpressure.
- Accurate drilling is necessary to maintain the designed blasting pattern. Too big a burden could cause venting in the collar part of the hole. Use setbacks to determine the burden of the next round.
- Bottom initiation decreases venting in the stemming area. See Chapter 5.8 Throw, flyrock.

- Reduction of the size of the round tend to reduce the air overpressure.
- If possible, the development of the benches should be such that the blasted material is thrown away from residential areas.
- The stemming material should be of sufficiant quantity and quality to confine the explosives on detonation. Crushed stone material size 4 to 9 mm gives better confinement than drill fines.
- Check the rise of the explosives column during charging to minimize the risk of overcharging in any void or fault.
- Avoid excessive delays between holes to prevent underburdening the holes.
- In multiple row blasting the delay between the rows should be longer then the delays between the holes in the row. In deep rounds, this promotes forward rather than upward movement of the burden.
- Do not use concussion charges in populated areas for secondary blasting and boulder blasting.
- If misfired underburdened holes have to be fired, use screening materials e.g. sandbags or loose sand to cover. The thickness of the cover has to be sufficient both to avoid flyrock and to damp the air shock wave.
- Surface lines of detonating cord should be avoided in residential areas. If electric firing is not allowed or possible the non-electric firing system NONEL should be used. If detonating cord is the only firing device available, trunk-lines and connecting lines should be covered with at least 600 mm of absorbent material, preferably sand.
- As speed and direction of the wind are major influences on the magnitude of air overpressure, blasting should, if possible be avoided when the wind is blowing towards critical areas.
- Blasting should be avoided in early mornings, late afternoons and evenings when temperature inversions are likely to occur.
- Schedule blasts to times when the noise level from surrounding sources is at its highest and when the neighbors are busy or expect blasting to occur.
- Employ an audible warning system immediately before every blast. If the blasting is an isolated occurrence, give specific warning indicating approximate time of the blast.
- Maintain good public relations. Give good and adequate information about the work, duration and disturbances to be expected. The most stringent measures against air overpressure can be rendered useless without good relationship between the blasting crew and the neighbors.

# 11. UNDERWATER BLASTING

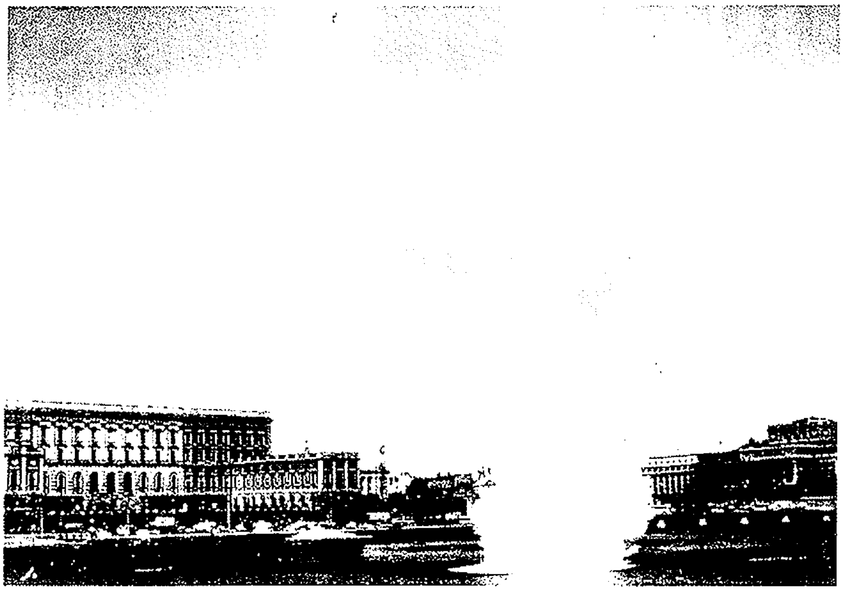


*Fig. 11.1 Underwater blasting in central Stockholm.*

## 11.1 General.

Underwater blasting includes removal of rock which is fully or partly covered by water.

The underwater blasting operation requires greater care and more thorough planning than similar operations above water. Both drilling and charging become more complicated and some factors which have to be considered for the successful underwater blasting operation are:

* special operation methods and drilling equipment
* different charging methods
* higher powder factor to displace rock, overburden and water
* use of explosives with good underwater properties
* use of safe and reliable initiation system
* keeping vibrations and water shock waves under control by using the right products and delay pattern.

## 11.2 Drill and blast methods.

The most common methods presently in use for underwater blasting are:

* drilling and blasting through rockfill
* drilling and blasting from platform
* drilling and blasting with divers
* blasting with concussion charges

Drill and blast methods.

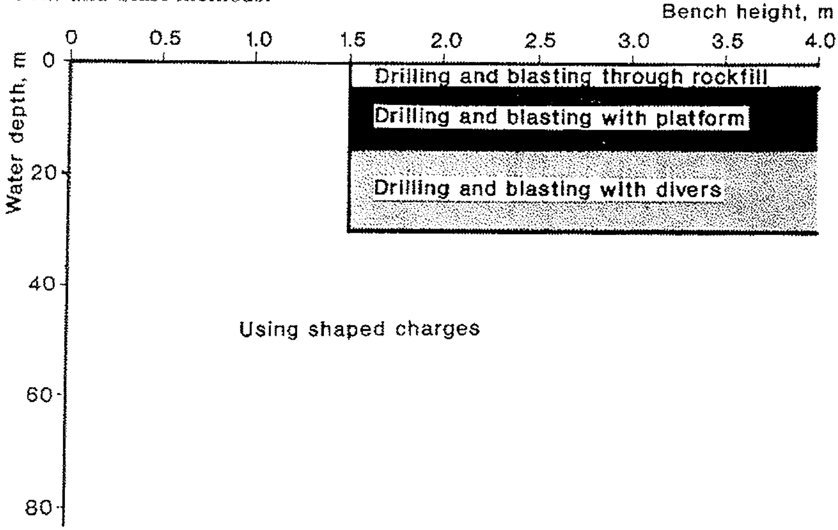



*Fig. 11.2 Economic bench heights and water depths for different methods.*

### 11.2.1 Drilling and blasting through rockfill.

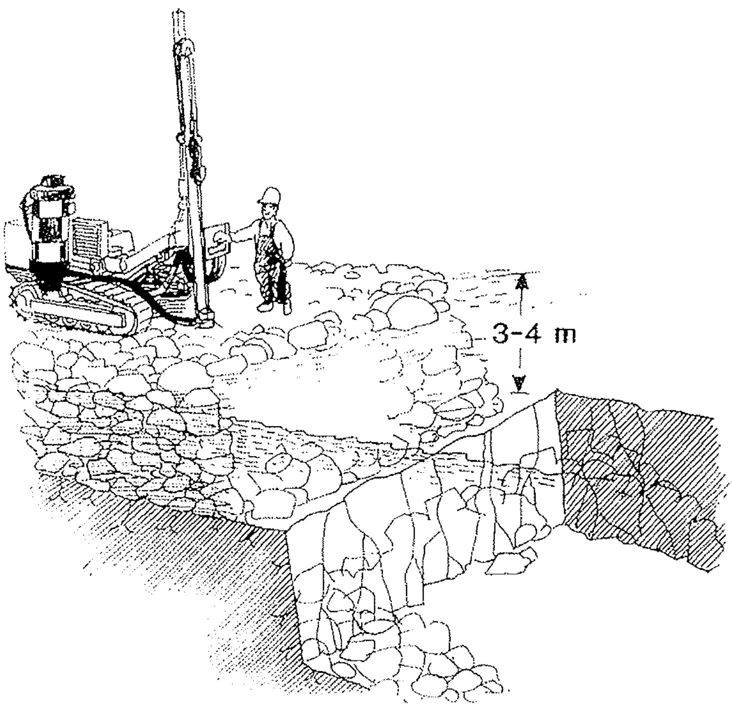



*Fig. 11.3 Drilling and blasting through rockfill.*

Where the water is not too deep, it may be economically advantageous to make a rockfill over the area to be blasted and drill and charge through the rockfill. The drilling and charge calculations are presented in Chapter 11.2.3 Charge calculations.

### 11.2.2 Drilling and blasting from platform.

Special equipment is required for the drilling and blasting of rock with this method. Usually a floating platform, supported above water with legs that can be extended down to the bottom, is used for the drilling and charging operations. The platform is normally equipped with several percussion or rotary drilling machines, which can be moved on tracks in four directions for greater flexibility.

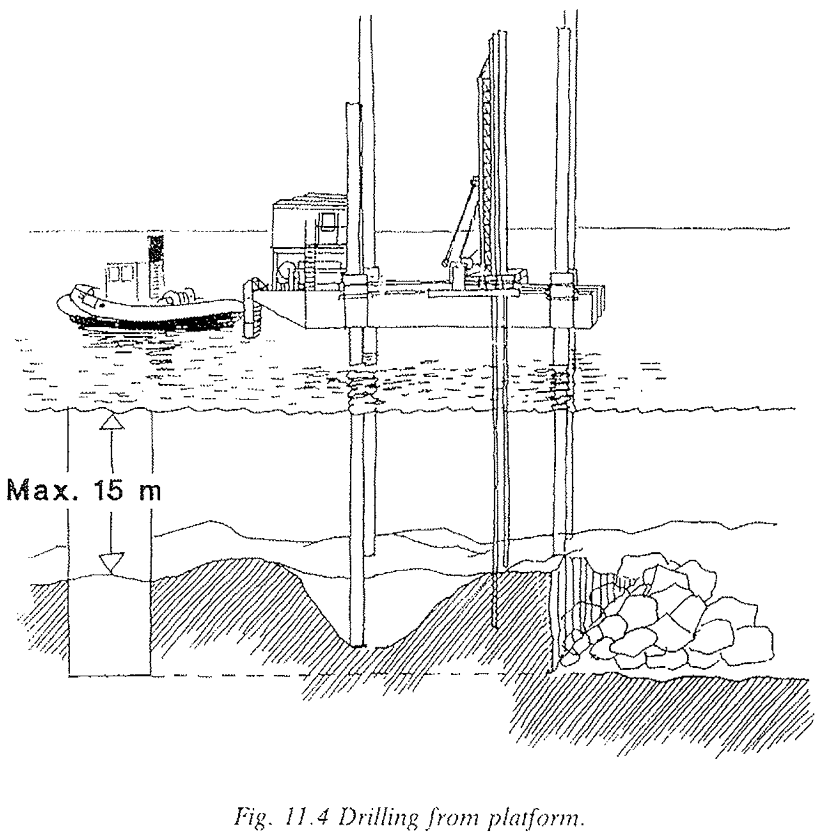



*Fig. 11.4 Drilling from platform.*

Where overburden covers the rock the following methods are widely used:

* OD method (OD – Overburden Drilling)
* Kelly bar method
* Charging through the drill rod.

**The OD method.**

The drilling is carried out by using powerful drilling equipment. The blastholes diameter can vary between 51 and 102 mm, but larger holes are occasionally drilled.
A set of steel tubes with ringbits is used to drill through the overburden and a few decimeters into the bedrock. (In case of underwater blasting without overburden, the tube is drilled into position at the collaring point in the same way.) Drilling is then carried out through the tube with extension rods and a drill bit. When the hole is completed, the set of drill rods is withdrawn and a plastic tube is inserted through the steel tube into the collar of the drill hole. The drilling tube is then withdrawn and the plastic tube remains in the blasthole for the charging operation.
Charging is mostly carried out by using pneumatic cartridge chargers, which gives a good tamping of the explosives. Each drill hole contains at least two detonators which are connected in different series to ensure maximum initiation dependability.

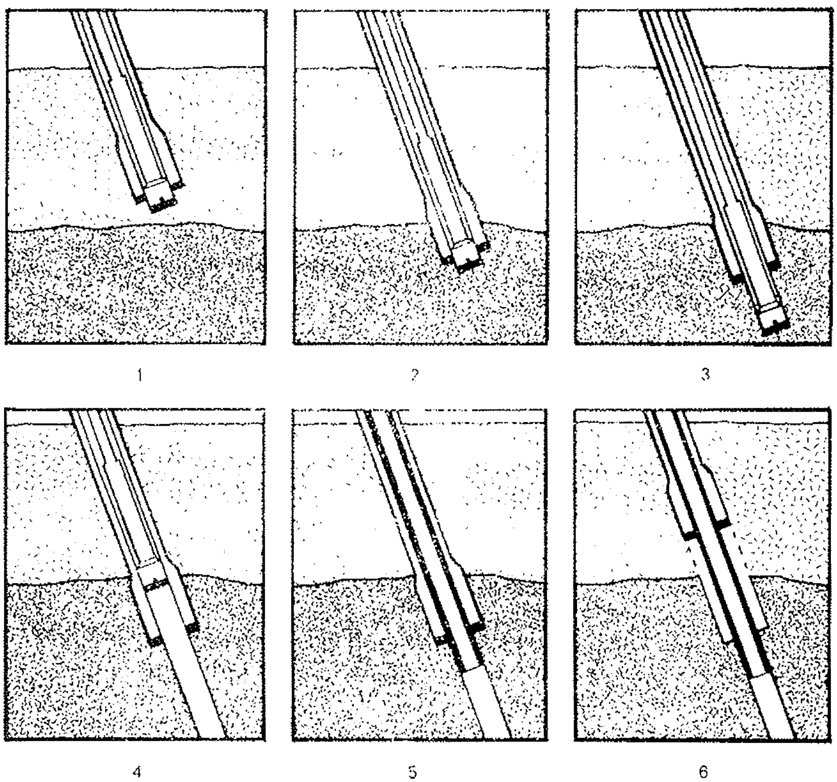



*Fig. 11.5 Principle of Overburden Drilling.*

1. Drilling tube and extension rods penetrate overburden.
2. Drilling tube penetrates a few decimeters into the bedrock.

3. Drill hole to be completed and cleaned.
4. Extension rods are withdrawn.
5. A plastic tube is introduced and fixed into the drill hole.
6. The drill tube is now withdrawn and charging can take place through the plastic tube.

**The Kelly bar method.**

Where overburden overlies the rock, a weighted steel pipe is first lowered from the platform. The diameter of the pipe is slightly larger than that of the drill bit and serves as a guide for the drilling. As drilling advances, the pipe sinks through the overburden to the surface of the rock. It then acts as a casing to keep soil from entering the hole and is used as a guide for the charging of the hole when drilling is completed.
(See Fig. 11.6 below.)

**Charging through the drill rod.**

In this method, which is similar to the Kelly bar method, a supporting leg is lowered from the platform to the overburden, serving as a guide for the drilling. When the drilling is completed, the drill bit is replaced by a ringbit and lowered into the hole. The charging of the hole is then carried out through the drill rod. As the charging proceeds, the drill rod is lifted.
(See Fig. 11.7 below.)

The typical drill hole range for the two methods described above is 102 to 152 mm.

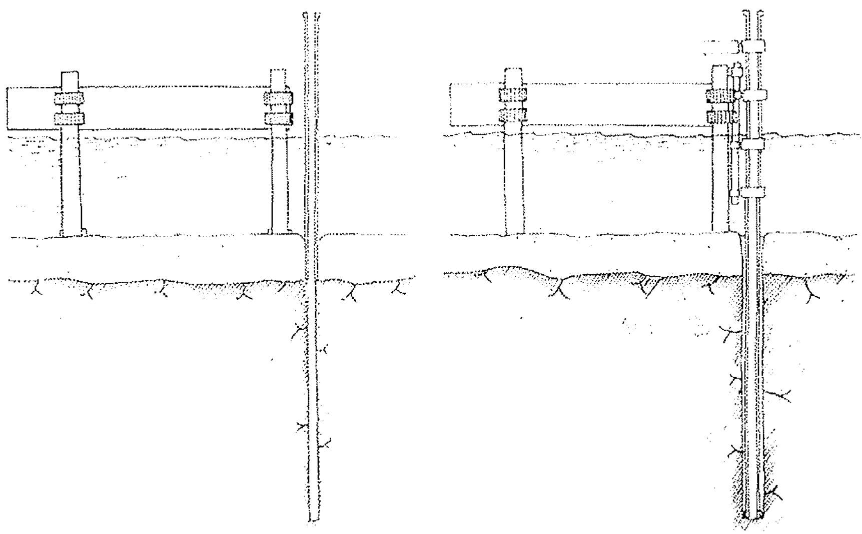


*Fig. 11.6 Kelly bar method.* *Fig. 11.7 Charging through drillrod.*

### 11.2.3 Charge calculations

The most important points in underwater blasting are to ensure good fragmentation and avoid stumps above the stipulated bottom.
Normal bench blasting requires a specific charge of approximately 0.50 kg/cu.m. (Emulite 150 or Dynamex AM) to ensure good fragmentation. Since misfires can be expected in some blastholes, the specific charge is doubled in the case of underwater blasting to 1.00 kg/cu.m., implying that if one of the blastholes does not detonate, but the adjacent holes do, the specific charge remains 0.50 kg/cu.m. in this area.

