ROCK EXCAVATION HANDBOOK

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PREFACE

The construction business is continuing to grow at a fast pace worldwide. The construction business is closely related to the world economy as well building of infrastructure. The building volume is increasing even though there are big regional variations. Interesting part of construction is rock excavation and the big variation of rock properties is challenging everybody involved; contractors, consultants, equipment suppliers etc. At the same time development of technology is offering new methods and solutions for surface and underground applications. Sandvik Tamrock has published this handbook to present these applications as well basic information to assist selection.

Thus this handbook will serve anyone who deals with the construction excavation business. It is also good reference tool for those studying related subjects.

Several people participated in writing this book and therefore I like to extend my thanks to Unto Ahtola, Karlheinz Gehring, Pekka Kesseli, Peter Kogler, Pasi Latva-Pukkila, Arne Lislerud, Maunu Mänttäri, Jukka Naapuri and Tuomo Niskanen. Special thanks to Kirsi Nieminen, Nordberg Inc, Aimo Vuento and his students at South Carelia Polytechnic, Raimo Vuolio, Finnrock Inc and Pekka Särkkä, Helsinki University of Technology. Finally I like to thank Leena K. Vanhatalo collecting and editing this handbook.

Matti Heiniö
Editor in chief

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1. INTRODUCTION

Excavation activities at construction sites are very diverse. Contracting customers also vary significantly in size. What is a typical construction project made up of? The answer is that all construction projects are time-limited. Project duration varies from one month to several years, and the schedules are almost always tight. Usually, immediate mobilization takes place once the contract has been awarded and often there are penalties involved if the original schedule does not keep. It is therefore important to choose the right excavation method together with the right equipment to keep the project on schedule. This handbook covers all modern excavation methods and also provides some recent case stories.

Excavation methods can be divided into groups. The following classification shows how it is handled in this book.

Both aggregate and limestone quarrying by the drill & blast method and mechanical method is discussed in the chapter called “Quarrying”.

General contracting includes a wide range of projects from rock foundation in buildings and roads to channel excavation in dams, road cuttings etc. Underwater excavation is also included in this section.

Today, demolition and recycling play an important role and are discussed in detail in the chapter on general contracting.

In the “Tunneling” chapter, which also includes underground excavation, both traditional drill & blast and mechanical method are discussed. Underground excavation varies significantly from sewage tunnels and powerhouses to railway and highway tunnels as well as from warehouses and parking halls to theaters, swimming pools and ice-hockey halls.

Dimensional stone quarrying is explained according to each method in use.
The last chapter concerning excavation methods describes water well and exploration drilling.

The final chapter is dedicated to project management. It describes issues that should be considered and remembered when handling an excavation project. Last but definitely not least is service support.

At the end of every chapter, there is a case description providing a real-life example of a typical excavation site.

2.1 GEOLOGICAL CLASSIFICATION OF ROCKS

Geological and mechanical properties of rock are interrelated; both must be taken into account when planning rock excavation, from designing underground openings and quarries to estimating drilling and blasting performance. Rock characteristics are determined primarily by origin, formation and mineral composition (FIGURE *2.1.-1*).

Geologically speaking, the earth is in a state of flux where both rocks and minerals are constantly being formed and altered (FIGURE *2.1.-2*). It is convenient to divide the rocks in the earth´s crust into three categories based on origin: igneous, sedimentary and metamorphic rocks.

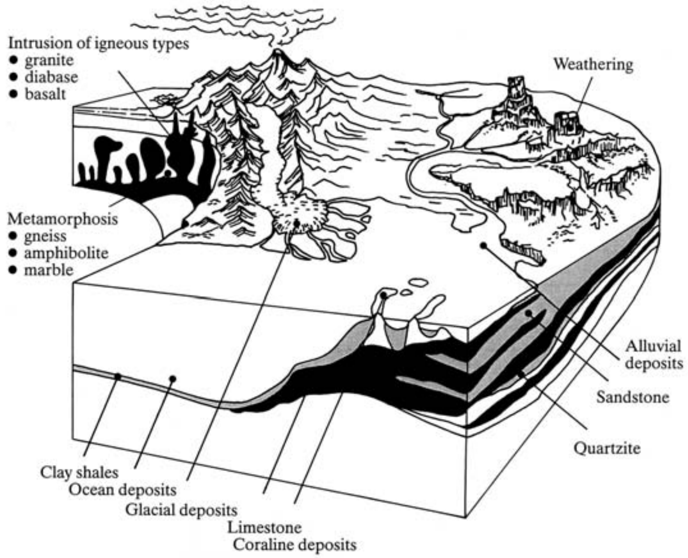


FIGURE *2.1.-1. Formation of minerals and rocks.*

MINERALS

All rocks consist of an aggregate of mineral particles. The proportion of each mineral in the rock, together with the rock's granular structure, texture and origin serves as a basis for geological classification.

A mineral may be defined as an inorganic substance that has consistent physical properties and a fixed chemical composition. With the exception of some carbon forms, sulfur and a few metals, all minerals are chemical compounds each containing two or more elements in fixed proportion by weight. Some elements are present in many minerals, the most common being oxygen and silicon, while others, including most precious and base metals, form an insignificant proportion of the rocks within the earth´s crust.

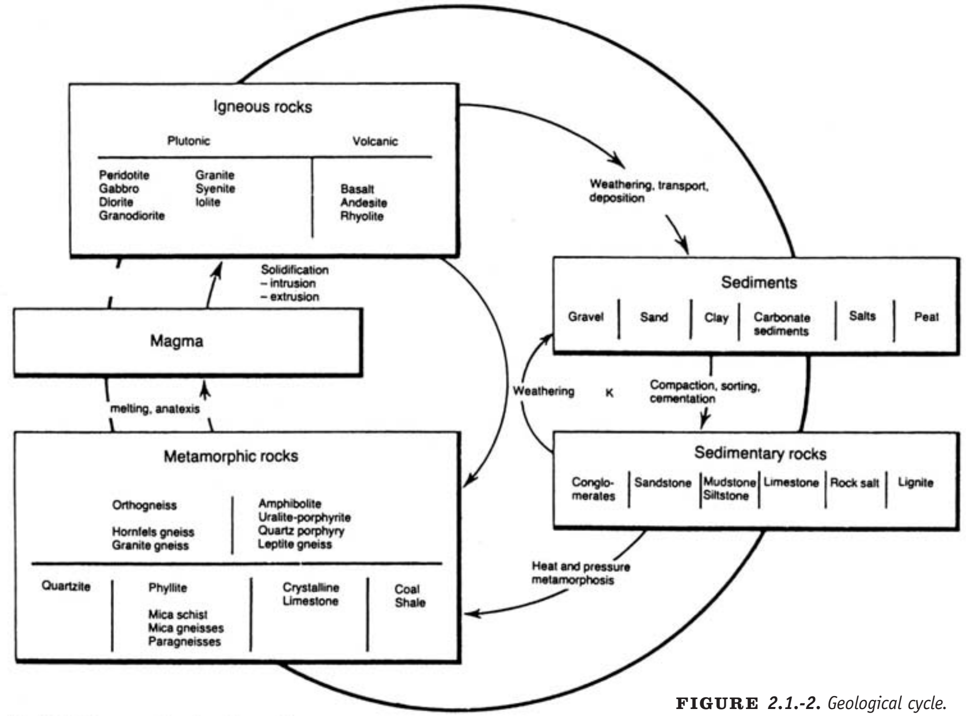


FIGURE *2.1.-2. Geological cycle.*

The way in which the composition of the earth´s crust is dominated by eight elements is presented in ***Table 2.1.-1.*** These elements, together with some others elements, form twelve common minerals which comprise approximately 99% of the earth´s crust. The remainder of the over 1,000 known rock-forming minerals make up less than 1% of the earth´s crust.

Table 2.1.-1. *Major chemical elements in the earth´s crust.*

Chemical Elements	*Weight Percent (%)*	*Volume Percent (%)*
Oxygen (O)	46.40	94.04
Silicon (Si)	28.15	0.88
Aluminum (Al)	8.23	0.48
Iron (Fe)	5.63	0.49
Calcium (Ca)	4.15	1.18
Sodium (Na)	2.36	1.11
Potassium (K)	2.09	1.49
Magnesium (Mg)	2.33	0.33

It can be assumed, therefore, that most, if not all, rocks encountered in mining and civil engineering consist of two or more minerals, each with its own particular set of physical properties that can affect the rock's engineering properties, such as the preferred cleavage direction and fracture. Hardness and crystal structure used to define minerals can, in some situations, determine the rock's reaction to outside forces, particularly where large amounts of a relatively soft mineral with marked fracture properties, such as mica or calcite, or of a particularly hard mineral such as quartz, are present.

ROCKS

Magma is essentially a hot silicate melt (600-1,200°C), and is the parent material of igneous rocks. Magmas and the formation of igneous rocks can be observed in volcanic regions. Usually, magma solidifies within the crust, and the formed rocks are later exposed at the surface due to erosion or earth movements - hence their classification as plutonic (intrusive), hypabyssal or volcanic (extrusive), depending on the depth and rate of cooling, which affects texture and crystal size.

Igneous rocks are also subdivided by composition into acidic, intermediate, basic (mafic) and ultrabasic (ultramafic) rocks depending on the amount of silica in the composition as presented in ***Table 2.1.-2.*** A relatively high hardness as to mineral constituents in igneous rock can immediately be seen. Mica content tends to be small.

Table 2.1.-2. *Geological classification of the most common igneous rocks.*

Texture	***Acidic*** ***> 66% silica***	***Intermediate*** ***66 - 52% silica***		***Basic*** ***< 52% silica***	***Ultrabasic*** ***< 45% silica***
PLUTONIC (coarse grained)	Granite	Syenite	Diorite	Gabbro	Peridotite Dunite Pyroxenite
HYPABYSSAL	Micro-Granite	Micro-Syenite	Micro-Diorite	Diabase	
VOLCANIC (fine grained)	Rhyolite	Trachyte	Andesite	Basalt	
Principal Mineral Constituents	Quartz Orthoclase (Mica)	Orthoclase Plagioclase (Mica)	Plagioclase Hornblende	Augite Plagioclase	Augite Olivine

Sedimentation is the result of atmospheric and hydrospheric interaction on the earth's crust. The original composition of the crust, igneous rock minerals, are more or less readily attacked by air and water. Having been formed at high temperatures, and occasionally high pressures, they do not remain stable under significantly varying conditions. Silicates vary considerably in chemical stability. Susceptibility to chemical attack of common rock-forming minerals can be ranked as follows: olivine, augite and calcium feldspar
> hornblende, biotite and sodium feldspar
> potassium feldspar > muscovite > quartz.

Quartz is the only common mineral in igneous rocks that is highly resistant to weathering processes. All minerals tend to be altered when attacked by oxygen, carbonic acid, and water; forming new minerals that are more stable under the new conditions. The altered rock crumbles under the mechanical effects of erosion and is transported by wind, water, or ice and redeposited as sediments or remain in solution.

Sedimentary rocks can be subdivided into three main groups according to whether they were mechanically formed, formed from organic remains or chemically deposited. (***Table 2.1.-3.)***

Table 2.1.-3. *Geological classification of the most common sedimentary rocks.*

Method of Constituents Formation	***Classification***	***Rock Type***	***Description***	***Principal Mineral***
MECHANICAL	Rudaceous	Conglomerate	Large grains in clay matrix	Various
	Arenaceous	Sandstone	Medium round grains in siliceous, calcareous or clay matrix	Quartz, Feldspar, Mica, Calcite
		Breccia	Coarse angular grains in matrix	
	Argillaceous	Clay	Micro-fine grained - plastic structure	Kaolinite, Quartz, Mica
		Shale	Harder - laminated compacted clay	
ORGANIC	Calcareous (siliceous, ferruginous, phosphatic)	Limestone	Fossiliferous, coarse or fine grained	Calcite
	Carbonaceous	Coal		
CHEMICAL	Ferruginous	Ironstone	Impregnated limestone or clay (or precipitated)	Calcite, Iron Oxide
	Calcareous (siliceous, saline)	Dolomitic Limestone	Precipitated or replaced limestone, fine grained	Dolomite, Calcite

In engineering, the most important sedimentary rocks are arenaceous (sand), argillaceous (clay) and calcareous (limestone) rocks. Typical arenaceous rock consist of discrete fragments of minerals, such as quartz and feldspars, held together by a matrix of clay, calcite or hydrothermal quartz. Thus, when a sandstone is broken, fractures follow the weaker clay or calcareous cement rather than propagating across the stronger grains. An argillaceous rock such as shale consists of minute particles held weakly together and comprising largely of kaolinite. Calcareous rocks consist of organic remains, or precipitates, mainly in the form of calcite.