In the case of vertical holes the specific charge should be increased with approx. 10 % to 1.10 kg/cu.m.

In underwater blasting, the rock movement is obstructed by the water pressure, the weight of the overburden and the weight of the rock itself. To ensure good breakage and displacement of the rock, the specific charge is increased to compensate for these conditions.

The water pressure is compensated for by increasing the specific charge by 0.01 kg/cu.m. per meter water depth ($K_w$).

For rock covered with overburden, the specific charge should be increased by 0.02 kg/cu.m. per meter of overburden ($K_{OB}$).

For the rock section, the compensation is 0.03 kg/cu.m. per meter of the bench height ($K_{rock}$).

The required specific charge should than be (for Emulite 150 or Dynamex AM):

$$q_{incl} = \mathbf{1.00+0.01 \times K_w + 0.02 \times K_{OB} + 0.03 \times K_{rock}}$$
$$q_{vert} = \mathbf{1.10+0.01 \times K_w + 0.02 \times K_{OB} + 0.03 \times K_{rock}}$$

The real specific charge will be even higher due to increased subdrilling.

When the required specific charge is known, the hole spacing may be calculated with regard to the possible charge concentration per meter blasthole. If Emulite 150 or Dynamex AM in paper cartridges are charged with a pneumatic charger or bulk Emulite is used, the charge concentration is found in table 1.a Chapter 5.2 Charge calculations.

$l_b$ = see table 1.a (for the explosive used) (kg/m)

For large diameter blastholes, where large diameter charges are used, the charge concentration is given by the actual unit of charge. See table 1.b Chapter 5.2 Charge calculations.

The drilling pattern in underwater blasting is usually square.

B = S (burden = spacing) (m)

In order to calculate the drilling pattern, the area which each hole will blast is computed by dividing the charge concentration with the required specific charge.

$$\text{Area} = \frac{l_b}{q} \qquad \text{(sq.m.)}$$

The burden and spacing are then:

$$B = S = \sqrt{\frac{l_b}{q}} \qquad \text{(m)}$$

The subdrilling should be at least of the same magnitude as the burden, but no less than 0.8 m.

$$U = B \qquad \text{(m)}$$

The hole depth is the bench height plus the subdrilling.

$$H = K+U \qquad \text{(m)}$$

The uncharged section of the hole should be 1/3 of the burden.

$$h_o = 1/3 \times B \qquad \text{(m)}$$

In deeper water, it is recommended that the holes are charged close to the blasthole collar to ensure displacement. If the holes are not charged close to the rock surface, the top of the rock is merely lifted and returns unbroken to its original position due to the weight of the water.

EXAMPLE

Underwater blasting to be carried out under the following conditions:

| | |
|---|---|
| Blasthole diameter: | 110 mm |
| Water depth: | 12 m |
| Overburden: | 2 m |
| Bench height: | 5 m |
| Vertical drilling. | |
| Explosive: | Emulite 150, 75×550 mm |
| Initiation: | VA/OD MS (milli-second delay) |

Required specific charge:

$q = 1.10+0.01\times12.0+0.02\times2.0+0.03\times5.0$
$q = 1.10+0.12+0.04+0.15=1.41$ kg/cu.m.

The charge concentration is:

$l_b$ = in accordance with table 1.b = 5.3 kg/m.

Area per drill hole = charge concentration/required spec. charge.

$$A = \frac{5.30}{1.41} = 3.76 \text{ sq.m.}$$

$B\times S=3.76$ sq.m.

$B = \sqrt{3.76}=1.94$ m approx. 1.90 m

$B = 1.90$ m

$S = 1.90$ m

Subdrilling = burden

$U = B$

$U = 1.90$ m

Hole depth = bench height + subdrilling

$H = K+U$

$H = 5.0+1.9=6.9$ m

Uncharged section of the hole = $1/3 \times B$

$h_o = 1/3 \times 1.90 = 0.63$ m approx. 0.60 m

Charge = charge concentration × height of charge.

$Q = l_b \times (H-h_o)$

$Q = 5.30 \times (6.90-0.60) = 33.4$ kg

$$\text{Specific charge} = \frac{\text{charge per hole}}{\text{blasted volume per hole}}$$

$$q = \frac{Q}{B \times S \times K}$$

$$Q = \frac{33.4}{1.9 \times 1.9 \times 5.0} = 1.85 \text{ kg/cu.m.}$$

SUMMARY OF IMPORTANT DATA.

| Bench height K | Hole depth H | Burden B | Spacing S | Charge Q | Specific charge q |
|---|---|---|---|---|---|
| 5.00 m | 6.90 m | 1.90 m | 1.90 m | 33.4 kg | 1.85 kg/cu.m. |

The charge calculations are based on the assumption that $K \geqslant 2B$. (The bench height is greater than twice the burden.)
On lower benches it may be feasible to increase the subdrilling and keep the same hole depth and drilling pattern.

**Opening the cut.**

The result of the blasting operation depends on a successful opening of the cut, thus drilling and charging must be carried out with utmost carefulness.
Often enough there is no free face against which the cut can be opened. Therefore, the first rows have to be drilled with a denser drilling pattern.
In the case of inclined holes the opening of the cut causes less problem than vertical holes.

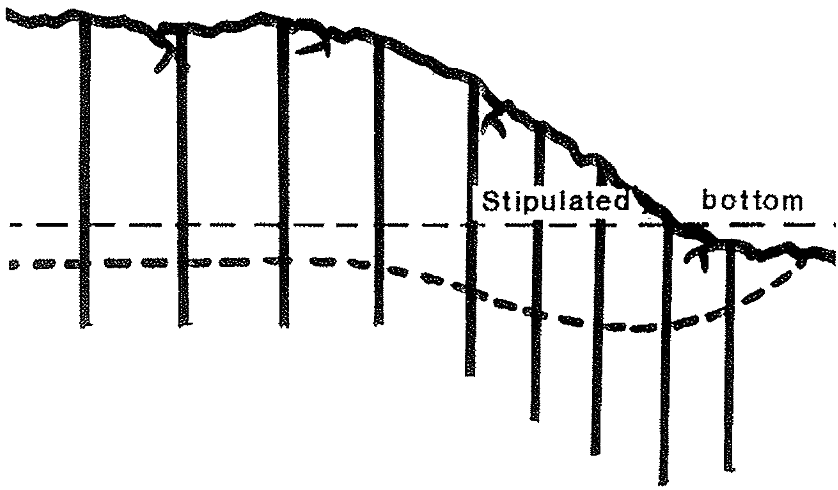



*Fig. 11.8 Opening the cut.*

**11.2.4 Explosives.**

Explosives for underwater blasting should have high strength, good water resistance and should retain their sensitivity when subjected to hydrostatic pressure. Emulite 150 and Dynamex AM meet these requirements even though less sophisticated explosives can be used if the water depth does not exceed 10 meters and the time of exposure to water is short.
The explosive must give full detonation even when it has been stored under water for a long time as it must be taken into account that more often than not, unpredictable circumstances in underwater blasting, add time to the operation.

Emulite 150 and Dynamex AM are guaranteed to resist water for at least one week.

The explosive may be charged into the blasthole by means of a pneumatic charger, which will result in a very good charge concentration.

Cartridges which are charged with tamping rods give a charge concentration which is more or less equal to the charge concentration of the cartridge.

The best charging result is obtained by pumping Emulite explosives into the blasthole. The Emulite will then fill up the hole completely, giving a very good charge concentration. In rock formations containing fissures and voids, pumpable products should not be used because of explosives migration into the voids, resulting in excessive local charge concentrations.

The low blasthole-to-blasthole propagation characteristics of Emulite is a great advantage in underwater blasting, where propagation between holes must be avoided from the point of view of both fragmentation and vibration.

### 11.2.5 Initiation.

It is imperative that a safe and reliable initiation system is used in underwater blasting. Since the blasting procedure requires millisecond delay blasting for proper breakage and displacement, electric or NONEL detonators are preferred to the use of detonating cord.

Electric detonators, specially designed for underwater blasting, should be used. VA-OD is a detonator with double aluminum capsules and double insulation on the legwires to withstand the considerable stresses to which the firing system is subjected during the charging operation. Two detonators should be used in each hole and be connected in different series. Continuous control of the detonators must be carried out during the charging of the hole and damaged detonators replaced.

In underwater blasting, water must be considered an electrolytic solution with current leakage as a hazard to successful operation. Thus, only detonators with good insulation should be used. If possible all connection work should be carried out above the water surface. If it is necessary to have the connections under water, they must be bedded into a box or sleeve which is absolutely impermeable to water; ordinary connecting sleeves or insulating tape are insufficient. It is advisable to make the detonator legwires long enough to allow for a number of detonators to be connected at one point. The ideal situation is when the detonators can be connected together above water level i.e. on small floats, ropes etc. As mentioned earlier, continuous control of the initiating system with regard to damage, current leakage and resistance must be carried out to check that the current leakage is on a tolerable level and that there are no broken circuits in the round.

The non-electric initiation system NONEL-OD, is similar to the VA-OD, except that the electric legwires are replaced by a plastic shock tube. (See Chapter 3b.2.4 Nonel.) The detonator has double aluminum capsules and reinforced plastic tube to withstand the rough handling which is normal in underwater charging works. The detonator and the shock tube are a closed system into which water can not penetrate.

The connection of the round has to be carried out with the greatest care, but the disadvantages of the electric initiation are eliminated. NONEL is delivered with the same time delay as VA-OD, 25 ms.

Detonating cord is occasionally used in underwater blasting operations, but the difficulties in obtaining accurate delays between the rows and problems with cutoffs caused by crossing cords, have deterred the blasters from its wider use.

### 11.2.6 Drilling and blasting by divers.

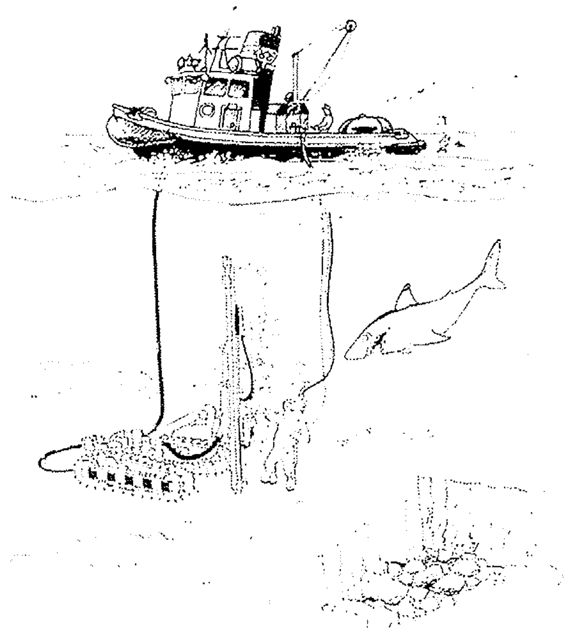


*Fig. 11.9 Drilling and blasting by divers.*

Underwater blasting with the assistance of divers is usually used for limited operations. As the visibility under water is mostly poor when drilling operation is taking place, special measures have to be taken to help the diver to orientate himself so he can drill the holes in the right place, i.e. a grill of steel where the locations of the holes are marked.

The charge calculations are the same as under Chapter 11.2.3 Charge calculations.

### 11.2.7 Blasting with concussion charges.

The FRAGMEX-system has been developed for rock blasting without drilling. The system is particularly adapted for underwater blasting, but is also used in surface applications i.e. boulder blasting and seismic blastings.

The technique of using shaped charges (lined cavity effect) has previously only been used in military applications. The effect of the shaped charge is obtained by directing the energy of the explosive. The best effect is secured if the charge has the shape of a cone with a certain angle in the top. If the cone is internally covered with a metal, the metal will be compressed on detonation creating a projectile of molten metal with a high penetration capability. To give the projectile time to form before it strikes the rock surface, the charge has to be placed at a certain distance from the surface. This distance (stand-off) is often built into the charges.

When shaped charges are used for underwater blasting, the "stand-off" must be kept free from water to obtain the desired effect.

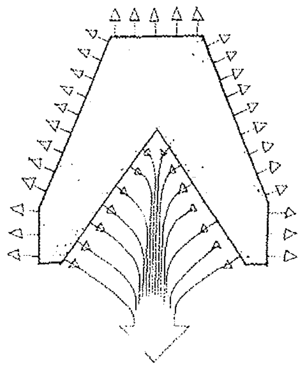


*Fig. 11.10 Shaped charge.*

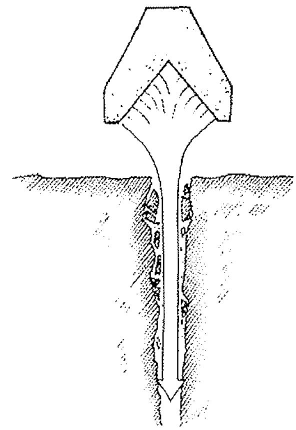


*Fig. 11.11 The effect of shaped charge.*

The FRAGMEX-system is simple to use and the charges are easily placed at the right locations by divers. As the specific weight of the charges are close to that of water, the charges tend to move unless they are weighed down by sinkers. Sinkers can easily be made of concrete with a shape to fit the FRAGMEX charge. Frequently the charges are prepared in metal frames onboard a barge and then placed at the right location by divers.

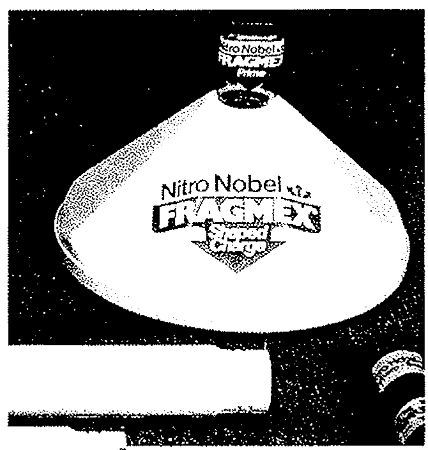



*Fig. 11.12 Fragmex 8 charge.*

The FRAGMEX charges are suitable for bench heights up to 1.5 m and a water depth up to 100 m. The charging pattern with FRAGMEX 8 depends on the rock characteristics, but some guide values will be given:

Hard rocks (granite, basalt, diabase etc.)
Charging pattern: 1.0×1.0 m to 1.5×1.5 m

Softer rocks (limestone, conglomerates etc.)
Charging pattern: 2.0×2.0 m to 2.5×2.5 m

To obtain better knowledge of the blastability of the rock, test blastings are

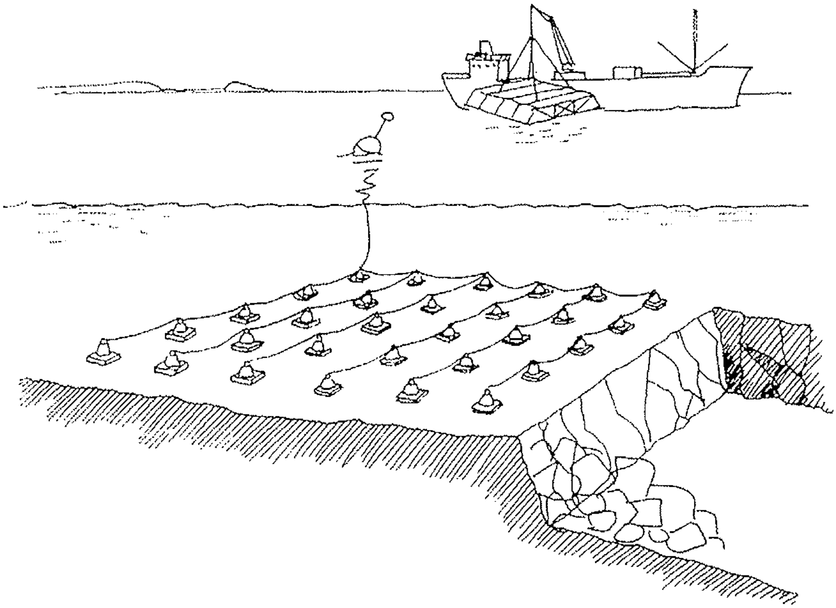


*Fig. 11.13 Charging with FRAGMEX charges.*

recommended. If the rock is covered with overburden, this should be removed before the blastings start.