Metamorphism is defined as the result of the processes that, beyond weathering, causes the recrystallization of either igneous or sedimentary rock material. During metamorphism, the rock remains essentially solid; if remelting takes place, magma is produced, and metamorphism becomes magmatism. Metamorphism is induced in solid rock as the result of pronounced changes in the temperature (200-800°C), pressure and chemical environment. These changes affect the physical and chemical stability of a mineral assemblage, and metamorphism results from the establishment of a new equilibrium. The rock's composition changes to minerals that are more stable under the new conditions and the minerals arrange themselves through the production of textures that are better suited to the new environment. Metamorphism thus results in partial or complete rock recrystallization, with the production of new textures and new minerals.

Heat, pressure, and chemically active fluids are the driving forces in metamorphism. Heat may be created by increasing temperature with depth or by contiguous magmas. There are two kinds of pressure: hydrostatic (uniform) pressure, which leads to a change in volume; and directed (shear) pressure, which leads to a distortion of shape. Uniform pressure results in the production of granular, non-oriented structures; directed pressure results in the production of parallel or banded structures. Uniform pressure affects the chemical equilibrium by promoting a volume decrease, i.e. the formation of minerals of higher density. The action of chemically active fluids is critical in metamorphism, since even when it does not add or subtract material from the rock, it promotes reaction by solution and redeposition. When it adds or subtracts material, the process is called metasomatism. It is likely that some degree of metasomatism accompanies metamorphism. Water is the principal chemically active fluid, and it is aided by carbon dioxide, boric acid, hydrofluoric and hydrochloric acids as well as other substances, often of magmatic origin.

Two major types of metamorphism are commonly recognized: thermal (contact) metamorphism, and regional metamorphism. Contact metamorphism is created around bodies of plutonic rocks. In this case, the temperature of metamorphism was determined mainly by proximity to the intrusive magma, which may also have given off chemically active fluids that stimulated recrystallization of the country rock. Regional metamorphism, as the name implies, is metamorphism developed over large regions, often over thousands of square kilometers in the root regions of fold mountains and in Precambrian terrain. (*Table 2.1.-4.)*

The earth´s crust is made up of 95% igneous rock, 5% sedimentary rock and an insignificant proportion of metamorphic rock. This does not, however, give a completely accurate picture of the kind of rock likely to be encountered in engineering projects. It is assumed that the earth's crust is 30 - 50 km thick. Virtually all major projects take place within the first few kilometers of the surface that contain the major part of sedimentary rocks. An engineer working on or near the surface must often contend with rock that is primarily sedimentary or metamorphosed. In addition, a high percentage of sedimentary rock is argillaceous, while the majority of the rest is arenaceous or calcareous.

Table 2.1.-4. *Geological classification of the most common metamorphic rocks.*

Classification	***Rock***	***Description***	***Principal Mineral Constituents***
Contact	Hornfels	Micro-fine grained	Feldspar, Quartz, Mica
Regional	Quartzite	Fine grained	Quartz, Feldspar
	Marble	Fine to coarse grained	Calcite or Dolomite
	Gneiss	Medium - fine grained	Feldspar, Hornblende
	Slate	Rock cleavage	Kaolinite, Mica
	Phyllite	Cleavage surfaces	Mica, Kaolinite
	Schist	Finely foliated	Feldspar, Quartz, Mica
	Felsic Gneiss	Coarsely foliated, banded	Feldspar, Quartz, Mica

Argillaceous rock is mainly comprised of two types of shale: consolidated and cemented. Both are normally closely bedded or laminated. The former is reasonably strong in a dry state, but weak when wet; the latter tends to have intermediate strength under most conditions, but is easily deformed under pressure. The problems encountered when mining, tunneling or building foundations in this rock type are immediately apparent.

ROCK MASS DISCONTINUITIES

A rock mass is generally considered to be a linear elastic material in the absence of specific information on rock mass discontinuities. Most rock formations are fractured to some extent; where fracture planes represent non-continuous structural elements in an otherwise continuous medium. The stability of rock slopes and underground excavations are two areas of geotechnical engineering in which the effect of intact rock properties is perhaps less dominant than the influence of rock mass discontinuities.

The structural mapping of rock formations consists of identifying the rock type, its distribution and degree of fracturing, and rating the predominant types of discontinuities. For practical use, this information must be accurately structured by geotechnical classification systems specially designed for predicting rock mass behavior regarding structural stability and excavation performance in rock. (**FIGURE** *2.1.-3.)*

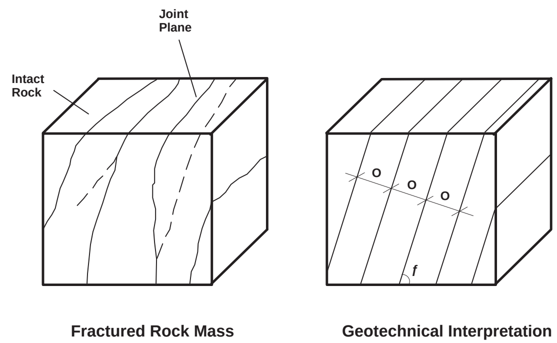


FIGURE ***2.1.-3.*** *Illustration of typically fractured rock mass by a single set of joints; and a simplified geotechnical model consisting of regularly spaced joints of similar strength.*

When two or more intersecting fracture sets are present in the rock mass, an equivalent or mean fracture spacing based on the accumulated volumetric fracture plane area is:

$$O_{mean} = (\sum 1 / O_{set})^{-1} = (\sum \text{fracture area per m}^3)^{-1} = [m^2 / m^3]^{-1}$$
$$O_{mean} = [\tfrac{1}{1} + \tfrac{1}{0{,}5} + \tfrac{1}{0{,}5}]^{-1} = 0{,}2m$$

In the NTH tunnel boring performance classification system, fracture types are grouped into four classes based on fracture strength (aperture or openness, persistence, surface roughness and waviness, and infilling material) :

- Systematically fractured rock mass characterized by:
  - parallel-oriented joint sets (rated Sp)
  - parallel-oriented fissure sets (rated St)
  - foliation or bedding planes, or parting sets (rated St)
- Non-fractured rock mass (rated St 0)
- Marked single joints (rated ESP)
- Shear zones - evaluation of necessary ground support work rather than increased net exca vation rates is required

The combination of fracture type or fracture strength rating, fracture set spacing and fracture plane orientation to the tunnel axis forms the basis of the rock mass fracture factor k_S. The fracture factor k_S for fissures and foliation planes is shown in **FIGURE** *2.1.-4.*

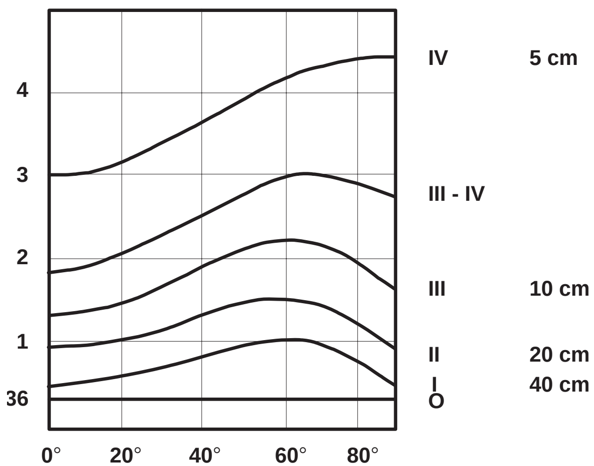


FIGURE *2.1.-4. Fracture factor ks for full-face tunnel boring performance prediction as a function of fissure class rating, angle α and the mean spacing between weakness planes.*

TBM advance rates are more or less proportional to the fracture factor k_S. However, unlike full-face tunnel boring machines, partial face cutting machines, like the TM60, are typically equipped with a profile cutting control system which maintains the tool depth of cut at a preset value. Thus the degree of rock mass fracturing does not affect the TM60's net cutting

rate (unless the operator changes the set-point values) but results in reduced mean tool forces when excavating increasingly fractured rock.

$$\propto \; = \; \arcsin [\sin f \bullet \sin (r - s)]$$

2.2 MECHANICAL PROPERTIES AND ROCK BEHAVIOUR

Rock strength, or rock resistance to failure under load, is a mechanical rock property mainly dependent on the nature of the rock itself. Rock cuttability, on the other hand, depends not only on the rock, but also on the working conditions as well as the cutting process (depth of cut, tool size, cutting speed, axial force, presence and extent of wetting, etc.). Therefore, the environment for rating rock cuttability/drillability is continuously changing as rock excavation methods improve.

Systems for rating rock cuttability and drillability for specific cutting/drilling methods (such as percussive drilling, rotary drilling, drag-tool and roller-disk cutting etc.) have been developed resulting in separate rating systems for each method. The rating systems are not directly connected, making it difficult to compare different cutting/drilling methods. Additionally, they tend to be outdated as cutting/drilling technologies develop.

A variety of apparatus and procedures has been developed for measuring mechanical rock properties. This has simplified the study of cutting/drilling processes including the effects various mechanical rock properties and other factors have on rock cutting/drilling performance.

Mechanical rock properties may be grouped as follows:

1. Strength
   - Resistance to (bulk) failure under elementary stresses such as compression, tension or shear
   - Effect of confining pressure, temperature, strain rates, pore- fluid pressure, specimen size, etc. on strength properties

2. Deformability
   - Resistance to change of shape or volume
   - Elastic and thermal expansion constants

3. Hardness
   - Resistance to a local (surface) failure by indentation or scratching

4. Fracture toughness
   - Resistance to fracture propagation

5. Coefficients of friction
   - Resistance to sliding of two bodies with planar surfaces in contact

6. Crushability and millability
   - Resistance to comminution (reduction of a substance to a powder)

7. "Extractability"
   - Resistance to fragmentation and disruption by different extraction processes such as rock cuttability, drillability, blastability, loadability of blast- rock and pumpability of cuttings under certain "idealized" or standard operating conditions

8. Abrasivity
   - Ability of rock to induce wear on mechanical tools and apparatus

Most physical tests involve tabulation of a series of readings and calculation of an average which represents the whole.
The question arises as to how representative this average is as the measure of the characteristic under investigation. Three important factors challenge the result:

- Instrumentation and procedural errors
- Variations in the rock specimens being tested
- Representability of selected rock specimens for the rock formation or zone under investigation as a whole

The largest source of error when determining mechanical rock properties for rock formations or zones is without a doubt the representability of the selected rock specimens.

2.3. RATING ROCK MASS CUTTABILITY AND DRILLABILITY

While the geological classification of rocks based on origin, mineral content and geological structure is generally useful for indicating certain strength parameters and trends, such classification provides little information to the engineer designing in or excavating rock. The engineer requires a functional geomechanical classification of rock mass properties for use as design and performance prediction criteria.

The following test methods for rating rock mass cuttability and drillability for performance prediction purposes is valid for the listed rock cutting tools:

- Roller-disk and studded roller-disk cutters
- Rotary tricone bits
- Drag tools
- Percussive drilling bits

Rock mass cuttability and drillability is, in its simplest form, defined as a factor proportional to net cutting or net penetration rates, or specific cutting/drilling energy. However, specific energy is closely linked to the apparatus or drilling equipment with which it was determined. Another, and perhaps more precise, definition for rock cuttability is rock resistance to tool indentation for a unit depth of cut, such as the critical normal force Fn_1 in roller-disk cutting, or K_1 for percussive drilling.