Caution is recommended if the water depth is less than 2.0 m. Throw from the material used as sinkers tends to occur. Furthermore the air shock wave has to be considered in built-up areas.

The deeper the water depth, the better the effect of the FRAGMEX charges as the confinement becomes better due to the water pressure.

The FRAGMEX charges must be initiated instantaneously. Connection of the charges with detonating cord is the best means of initiation.

For the blasting of occasional boulders or stumps above the stipulated grade, a box (approx. 25 kg) of Dynamex or Emulite can be placed on the object. The charge must be securely anchored using available material as ballast. Note the risk of throw from the ballast in shallow waters. If more than one charge is blasted, the charges should be connected with detonating cord for instantaneous detonation.

## 11.3 Ground vibrations and water shock waves.

In underwater blasting, the ground vibrations are more unpredictable than in ordinary blasting due to the greater risk of propagation between the holes. This makes the estimation of the ground vibrations uncertain as there is no certainty how and to what extent propagation will occur. Propagation between the holes not only causes larger total charges to be detonated at the same instant, but also stronger ground vibrations, due to greater fixation caused by the faulty ignition sequence.

Water shock waves from underwater blastings may cause problems not only to adjacent constructions like lock gates, water intakes etc. but also to shipping and people in the water.

The pressure of the shock waves is considerably higher if the explosive is detonated freely in the water than if it is confined in a drill hole in the rock.

The maximum pressure in the water is approx. 10 times higher if the explosive is placed on the rock surface than if the same amount of explosive is charged inside the rock.

The water shock wave is not such a substantial risk when the rock is drilled and the explosives are confined in the rock. The duration of the shock wave is short, the peak value is reduced to half in a fraction of a millisecond. This implies that co-operation between charges with different delays does not occur (as the time delay is normally 25 ms). Neither should co-operation between different charges within the same delay period occus as the scatter within each period is ±5 ms.

Concussion charges such as FRAGMEX cause higher pressure in the water, as the charge is placed freely on the rock surface. The charges are often located in patterns of 1.0×1.0 m to 2.5×2.5 m and connected with detonating cord. The detonating cord detonates with a velocity of approx. 7000 m/sec., meaning that the charges are ignited with a delay of 0.15 to 0.3 ms. In this case it has to be anticipated that co-operation of the water shock waves from the different charges occur.

The great difference in peak pressure values between a free charge and one confined in rock indicates that drilling and charging must be the first choice close to delicate structures.

Different measures have been suggested for reducing the pressure of water shock waves, for example an air bubble curtain. The air bubble curtain is produced by placing perforated steel pipes on the bottom, through which air is pumped and bubbles up to the surface. When the water shock wave arrives at the air bubble curtain, part of it is absorbed in the bubbles. These are compressed during fractions of milliseconds and emit, for some fractions of a millisecond to some milliseconds, compression waves, with lower peak value, in all directions. The part of the shock wave that passes between the bubbles receives a reduced peak value and a more rapid fade out than the original undisturbed one. The air bubble curtain reduces the peak pressure of the shock wave, but does not essentially affect the impulse of the wave. In practical terms it can be said that the air bubble curtain may be effective in reducing structural stress in compliant structures, but may not be particularly effective in reducing stress in massive structures.

The most effective way of reducing both the pressure and the impulse of the water shock wave is to reduce the charges and confine them in the rock. The reduction of the charges can be done by reducing the spacing between the blastholes and charging less explosive in each hole (the specific charge should not be altered).

With a reliable short delay initiation system and an explosive with low propagation characteristics it is possible to keep both ground vibrations and water shock waves under control.
The following diagrams show the safety distances for different methods of underwater blasting.

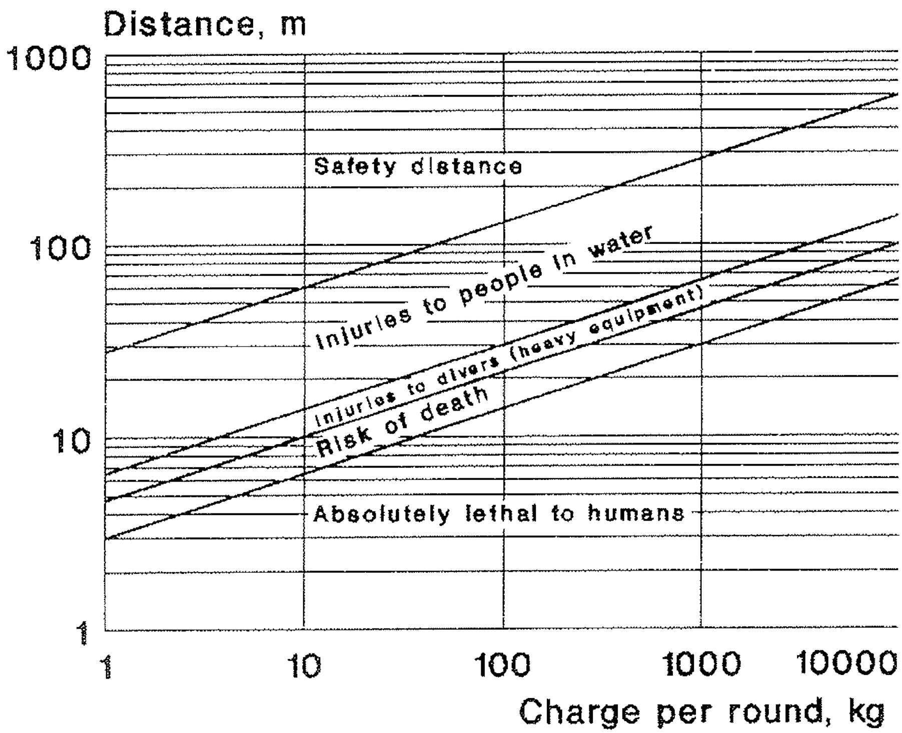



*Fig. 11.14 Safety distances for charges confined in blastholes, total charge per round.*

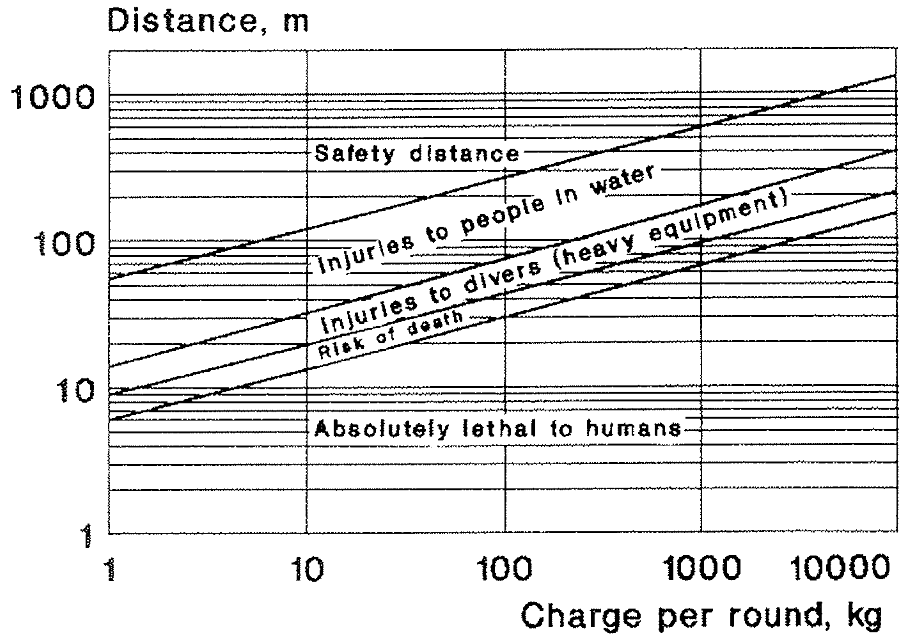



*Fig. 11.15 Safety distance for shaped charges in underwater blasting.*

# 12. SPECIAL FORMS OF BLASTING

## 12.1 Blasting of natural boulders.

Two methods are used for boulder blasting:

* Drilling and blasting.
* Concussion charges.

The latter method has its limits for technical and safety reasons. Two types of boulders can be distinguished, natural boulders and boulders originating from previous blasts. (See Chapter 5.4 Secondary blasting.)
The natural boulders require more explosives as they have not been subject to strains from previous blasts. Furthermore, the natural boulders are often partly covered by earth.

Factors to consider when blasting natural boulders:

* Kind of rock, size and shape.
* Desired fragmentation.
* Free-lying or partly buried in earth.
* The availability of covering material with regard to permitted traveling distance of flyrock.

**Drilling and blasting.**

The method of drilling one or more blastholes in the boulder is the most widely used method of fracturing boulders.
The blasting requires approximately double the charge used in secondary blasting for free lying boulders and around three times as much for buried boulders.

The specific charge for blasting natural boulders is approx. 0.1 kg/cu.m., which gives a good fragmentation of the boulder. However, the risk of flyrock is great so careful covering with heavy blasting mats must be done in built-up areas. In the vicinity of buildings it is also recommended to reduce the specific charge to 0.08 kg/cu.m., still using careful covering.

Charging table for free-lying natural boulders:
The charge should be placed in the center of gravity of the boulder.

| Size of boulder cu.m. | Thickness of boulder m | Depth of blasthole m | Number of blastholes | Charge of Emulite or Dynamex kg/hole |
|---|---|---|---|---|
| 0.5 | 0.8 | 0.45 | 1 | 0.05 |
| 1.0 | 1.0 | 0.55 | 1 | 0.10 |
| 2.0 | 1.0 | 0.55 | 2 | 0.10 |
| 3.0 | 1.5 | 0.85 | 2 | 0.15 |

Natural boulders which are partly or completely buried in earth are more difficult to blast than those lying on the ground.

The charges must be estimated considering the shape and extent of the boulder. The charge should be placed in the center of gravity of the boulder, which may be difficult to determine.
One way to determine the thickness of the boulder is to drill through it and than fill the hole with sand up to the suitable location of the charge.

Charging table for buried boulders:

| Size of boulder cu.m. | Thickness of boulder m | Buried section m | Depth of blasthole m | Number of blastholes | Charge of Emulite or Dynamex kg/hole |
|---|---|---|---|---|---|
| 1.0 | 1.0 | 0.5 | 0.6 | 1 | 0.15 |
| 1.0 | 1.0 | 1.0 | 0.6 | 1 | 0.20 |

When the boulder is fully buried in earth the specific charge can be increased to 0.2 kg/cu.m.

**Concussion charges.**

Concussion charges must be used only outside built-up areas. The air shock wave created by the concussion shot may cause severe damage to surrounding buildings. See Chapter 10.6 Air shock waves, Fig. 10.25.

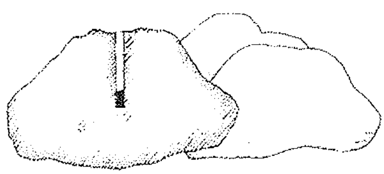


*Fig. 12.1 Drilling and blasting.*

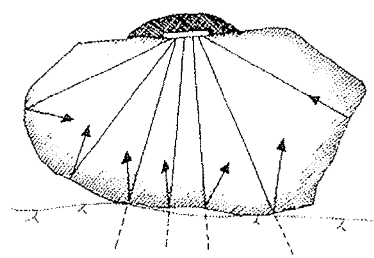


*Fig. 12.2 Concussion charge.*

The specific charge for a boulder fully above ground level is 1.0 kg/cu.m. The concussion shot is placed on the top of the boulder and stemmed with wet clay or similar material in abundance. No larger particles which may form projectiles should be included in the stemming.
When the shot is fired a shock wave goes through the boulder and is reflected against the free surfaces of the boulder. The colliding reflected shock waves within the boulder give rise to tensile stresses in the boulder which break it if the charge is correctly calculated.

Buried boulders are close to impossible to blast with concussion charges, as no reflection against any free surface can occur because the shock waves are transmitted further out into the ground. For concussion blasting high velocity explosives like Emulite or Dynamex should be used for the best result.

## 12.2 Blasting of ditches.

The blasting of ditches in earth is often resorted to when mechanical excavation is impossible or impractical i.e. in swamps and forests.
The most used method is to make bundles of explosives with a weight of 0.2 to 0.3 kg each which are placed in holes with a distance of 0.6 to 0.8 m. The charge is placed slightly below half the desired depth. Higher charge weights give an increased width of the ditch.
The charge weight, hole depth and distance between the charges must be adjusted to the actual conditions on the site. A test blasting over a shorter distance would give indications of the design of the blast.
When blasting ditches a water resistant explosive should be used.

The firing of the blast must be instantaneous, as no forward movement of the blasted material is permissible. The displacement of the material should be sideways.
Detonating cord is best for the firing, but of course VA and NONEL detonators can be used but with the same period number, preferably between No. 1 and 10.

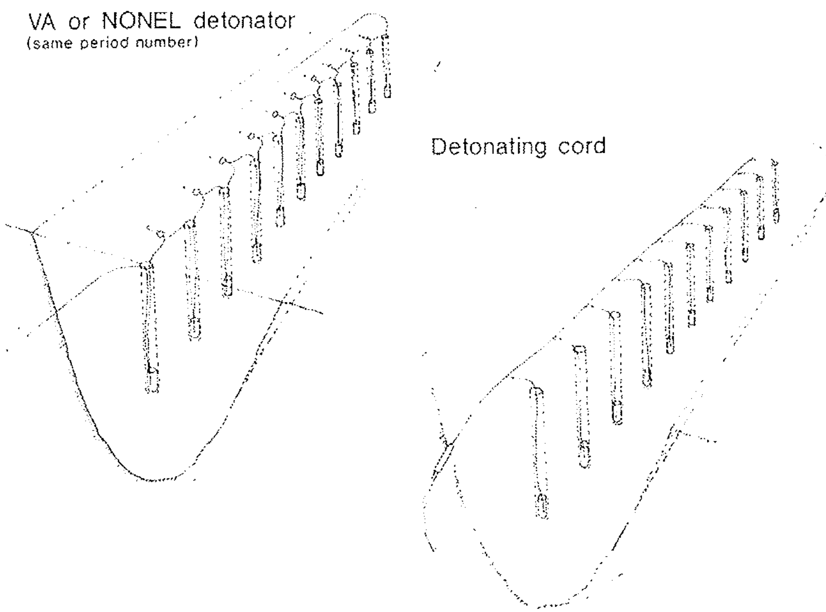



*Fig. 12.3 Ditch blasting.*

## 12.3 Blasting of stone blocks.

The blasting of stone blocks which are intended to be cut into tiles for floor and wall coverings requires a special and different technique.

The block must not be damaged and microcracks around the blastholes should be of very limited extent.

The size of the block is normally 1.5×1.5×2.0 m.

The contour of the block is drilled with a spacing of 0.2 m and the holes charged with 2 strings of 10 gr/m Primacord. The sides and the bottom are fired simultaneously and the combined force from the charges cuts the block loose from the rock, just moving it a few centimeters. Normally no stemming is used but water may be used as stemming in the vertical holes which seems to decrease the amount of microcracks around the holes.