Today, several empirical test methods are used for rating rock mass cuttability and drillability for performance prediction purposes. These methods can be divided into the following groups:

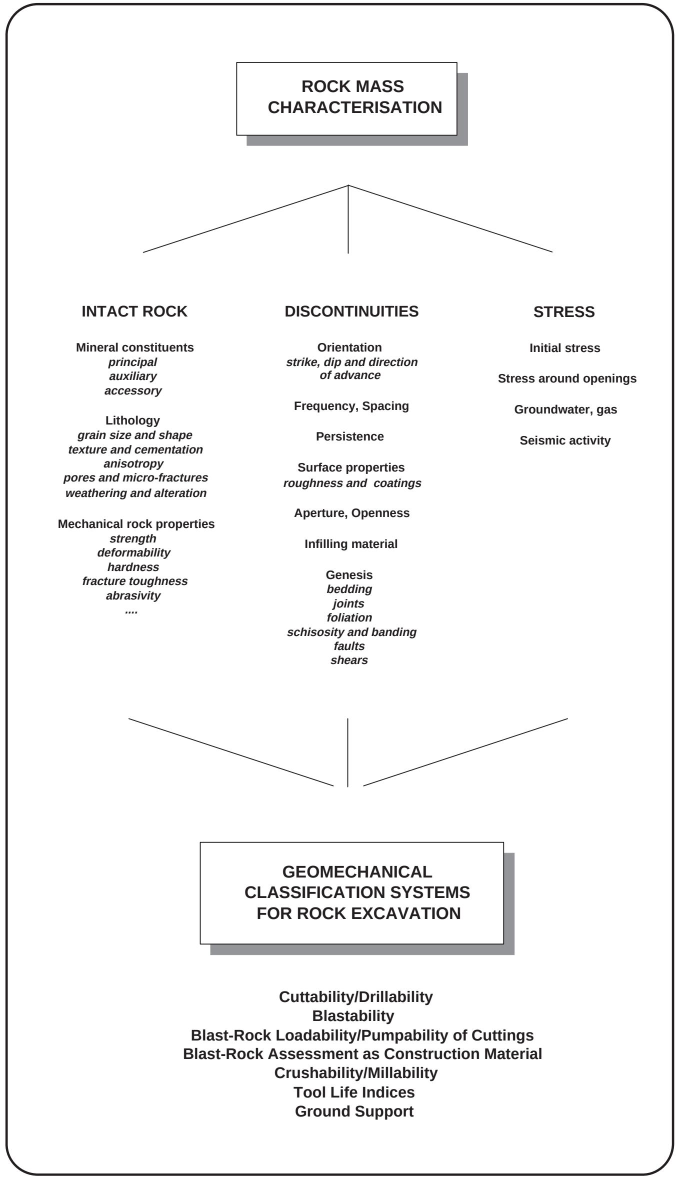


FIGURE *2.2.-1. Relationship between rock mass characterization and geomechanical classification systems for rock excavation.*

1. Compiled historic performance data (generally net cutting or net penetration rates) for a given cutting/drilling equipment and tool combination by referencing *net penetration rates to results obtained in a standard rock type* as a means of rating rock cuttability and drillability. The most commonly used standard rock types are:
   - Barre Granite from Vermont, USA
   - Dresser Basalt from Wisconsin, USA
   - Myllypuro Granodiorite from Tampere, Finland

2. Compiled historic performance data including the utilized power levels for a given cutting/drilling equipment and tool combination by correlating the *specific cutting energy to mechanical properties of rock* as a means of rating rock cuttability/drillability. The most commonly used mechanical rock properties are:

- Uniaxial compressive strength, UCS
- Brazilian tensile strength, BTS
- Point load index, I_S

3. *Stamp tests* based on impact loading and crushing of a confined solid or aggregated specimen of intact rock. Due to the impact loading and crushing nature of stamp tests - they represent the relative energy required to break a given rock volume; thus allowing for the cutting/drilling performance or specific energy in the field to be related to stamp test indice values.
The most commonly used stamp tests for rating drillability are:
- Drilling Rate Index, DRI
- Protodyakonov Rock Hardness, *f*
- Rock Impact Hardness Number, RIHN

Performance prediction models based on rock cuttability/drillability indices often include the effects of porosity and rock mass discontinuities by incorporating correction factors or modifiers for these rock mass characteristics using back analysis of experimental field performance data.

4. Laboratory *linear cutting tests* for roller-disk and drag-tool
cutting for rating rock cuttability. In addition, cutterhead force prediction as a function of net cutting rates in non-fractured rock mass conditions can be made using analytical models by combining linear cutting test results with cutterhead lacing designs. Refer to Chapter 3.7.

5. Numerical simulation with *finite element and particle flow codes*. Rock loading by roller-disk cutters causes macro-fractures to initiate from the corners of the tool rim, and to propagate sideways and upwards in curved trajectories. Preliminary results also indicate that a small shear load of approx. one tenth the normal force significantly modifies the stresses in the rock around the tool path. More importantly, in kerf cutting, tensile stresses may develop from the adjacent kerf; hence it is possible for macro-fracture propagation to occur from an adjacent kerf as well as from the kerf currently being cut.

6. *Analytical analysis and simulation of stress wave propagation* combined with *bit indentation tests* (static or dynamic K_1 values) to incorporate the dynamic nature of rock loading and bit indentation encountered in percussive drilling. An example of this method is the CASE program developed by Sandvik Mining and Construction.

INDENTATION ROCK CUTTING

When elastic deformation leads to failure, the material loses cohesion by developing a fracture or fractures across which the continuity of the material is broken. This type of behaviour is called brittle behavior and governs the development of faults, joints and macro-fractures. Ductile behavior, in contrast, produces permanent strain that exhibits smooth variations

across the deformed rock without any marked discontinuities. Most rock materials are capable of exhibiting either brittle or ductile behavior depending on factors such as the size of differential stress, confining pressure, temperature, strain rate and pore-fluid pressure. Brittle failure is typical of rocks at low confining pressure and low temperature. Pore-fluid pressure has the effect of reducing the shear stress required for slip, for example, it reduces the shear strength of the rock since the direct pressure between adjoining grains caused by the confining pressure is countered by the effect of the pore-fluid pressure.Most mechanical tools break rock by indenting the surface. Rock crushing, macro-fracture propagation and chip formation occur under a loaded indentation tool, but the sequence, relationship and amount of each is largely unexplored. Thus the parameters controlling rock cuttability or rock resistance to tool indentation can not be readily related to any single mechanical rock property since the indentation process (**FIGURE** *2.3.-1*) is a combination of the following failure modes:

- *Initial tool indentation of rock surface with crushing and compacting of rock material under the tool tip*
- *Development of macro-fracture propagation patterns resulting in rock chip formation, chip loosening and stress release*

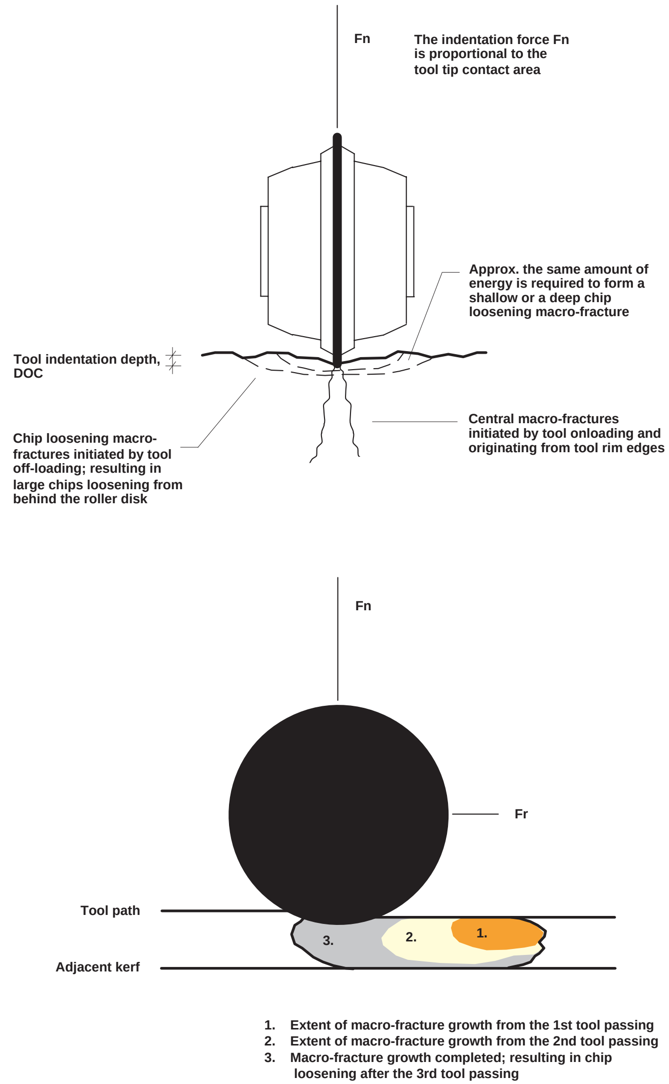


FIGURE *2.3.-1. Roller disk indentation on a rock surface with crushing under the tool tip produced macro-fracture growth patterns, and consequent stages of chip formation, chip loosening and stress release in multiple-tool pass cutting.*

- *Multiple pass cutting if chip loosening does not occur for every tool pass or load cycle*
- *Efficient chip and fines removal so as to avoid recutting and recompacting of broken material in the tool path.*

Rock cutting or drilling is therefore the art of maximizing chip formation and rock material removal as cuttings. It is not the development of extensive macro-fracture propagation patterns under a tool. The influence of rock mass discontinuities on rock mass cuttability is generally on a larger scale than one individual tool. It typically affects several tools simultaneously and the cutting performance of the cutterhead as a whole.

EVALUATION OF CLASSIFICATION SYSTEMS FOR ROCK MASS CUTTABILITY AND DRILLABILITY

The *Drilling Rate Index DRI*, developed by R. Lien in 1961, is a combination of the intact rock specimen brittleness value S_{20} and Sievers miniature drill-test value SJ. The SJ miniature drill test is an indirect measure of rock resistance to tool indentation (surface hardness); the brittleness value, S_{20}, is an indirect measure of rock resistance to crack growth and crushing. S_{20} is determined by the Swedish Stamp Test (**FIGURE** *2.3.-2*).

The rock aggregate is placed in a mortar and then struck 20 times with a 14-kg hammer. The mortar aggregate volume corresponds to that of a 0.5 kg aggregate with a density of 2.6 5 tons/m^3 in the fraction 11.2 - 16.0 mm.

S_{20} equals the percentage of undersized material that passes through a 11.2 mm mesh after the droptest. S_{20} is presented as a mean value of three or four parallel tests.
The second DRI parameter is the SJ value. The SJ value is obtained from a miniature drill test (**FIGURE** *2.3.-3*). The hole depth in the rock sample is measured after 200 revolutions in 1/10 mm. A mean value of four - eight test holes is used.

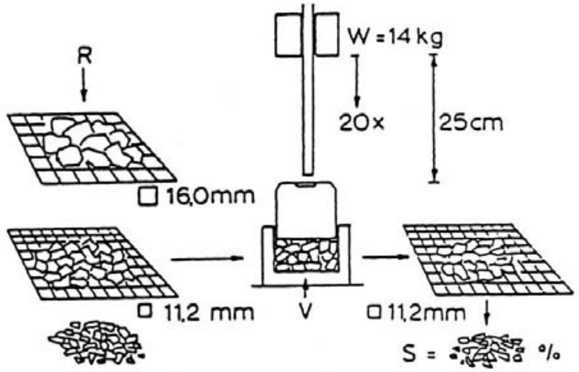
The orientation of the rock specimen can affect test results. Therefore, the SJ value is always measured for holes parallel to rock foliation. In coarse grained rocks, care must be taken to ensure that a representative number of holes is drilled in the different mineral grain types. The drilling rate index (DRI) is determined by the diagram shown in **FIGURE** *2.3.-4*. The DRI can also be seen as the brittleness value corrected for its SJ value.

A qualitative DRI drillability rating scale is shown in the following table.

Table 2.3.-1

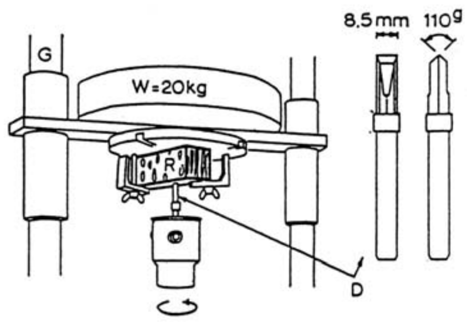
Rating	***DRI***
Extremely low	21
Very low	28
Low	37
Medium	49
High	65
Very high	86
Extremely high	114

Table 2.3.-1 presents typical DRI values for various rock types and is used for general DRI estimates. When increased accuracy is required to determine rock drillabilitiy, S_{20} and SJ testing is performed.