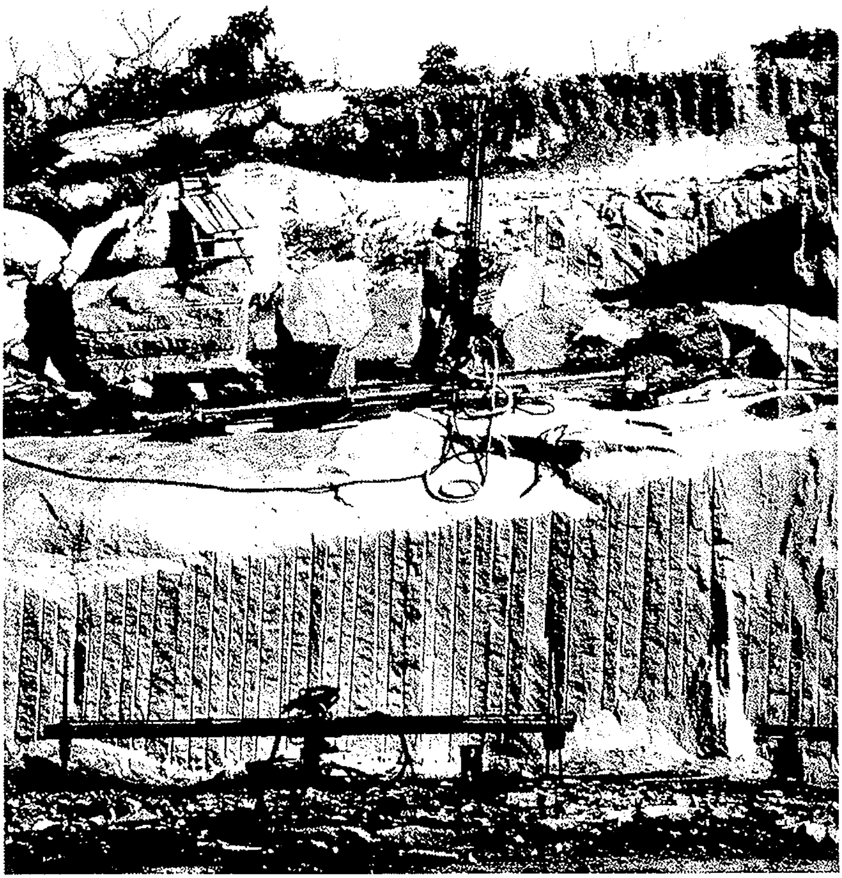


*Fig. 12.4 Blasting of granite blocks for tiles.*

## 12.4 Blasting of pole holes.

Blasting of pole holes is a major concern for many. The blast often results in a big crater causing problems when the pole is to be erected.
Well planned and executed blasting decreases the total work for the erection of the pole. No bolting or staying is necessary and the refill is kept to a minimum.

The blasting of pole holes is similar to tunnel blasting as only one free face is available. To create a second face, towards which the blasting is done, some kind of cut is drilled.
High specific drilling and charge are characteristic for this kind of blastings.

The most common cut for pole holes is the parallel hole cut. The calculation for this type of cut is the same as the charge calculation for tunneling in Chapter 7.1.1 The cut.
The center hole which is the opening for the blast should have a diameter of no less than 28 mm.
In small pole holes (diameter around 0.5 m) the blast consists of the cut hole, the 1st square and the perimeter holes. The perimeter holes are smooth blasted.
The blast can be charged with 22 mm Emulite 150 or Dynamex cartridges.
The firing is done with millisecond delay and either VA or NONEL detonators may be used. Due to the high specific charge and the concentrated direction of the blast, it must be carefully covered.
See also Chapter 12.10 Mini-hole blasting.

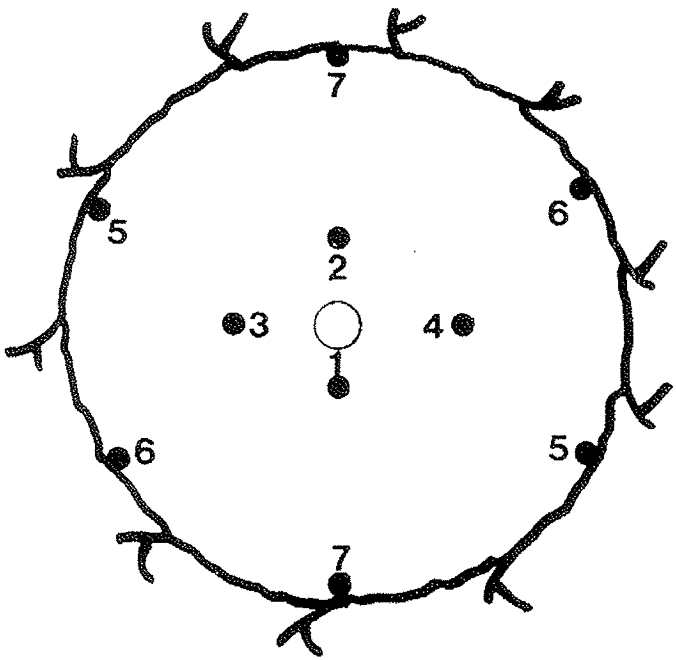



*Fig. 12.5 Firing pattern for pole hole.*

## 12.5 Blasting of stumps.

Of the two methods which are used to blast stumps, the one to place a charge under the stump to blow it away completely is the most common, but occasionally smaller charges are placed in drilled holes in the stump and thicker roots.

Factors to consider when blasting stumps:

* The diameter of the stump
* The age and species of the tree
* The nature of the underlying ground
* Permitted throw distance

When blasting a stump by placing an explosives charge underneath it, a correct quantity of explosives should be placed in the right position. If the first attempt is unsuccessful, no further blast is possible as usually too large a cavity is created under the stump by the first blast.
The charge should be placed approx. 0.5 m under the center of the stump and consist of 0.2 to 0.3 kg per decimeter of the stump diameter.
This guide value is for soft ground and fresh stumps. Oak and beech stumps need twice the charge. In harder earth, half the above recommended charge is sufficient.
To place the charge under the stump, a hole is made with a crowbar. For bigger stumps, the hole might not accommodate all the explosives needed and a smaller chamber must be created by blasting 1/3 of a cartridge.

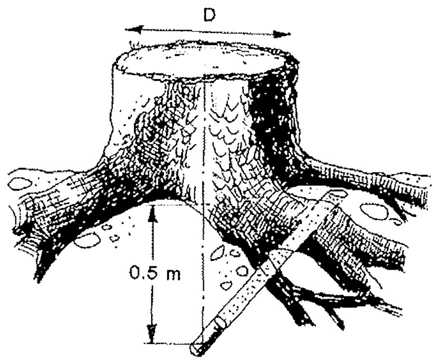


*Fig. 12.6 Blasting of stump with underlying charge.*

The explosives are placed in the chamber, well bunched together and the hole is stemmed, preferably with moist stonefree earth. If the blast is unsuccessful, the remaining roots are normally laid bare and can be blasted with 1/3 of a cartridge.

One disadvantage with the method is that it creates a large hole where the stump has been. The softer the ground, the larger the hole.
Another disadvantage is the control of the throw of the stump, which is virtually impossible to predict. The method should not be used close to installations and residential areas.

A gentler method is to break the stump with small charges placed in drilled holes in the stump. The holes are normally drilled from above and charged with 1 to 2 cartridges of Emulite 150 or Dynamex M. Thick roots may also be drilled but can be blasted off with 1 to 2 cartridges as surface charges. If the method is used close to residential areas, the blast should be covered and the air shock wave considered, as the confinement of the charges is not as good as when blasting rock.

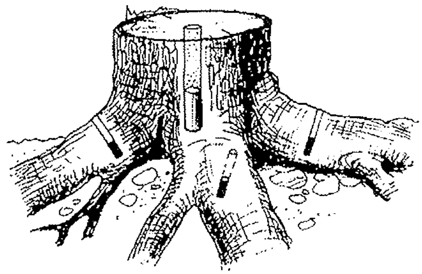


*Fig. 12.7 Blasting of stump with drilled-in charges.*

## 12.6 Blasting of soil frost.

As soil frost is a material, which varies in composition, it is difficult to give generally applicable recommendations with regard to suitable drilling and charging patterns. One important factor to consider when planning blasting of soil frost and selection of appropriate blasting method is the depth of the soil frost. Considering the depth of the soil frost, blasting may be subdivided into two methods:

* blasting with the charges placed under the soil frost
* blasting with the charges in the soil frost

Blasting with the charges placed under the soil frost works best when the frost is of limited depth, up to 1.0 m. By placing the charges under the soil frost the pressure from the explosive will force the soil frost to break upwards. As the charges are placed in the unfrozen ground, relatively large charges are needed for which chambers have to be blasted before the final charge is introduced. The chamber may be blasted with 1/3 to 1/2 cartridge of Emulite 150 or Dynamex M, 22×200 mm. The size of the final charge is determined by the material under the soil frost, required fragmentation and risk of undesired throw.


*Fig. 12.8 Charging under the soil frost.*

Before larger areas are blasted, a test blasting should be executed.

The following table gives guide values for charges under the soil frost:

| Depth of soil frost m | Charge depth m | Charge distance m | Charge kg/hole |
|---|---|---|---|
| 0.3 | 0.5 | 0.8 | 0.15 |
| 0.5 | 0.7 | 1.1 | 0.25 |
| 0.7 | 0.9 | 1.5 | 0.50 |
| 1.0 | 1.4 | 2.0 | 1.00 |

If larger holes are drilled (≥51 mm), no chamber blasting is needed, but the charge depth should be increased to about 1.5 times the soil frost depth. The drilling pattern is square with a spacing of 2 times the soil frost depth. The initiation can be instantaneous, but where risk for damage by ground vibrations may occur short interval detonators should be used.

The breakage of soil frost by placing the charges under it becomes difficult when its depth exceeds 1.0 m. Blasting with the charges placed in the soil frost is preferable in this case, especially if the ground under the soil frost is soft and provides little or no resistance.

The drilling and charging pattern depends on factors like type of soil frost, permitted throw, nearness to buildings with regard to ground vibrations, desired fragmentation etc.

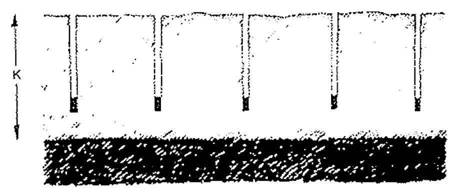



*Fig. 12.9 Charging in the soil frost.*

As soil frost may be difficult to drill and good air flushing is needed, it is advantageous to use heavy drill equipment (crawler drills etc.) with larger holes. However, the larger hole diameter can not be utilized in cautious blasting of soil frost when the charge weight and drilling pattern must be limited.

Soil frost is normally a homogeneous material which may cause sizable ground vibrations. Furthermore, good contact between soil frost and house foundations may cause damage due to displacement of the blasted soil frost, which must be considered when blasting is performed close to buildings.

The limitations of the charge weights which apply to cautious blasting in rock must also be considered when blasting soil frost.

Recommended drilling and charging pattern for cautious blasting of soil frost. Charges in the soil frost.

| Depth of soil frost m | Drilling depth m | Burden m | Spacing m | Charge kg/hole |
|---|---|---|---|---|
| 0.9 | 0.7 | 0.6 | 0.6 | 0.15 |
| 1.1 | 0.9 | 0.8 | 0.8 | 0.3 |
| 1.3 | 1.1 | 0.9 | 0.9 | 0.6 |
| 1.5 | 1.3 | 1.1 | 1.1 | 0.8 |

Recommended drilling and charging pattern for "free" blasting of soil frost. (No limitations for ground vibrations etc.) Charges in the soil frost.

| Depth of soil frost m | Drilling depth m | Burden m | Spacing m | Charge kg/hole |
|---|---|---|---|---|
| 0.9 | 0.7 | 0.7 | 0.7 | 0.25 |
| 1.1 | 0.9 | 0.9 | 0.9 | 0.5 |
| 1.3 | 1.1 | 1.1 | 1.1 | 0.8 |
| 1.5 | 1.3 | 1.3 | 1.3 | 1.2 |
| 2.0 | 1.8 | 1.5 | 1.5 | 2.5 |

## 12.7 Blasting of ice-holes.

The most common and safest way of blasting ice-holes is to place the charge under the ice. The charge should be placed 1.25 m under the ice for the best result. If the water depth is less than 2.50 m, the charges should be placed in half the water depth.

The following tables show the Swedish National Defense Forces' experience of blasting ice-holes:

| Thickness of the ice m | Width of the ice-hole m | Charge kg | Spacing m |
|---|---|---|---|
| Up to 0.4 | 5 | 1 | 4 |
| Up to 0.4 | 6 | 2 | 5 |
| Up to 0.4 | 8 | 3 | 8 |
| 0.4–0.6 | 8 | 4 | 8 |
| 0.6–1.0 | 8–10 | 5 | 8 |

If the water depth is less than 2.5 m, the diameter of the ice-hole will be smaller and the spacing between the holes reduced accordingly:

| Water depth m | Spacing at different charge weights (m) | | |
|---|---|---|---|
| | 3 kg | 4 kg | 5 kg |
| 2.0 | 5 | 7 | 8 |
| 1.5 | 4 | 6 | 8 |
| 1.0 | 4 | 5 | 6 |
| 0.5 | 3 | 4 | 5 |

If the charges in the above tables are increased, the ice-hole will be cleaner from ice but not wider.

Suitable explosives are Emulite 150 and Dynamex M.

To place the charges under the ice, a hole is made with an ice-pick or an ice-drill. The charges are connected in bundles and sunk to the intended level. In waters with currents it might be necessary to use a stone sinker or similar to ensure that the charges stay at the intended level.

The charges should be instantaneously initiated, preferably with a detonating cord. To keep the charge at the right level under the ice, the detonating cord downline is connected to a cross-bar on the ice. The downline is then connected to a trunkline, to which a detonator is connected.

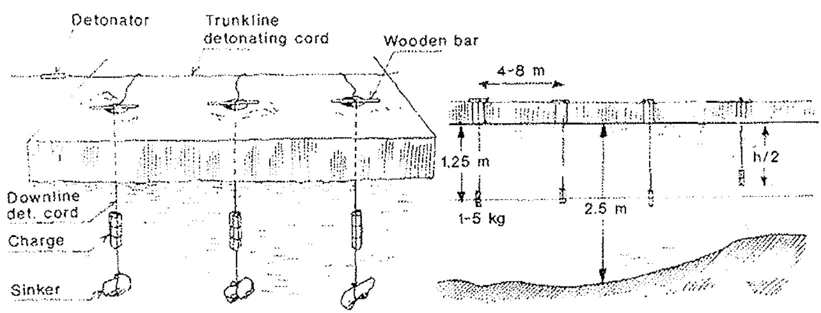



*Fig. 12.10 Charging for blasting of ice-hole.*

## 12.8 Blasting of openings in concrete.

One of the most difficult problem in renovation work is making openings in concrete. Even though experimental methods using lasers, micro waves, and high pressure water jets have been tried, blastings has since the late 1960s been developed into a technique with very high precision. The blasting is executed with small charges, causes little damage and is well controlled.

The most common method is to blast with drilled-in charges, but for thinner constructions, shaped explosive strips (lined cavity effect) may be used. With drilled-in charges, the concrete in the whole opening can either be blasted into small fragments or the opening can be divided into larger parts by slot blasting.

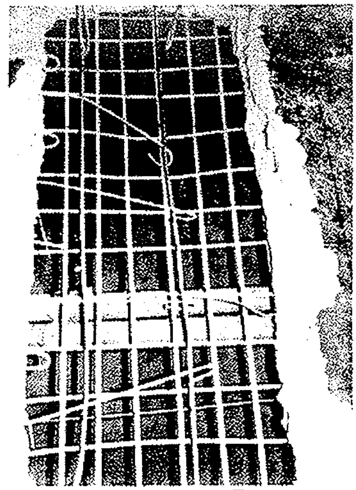


*Fig. 12.11 Opening in concrete made by blasting.*

In the contour, there is an increasing demand for smoothness and minimal formation of cracks. To obtain this effect, principally two methods are used, smooth blasting and presplitting. These two methods are well known and tested, so a desired contour can be obtained.

### Drilling.

The blasting method for openings in concrete can be compared to that of mini-hole blasting (Chapter 12.10 Mini-hole blasting).

When drilling the blastholes, it is convenient to use light pneumatic drilling machines with a 22 mm drill steel.

Electric hammer drilling machines can also be used, but with somewhat smaller diameters. If the concrete is more than 150 mm thick the capacity of electric drilling machines is considerably lower than that of pneumatic ones.

Both types should be equipped with dust collectors if respirators are not used.

### Charging.

The most suitable explosive is Primex A 17×150 mm in plastic tubes. These tubes may be cut into suitable lengths (30 mm=10 g) and be initiated with a detonator or detonating cord.

The charge should always be placed in the center of the construction to be blasted.

### Initiation.

Initiation is done with millisecond detonators (25 ms delay) or detonating cord (contour blasting). Suitable detonators are electric VA-detonators or NONEL GT detonators.

**Covering.**

Covering to prevent throw of concrete is done with small rubber mats (tire mats) and industrial felt.
When walls are blasted, both sides are covered.
It is often enough to cover the blast with a new type of blasting felt, "Lotrak 45/45", which is substantially stronger than industrial felt.

**Dust protection.**

To minimize the formation of dust during the blasting, water can be sprayed on the blasting site or the room be filled with a light foam (1 liter of water + 8 centiliter of detergent give 1 cu.m. of foam).
Covering the opening with plastic bags filled with water or with a small "pool" made of wooden planks and a plastic foil, efficiently decrease the formation of dust, the air shock wave and the throw of concrete.