R = Rock sample aggregate
W = Weight (14 kg)
S_{20} = Brittleness value after 20 impacts

FIGURE *2.3.-2. Measuring rock brittleness by the stamp test.*



G = Guide
W = Weight (20 kg)
D = Miniature drill tipped with tungsten carbide
R = Rock sample, precut

FIGURE *2.3.-3. Sievers miniature drill test.*

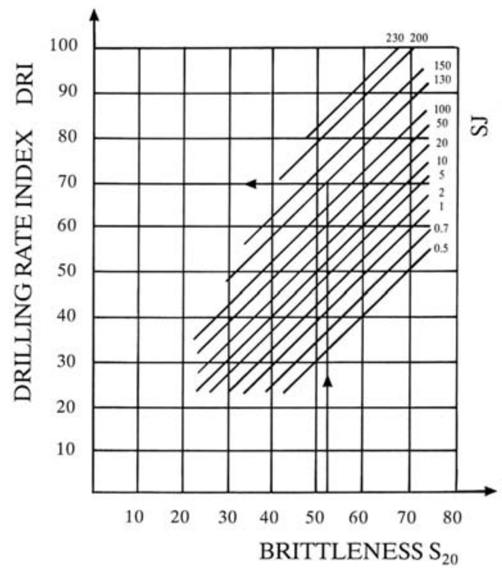


FIGURE *2.3.-4. Diagram used to determine the drilling rate index, DRI.*

Table 2.3.-1. Typical range of DRI values for some common rock types.

Rock type	DRI	Rocktype	DRI
Andesite	30...80	Graywacke	25...65
Anhydrite	85...115	Hematite ores	25...85
Anorthosite	30...50	Hornfels	30...50
Amphibolite	15...75	Limestone	30...100
Arkosite	29...75	Marble	40...110
Augen Gneiss	30...70	Magnetite ores	15...50
Basalt	20...75	Meta-Peridoties	40...105
Black Shale, Alum	40...70	Mica gneiss	25...75
Claystone, Slate	40...90	Mica schist	25...85
Coal *	110...120	Nickel ores	40...80
Concrete, C30	115	Norite	20...30
Conglomerate	25...75	Olivine basalt	20...60
Copper ores	30...90	Pegmatite	40...80
Chromite	70...125	Phyllite	35...75
Diabase, Dolerite	30...50	Porphyrite	30...80
Diorite	25...65	Quartzite	25...80
Dolomite	40...55	Rhyolite	30...65
Epidotite	25...40	Sandstone	15...90
Gabbro	30...65	Siltstone	30...145
Gneiss	25...75	Skarn	20...70
Granite	30...80	Sphalerite ores	90...105
Granite, Gneiss	25...80	Syenite	30...80
Granodiorite	30...55	Tonalite	30...70
Granulite, Leptite	20...45	Tuff	30...80
Green schist	40...70	Tuffites	35...145
Greenstone	20...75	TAMROCK**	43...49
		Granodiorite	

* Two rock samples only; coal is too brittle for the stamp test.
** Typical value for the TAMROCK test mine in Myllypuro.

A relationship between the unconfined, or uniaxial compressive strength (UCS) and the DRI has been established for 80 parallel tests (**FIGURE** *2.3.-5.*) by grouping scattered plotted values according to rock type. Envelope curves clearly illustrate that when the uniaxial compressive strength is used for rating rock cuttability / drillability - the following should be noted:

- *Cuttability of foliated and schistose (anisotropic) rock types* such as phyllite, micaschist, micagneiss and greenschist generally tend to be underestimated
- *Cuttability of hard, brittle rock types* such as quartzite generally tend to be somewhat over-estimated

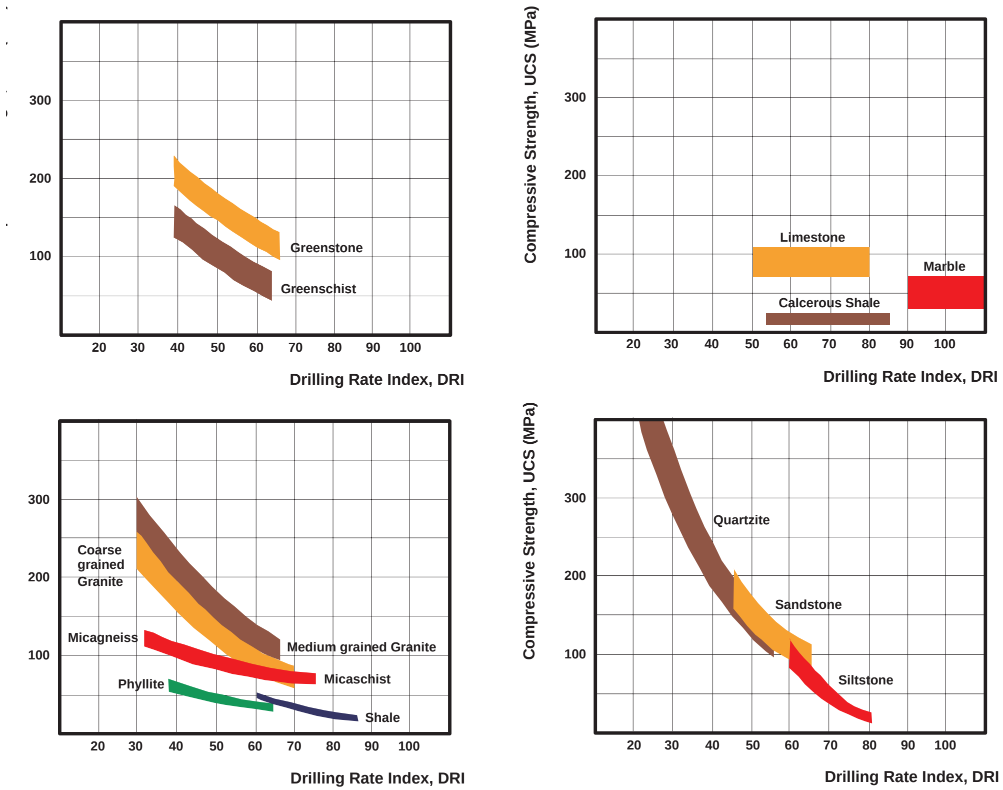


FIGURE *2.3.-5. Relationship between the DRI and UCS for some common rock types.*

In performance prediction models based on UCS-rated rock cuttability, correction factors or modifiers for rock type are commonly used to incorporate the effect of rock "toughness". The SJ value represents the aggregate rock surface hardness.
A useful correlation between SJ and the Vickers Hardness Number Rock (VHNR) for determining the degree of rock weathering is shown in **FIGURE** *2.3.-6.* (typical VHN values for minerals are shown in ***Table 2.4.-2.***). The S_{20} value represents rock brittleness, which comprises grain size and grain bonding strength. Unfortunately, rock porosity has very little effect on the brittleness value. Field performance follow-up in vesicular basalt indicates that 3 - 12% porosity has a considerable effect on both the critical normal force, Fn1, and net penetration rates for TBMs, and in addition the degree of rock fragmentation caused by blasting.

The brittleness value, S_{20}, when combined with the stamped rock specimen flakiness value f, is commonly used for assessing blast-rock suitability for road and highway construction purposes, and as crushed aggregates in asphalt and concrete.

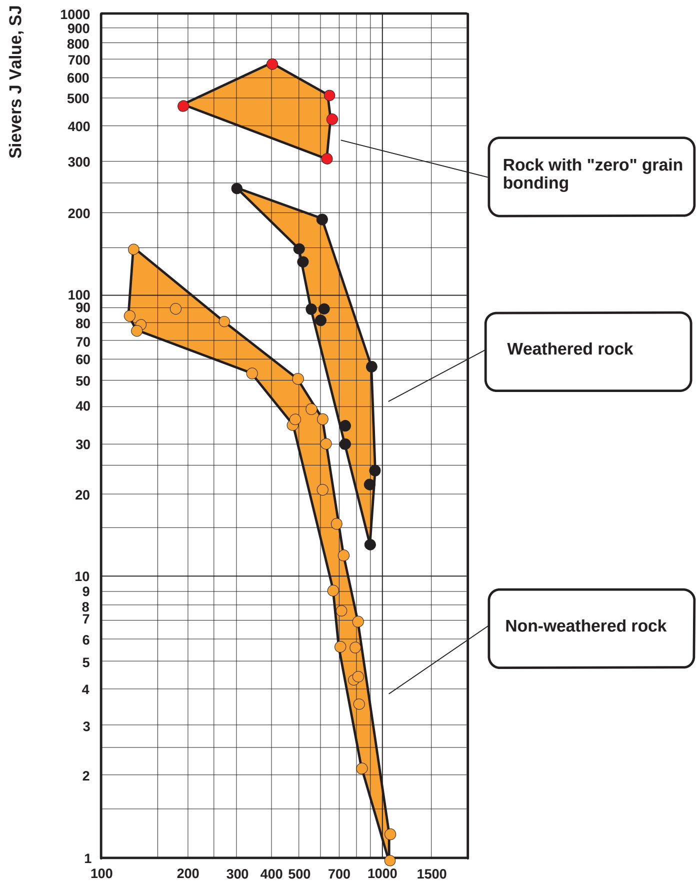


FIGURE 2.3.-6. *Relationship between Vickers Hardness Number Rock (VHNR) and Sievers J value for some common rock types.*

A comparative scale for rock resistance to breakage is the stamp test and rock hardness ratio f, which was developed by M.M. Protodyakonov Sr. in 1926. This scale is primarily used in Russia for assessing both rock drillability and blastability. Protodyakonov established the following relationship between the relative rock hardness scale and the uniaxial compressive strength:

$$f = 0.1 \times UCS$$

Unfortunately the Protodyakonov rock hardness scale, (***Table 2.3.-2***), does not differentiate between the hardness of rocks beyond 200 MPa.

Table 2.3.-2. *Protodyakonov classification of rock hardness.*

Category	*Hardness Level*	*Description of Rock*	*Rock Hardness f*
I	Highest	The hardest, toughest and most dense quartzites and basalts.	20
II	Very hard	Very hard granitic rocks, quartz porphyry, siliceous schist, weaker quartzites. Hardest sandstone and limestone.	15
III	Hard	Granite (dense) and granitic rocks. Very hard sandstones and limestones. Quartz veins. Hard conglomerate. Very hard iron ore.	10
IIIa	Hard	Limestones (hard). Weaker granites. Hard sandstones, marble, dolomites and pyrites.	8
IV	Rather hard	Ordinary sandstone. Iron ore.	6
IVa	Rather hard	Sandy schists. Schistose sandstones.	5
V	Moderate	Hard shale. Non-hard sandstones and limestones. Soft conglomerates.	4
Va	Moderate	Various schists (non-hard). Dense marl.	3
VI	Rather soft	Soft schists. Very soft limestones, chalk, rock-salt, gypsum. Frozen soil, anthracite. Ordinary marl. Weathered sandstones, cemented shingle and gravel, rocky soil.	2
VIa	Rather soft	Detritus soil. Weathered schists, compressed shingle and detritus, hard bituminous coal, hardened clay.	1.5
VII	Soft	Clay (dense). Soft bituminous coal, hard alluvium, clayey soil.	1.0
VIIa	Soft	Soft sandy clay, loess, gravel.	0.8
VIII	Earthy	Vegetable earth, peat, soft loam, damp sand.	0.6
IX	Dry Substances	Sand, talus, soft gravel, piled up earth, substances extracted coal.	0.5
X	Flowing	Shifting sands, swampy soil, rare-fractioned loess and other rare-fractioned soils.	0.3