**Ground vibrations.**

The small quantities of explosives which are used in combination with milli-second delay blasting mean that the ground vibrations fade very fast with the distance, see Fig. 12.12.
Generally, the adjacent construction is not damaged by the vibrations. Sometimes installations and equipment sensitive to vibrations (e.g. computers) are close to the blasting site and the charge calculations must thus be adjusted accordingly.
However, these problems are well known to those blasting in built-up areas and measures to decrease the vibrations are well tested. (See Chapter 12 Cautious blasting.)

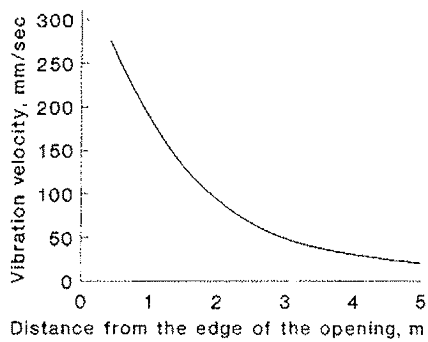



*Fig. 12.12 The vibrations decrease rapidly with distance.*

**Air shock waves.**

The air shock waves can, in smaller confined spaces, create problems, with cracked windows as a consequence. Measures to decrease the problems are on one hand to decrease the number of holes to be blasted each time and on the other hand to reinforce the windows by taping them.

As mentioned earlier both water and light foam are excellent means of reducing air shock waves.

**Crack formation in remaining concrete.**

Blasting produces a certain crack formation in concrete constructions and a smaller crushed zone in the contour. The cracks are judged as being of no importance in normal concrete floors and walls. The extent of the cracks is both perpendicular and parallel to the construction plane. The former type, which is common for other reasons, has a crack width of 0.1 to 0.3 mm and arises only occasionally. The second type, the parallel cracks, are nearly always in the same plane as the reinforcement and where they occur, adhesion between iron bars and concrete is lost. Investigations show that the extent of the cracks on average is limited to about 240 mm in the walls and about 390 mm in the floors. If the construction is heavily utilized, reinforcing measures may be necessary.
Nitro Consult AB, a subsidiary of Dyno Industries, Norway, has a vast experience of this kind of work and should be contacted when any doubt arises.

## 12.9 Blasting of concrete.

Blasting of concrete is often carried out in connection with the demolition of buildings but also in the cleaning up of a building site where foundations for cranes etc. must be taken away. In the latter case it may be practical to prepare the foundation with plastic tubes when it is constructed. They can then be charged with explosives when it is time for blasting.
General drilling and blasting patterns are difficult to design as such important parameters as strength, reinforcement and geometry of the object should be known in order to carry out exact charge calculations. As an example, thin heavily reinforced objects may require specific charges of 3 to 6 kg/cu.m. while un-reinforced objects could be blasted with 0.3 to 0.7 kg/cu.m.

The following table serves as guidance for the blasting of thicker concrete objects.

| Object of concrete | Specific charge kg/cu.m. | Square drilling pattern | Type of explosive |
|---|---|---|---|
| Poor quality no reinforcement | 0.3 | 0.7–0.8 | 17 mm Gurit |
| Good quality no reinforcement | 0.4–0.5 | 0.6–0.7 | 17 mm Gurit |
| Reinforced | 0.8–1.0 | 0.5 | 22 mm Em 150 |
| Heavily reinforced | 1.0–2.0 | 0.4–0.5 | 22 mm Em 150 |

When blasting thin concrete objects like floors, walls etc. the drilling pattern must be relatively dense. Too widely spaced blastholes can result in an unsuccessful blast as only craters will occur around the holes.
Requisite specific charge for concrete walls, floors and columns are:

- walls and floors 1.5–3.0 kg/cu.m.
- columns 1.0–1.5 kg/cu.m.

The following table lists the guide values for drilling and charging of thin concrete objects with no reinforcement:

| Thickness of concrete | Hole depth m | Square drilling m | Charge weight/hole kg |
|---|---|---|---|
| 0.20 | 0.14 | 0.25 | 0.03 |
| 0.30 | 0.20 | 0.35 | 0.04 |
| 0.40 | 0.30 | 0.50 | 0.06–0.1 |

It can generally be said about blasting concrete that the explosives should be distributed in the object in as many holes as possible. It is preferable to use many small-diameter holes instead of a few larger and many small charges instead of a few larger ones.

## 12.10 Mini-hole blasting.

Mini-hole blasting has been found to be the most effective method for rock breaking in sensitive environments.
With all that blasting entails, the choice of blasthole diameter is one of the most important factors for achieving a succesful result, both from economic and safety points of view. Certain types of blasting demand significant reductions in the amount of explosives which can be used, e.g. due to the close proximity of buildings, low bench heights, etc.
Reductions in the amount of explosive should in turn mean reduced blasthole diameters. However, this is not common practice today, as the majority of blasters use a minimum hole diameters of 34 mm. The 34 mm blasthole is not fully utilized when the charge weights are reduced and therefore the mini-hole technique has been developed with a blasthole diameter of 22 mm.

**Drilling.**

By using 22 mm diameter drill steels, the drilling time is substantially reduced (see table below).

**Drilling machines:**

**Atlas Copco Cobra**
Drilling depth 0.6 m
Diam. 28 mm – 8 min. 40 sec.
Diam. 22 mm – 3 min. 33 sec.

**R.H.65**
Drilling depth 0.75 m
Diam. 34 mm – 2 min. 45 sec.
Diam. 22 mm – 1 min. 20 sec.

For ergonomic reasons it is unnecessary to load the drilling machines heavily, which facilitates the hole inclination and also gives greater flexibility when positioning the blasthole. Otherwise, the blastholes are drilled in the traditional manner in accordance with the indicated drilling patterns. It is generally the case that the burdens do not exceed 0.5 m.

**Charging.**

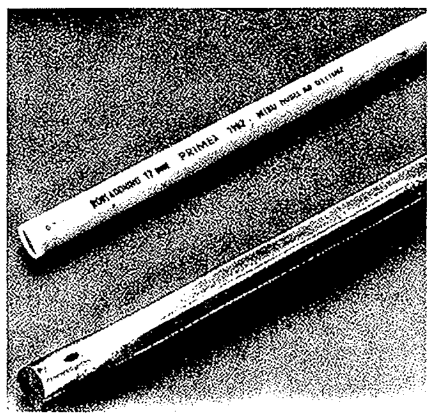
To facilitate mini-hole charging, Nitro Nobel has developed a charging unit that makes it possible to safely initiate charge quantities down to 4 grams.

**Charging unit.**

A special pipe charge has been developed with the dimensions 17×150 mm and a weight of 52 grams, which can be cut into suitable charge quantities. The pipe charges contain the explosive PRIMEX, an explosive with high detonation velocity. If difficulties are encountered with direct insertion of the detonator into the pared off section, then that section can be placed inside a plastic capsule, whereupon the adaptation of the detonator can be completed.
The mini-hole technique complements the drill hole with an ideally suited charge diameter, which makes better use of the explosive energy. The quantity of the charge can therefore be reduced, usually by half the normal amount, resulting in a reduced risk of flyrock and reduced ground vibrations.
When using the above mentioned charges (capsule or pipe charges), greater safety is ensured because the explosive and detonator are kept in an integral state and cannot come apart during the charging operation.

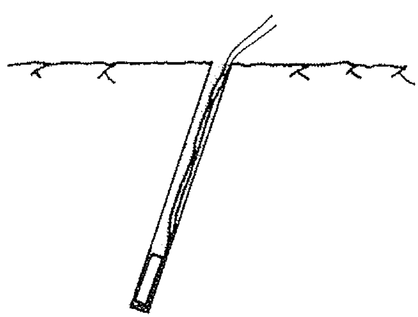
**Stemming.**

The best stemming material is well packed dry and relatively coarse sand. Practical trials have shown that sandplugs function more effectively in mini-holes. It is therefore possible to drill shallow holes, from 0.3 to 0.4 m, if mini-holes are utilized, and thus meet the demands of building regulations by not blasting below a theoretical level. The rule that the burden should be less than the hole depth, is as always valid.

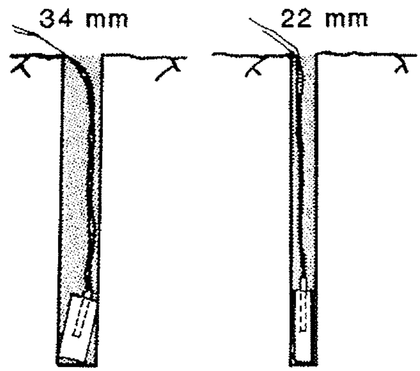


The advantages of the mini-hole technique over conventional techniques can be summed-up as follows:

- reduced drilling times
- simplified charging procedure with safer adaptation of the detonator
- reduction of flyrock
- reduced ground vibrations due to lower quantities of explosives.



To summarize, this means better blasting economy, increased safety for those taking part in the operation and for the immediate surroundings.

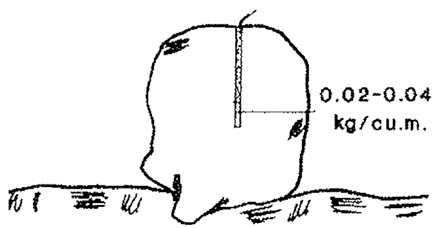



**Areas of usage.**

The mini-hole technique can be advantageously utilized in the following types of blasting.

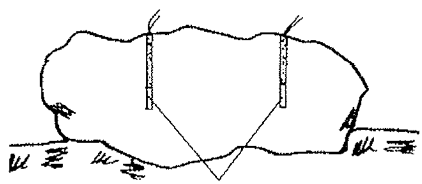
- natural boulders and secondary blasting
- leveling by blasting
- cable trenches
- electricity/telephone/fencing poles with footings in rock
- other cautious blasting operations.

**Natural boulders and secondary blasting.**



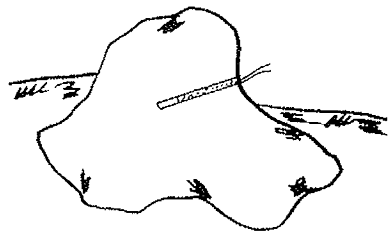

When blasting natural boulders, a specific charge of 0.02 to 0.04 kg/cu.m. is recommended, depending on the boulder's character and the desired fragmentation. To reduce unnecessary risks, boulders should not be blasted into pieces smaller than necessary

The specific charge for secondary blasting can be reduced to 0.01 to 0.02 kg/cu.m.




Earth-bound boulders are considerably more difficult to blast than free-lying ones, and become increasingly so the deeper they are buried. In built-up areas, the blasting of earth-bound boulders should be avoided, because it is often difficult to determine the size of the boulder and thus also to control the blasting operation. In such cases, the best thing is to excavate partially or fully around the boulder so that the size can be ascertained, and a well-poised charge quantity determined.

This will bring about a more successful blasting operation, from both technical and safety points of view. It is often sufficient to blast away the part of the boulder that appears above ground level.



**Leveling by mini-hole blasting.**

When leveling by blasting, flyrock stands out as a negative factor. The reason for this is that the explosive lies close to the rock surface with only a thin protective layer of rock to cover it. In order to prevent fly-rock, the quantities of explosives must be limited, which in turn means smaller burdens. Leveling therefore demands high specific drilling (3.0 to 4.0 drilled meters per cubic meter). By using a 22 mm drill steel the drilling time can be reduced and greater cost savings made.

To avoid upward break-out, the burden must be less than the hole depth. To obtain less constriction the holes should be inclined towards the vertical plane with an inclination of 3:1.

It is a known fact that rock moves more quickly with small burdens, therefore when blasting multiple rows, delay time between the rows ought to be limited to 25 ms.

The table gives recommendations for suitable hole distances and charge quantities for leveling in rock. As can be seen, the charge quantities can be reduced to half that of conventional techniques. The values should be taken as guide values, which sometimes require adjustment to suit the various rock characteristics.

LEVELLING

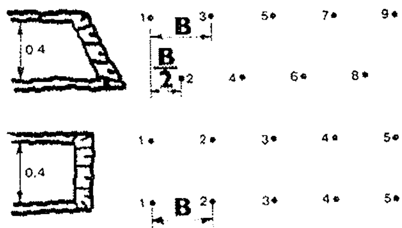
Hole diameter 22 mm.
Explosives Primex® 17 mm

| Bench height K m | Hole depth Incl. 3:1 H m | Burden B m | Spacing S m | Charge $Q_b$ kg | Uncharged part $h_o$ m |
|---|---|---|---|---|---|
| 0.3 | 0.5 | 0.3 | 0.5 | 0.01 | 0.4 |
| 0.4 | 0.6 | 0.4 | 0.5 | 0.02 | 0.45 |
| 0.5 | 0.7 | 0.45 | 0.6 | 0.03 | 0.55 |
| 0.6 | 0.9 | 0.5 | 0.65 | 0.05 | 0.65 |

### Cable trenches.

Cable trenches often have limited depths and are therefore ideally suited to the mini-hole blasting method. Drilling and charging should be carried out in accordance with the adjoining diagram and table.

| Depth m | Hole depth incl. 3:1 m | Burden m | Charge kg | Uncharged part m |
|---|---|---|---|---|
| 0.3 | 0.5 | 0.3 | 0.06 | 0.3 |
| 0.4 | 0.6 | 0.4 | 0.08 | 0.4 |
| 0.5 | 0.7 | 0.4 | 0.1 | 0.4 |
| 0.6 | 0.8 | 0.4 | 0.12 | 0.4 |




### Pipeline trenches.

If the trench depth is to be less than 0.8 m, the mini-hole method can be used in accordance with the adjoining recommendations.
The blastholes are set out in accordance with a conventional pattern.
The mini-hole technique complements the blasthole with an ideally suited charge diameter, which makes better use of the explosive energy. The quantity of the charge can therefore be reduced, usually by half the normal amount, resulting in a reduced risk of flyrock and reduced ground vibrations.

| Trench depth m | Hole depth incl. 3:1 m | Burden m | Charge kg | Uncharged part m |
|---|---|---|---|---|
| 0.3 | 0.5 | 0.3 | 0.03 | 0.3 |
| 0.4 | 0.6 | 0.4 | 0.05 | 0.4 |
| 0.5 | 0.7 | 0.4 | 0.08 | 0.4 |
| 0.6 | 0.8 | 0.4 | 0.1 | 0.4 |

### Pole/pylon footings.

Blasting footings for poles and pylons in rock is a great problem. The resultant blast often leaves a crater, where the objective of securely wedging the pole still remains.
A well planned and executed blasting results in the total operational time for setting the pole being greatly reduced because stays and anchors are

not required. Furthermore, only a minimum of back-fill material is needed.

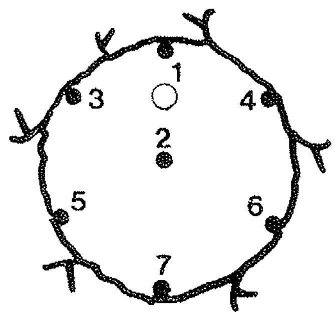
Blasting for pole footings can be compared with a tunnel blasting, as there is only one free face for displacement. This type of blasting operation has the characteristics of high specific drilling and charging.

The introduction of the mini-hole technique has opened new perspectives for this type of blasting operation. The drilling time, as mentioned above is halved.

When blasting pole footings with a diameter of ≤0.6 m, the parallel hole cut is the most favorable method. This is because the cut may be placed in various positions in order to limit the number of holes to a minimum. To maintain good drilling precision, a drill-guide should, if possible, be utilized. Another method is to use a guide-leg, which is let into the first drill hole, and subsequently used for the remaining holes.

**Charging.**

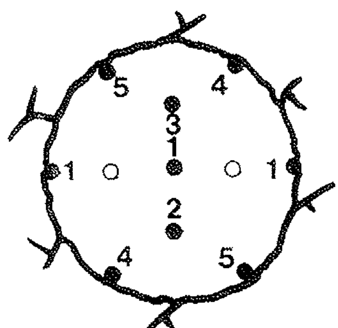
PRIMEX A 17×150 mm charges are used for charging. If necessary, the last charge should be cut off to leave an uncharged section of approx. 6 cm. A high specific charge is important with parallel hole cuts. Stem with sand to prevent the charges from being blown out of the hole.




**Ignition pattern – blasting.**

The best result is obtained if the pole hole is blasted in one round with delay detonators, in accordance with the adjoining firing patterns.

In order for the rock to achieve the desired swelling, a minimum of 50 ms delay between the holes is necessary. If the holes are to be blasted individually, they should be blasted in the



same order as stated in the firing patterns.

Other specialized areas where the mini-hole technique can be used are:

- blasting concrete
- blasting in furnaces.

## 12.11 Demolition blasting.

In connection with the clearing of shut down industries and older housing areas, blasting has proved to be a very competitive demolition method.
The method was introduced in the 1960s and many buildings have been blasted since then, both brick buildings and reinforced concrete buildings.
The principle of demolition blasting is to blast the vital supporting framework in the ground floor. The tare weight of the building will then break the rest of the framework.
In order to minimize the number of blastholes in the cut it is important to utilize all the openings (e.g. windows, doors) which are available on the floor where the cut is placed.
In order to obtain maximum breakage and the desired direction of felling, the design of the firing pattern is of the utmost importance.
It is difficult to give general advice on how a building should be demolished by blasting, as experience shows that each building requires careful charge calculation and design of the firing pattern owing to its construction, design, location in relation to other buildings, traffic routes etc.
Nitro Consult AB, a subsidiary of Dyno Industries, has all the necessary competence for demolition work.

*Fig. 12.14 Demolition blasting.*

# 13. DESTRUCTION OF EXPLOSIVES AND FIRING DEVICES

Frequently explosives and firing devices have to be destroyed. The reason may be deterioration due to poor storage, age or as is the case in many countries, broken package of explosives may not be returned to the store. In such cases, the explosives must be destroyed.

**Explosives, detonating cord and blasting agents.**

Blasting agents like ANFO are best destroyed by mixing them with water. Water destroys them very quickly.

Dynamite type explosives and emulsion and slurry type explosives should be destroyed by burning. The latter two types contain a substantial amount of water and are not easily set on fire.

Procedure for destruction:
* Find a stone free location for the burning. The location should be far away from inhabited areas. See Graph 10.25 Chapter 10.6 Air shock waves.
* Do not burn more than 5 kg each time.
  When larger amounts of explosives are to be destroyed, the manufacturer should be contacted to provide the necessary experts for such operations.
* Take the explosives out of the box. No confinement.
* Spread the explosives on a bed of paper or wood wool.
* Pour diesel (fuel oil) or kerosene over the explosive and the bed and set fire.
* Go to protected place and stay there until all explosives are burnt.
* Check the ashes for fragments of explosives, if explosives remain, repeat the procedure.

**SEE THAT THERE ARE NO DETONATORS WITH THE EXPLOSIVES TO BE BURNT.**

The destruction of detonating cord follows the same procedure as the destruction of dynamites. Detonating cord should not be destroyed together with dynamites or other explosives, but separately.

**VA, NONEL and plain detonators.**

The destruction of detonators can in the case of VA and NONEL detonators be done by dropping the detonators into blastholes after the legwires or NONEL tube have been cut.

Plain (safety fuse) detonators have to be blasted. The best way is to tape the detonators to a stick of dynamite or emulsion explosive with a good detonator and blast them away.
VA and NONEL detonators can be destroyed in the same way if no blasthole is available for their destruction.

As for the destruction of explosives, the location for the blast of detonators must be selected in such a way that no damage is caused to people or property. The location should be stone-free and well away from buildings.

Plain detonators MUST NOT be dropped into blastholes as sand or other impurities may enter into the detonator and cause unintended firing.

Plain detonators or other detonators MUST NOT be thrown into water as they may be found and will after drying have the same dangerous properties as before.

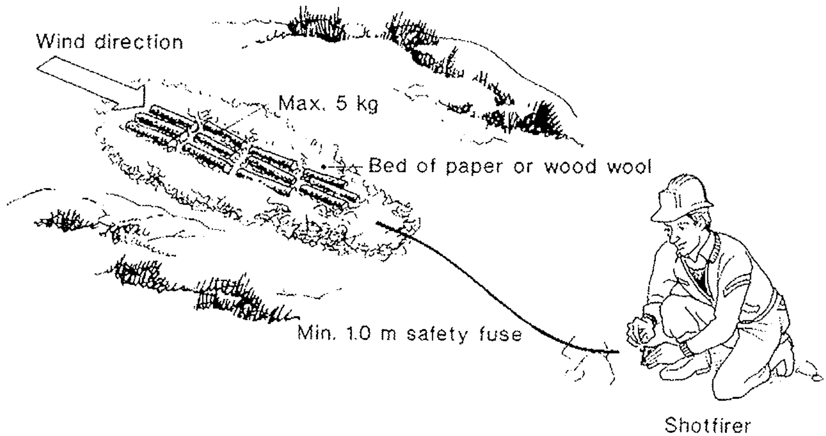



*Fig. 13.1 Destruction of explosives by burning.*

**NOTE! If any local regulation contradicts any of the above, the local regulation applies.**

# 14. CODE OF GOOD PRACTICE

The Federation of European Manufacturers (FEEM), is continuously working to improve the safety record related to the use of explosives and has produced a "Code of Good Practice", which is a set of recommendations for the execution of all stages of the work associated with blasting.

The standards proposed by FEEM are high and the Code can therefore be used in its entirety in countries without established legislation and also as a reference document in other countries.

**IT MUST, HOWEVER, BE ENPHASIZED THAT NATIONAL AND LOCAL LEGISLATION MUST ALWAYS TAKE PRECEDENCE OVER THIS CODE IN CASES WHERE THERE APPEARS TO BE A CONFLICT.**

For the Code to be effective, it is important that the areas of responsibility of personnel engaged in planning and carrying out blasting work cover all the recommendations. At some operations with infrequent blastings, e.g. a small quarry, it is likely that the person who charges the blast will also plan the drilling, decide on the type of explosives etc. He will be responsible for ensuring that all the recommendations are observed. At the other extreme, e.g. a large open pit using several tons of explosives per day, it is likely that one or more professional engineers will be appointed, and that they will undertake the planning of the blast and the selection of the explosives, while still leaving the actual charging to the shotfirer. Many variations are possible within the two extremes, and it is important that areas of responsibility should be defined in all cases where the possibility of confusion exists.

Correctly used, explosives provide a safe and economic tool for many of the operations essential for the functioning of a modern community. It is FEEM's hope that this Code will help to make them even safer.

## Section 1. Qualification and appointment of shotfirers and of assistant personnel.

1.1. To qualify for appointment as a shotfirer, a person should
- be preferably at least 21 years old and be of stable temperament,
- have practical experience, working with a shotfirer, amounting to at least one year or at least 50 blasts,
- have attended a recognized course of training covering the theoretical, practical and, where applicable, legal aspects which have general application for the use of explosives for all types of blasting work,
- have attended additional courses covering specialized requirements, where this is necessary because of the nature of the work being undertaken, e.g. blasting in hot masses, underwater blasting.

1.2. To qualify for appointment as assistant personnel, a person should
- be at least 18 years old and be of stable temperament,
- have attended a short course dealing with the main hazards in the handling and use of explosives.

1.3. All appointments of shotfirers and of assistant personnel should be made in writing.

1.4. Where the blasting operation is so extensive that several shotfirers are required, one should be nominated as having overall control. Where necessary, the nominations should cover different areas and different times of the day.

## Section 2. Choice of explosives and initiating systems.

2.1. Explosives and initiating systems should be selected for their suitability for the work, or for different parts of the work, to be undertaken. The following factors should always be taken into account, but additional factors may have to be taken into consideration in special circumstances:

2.1.1. Explosives:
- nature of blasting operation and rock type including hole diameter, depth and presence of cavities,
- whether the explosive is to be a bottom or column charge,
- presence of water, including water pressure and time of exposure,
- temperatures likely to be encountered in use,
- ventilation conditions,
- storage conditions.

2.1.2. Initiating system:
- nature of blasting operation,
- requirement of initiation power for the selected explosive in the conditions of use.
- presence of water, including water pressure and time of exposure,
- presence of potential sources of premature initiation,
- temperatures likely to be encountered in use.

## Section 3. Site storage.

3.1. Normally, all site storage of explosives materials will be governed by national and/or local regulations detailing construction, safety, distances etc. Where this is not the case, the construction of the store should be of fire resistant material and capable of affording a degree of security against illegal entry.
Reference should be made to "Code of Good Practice – Storage of explosives" (publication No. 10), published by FEEM.

3.2. Premises for storing explosives materials should be kept locked and should be under the control of the Site Manager.

3.3. Records should be kept, detailing the quantities of explosives materials received and issued.

3.4. Premises used for storing explosives materials should not be used for any other purpose. They should be maintained clean and dry, and be well ventilated. Where extremes of high or low temperatures may be encountered, consideration should be given to the need for temperature control.

3.5. Detonators should be stored physically separately from any explosive, and in such a way that there is no possibility of an accidental initiation of the detonators being transmitted to the explosive.

3.6 All explosives issued from the Magazines for use should be under the supervision and responsibility of the shotfirer.

## Section 4. Site transport of explosives materials in powered vehicles.

4.1. Transport of explosives materials and personnel should be separate, where this is practicable.

4.2. Vehicles used for transport of explosives materials should be included in a scheme of planned maintenance. The intervals between examinations should be decided after consideration of the types of duty to which the vehicle is exposed.

4.3. Where the explosives transport vehicle is also used for other purposes, it should be emptied, cleaned and checked before loading of explosives materials is started, and again at end of shift to ensure all explosives have been removed.

4.4. The vehicle should be readily distinguishable from other site vehicles, and should preferably be fitted with an additional identification e.g. flashing light, when actually engaged in explosives transport.

4.5. Vehicles transporting explosive materials should not carry any other materials, with the exception of the equipment required for the blasting operation being undertaken.

4.6. Transport of explosive materials should be undertaken under the supervision of the shotfirer who is familiar with the hazards involved, and the actions required in the event of fire or other abnormal occurrence.

4.7. The load on the vehicle should be secured to avoid any part being dislodged. Special care should be taken with the contents of cases which have already been opened.

4.8. The vehicle should be fitted with at least one fire extinguisher.

4.9. Small fires not directly in contact with the load may be tackled with fire extinguishers. In cases where the fire is in contact with the load, the vehicle should be abandoned and the area cordoned off.

4.10. When detonators and other explosives materials are transported on the same vehicle, provision should be made for the detonators to be kept in a separate protective container.

## Section 5. Planning of blasts and general safety considerations.

5.1. Before a new operation involving blasting is started, local authorities and utilities, gas, water, electricity etc., should be consulted to check on the presence of services which could be damaged. The distances to the nearest structures (houses, offices, factories) must also be taken into consideration when designing blasts so that ground vibration and air overpressure may be kept within acceptable levels. Please refer to "Code of Good Practice – Ground and Airborne Vibration from the use of Explosives" (Publication No. 4) published by FEEM.

5.2. The Blasting Plan once agreed should be adhered to and any changes reported and approved.

5.3. Records, including details of the drilling pattern and of the explosives materials used in each blast, should be kept. In critical situations e.g. in built-up areas, consideration should be given to the advisability of keeping more detailed records.

5.4. Every effort should be made to avoid exposing explosive materials to excessive pressure, shock, heat, or friction. The following specific rules should be observed, but should not be taken as limiting the generality of this section.

5.4.1. Smoking, and the use of naked flame, for example, electric arc welding, oxy-acetylene cutting etc., should not be allowed within a safe horizontal distance of at least 10 m from explosives materials, in stores, during transport, and at the blast site. All personnel on the site, permanently or temporarily employed or visiting, should be informed of this rule, and steps taken so that they can easily identify the prohibited zone.

5.4.2. Use of explosives should be avoided in areas exposed to flammable gases or dust.

5.5. Only approved non-explosive blasting accessories (cables, tools, instruments etc.) should be used in blasting work. Substituting standard, apparently similar, accessories may increase the chance of misfire and/or premature firing.

5.6. When using electric detonators the following precautions should be observed:

5.6.1. Before electric detonators are removed from the packaging, the immediate area should be examined for possible sources of extraneous electricity not taken into account when the choice of detonator type was made. Any such source should be removed or rendered harmless before work on the preparation of primers is begun.

5.6.2. Charging with electric detonators should not be started if there are any indications of electrical storms in the area. Where part or all of a blast has been charged when the first signs of an electric storm appear, it should be fired immediately if this is possible. Failing this, the area should be evacuated and fenced off. Special consideration should be given to installing warning systems in areas specially prone to electrical storms, for both surface and underground blasting operations.

## Section 6. Preparation of primer.

6.1. General.

6.1.1. Consideration should be given to the handling which the assembly will receive before and during the charging operation, and the method for securing detonator or detonating cord to the explosive should be such that they can not become detached.

6.1.2. A minimum of force should be used in preparing primers. The primer cartridge should preferably be prepared by forming a hole with a brass or wooden rod to take the detonator and allow easy insertion.

6.1.3. Where explosive cartridges are used, the detonator should be located centrally in the cartridge and with the base of the detonator in the explosive. Wherever possible, the complete length of the detonator shell should be inside the cartridge.

6.1.4. Where cast primers are used, the complete length of the detonator shell should be contained in the detonator recess.

6.1.5. Where the system of work requires the preparation of primer or charges containing detonators in advance of the actual charging operation, special rules should be prepared detailing the precautions to be taken. On no account should primers be assembled in the magazine.

6.2. Priming with Plain Detonators.

6.2.1. Where plain detonators are used, they should be assembled to the fuse at a place remote from the blasting site.

6.2.2. Before the primer cartridge is prepared, the burning time of the fuse should be calculated, and be adequate to allow the safe firing of the charge/s.

6.3. Priming with Detonating Cord.

6.3.1. Consideration should be given to the need to protect the ends of the detonating cord against the ingress of water, taking into account the exposure times and the water pressures likely to be encountered.

6.3.2. Wherever possible, the length of detonating cord within the borehole should be free of joints. Where it becomes necessary to use two or more lengths of detonating cord for a downline in a hole, the joint should be protected against mechanical damage and separation, and the ends protected against ingress of water.

6.4. Priming with other systems.

Manufacturers' recommendations should be followed.

## Section 7. Charging of boreholes.

7.1. General.

7.1.1. Where charging is started before all the holes in a blast have been drilled, it is recommended that the minimum horizontal distances between the two operations should be:
Holes less than 5 m depth .... 2 m at full length of hole.
Holes more than 5 m depth .... Not less than the length of the borehole at any point.

7.1.2. Boreholes should be checked for freedom from obstructions before charging is started, to minimize the possibility of the cartridges becoming fast in the borehole.

7.1.3. Where mechanical loading equipment is used, it is important that all conditions attached to its use are observed.

7.1.4. At least one primer assembly should be located at the bottom of the borehole, or, where exceptions are made, consideration should be given to the possibility of having to deal with undetonated explosive remaining in the borehole after firing.

7.1.5. In conditions where cut-offs may occur, a top and bottom primer should be used.

7.1.6. The primer cartridge should be loaded so that the base of the detonator points towards the main charge in the borehole.

7.1.7. Whatever initiating method is used the detonator lead lines should be straightened out before charging is started, without putting strain on the lead lines, especially where they enter the detonator. In loading vertical holes, the cartridge or booster containing the detonator should be lowered at an even rate to avoid snatch loading.

7.1.8. Care should be taken to avoid damaging the wire and/or initiating lines by over-vigorous use of the stemming rod.

7.1.9. Where detonators with additional protection (e.g. shorted and/or sealed ends) are used, care should be taken not to remove this during the charging operation.

7.2. Charging where Plain Detonators and Fuse are used.

7.2.1. The choice of direct or inverse priming, and the direction in which the detonator points, should take into account the type of fuse being used.

7.2.2. Care should be taken to avoid damaging the fuse by over-vigorous use of the stemming rod.

7.3. Charging where Detonating Cord is used.

7.3.1. Care should be taken at all times to avoid sharp bends or kinks in the cord.

7.3.2. Immediately after the detonating cord has reached the desired position, the cord should be cut from the reel. Sufficient free length should be secured over the hole to allow for any possible settlement in the hole and for subsequent connecting.

7.4. Charging where other systems are used.

7.4.1. The manufacturers' recommendations should be followed at all times.

## Section 8. Preparations for firing.

8.1. General.

8.1.1. Connecting of charges should be started in good time to allow the work to be completed and the blast fired within any time limitations existing at the site. Allowance should be made for dealing with unexpected difficulties.