2.4 ROCK ABRASIVITY - TOOL SERVICE LIFE

WEAR CLASSIFICATION

Tool wear can be defined as the microscopic or macroscopic removal or fracture of material from the working surface of a tool or wearflat by mechanical means. In other words, any degradation that reduces tool life. Wear classification is based on the relative movement between the materials on contact, including sliding, rolling, oscillation, impact and erosive wear. Generally, the kind of tool wear encountered in rock cutting is a combination of various types of wear. Some types of wear are more predominant than others. Wear types are influenced by several parameters, many of which are interdependent, such as hardness and fracture toughness of wear materials, contact mo-tion (for example, sliding, im-pact), wearflat temperature and contact stresses.
Tool wear is, therefore, a process in which the outcome is determined by the material properties of the tool tip, rock mass, and force-related interactions on the contact surfaces of

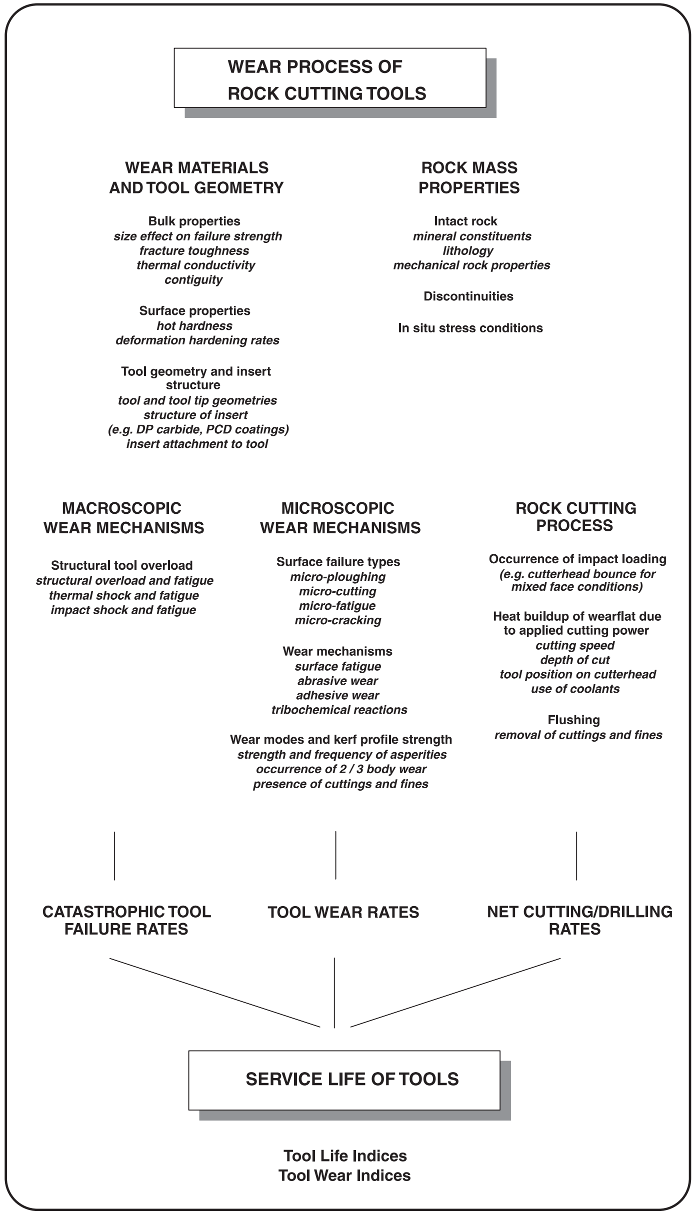


FIGURE *2.4.-1. Characterization of rock cutting tool degradation and tool service life.*

these materials. The wear capacity of a rock mass, (**FIGURE** *2.4.-1*) is a combination of:

- Mineral constituents, including size and hardness of mineral grains
- Strength and toughness of intact rock
- Tool depth of cut and cutting speed
- Occurrence of impact loading of tools (cutterhead bouncing, i.e. cutting in broken rock and mixed face conditions or through shears)
- Cutting type or contact motion in question (impacting, scraping, rolling, grinding, etc.)
- Presence of coolants at the tool tip/rock interface
- Efficient cuttings and fines removal
- Strength, wear resistance and quality of the cutting tool

Various indices for tool life and wear rates are typically used for measuring the rock's wear capacity. The established relationships are mainly based on the correlation of historic field performance data for predicting tool wear in the field. However, when new laboratory methods are developed, relevant field data is often not available. Consequently, relationships between new and old tool life and wear rate indices are often established so that previously reported field data can be used indirectly.

CLASSIFICATION OF WEAR MECHANISMS

The importance of wear mechanisms for cemented carbides may be classified according to scale of damage they cause, for example, macroscopic and microscopic failure.

MACROSCOPIC FRACTURE AND STRUCTURAL FAILURE

Cemented carbides comprise a range of composite materials with hard carbide particles bonded together by a metallic binder. The proportion of carbide phase is generally between 70 - 97% of the total composite weight. Its grain size averages between 0.4 - 14µm. Tungsten carbide (WC), the hard phase, together with cobalt (Co), the binder phase, forms the basic cemented carbide structure from which other types of cemented carbide are developed. In addition to straight tungsten carbide-cobalt compositions, cemented carbide may contain various proportions of titanium carbide (TiC), tantalum carbide (TaC) and niobium carbide (NbC). Cemented carbides, which have the cobalt binder alloyed with or completely replaced by other metals such as iron (Fe), chromium (Cr), nickel (Ni), molybdenum (Mo) or alloys of these elements, are also produced.

Structural overload and fatigue refer to the macroscopic failure or degradation of the tool tip material structure caused by stresses induced in the bulk of the wear material. Voids and flaws in materials serve as fracture-initiation sites due to stress concentrations at these sites. In cemented carbides, such voids or defects can result from inherent porosity caused by incomplete densification during the sintering process. They can also form during service as a result of the stress history of the tool. In the presence of shear stresses, such as those caused by friction at a wearflat, microscopic voids can nucleate at WC grain boundaries due to the separation of WC grains from the Co binder and other WC grains.

Toughness is defined and determined in many ways. Modern fracture mechanics provides a means of explaining toughness as it deals with the conditions of micro-crack initiation and growth in non-homogeneous materials under stress and where the material's fracture toughness is represented by the critical stress intensity factor K_{IC} . An indirect method commonly used for determining the toughness of cemented carbides is the Palmqvist method, in which the sum of corner crack lengths for a Vickers hardness indentation is used to derive the fracture toughness. The critical stress intensity factor for cemented carbides can be expressed as:

$$K_{IC} = 6.2 \times (HV_{50} / \sum L)^{1/2} \qquad [MN/m^{3/2}]$$

Toughness tests on cemented carbides show that the critical stress intensity factor increases with Co content and WC grain size. The range for critical stress intensity factors for the following materials is:

Cemented carbides $K_{IC} = 5 - 30\ MN/m^{3/2}$
Intact rock specimens $K_{IC} = 0.05 - 3\ MN/m^{3/2}$

Fracture toughness is substantially reduced at elevated temperatures. Due to the reduced fracture toughness with temperature, cemented carbides may exhibit a decrease in strength during cyclic loading at elevated temperatures. (**FIGURE** *2.4.-2.)*

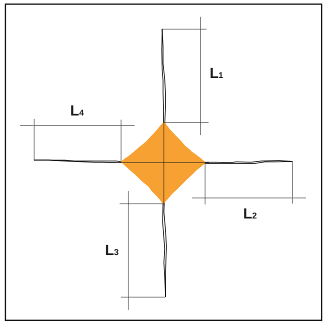


FIGURE ***2.4.-2.*** *Illustration of Vickers pyramidal indentation impression and resulting corner cracks used in the Palmqvist method for determining the critical stress intensity factor K_{IC} for cemented carbides.*

Cemented carbides are classified as brittle materials because practically no plastic deformation precedes fracture. However, cemented carbides show large variations in toughness behavior due to their microstructure. The types of fracture seen are cleavage fractures in carbide grains, grain boundary fractures between carbide grains and shear fractures in the binder. Generally, the amount of cleavage fractures increases with increased grain size and the amount shear fractures with increased binder content. Expressed as fracture energy, the major contribution is from the latter, for example, the crack propagation through the binder. Thermal fatigue of cemented carbides is most noticeable in non-abrasive rocks since the low abrasive wear preserves greater visual evidence of thermal cracks. These cracks penetrate deeply into the bulk of the material, run in an intergranular fashion, and branch readily. Fractures intersect, removing large flakes of material and forming relatively steep angular craters. Once this process has started, the tool rapidly becomes useless for rock cutting.

Wear resistance (a surface property) and toughness (a bulk property) are two complex properties, both of which provide a material the ability to withstand destruction. High wear resistance for cemented carbides can only be achieved if the demand for high toughness is reduced and vice versa. However, both high wear resistance and high toughness can be achieved simultaneously, provided these properties can be re-distributed. There are two ways of doing this: Dual Property (DP) cemented carbides, or coatings of highly wear resistant materials such as polycrystalline diamond (PCD) on a cemented carbide substrate.

In an ideal case, tool life and tool wear rates are inversely proportional. However, a tool's service life is also determined by structural overloading, and the interval and rate of catastrophic tool failures. The generalized distribution curve in **FIGURE** *2.4.-3*

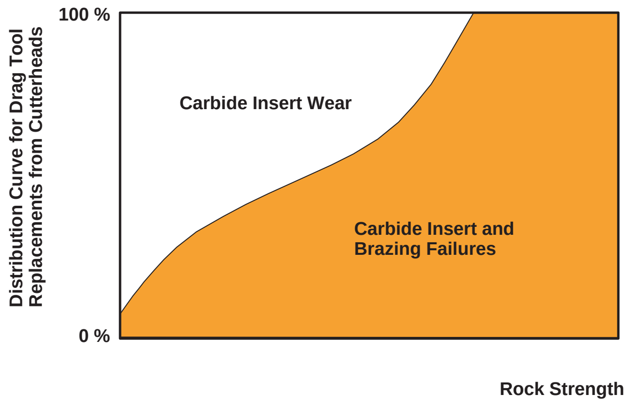


FIGURE ***2.4.-3.*** *Generalized distribution curve identifying the main reasons for drag tool replacements on cutterheads in service as a function of rock strength.*

for drag tool replacements on a cutterhead in service illustrates the increased sensitivity to tool impact failures in harder rock formations as well as the detrimental effect of increased tool loading required to cut harder rock. However, conical drag tools are not as sensitive to catastrophic failures as radial drag tools.

Catastrophic tool failure caused by impact loading is typically a result of both tool and cutterhead bouncing which occurs for an unfortunate combination of rock mass structure, cutterhead lacing design, and selected rotary speed. The impact force on the tool is caused by the striking action as it re-enters the kerf or harder portions of the rock structure. This leads to progressive tool tip chipping and ultimately catastrophic failure of carbide inserts and disk rims. For single-rowed carbide insert studded disks, a ripple breakage effect of the studs is often experienced. Some typical examples of rock structure leading to reduced tool life are:

- Fractured rock mass resulting in rock fallout and voids in the face
- Variable rock structure hardness or mixed face conditions

The severity of tool damage caused by impact loading is increased by the hardness ratio for mixed face conditions, for example $VHNR_{mineral-2} / VHNR_{mineral-1}$ (as illustrated in **FIGURE** *2.4.-4*).

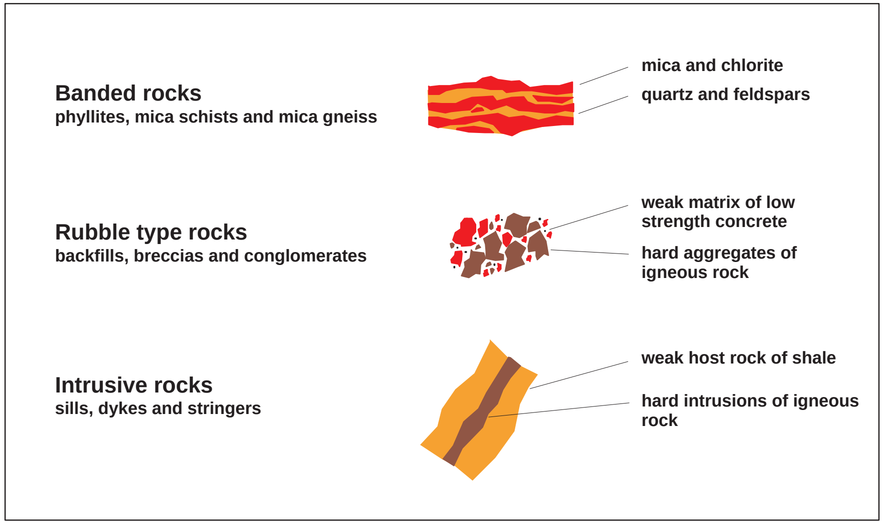


FIGURE *2.4.-4. Variable rock structure hardness or mixed face conditions which typically lead to impact loading and shattering of drag tools in particular.*

MICROSCOPIC FRACTURE AND WEAR MECHANISMS

Tool wear on the microscopic scale is the result of four basic wear mechanisms: Surface fatigue, tribochemical reaction, adhesive and abrasive wear. Plastic deformation as such is generally not regarded as a wear mechanism, but plays an important part in many wear processes. Abrasive and adhesive wear mechanisms dominate the tool wear process during of cutting rocks that contain minerals harder than the tool tip itself. Surface fatigue wear mechanisms only play a role if the wear rates are low, thereby giving necessary time for these processes to take place.

ABRASIVE AND ADHESIVE WEAR

Abrasive and adhesive wear mechanisms are the primary cause of the total wear encountered by tools sliding across abrasive rock surfaces. Wear caused by sliding abrasion is divided into four basic material failure types: Micro-ploughing, micro-cutting, micro-fatigue and micro-cracking (**FIGURE** *2.4.-5*).

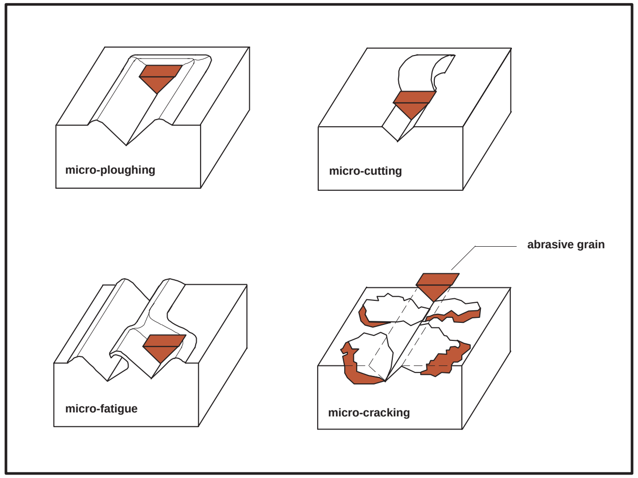


FIGURE *2.4.-5. The four basic types of material failure for abrasive wear.*

One of the main properties of metallic materials required to resist abrasive wear is surface hardness. Studies of tool wear rates show that abrasive wear mechanisms is a function of the relative hardness of materials with sliding contact. It has been established that one material will scratch another provided the difference between the respective surface hardnesses is greater than ~20%.

Abrasive wear can be divided into two categories: Soft and hard abrasive wear.

Soft abrasive wear $H_{rock} / H_{tool} < 1.2$

Wear rates are relatively low and do not greatly depend on the actual hardness ratio. Soft abrasive wear in cemented carbides occurs when the abrasive particles (e.g. quartz at room temperature) which are softer than WC grains yet harder than the Co binder, preferentially remove the Co binder, leaving the WC particles free to be dislodged from the structure. In the absence of thermal effect, soft abrasive wear rates are relatively low.

Hard abrasive wear $H_{rock} / H_{tool} > 1.2$

Wear rates increase significantly and become very sensitive to the hardness ratio. Hard abrasive wear in cemented carbides occurs when the abrasive particles harder than WC grains strike the composite and fracture WC grains on impact. This action causes a large degree of plastic deformation as the particles cut grooves or craters into the wearflat surface, resulting in voids and residual stresses that lead to additional fragmentation of WC grains. (**FIGURE** *2.4.-6.)*

Silicates typically cause most of the abrasive wear on rock cutting tools. A range of typical room-temperature Vickers hardness values for some selected materials are:

- Feldspars 730 ... 800 kgf/mm ^{2}
- Quartz 1,060 kgf/mm ^{2}
- Cast iron and steels 200 ... 750 kgf/mm ^{2}
- WC-Co mining grades 800 ... 1,700 kgf/mm ^{2}
- Polycrystalline diamond, PCD 4500 ... 7,000 kgf/mm ^{2}

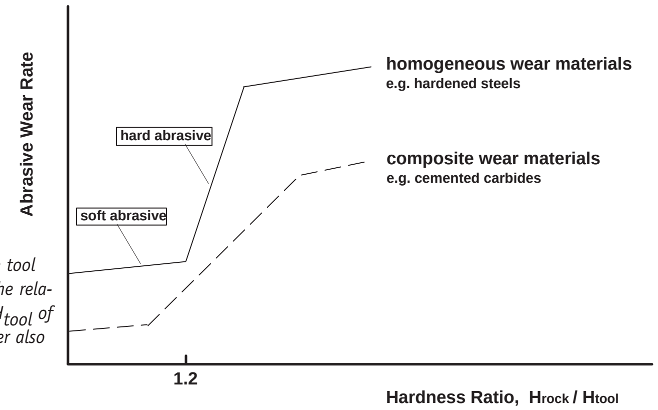


FIGURE *2.4.-6. Abrasive tool wear rates as a function of the relative hardness ratio,* H_{rock} / H_{tool} *of the materials in contact. Refer also to Figure 2.4.-9.*

Both rock and tool tip materials are often non-homogeneous in hardness testing and may consist of several components of varying hardness. The “aggregate surface hardness” of rock and wear materials is the mean value based on the hardness of their components. However, some components influence aggregate hardness more than others:

- Carbides in steel, for example, have a significant effect on the wear resistance of steel cutting tools, but do not influence the overall composite material hardness since they are too small to be significant for the Vickers microindentation hardness.

- Quartz would be considered a soft abrasive relative to WC-Co composites. Yet rock cutting tool wear in quartzitic rock occurs rapidly consistent with that produced by hard abrasives. This behavior suggests that thermal effects are important. With increasing temperature, the hardness of the wearflat drops more rapidly than that of quartz, therefore increasing the H_{rock} / H_{tool} ratio. Quartz particles may not attain the same temperature rise as the tool tip due to the limited period of time that individual quartz particles are subjected to frictional heating.

Thus, relative hardness between tool tip materials and rock mineral grains is insufficient to describe their behavior in a wear system. This is partly true due to the different nature of rock and tool materials, and the mechanical response in hardness testing and wear systems. Adhesive wear contributes to the total wear when the wearflat temperature and contact stress are high enough to weaken the tool tip material so that the cutting tool becomes worn by hard abrasives. The ability to retain hardness at high temperatures, or hot hardness, is one function of the WC-Co composite structure. WC grain hardness is not appreciably affected by temperatures reached during normal cutting operations. Critical hardness loss results when the Co binder absorbs sufficient heat to transform it into the plastic range where deformation and creep of WC-Co composites readily occurs. Sintered cobalt within cemented carbides melts at approx. 1,350°C. Bearing this in mind and due to the presence of asperities, localized peak contact temperatures may be as high as 2,000°C. (**FIGURE** *2.4.-7.)*

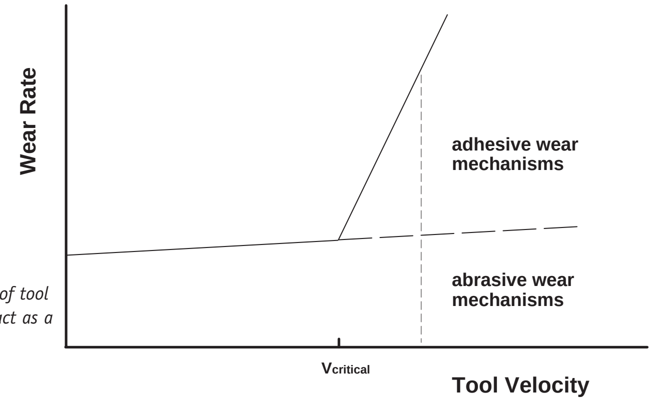


FIGURE *2.4.-7. Typical trend of tool wear rates for sliding motion contact as a function of tool cutting velocity.*

For wearflat temperatures below a threshold limit, WC-Co composites in rock cutting experience in wear produced by soft abrasives; while at higher temperatures wear is accelerated and occurs by mechanisms associated with hard abrasives and adhesion.

The temperature at which tool tip materials first start to weaken is called the critical temperature, Tcritical, and the corresponding tool cutting velocity, vcritical. Critical velocity is affected by several factors such as tool tip geometry, tool tip material properties (especially WC grain size since coarse WC grains improve thermal conductivity and thus enhance the transfer of heat away from the wearflat), use of waterjets for cooling and rock wear capacity (**FIGURE** *2.4.-8.*).

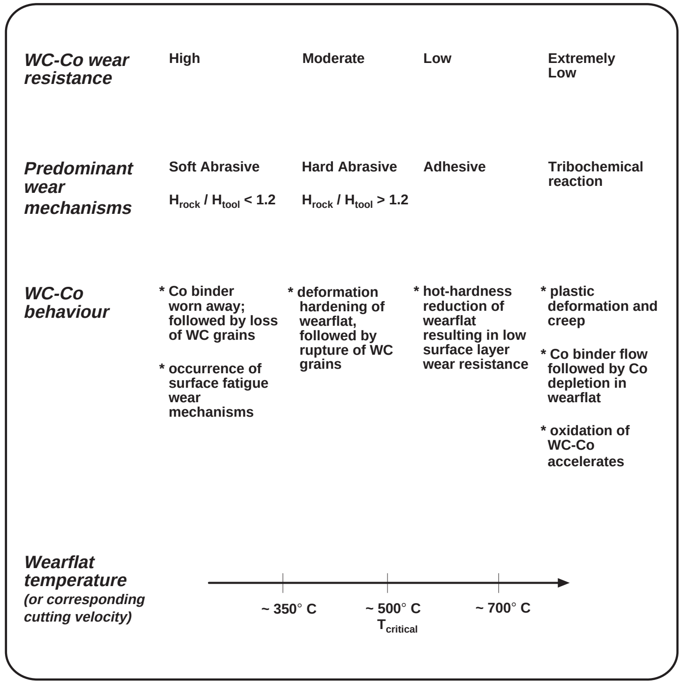


FIGURE *2.4.-8.*
Generalized summary of the behaviourand wear resistance of cemented carbides as a function of temperature.

METHODS FOR RATING WEAR CAPACITY OF A ROCK MASS

Parameters for characterizing and quantifying properties of intact rock specimens can be divided into two groups:

1. Physical rock properties such as grain size, density and porosity. These parameters describe insintric rock properties, which are inherent only to the rock itself.
2. Mechanical rock properties such as *strength, deformability, hardness, toughness, wear capacity* etc. These properties are influenced by the test method.

Tool consumption depends on the following wear process parameters:

Tool tip material
- Carbide grade wear resistance to thermal and surface fatigue
- Carbide grade resistance to catastrophic failure due to structural overload, thermal shock and shattering
- Carbide insert size, shape and arrangement of attachment to tool holder

Kerf profile
- Fragment size and strength of kerf rock powder (both dependent on mineral grain surface hardness)
- Tool indentation depth (defines both the tool/rock contact area, i.e. where wear takes place and which abrasive wear mode predominates)
- Effect of rock cutting mode (relieved/unrelieved cutting) on tool force levels

Tool service conditions
- Actual cutting velocity relative to the critical velocity $v_{critical}$ for the selected tool tip material
- Presence of tool tip cooling (waterjets etc.)
- Cut length per revolution for drag tools
- Occurrence of structural overloading of tools and cutterhead bouncing
- General handling of tools during transport, tool change, etc.

The most common laboratory methods used for determining the *wear capacity of rock specimens* are:
- (Rosiwal Mineral Abrasivity Rating)
- Wear Index F
- CERCHAR Abrasivity Index, CAI
- Vickers Hardness Number Rock, VHNR
- Cutter Life Index, CLI (a combination of the Abrasion Value, AV and Sievers miniature drill-test value, SJ)
- Hardgrove Grindability Index

ROSIWAL MINERAL ABRASIVITY RATING

A relative mineral abrasivity rating based on grinding tests was introduced in 1916, by A. Rosiwal in which the mineral specimen volume loss relative to corundum was used as an abrasivity rating:

Rosiwal = 1000 • volume loss corundum / volume loss mineral specimen

Typical Rosiwal abrasivity ratings for some common non-weathered minerals that do not contain impurities are listed in ***Table 2.4.-2***.