8.1.2. Blasts should be kept under the supervision of a shotfirer. Where this is impracticable, the blast area should be clearly identified. and when necessary fenced off.

8.2. When using Electric Detonators.

8.2.1. All detonators used in a blast should be of the same electrical sensitivity and be produced by the same manufacturer.

8.2.2. The choice of detonator type, shotfiring cable, connecting wire, connecting system (series, series/parallel etc.) and firing device (exploder, etc) should be made at the planning stage of the blast.

8.2.3. All connections in the shotfiring circuit should be clean and tight and insulated from contact with each other and the ground. The type of insulation should be chosen after consideration of local conditions, i.e. presence of water, conducting rock, firing voltage, and checks should be made of insulation to ground where conditions are known to be difficult.

8.2.4. Only approved instruments should be used at the blast site for checking circuit values.

8.2.5. The resistance of the firing circuit should be measured and the result should be consistent with the calculated value in accordance with the number and type of detonators. In the case of series/parallel connection, the resistance of each circuit should be balanced to the limits appropriate for the exploder and the detonators being used.

8.2.6. The shotfiring cable should be checked visually for mechanical damage before every blast. It should also be checked for correct open and closed circuit resistance before it is connected to the detonator circuits.

8.2.7. Exploders and testing equipment should be regularly tested for correct performance. The intervals between tests should be decided after consideration of the local factors, but the tests should always be carried out if the exploder and/or tester equipment have been subjected to abnormal conditions, or following a misfire.

8.2.8. The means for controlling the discharge of the electrical energy into the firing circuit should always be under the control of the shotfirer.

8.3. When using Fuse and Plain Detonators.

8.3.1. The method for lighting the fuses should be decided at the planning stage, after consideration has been given to the number of fuses, the burning time of the fuses, and the time required to come to safety after the last fuse has been lit.

8.3.2. Where fuses are lit individually by the shotfirer, fuse lighters should be used. It is desirable that a second person should control the time which has elapsed from the lighting of the first fuse.

8.3.3. Where igniter cord is used in conjunction with fuse, the delay pattern should be drawn up so that all the fuses are burning in the holes before the first charge detonates.

8.4. When using Detonating Cord.

8.4.1. All connections between lines of detonating cord should be made at points where the core load is known to be dry. Where the ends have been, or may be, exposed to wet conditions, connections should be made at least 50 cm from the ends, and the excess folded over and secured to the main line.

8.4.2. The angle between branch lines and the part of the main lines carrying the incoming detonation waves should be 90° or more, and the connections should be made firmly, e.g. by taping or by slip-proof knots. Care should be taken that branch lines do not loop back and cross over the main lines before entering the hole.

8.4.3. Lines of detonating cord should be laid or hung in straight lines or smooth curves, without excessive slack or tension and should be kept at least 20 cm apart. For core loads over 12.5 g/m, greater separation may be required, and the manufacturers' instructions should be followed.

8.4.4. When detonating relays are used to achieve the desired delay pattern, the manufacturers' instructions for the type in use should be followed. Care should be taken to ensure that detonating relays are properly connected to ensure protection.

8.4.5. It is important that the full length of the detonator is firmly taped to the line of detonating cord, with the base of the detonator pointing in the desired direction of detonation in the cord.

8.4.6. To ensure that the desired delay pattern is achieved, it is important that the manufacturers' recommendations on the required separation between detonators and/or detonating relays, and adjacent lines of cord, are followed.

8.5. When using other methods.

The manufacturers' instructions should be followed at all times. Where any doubt exists, advice should be sought at the planning stage.

## Section 9. Precautions before and after firing.

9.1. The shotfirer should determine the danger area for the blast being fired, having regard to the type of operation and to local conditions, e.g. visibility. He should be responsible for checking that the danger area is clear of all personnel before going to the place of safety from which the blast is to be fired.

9.2. Sentries should be posted to guard all possible entries to the danger area, where it is large and/or not completely visible from the shotfirer's place of safety. Sentries should carry an unmistakable form of identification e.g. red flags, and should prohibit entry into their sector of the danger area until the shotfirer has checked that it is clear and has relieved him of his duty.

9.3. The post-firing examination of the blast area should not be undertaken before sufficient time has elapsed to allow for the dilution of the shotfiring fumes to a safe level. The time required should be decided after consideration of the local conditions.

9.4. The shotfirer's duty should include an examination of the blasting site, and this should be completed before work is resumed. This examination should include inspections for:

- dangerous rock conditions,
- presence of undetonated explosives and/or accessories in the rock pile,
- presence of undetonated explosives and/or accessories in blastholes,
- abnormal appearance of the blast, which could suggest that not all the blastholes had detonated correctly.

9.5. In many types of blasting work, a considerable accumulation of empty cases can be built up during the charging operation. The cases should first be checked for freedom from explosives material, and then destroyed by burning on the surface, preferably after the blast has been fired. The burning operation should preferably take place at a designated site removed from the charged boreholes, explosives materials, or explosives stores, and in accordance with manufacturers' instructions.

## Section 10. Dealing with misfires.

10.1. General.

10.1.1. No attempt should be made to fire a hole or holes which have misfired before a very careful examination of the conditions has been made. It is possible that the burden/s on the misfired hole/s will have been reduced to an unsafe level, and that blasting could result in rock projection over many times normal distances.

10.1.2. All normal rules detailed in section 9 should be observed when firing misfired holes.

10.1.3. When misfires occur, action should be divided into two parts. The first is dealing with the immediate situation. The second is an analysis of why the misfire occurred, and what can be done to prevent repetition. Consideration should always be given to seeking qualified professional assistance in the second phase, and also in the first phase if the misfire has resulted in a specially difficult situation.

10.1.4. Where the charge could be damaged by water, or where the hole cannot safely be fired, the charge should be retrieved. In horizontal holes, this is most easily done by washing the charge out. In vertical holes, it may as a last resort be necessary to fire a succession of short small diameter holes or plaster shots to dislodge the rock, and recover the explosives from the rock. Great care must be taken that the holes drilled for this purpose are angled away from the charge.

10.2. When using Electric Detonators.

10.2.1. Where the detonator leads are accessible, the individual assemblies should be checked for continuity. If the test gives a positive result, the hole/s can be fired. It is advisable to use another, tested, exploder and the cable should be examined carefully for faults before the attempt to fire is made.

10.2.2. Where the leads are inaccessible, or where the test for continuity gives negative results, the hole/s should be reprimed. The stemming should be washed out with water, and new primers inserted. This method should be used only if the main charge will not be seriously affected by the water used to remove the stemming.

10.3. When using Plain Detonators.

10.3.1. Where the fuses are accessible, they can be trimmed and the holes fired.

10.3.2. Where the fuses are inaccessible or damaged, the principles detailed in 10.2.2. should be observed.

## Section 11. Environmental factors.

11.1. General.

11.1.1. The use of explosives may result in one or more of the undesirable side effects of blasting leading to environmental pressures to have a blasting operation restricted or stopped. The extent of the problem will vary with the location of the site and with the nature of the blasting operation, but it is increasingly true to say that few blasting works can be carried out without some attention being paid to the question of environment.

11.1.2. Good blasting practice must always be the main defence against complaints about environmental factors, and many of the preceding sections in this Code are aimed, directly or indirectly, at avoiding complaints by maintaining good standards. The remainder of this section deals with specific points which do not fit naturally into the earlier points of the Code.

11.2. Nuisance.

11.2.1. No one likes to have a blasting operation near their house or their place of work, and this factor in itself gives rise to nuisance. Experience has shown clearly that most people react negatively if an operation is started without their knowledge, and also that this reaction will take the form of complaints.

Experience has also shown that the complaints will take the form of claims about specific factors, such as vibration, fly rock or air blast, rather than general nuisance. For this reason, it is advisable that every effort should be made to pre-empt nuisance complaints with good public relations BEFORE blasting starts, as it may be difficult, time-wasting, or even impossible to resolve vague complaints at a later date.

11.2.2. Suggested methods to avoid this type of complaint are:

a/ a publicity campaign before blasting is started, explaining what is being done and why,

b/ the firing of blasts at pre-arranged times, to reduce the "startle" effect,

c/ informing people individually of the blast times where this is practicable or specially requested.

11.3. Fly rock.

Fly rock is potentially one of the most serious hazards of blasting, and will inevitable give rise to complaint. Good blasting practice and unremitting attention to the detailed daily work during charging is the only protection against fly rock.

11.4. Vibration.

This subject is given proper and complete covering in "Code of Good Practice – Ground and Airborne Vibration from the use of Explosives" (Publication No. 4) published by FEEM.

11.4.1. The use of explosives in rock inevitably results in some vibration in the ground, and this is transmitted to houses and buildings where people live and work. For the blasting engineer, the main problem is that levels clearly perceptible to humans are not damaging to buildings but it is very difficult to convince complainants of this fact.

11.4.2. It is clearly in the interests of the users of explosives to reduce the unwanted effects of vibrations to a minimum, and there are many techniques which can be applied to this end. Some of these are easily applied without any cost penalty, but others will require additional drilling (e.g. smaller holes) and/or sophistication (e.g. sequence switch) which will inevitably increase the cost of the operation. It is, therefore, important at the planning stage of the blasting job that the relation cost versus complaint level, is given careful consideration. It is also important that any agreed level of vibration will be realistic for the life of the work, and not just the immediate period.

11.4.3. Good blasting practice is also good defense against high levels of vibration. Poorly executed drilling patterns can give rise to excessive confinement of some holes, which will result in higher level of vibration than expected for the weight of explosive detonated.

11.5. Air blast/Air Overpressure.

This subject is given proper and complete coverage in "Code of Good Practice – Ground and Airborne Vibration from the use of Explosives", (Publication No. 4) published by FEEM.

11.5.1. Air blast and noise can give rise to complaints when blasting is carried out in or near built-up areas. These complaints may be registered as vibration, as the air blast may cause the house to react, and therefore vibrate. It is important that the true nature of the complaint is identified, to avoid costly and time-wasting investigations.

11.5.2. Complaints resulting from air blast can be reduced by paying attention to the following factors:

a/ Avoid, wherever possible the firing of explosives without confinement. Plaster shooting is almost certain to cause complaints over a large area round the blasting site. The use of detonating cord should be critically examined in difficult situations and where there is a problem with noise the cord should be adequately covered with sand if an alternative initiation system cannot be used.

b/ Use sufficient good quality stemming for the hole diameter in use. In difficult conditions, the use of specially selected material, rather than drill fines, should be considered.

c/ Use delay blasting to reduce the amount of explosive detonating at any time.

d/ Try to fire blasts at times when the ambient noise level is high, so that any air blast becomes less noticeable against the background.

e/ Be on the watch for temperature inversions, which may cause reflection or even focusing of the air blast.

## Section 12. Underground blasting (mining and tunneling).

12.1. General.

12.1.1. In tunneling and shaft sinking work, where many holes are drilled in a limited area, the possibility of drilling into a socket containing explosive is greater than in most other types of blasting. It is important that the face is closely inspected before drilling is started, to check for undetonated charges from the previous round, and that holes are not started in sockets.

12.1.2. It will not normally be possible to examine the complete working face for undetonated charges immediately after firing. This part of the examination can only be carried out as loading of the rock proceeds.

12.1.3. Careful checks should be maintained on the position of the tunnel face in situations where the possibility exists of a blast from one working place breaking through to another.

12.1.4. Electric Storms – When using electric initiation systems the distance from openings, the thickness of overburden and the covering of surface vegetation should all be taken into consideration.

An adequate warning system should be installed where lightning discharge may be conducted to the underground working through water bearing fissures, rails, pipes etc.

12.1.5. The possibility of encountering flammable gas mixtures in the strata must be considered when choosing explosives, detonators and all other accessories. If in doubt, expert advice should be obtained.

12.2. Fumes.

12.2.1. Special consideration should always be given to the fume characteristic of any explosives in the expected conditions of use, and advise sought from the manufacturers in cases of doubt.

12.2.2. The use of explosives will always give rise to a certain amount of fumes, and it is important that these are diluted to a safe level before work is re-started. The most important factor in ensuring a safe level is the efficiency of the ventilation system. It is not possible to give limits for the necessary waiting time, as this will vary with the quantity of explosive, the ventilation and mining system, the rock type etc. Best results will always be obtained by a combination of good blasting and good mining practice. Measurements can be taken to establish when safe conditions have been reestablished.

12.3. Storage.

12.3.1. Mine and tunnel atmospheres are frequently humid, and may also be warm or hot. These conditions represent poor storage conditions for explosives materials, and every effort should be made to adhere to a rigid system of stock rotation and to keep the case contents in unopened original packaging.

12.3.2. Stores should be remote from, or protected from, any site where blasting will take place.

## Section 13. Surface bench blasting.

13.1. Drilling pattern.

Before drilling is started, a careful examination should be made for areas where the designed pattern would result in reduced burdens, or where the material in the burden is weak, or where planes of weakness exist. The drilling pattern must take these factors into account, and any abnormalities reported to the person responsible for charging the blast.

13.2. Charging.

A check should be made before charging is started to ensure that the holes have been drilled to the designed pattern, and that there are no areas of weakness and/or light burdens. Where these conditions are not fulfilled, consideration should be given to having the face surveyed, and special care must be exercised in charging the holes.

Where rock conditions are such that open fissures and caverns may be encountered, bulk explosives should be used only in conjunction with a system, e.g. borehole liners, for avoiding excessive concentrations of energy and the resultant possibility of fly rock.

13.3. Precautions before and after firing.

13.3.1. An agreed code of audible or visible signals should be used at all sites where surface blasts are fired and, notices detailing the code set up at all access points to such sites. It is important that visitors are made aware of the code.

13.3.2. It is suggested that the code of signals should contain three signals:

1. A warning signal before blasting. It is suggested that a suitable time for the warning signal is that sufficient for people to reach a place of safety before the blast is fired.
2. A firing signal.
3. An all-clear signal.

13.4. Fumes.

In open pit mining or in deep quarries fumes from blasting which may be trapped in low lying areas take considerably longer to clear than in normal surface operations. The re-entry time should be extended having regard to the prevailing weather conditions and their effect on the dispersal of the fumes. When in doubt, consideration should also be given to the regular monitoring and recording of toxic fumes, and possible forced ventilation.

13.5. Environmental factors.

Reference should be made to the "Code of Good Practice – Ground and Airborne Vibration from the use of Explosives" (Publication No. 4) published by FEEM.

## Section 14. Blasting in civil engineering.

This section includes such work as trenching, foundation excavations, roadway cuttings, but excludes tunneling and bench blasting which are covered in Sections 12 and 13 respectively.

14.1. Planning.

14.1.1. As civil engineering blasting often requires work to be carried out in urban areas, special attention should be paid to the location of gas water mains as well as overhead and underground power lines and telephone cables. The possibility of radio frequency energy from radio transmitters should also be investigated.

14.1.2. In urban areas, there may be disruption to road traffic and fore-thought will be required in dealing with members of the public and securing the co-operation of the local authorities and possibly the police. Prior notice of blasting should always be given in residential areas.

14.1.3. In urban areas blasts should be planned so as to inconvenience the general public as little as possible, and the use of blast retaining mats should be considered when the blast site is close to buildings or plant. Where blasting mats are required these should be sufficiently heavy or weighted to prevent fly during the blast. When locating mats, care should be exercised so as not to damage leads of detonators or other initiation system. Particular care should be given to ensure that initiating circuitry is not adversely affected.

14.1.4. When blasting close to overhead cables, blasting mats and blasting cables should be securely staked to the ground to prevent fly and possible fouling of the cables.

14.1.5. The choice of initiation systems should take into account any possible hazards from stray currents etc.

14.1.6. Any blasting plans should recognize areas of weakness, fissures etc. and charges adjusted accordingly.

14.2. Charging.

14.2.1. All blasts should be charged and fired within the working day so as to remove the necessity for providing overnight sentries and guards.

14.2.2. In urban areas, the initiation system should be carefully chosen so as to reduce the noise and air blast to an absolute minimum.

14.2.3. All explosives used on the site should be carefully accounted for, and no explosives left on the site except under the control of the shotfirer or other competent person.

14.3. Firing the blast.

14.3.1. A code of signals should be established and all personnel advised as to the signals and times of blasting. Sentries should be appointed and made aware of their duties.

14.3.2. After firing the shot, a post blast examination should be made by the shotfirer before workmen are allowed to return to the site. Any misfires must be dealt with immediately or the area fenced off and suitable warning notices erected and remedial action taken later.