WEAR INDEX F FOR DRAG-TOOL CUTTING

The wear index F, proposed by J. Schimazek and H. Knatz in 1970, was the result of pin-on-disk wear tests on carboniferous rock from the coal mining districts in Germany. The Wear Index F is linearly related to pin wear rates; and increases with relative mineral abrasivity, mean quartz grain size and tensile strength of the rock specimen, i.e.

$$F = Q \cdot D \cdot Z \cdot 10^{-2}$$

Q	=	Equivalent quartz percentage	[%]
D	=	Mean quartz grain size	[mm]
Z	=	Brazilian tensile strength	[MPa]

The equivalent quartz percentage takes both the amount and relative mineral grain abrasivity to quartz into consideration. The Rosiwal mineral abrasivity rating used by Schimazek and Knatz for determining the equivalent quartz percentage is:

Carbonates	3%
Mica, chlorite, clay	4%
Feldspars	30 - 33%
Quartz	100%

Determining the equivalent quartz percentage for a typical sandstone is exemplified in the following table:

Mineral	*Mineral Content (%)*	*Equivalent Quartz Percentage (%)*
Quartz	63	$63 \cdot 1.0 = 63.0$
Feldspar	9	$9 \cdot 0.32 = 3.0$
Carbonate	3	$3 \cdot 0.03 = 0.1$
Mica, clay	25	$25 \cdot 0.04 = 1.0$ => 67.1

The relationship between the Wear Index F and the CERCHAR Abrasivity Index, CAI for the Saar Coal District in Germany, has been established as:

$$CAI = 0.6 + 3.32 \cdot F$$

The wear index F has been successfully used in very fine-grained and porous sedimentary rocks in Central Europe. Unfortunately, using the Wear Index F in coarse grained metamorphic and igneous rocks leads to highly misleading results. The Wear Index F was consequently modified by G. Ewendt in 1989.

CERCHAR* ABRASIVITY INDEX, CAI

* CERCHAR is an acronym for the Centre d´Etudes et Recherches des Charbonnages de France.

The CERCHAR scratch test for rating rock wear capacity was introduced in 1971. It is defined as follows: A pointed steel pin with a cone angle of 90° is applied to the surface of a rock specimen, for approx. one second under a static load of 7 kgf to scratch a 10mm long groove. This procedure is repeated several times in various directions always using a fresh steel pin. The abrasivity index is obtained by measuring the resulting steel pin wearflat diameter d in millimeters using an average value of 3 - 6 scratch tests depending on the variability of the individual scratch test results:

$$CAI = 10 \cdot \sum d_{wearflat} / n$$

Steel pin volume loss is proportional to the pin wearflat diameter as d^3, and therefore to the abrasivity index as CAI^3. The pin steel is specified by CERCHAR only as having a strength of 200 kgf/mm^2.

Typical CERCHAR abrasivity ratings for some common non-weathered minerals that do not contain impurities are listed in ***Table 2.4.-2.***

The abrasiveness of a rock specimen is not necessarily the same as the aggregate abrasiveness of its mineral constituents; factors such as grain size and angularity, grain cementation and degree of weathering all affect wear capacity of rock.

The CERCHAR Abrasivity Index scale ranges from 0 to 7. Typical ranges for some common rock types are given in ***Table 2.4.-1.***

Table 2.4.-1. *CERCHAR Abrasivity Index CAI for some common rock types.*

Rock Type	***CAI***
Igneous Rock	
Basalt	1.7 - 5.2
Diabase	3.8 - 5.4
Andesite	1.8 - 3.5
Diorite/Syenite	3.0 - 5.6
Granite	3.7 - 6.2
Sedimentary Rock	
Limestone	0.1 - 2.4
Sandstone 1)	0.1 - 2.6
Sandstone 2)	2.3 - 6.2
Metamorphic Rock	
Phyllite	1.3 - 4.3
Mica schist and mica gneiss	1.8 - 5.0
Felsic gneiss	3.7 - 6.3
Amphibolite	2.8 - 3.7
Quartzite	4.8 - 7.3

1) with carbonate and/or clayey cementation of mineral grains
2) with SiO_2 cementation of mineral grains

The following relationship between the CERCHAR Abrasivity Index, CAI and Vickers Hardness Number Rock VHNR for non-weathered rocks has been established for CAI > 0.7 as:

$$CAI = VHNR / 145$$

VICKERS HARDNESS NUMBER ROCK (VHNR)

A simplified approach to rating rock wear capacity is the use of rock surface hardness or mineral microindentation hardness. The most commonly used diamond-tipped microindenters are Vickers (a square based pyramid) and Knoop (an elongated based pyramid). Most systematic studies of ore minerals have employed Vickers microhardness determination and this technique has been widely adapted in ore microscopy.

The hardness number is defined as the ratio of the applied indenter load (kilogram force) to the total (inclined) area of the permanent impression. Microindenter hardness tests on minerals normally employ loads of 100 ... 200 gf; resulting in indentations with diagonal lengths of 5 ... 100 µm. For precise results, the load employed should be stated since VHN values obtained are not independent of load. For comparison, test loads and notation used for rating cemented carbides are:

Test	*Test Load*	*Notation for Metal Testing*
Hot hardness rating	500gf	$HV_{0.5}$
Hardness rating	30kgf	HV_{30}
K_{IC} determination	50kgf	HV_{50}

The rock matrix is typically non-homogeneous on the scale of testing and may consist of several minerals of widely varying individual grain hardnesses. The Vickers Hardness Number Rock, VHNR or the "surface hardness" of the rock is an aggregate value based on the weighted hardness values of its mineral constituents, i.e.

$$VHNR = \sum (VHN_j \, (\% \, mineral_j \, / \, 100) \qquad [\, kgf/mm^2 \,]$$

VHNR = Vickers Hardness Number Rock [kgf/mm^2]

% $mineral_j$ = percentage content of mineral j in rock specimen [%]

VHN_j = Vickers Hardness Number for mineral j [kgf/mm^2]

Typical mean values for the Vickers (VHN) and Knoop Hardness Numbers, Rosiwal and CERCHAR Abrasivity Indices for a selection of non-weathered rock-forming minerals without impurities are listed in ***Table 2.4.-2.***

Table 2.4.-2. *Typical mean values for Vickers (VHN) and Knoop Hardness Numbers, Rosiwal and CERCHAR Abrasivity Indices for a selection of non-weathered rock-forming minerals.*

Mineral	*Chemical Composition*	*Vickers*	*Knoop*	*Rosiwal*	*CERCHAR*
Corundum	Al_2O_3	2300	1700	1000	
Quartz	SiO_2	1060	790	141	5.7
Garnet	Fe-Mg-Al-Mn-Ca-Cr silicates	1060			
Olivine	$(Mg, Fe)_2SiO_4$	980			
Hematite	Fe_2O_3	925			
Pyrite	FeS2	800			4.7
Plagioclase	$(Na, Ca)(Al, Si)AlSi_2O_8$	800			4.7
Diopside	$CaMgSi_2O_6$	800			
Magnetite	Fe_3O_4	730			
Orthoclase	$KAlSi_3O_8$	730	560	52	4.4
Augite	$Ca(Mg, Fe, Al)(Al, Si)_2O_6$	640			
Ilmenite	$FeTiO_3$	625			
Hyperstene	$(Mg, FE)SiO_3$	600			
Hornblende	$NaCa_2(Mg,FE, Al)_5(Al,SI)_8O_{22}(OH)_2$	600			
Chromite	$(Mg, FE)Cr_2O_4$	600			
Apatite	$Ca_5(PO_4)_3(F, Cl, OH)$	550	395	7.3	3.1
Dolomite	$CaMg(CO3\ )_2$	365			3.3
Pyrrhotite	$Fe_{1-x}S$	310			
Fluorite	CaF_2	265	163	4.3	1.9
Pentlandite	$(Fe, Ni)_9S_8$	220			
Sphalerite	(Zn, Fe)S	200			
Chalcopyrite	$CuFeS_2$	195			
Serpentine	$Mg_6Si_4O_{10}(OH)_8$	175		0.8	
Anhydrite	$CaSO_4$	160			
Calcite	$CaCO_3$	125	85	4.08	0.8
Biotite	$K(Mg, Fe)_3(AlSi_3O_{10}\ )(OH)_2$	110			
Galena	PbS	85			
Chalcocite	Cu_2S	65			
Chlorite	$(Mg, Fe, Al)_6(Al, Si)_4O_{10}(OH)_8$	50			
Gypsum	$CaSO_4 \bullet 2H_2O$	50	32	0.85	0.3
Talc	$Mg_3Si_4O_{10}(OH)_2$	20	12	0.82	0
Halite	NaCl	17			
Sylvite	KCl	10			

Table 2.4.-3. *Vickers Hardness Number Rock (VHNR) for some common rock types.*

Rock type	*VHNR*	*Rock type*	*VHNR*
Amphibolite	500...750	Marble	125...250
Andesite	550...775	Metadiabase	500...750
Anortosite	600...800	Metagabbro	450...775
Basalt	450...750	Micagneiss	500...825
Black shale	300...525	Micaschist	375...750
Chromite	400...610	Nickel ores	300...550
Copper ores	350...775	Norite	575...725
Diabase/dolerite	525...825	Porphyrite	550...850
Diorite	525...775	Pyrite ores	500...1450
Epidotite	800...850	Phyllite	400...700
Gabbro	525...775	Quartzite	900...1060
Gneiss	650...925	Rhyolite	775...925
Granite/			
Granite gneiss	725...925	Sandstone	550...1060
Granodiorite	725...925	Serpentinite	100...300
Granulite/leptite	725...925	Shale and silstone	200...750
Green schist	625...750	Skarn	450...750
Greenstone	525...625	Sphalite ores	200...850
Hornfels	600...825	Tonalite	725...925
Limestone	125...350	Tuffite	150...850

ROCK CUTTABILITY "WINDOWS"

One of the main objectives for testing rock specimens and field follow-up for rating jobsite rock mass cuttability and machine performance is to visualize a geotechnical excavator work area or rock cuttability "window" for the evaluation of rock cutting productivity and economic excavation range of rock by tunneling machinery.

The Rock Cuttability Window for Intact Rock (**FIGURE** *2.4.-9*) is a scatter plot of rock wear capacity versus rock strength for rock specimens tested during a recent R&D program. In essence, **FIGURE** *2.4.-9* is a scatter plot of rock surface hardness versus rock specimen bulk strength.

There is an obvious trend illustrating that rock wear capacity increases with rock bulk strength and mineral surface hardness. However, there are some important exceptions as noted in **FIGURE** *2.4.-9* such as:

- *Ultramafic rocks* characterized by relatively high bulk strength and low rock wear capacity values. Ultramafic rocks have relatively high bulk strength values since fractures primarily propagate through mineral grains; and not along grain boundaries.

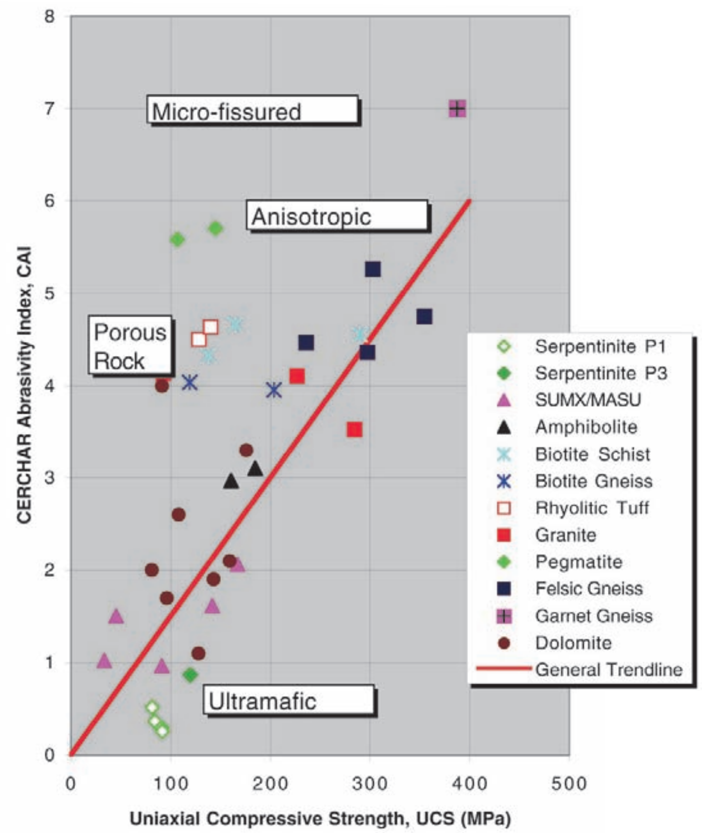


FIGURE *2.4.-9. The rock cuttability window for intact rock - a scatter plot of rock wear capacity versus the bulk strength of rock specimens tested during a recent R&D program.*

- *Anisotropic rocks* characterized by low bulk strength and high rock wear capacity values. Anisotropic rocks have relatively low bulk strength values due to fracture propagation primarily along schistosity planes. This effect is especially pronounced in uniaxial compression tests of rock specimens.