14.3.3. In pipeline blasting operations, where close to active pipelines, age and condition of the existing pipeline should be taken into consideration. Care should be taken when sheltering behind stacked pipes since ground vibrations may upset the stability of such stacks and present danger to personnel.

## Section 15. Demolition blasting.

15.1. Planning.

Planning of the blast may require the services of specialists, e.g. structural engineers, to advise on the stability of the building to be demolished and of neighboring buildings. If possible dynamic models should be employed to simulate the demolition in advance in the case of high rise buildings.

15.2. Initiation.

Plain detonators and fuse should not be used in demolition work.

15.3. Precautions before and after firing.

15.3.1. An agreed code of audible or visible signals should be used at all sites where demolition blasts are fired and notices detailing the code set up at all access points to such sites. It is important that visitors are made aware of the code.

15.3.2. It is suggested that the code of signals should contain three signals:

1/ A warning signal, before blasting. It is suggested that a suitable time for the warning signal is that sufficient for people to reach a place of safety before the blast is fired.

2/ A firing signal.

3/ An all-clear signal.

15.3.3. Determining the danger areas for demolition work can be very difficult especially where cutting charges are used, and it is important that all personnel have taken cover when the blast is fired. The proximity to properties and services should be assessed together with the wind direction since material will be ejected. In addition the danger area can be limited by providing sufficient cover and protection from projected material. The definition of the danger area is the responsibility of the engineer in charge.

15.3.4. It is important that a last minute check is made **inside** the building to ensure all personnel have been evacuated.

## Section 16. Underwater blasting.

16.1. Planning and general considerations.

16.1.1. Advance notice of intention to carry out blasting operations should be given to the relevant civil and military authorities.

16.1.2. In the initial stage of planning, areas with underwater pipelines, or alluvial sediments should be evaluated for danger of slippage.

16.1.3. In areas close to military installations the planning of the blasting should be carried out jointly with the Military Authorities.

16.1.4. Where plaster charges are used, great care must be taken during placement to ensure that they cannot be dislodged by current or tides, or by any other mechanical forces, e.g. boats which may be present at the site.

16.1.5. The responsible shotfirer should continuously check the area around the blast site. He should therefore not be the diver.

16.1.6. The decision to pre-assemble on shore, or to prime charges in a boat or in the water, should be arrived at from considerations of minimizing the overall risk of the operation.

16.1.7. All vessels carrying explosives must carry the internationally accepted code of visual signals.

16.2. Charging.

16.2.1. Explosives and detonators for underwater use should be carefully chosen to ensure that they have the requisite degree of water resistance and that they will reliably detonate under known depths of water.

16.2.2. Where plaster charges comprise a number of individual charges these should be securely bound together to ensure complete simultaneous detonation.

16.3. Firing the blast.

16.3.1. Before firing the blast, the shotfirer should ensure that all divers and swimmers have been withdrawn from the water.

16.3.2. Before firing, a final examination of the blast area should be made by divers paying particular attention to the firing vessel itself to ensure that no explosives have become dislodged and are fouling the vessel.

16.4. Shock waves.

16.4.1. Shock is transmitted very efficiently in underwater blasting. The assessment of the danger area must take this into account to avoid claims of damage and/or injury, including damage to marine life. Plaster charges produce the greatest intensity of shock waves compared to borehole charges, and consideration should be given to the use of air bubble curtains, but in all cases, expert advice should be sought at the planning stage of the operation.

16.4.2. In underwater blasting there is always the risk of sympathetic detonation and explosives should be selected accordingly.

## Section 17. Blasting in hot materials.

17.1. General rules when blasting hot rock from +70°C to +100°C and all hot masses over +100°C.

17.1.1. In the majority of cases where blasting must be carried out in hot materials the system used will be specially designed to suit the local site requirements.

17.1.2. The system chosen should take into account the expected temperature of the material at the time of charging and if this exceeds +70°C the detonators and explosives manufacturers, advice should be sought if protection against heat is not possible and time exposure cannot be sufficiently limited.

17.1.3. Every effort should be made to reduce the time to the practical minimum from the start of the operation to the firing of the blast.

17.1.4. In areas with hazards of high stray currents such as steel furnaces, high energy electric detonators, non electric detonators, or transformer coupled should be used.

17.2. Additional rules when blasting hot masses over +100°C.

17.2.1. Only personnel with special training or under direct expert supervision should be employed. The explosive charge should be prepared at a distance where there will be no effect from the heat.

17.2.2. The charge should be inserted into a special loading tube (for example, a thick walled cardboard tube). For ease of handling it is advisable that the tube should be longer than the borehole depth. The charge should also be stemmed within the tube.

17.2.3. Precautions should be taken to ensure that the charge does not slide out of the borehole.

17.2.4. When firing electrically, before charging is commenced, the firing circuit should be laid out and checked, including connecting to the exploder. All detonator leads and firing cables should be so placed that they will not suffer any damage from the heat.

17.2.5. When firing non-electrically similar procedures should be adopted as specified in 17.2.4. with the purpose of ensuring that the firing circuit is complete before charging is commenced and that the lead lines and connectors will not suffer any damage from the heat.

17.2.6. The second warning signal should be given and the area evacuated by all personnel not needed for loading the holes before making the final connection to the exploder.

17.2.7. The temperature within the borehole should be reduced as much as possible by using cold air or water.

17.2.8. Borehole liners should be employed when blasting unconsolidated material

17.2.9. Both lined and unlined boreholes should be checked before loading with a test tube having at least the same diameter as the loading tube.

17.2.10. When loading more than one borehole all charges should be loaded at the same time under the direct supervision of the shotfirer in charge. A maximum number of six persons is advised with each person having no more than one charge in each hand.

17.2.11. Immediately the holes are loaded the area must be evacuated by a predetermined route.

17.2.12. The blast must be fired without any further delay.

17.2.13. In the event of a misfire, the area should not be approached until the explosive has detonated through the effect of the heat.

## Section 18. Blasting with black powder.

18.1. General considerations.

18.1.1. Special attention should be given to ensure that smoking or the use of naked flame does not occur in the vicinity of the charging operation, and a minimum safe horizontal distance of 50 m is recommended.

18.2. Charging.

18.2.1. When using black powder precautions should be taken to prevent it from entering into cracks, fissures, voids or other boreholes which do not form part of the overall blast pattern. This may require the use of anti-static loading hoses, lay flat plastic tubing or pre-formed channels which would be capable of reaching the deepest parts of the boreholes and/or fissures.

18.2.2. If holes are to be "sprung" an interval of at least one hour should be allowed to elapse between the firing of the charge and the loading of the next charge. This is to permit cooling of any hot spots within the sprung hole. However, flushing by compressed air may reduce the time. Springing should not be carried out in the vicinity of any charged holes because of the danger of exploding sympathetically.

## Section 19. Blasting in ice and snow.

19.1. Blasting in ice.

19.1.1. General considerations.

19.1.1.1. The shotfirer in charge should have received specialized training in the subject.

19.1.1.2. Before charging commences rescue equipment should be checked and ready to hand. This includes boats, ladders, poles, life belts, life vests etc.

19.1.1.3. In the situation where there is danger of the shotfirer falling into the water he should be provided with a life vest and be attached to a safety rope.

19.1.2. Charging.

19.1.2.1. Explosives with good water resistance should only be used.

19.1.2.2. Precautions should be taken to ensure that the ice flow being blasted together with the explosives does not go adrift before the blast is fired.

19.1.2.3. When using plaster charges ensure that they are well covered before detonation.

19.1.2.4. When clearing an ice jam with sheets of ice obtruding at an angle from the water the explosive charges should be placed on the underside of the plates.

19.1.2.5. When metal tubes are employed for the purpose of placing charges in otherwise inaccessible positions the tubes should always be removed before firing.

19.2. Blasting in snow for avalanche control.

19.2.1. General considerations.

19.2.1.1. The shotfirer in charge should have received specialized training in the subject.

19.2.1.2. The shotfirer in charge should have a thorough knowledge of the local terrain.

19.2.1.3. The shot firing team should have received training in first aid in the event of accidents.

19.2.1.4. Due to difficulty of access, transport of explosives by rucksack or back pack is permitted provided explosives and detonators are carried by separate persons.

19.2.1.5. Before any charges are laid the entire area likely to be affected by the expected avalanche should be evacuated.

## Section 20. Agricultural blasting.

Blasting operations may be required in agriculture for land clearance including the removal of tree stumps, boulders and the reduction of rocky outcrops, ditching and the excavation of wildlife ponds.

20.1. Planning.

20.1.1. Tree stumps should be examined to assess the root system and the general condition of the wood. Allowance should be made for any obvious signs of decay, and every effort should be made to locate the main charges under the heaviest roots.

20.1.2. Buried or partially buried boulders should be surveyed in order to assess their size and charges calculated to ensure the removal of the boulder from the ground without excessive ejection.

20.1.3. When breaking boulders, the size of the boulder should be assessed before the charge is calculated. The method of boulder blasting should reflect the environmental considerations and plaster or lay-on charges only used in remote areas.

20.1.4. Ditching operations and wildlife pond formation should recognize the type of ground and only waterproof explosives should be used in wet or marshy areas. The proximity to properties and services should be assessed together with the wind direction since material will be ejected.

20.1.5. All personnel should be withdrawn from the blast area which should be extended on the down wind side of ditches or wildlife pond blasts.

20.2. Charging.

20.2.1. Holes should be excavated, drilled or punched into the subsoil and the diameter should allow easy loading of the explosive.

20.2.2. In ditching and wildlife pond formation blasting, small scale test blasts may have to be undertaken in order to assess the type of ground conditions and the final blast based upon the results of these test blasts.

# 15. REFERENCES


| | |
|---|---|
| Abramsson, Eddy | Låga tryck från explosioner |
| Atlas Copco<br>Atlas Copco | Manual Fourth Edition<br>Construcción y Mineria Subtérranea |
| Atlas Powder Co<br>Atlas Powder Co | The Basic Principles of Priming ANFO<br>Explosives and Rock Blasting |
| Bergqvist, Ingvar | Technical Reports:<br>En jämförelse mellan olika metoder att enligt "Rock Blasting" beräkna försättingar vid pall-sprängning<br>Bottenladdningens ekonomiskt optimala höjd vid pallsprängning<br>Nitro Nobel AB, Sweden |
| Brännfors, Sten | Bergsprängningsteknik<br>Svenska Bokförlaget/Bonniers |
| Bååth, Jan<br>Daugaard, Thomas<br>Hermansson, Michael<br>Noren, Jan | SPRÄNGTEKNIK<br>Speciella sprängningar<br>Nitro Nobel AB, Sweden |
| Dick, Richard A<br>Fletcher, Larry R<br>D'Andrea, Dennis V | Explosives and Blasting Procedures Manual<br>Bureau of Mines Information Circular 1983<br>United States Department of the Interior |
| Dobrin, Milton B | Introduction to Geophysical Prospecting |
| DuPont | Blasters' Handbook |
| Ensign Bickford | Primacord, detonating cord |
| Gustafsson, Rune | Bergsprängningsboken<br>SPI Göteborg |
| Harries, G<br>Mercer, J.K. | The science of blasting and its use to minimize costs<br>Proc. Australian Inst. Min. Metal.<br>Annual Cinf. 1975 |
| Hermansson, Mikael | Försiktig Sprängning<br>Nitro Nobel AB, Sweden |
| Holmberg, Roger<br>Holmberg, Roger | Kast och buller vid sprängning<br>Flyrock<br>Swedish Detonic Research Foundation, Sweden |
| Holmberg, Roger et al | Vibrations generated by traffic and building construction activities<br>Swedish Council for Building Research, Sweden |
| Jonsson, Bert | Seismological Prospecting |

| | |
|---|---|
| Palm, Hans | Nitro Nobel AB, Sweden |
| Johansson, C.H.<br>Persson, P.A. | Detonics of High Explosives<br>Academic Press, London |
| Langefors, Ulf<br>Kihlström, Björn | The Modern Technique of ROCK BLASTING<br>Halsted Press |
| Larsson, Bernt | The role of correct interval times in rock blasting<br>Nitro Nobel AB, Sweden |
| Larsson, Bernt<br>Clark, Dennis A | Cost savings and improved stability through optimized rock blasting<br>VME – Nitro Consult, U.S.A. |
| Lundborg, Nils | En metod för beräkning av sannolikheten för stenträff på stora avstånd från sprängplatsen |
| Lundborg, Nils | The Probability of Flyrock |
| Lundborg, Nils | Något om luftstötvågor, ljudbangar och buller<br>Swedish Detonic Research Foundation |
| Meyer, Rudolf | Explosives<br>Verlag Chemie |
| Nitro Nobel | Rock Blasting Technique<br>Blasting Techniques Dept |
| Parasnis, D.S. | Principles of Applied Geophysics |
| Persson, A | ANFO-explosives, initiation, detonation, toxic fumes |
| Persson, A | Stötvågor från detonerande laddningar<br>Swedish Detonic Research Foundation |
| Persson, P.A. | Bench drilling – an important first step in the rock fragmentation process<br>Atlas Copco Bench Drilling Symposium 1975 |
| Rosell, Sven<br>Nitro Nobel | Airborne vibration<br>Nitro Nobel INFO & NEWS # 8 Sept 1985 |
| Sjöberg Conny | Cautious tunnel blasting<br>World Construction June 1975 |
| Sjöberg, Conny | Håltagning i betong och partiell rivning genom försiktig sprängning<br>Husbyggaren nr 2 mars 1984 |
| Sjöberg, Conny | Totalrivning av höga byggnader och andra större konstruktioner medelst sprängning |
| Sjöberg, Conny | Klenhålssprängning tillämpad inom ROT-sektorn<br>Nitro Consult AB, Sweden |
| Svärd, Johan | ANFO-sprängämnets genombrott och fortsatta utveckling<br>Nitro Nobel AB, Sweden |

# Appendix 1
# CONVERSION TABLE

Conversion factors for units used in this book.

| **Multiply** | | **By** | **To obtain** | |
|---|---|---|---|---|
| Atmosphere | (atm) | 101.3 | kilopascal | (kPa) |
| Cubic centimeter | (c.c.) | $61.02 \times 10^{-3}$ | cubic inch | ($in^3$) |
| Cubic meter | ($m^3$) | 35.32 | cubic foot | ($ft^3$) |
| Cubic meter | ($m^3$) | 1.308 | cubic yard | ($yd^3$) |
| Gram | (g) | 15.43 | grain | (gr) |
| Gram | (g) | $35.27 \times 10^{-3}$ | ounce | (oz) |
| Gram/cubic centimeter | (g/c.c.) | 62.43 | pound/cubic foot | ($lb/ft^3$) |
| Gram/meter | (g/m) | 4.704 | grain/foot | (gr/ft) |
| Kilogram | (kg) | 2.205 | pound | (lb) |
| Kilogram/cubic meter | ($kg/m^3$) | $62.43 \times 10^{-3}$ | pound/cubic foot | ($lb/ft^3$) |
| Kilogram/cubic meter | ($kg/m^3$) | 1.686 | pound/cubic yard | ($lb/ft^3$) |
| Kilogram/square meter | ($kg/m^2$) | $1.422 \times 10^{-3}$ | pound/square inch | ($lb/in^2$) |
| Kilogram/sq. centimeter | ($kg/cm^2$) | 14.22 | pound/square inch | ($lb/in^2$) |
| Kilometer | (km) | 0.6214 | mile | (mile) |
| Kilometer/hour | (km/h) | 0.6214 | mile/hour | (mile/h) |
| Kilopascal | (kPa) | 0.1450 | pound/square inch | ($lbf/in^2$) |
| Liter | (l) | 0.0353 | cubic foot | ($ft^3$) |
| Liter | (l) | 0.220 | gallon | (gal) |
| Liter | (l) | 0.2642 | US gallon | (US gal) |
| Meter | (m) | 3.281 | foot | (ft) |
| Meter/second | (m/sec) | 2.237 | mile/hour | (mile/h) |
| Millimeter | (mm) | $39.37 \times 10^{-3}$ | inch | (in) |
| Square meter | ($m^2$) | 10.76 | square foot | ($ft^2$) |
| Square centimeter | ($cm^2$) | $0.155 \times 10^{-3}$ | square inch | ($in^2$) |

Degress Celsius, °C, to degrees Fahrenheit, °F:
$°F = 9/5 \bullet °C + 32 = 1.8 \bullet °C + 32$

Notes