- *Porous rocks* characterized by low bulk strength and high rock wear capacity values. These rock types have relatively low strength values due to rapid fracture propagation originating at and radiating from voids in the rock matrix under stress, thus enhancing the cuttability or drillability of intact rock.

- *Micro-fractured igneous rocks* characterized by low bulk strength and very high rock wear capacity values. Observations show that this micro-fracturing seldom, if ever, enhances rock cuttability or drillability of intact rock. The phenomenon is typical for Pre-Cambrian granites, granodiorites and felsic gneisses in the Fenno-Scandian Shield.

- *Weathered and decomposed rocks* characterized by low bulk strength and low rock wear capacity values due to chemical alteration of the mineral grains.

2.5. ROCK MASS CHARACTERIZATION

2.5.1. Importance

Together with rock specimen properties, all rock mass irregularities are categorized systematically. The most important features are:

Structural make up of a rock mass includes:
- Rock types
- Frequency of variation
- Geometrical boundaries of structural members

Rock mass discontinuities includes:
- Bedding planes
- Schistocity planes and cleavage
- Fractures and fissures
- Faults

Rock pressure due to gravity and/or tectonic stress; including the excavation process.

2.5.2. Interbedding

Interbedding describes the alternation of different rock types in a rock mass.

- Regular interbedding means a sequence of different rock types with approximately parallel boundaries; and is typical for sedimentary rocks.
- Irregular interbedding stands for rock formations with non-parallel boundaries. It can also be the result of interruptions in the sedimentation process, including larger inclusions of other rock types, non-regular changes of rock type and local inclusions. (Typical in igneous rocks and metamorphic rocks.)

Any technical process influenced by different rock type behavior (excavation stability of structures) requires a thorough evaluation of the nature of interbedded rock structures regarding:

- Frequency of rock type occurrence and a particularly interesting behaviour
- Potential development of interbedding structures and their behavior throughout a project – also in areas where the rock mass is unavailable for investigation
- Forecast of rock types or conditions, which might become critical or even limiting for the applied equipment or cutting technologies.

Surface mapping and well-planned drilling programs as well as area mapping and geophysical preinvestigations are used to gain the necessary data and achieve a comprehensive picture complex geological structures.

2.5.3. Rock Mass Discontinuities

GENERAL

Another distinguishing feature in a rock mass are parting planes that intersect the massive rock. Depending on their frequency and relative spacing, they can either be of no importance or dominate the behavior of the rock during the excavation process.

Depending on their origin, the following parting planes can be found:

- Bedding planes
- Schistocity planes and cleavage
- Fractures and fissures

BEDDING PLANES

Bedding planes are the result of disturbances in the sedimentation process and are characteristic for all types of sedimentary rock, particularly mechanical sediments.
The following terms are widely used for characterizing the spacing between bedding planes:

Spacing between bedding planes (cm)	*Classification*
< 200	Massive
50 - 200	Thickly bedded
20 - 50	Bedded
10 - 20	Thinly bedded
2 - 10	Layered
< 2	Thinly layered

Bedding planes can be characterized as:

Open	-	with partial or no contact connection between strata
Closed	-	with no connection of strata, but full contact
Stained	-	plane surface coated by a gouge material -such as clay
Slickensided	-	visual evidence of polishing exists
Filled	-	partings filled with gouge material
Cemented	-	partings filled with adhesive materials (mineralogical bonds)

SCHISTOCITY PLANES

Schistocity planes are typical in most metamorphic rocks. During metamorphism, mineral grains are rearranged resulting in a predominant mineral grain orientation due to rock pressure. Thus “weakness” planes are generated especially if such planes are formed by mica or other lamellar minerals. Pressure release and/or tectonic activity can open such planes. As opposed to bedding planes, schistocity plane surfaces are mostly rough or undulating. If schistocity planes have opened throughout the metamorphic process, they are often filled with hydrothermal quartz and occasionally carbonates.

FRACTURES AND FISSURES

The following terms can be used for characterizing spacing between fractures and fissures:

Fracture/fissure planes per m	*Spacing between planes per m*	*Classification*
< 1	> 1	Massive
1 - 2	0.6 - 1	Slightly fissured or fractured
2 - 4	0.3 - 0.6	Moderately fissured
4 - 6	0.2 - 0.3	Fissured
6 - 10	0.1 - 0.2	Highly fissured
10 - 20	0.05 - 0.1	Extremely fissured
> 20	> 0.05	Mylonitic

The same terms used for describing bedding planes can also be used to describe the appearance of fracture/fissure plane surfaces.

JOINT SETS

Bedding, schistocity, fracture and fissure planes form the joint set of a rock mass. Joint sets are characterized by frequency of occurrence and orientation.
In practice, most rock masses have a minimum of 2 joint sets. In sedimentary rocks, the second joint set is typically perpendicular to the bedding planes.

2.5.4. Classification of Rock Mass Properties

To utilize the mapping of rock mass discountinuities, it is necessary to classify and quantify their effect on the rock excavation process.
The 3 most widely used geotechnical classification systems for ground support are:

- Rock Quality Designation (RQD-Index)
- Rock Mass Rating System (RMR-Value)
- Rock Mass Quality (Q System)

These systems have been developed primarily to assess the reduced stability of a rock mass intersected by parting planes. However, they can also provide assistance for estimating the influence of rock mass features during the excavation process.

ROCK QUALITY DESIGNATION (RQD)

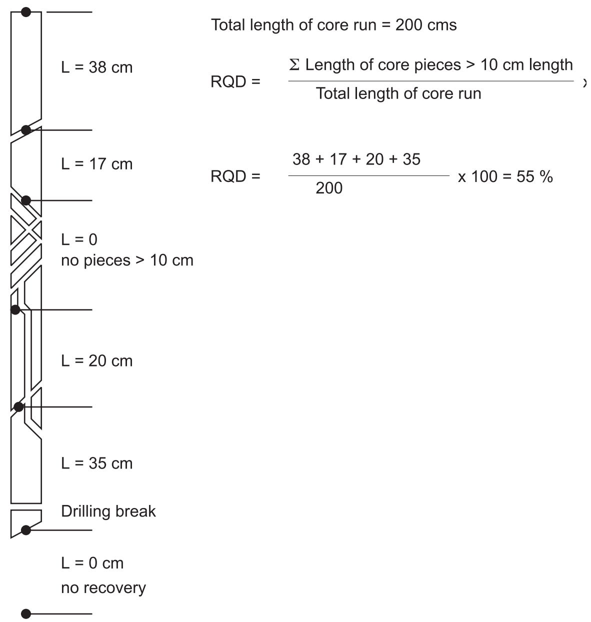


FIGURE ***2.5.-1.*** *Procedure of RQD measurements and calculation.*

Deere proposed in 1964 a quantative index based on a modified core recovery procedure only sound pieces of core that are 100 mm or greater in length. The principle is illustrated in **FIGURE** ***2.5.-1.***

This basic approach of calculating the effect of a joint set is also used in many rating systems, but can also be used alone to evaluate the influence of parting planes and frequency of occurance for roughly estimating the rock mass behaviour.

RQD (%)	*Rock mass classification*
90 - 100	Excellent
75 - 90	Good
50 - 75	Fair
25 - 50	Poor
> 25	Very poor

ROCK MASS RATING SYSTEM (RMR)

The rock mass rating (RMR) system was developed by Bieniawski in 1973. This engineering classification of rock masses, utilises the following six parameters, all of which are measurable in the field and can also be obtained from corchole data:

1. Uniaxial compressive strength of rock material.
2. Rock quality designation (RQD)
3. Spacing of discontinuities.
4. Condition of discontinuities.
5. Groundwater conditions.
6. Orientation of discontinuities.

The RMR value is calculated as follows:

RMR = [①+②+③+④+⑤] +⑥
❍ Rating of an individual parameter.

The following rock mass classes are defined by the RMR value:

RMR	*Class No.*	*Classification*
100 – 81	I	Very good rock
80 – 61	II	Good rock
60 – 41	III	Fair rock
40 – 21	IV	Poor rock
< 20	V	Very poor rock

ROCK MASS QUALITY (Q SYSTEM)

The Q - system of rock mass classification was developed in Norway in 1974 by Barton, Lien and Lunde, all of the Norwegian Geotechnical Institute.

The Q Method is a rating system based on the study of some 1000 tunnel case histories. It includes the following parameters for rating rock mass properties:

Table 2.5.-1. *Classification of jointed rock masses through the RMR system.*

Parameter		*Range of values*				
1 Strength of intact rock material	Pointed load strength index (Mpa)	>10	4-10	2-4	1-2	For this low range, uniaxial compressive test is preferred
	Uniaxial compressive strength (Mpa)	>250	100-250	50-100	25-50	5-25 1-5 <1
Rating		15	12	7	4	2 1 0
2 Drill core quality RQD (%)		90-100	75-90	50-75	25-50	<25
Rating		20	17	13	8	3
3 Spacing of discontinuities		>2m	0.6-2 m	200-600 mm	60-200 mm	<60 mm
Rating		20	15	10	8	5
4 Condition of discontinuities		Very rough surfaces Not continuos No separation Unweathered wall rock	Slightly rough surfaces Separation<1 mm Slightly weathered walls	Slightly rough surfaces Separation<1 mm Highly weathered Continuous	Slikensided surfaces or Gouge<5 mm thick Continuous	Soft gouge >5 mm thick or Separation>5 mm
Rating		30	25	20	10	0
5	Inflow per 10 ml tunnel length (L/min)	None	<10	10-25	25-125	>125
Ground water	Ratio Joint water pressure / Major principal stress	or 0	or <0,1	or 0,1-0,2	or 0,2-0,5	or >0,5
General conditions		or Completely dry	or Damp	or Wet	or Dripping	or Flowing
Rating		15	10	7	4	0
6 Strike and dip orientation of joints		Very favorable discontinuity	Favorable	Fair	Unfavorable	Very unfavorable
Ratings for	Tunnels and mines	0	-2	-5	-10	-12
	Foundations	0	-2	-7	-15	-25
	Slopes	0	-5	-25	-50	-60

- Rock quality designation (RQD)
- Number of joint sets (J_n) indicating the "freedom" of rock mass
- Roughness of the most unfavorable joint set (J_r)
- Degree of alteration or filling of the most unfavorable joint set (J_a)
- Degree of joint seepage, or joint water reduction factor (J_w)
- Stress reduction factor, SRF, which calculates load reduction due to excavation, apparent stress, squeeze and swelling.

The above six parameters are grouped into three quotients to give the overall rock mass quality Q as follows:

$$Q = (RQD/J_n)x(J_r/J_a)x(J_w/SRF)$$

Rock mass classification using the Q index:

Rock mass quality Q	***Behavior of rock mass in tunneling***
1000 ⟵ 400	Exceptionally good
400 ⟵ 100	Extremely good
100 ⟵ 40	Very good
40 ⟵ 10	Good
10 ⟵ 4	Fair
4 ⟵ 1	Poor
1 ⟵ 0,1	Very poor
0,1 ⟵ 0,01	Extremely poor
0,01 ⟵ 0,001	Exceptionally poor

2.5.5. Rock Pressure

One important factor primarily in underground construction is rock pressure – the in situ state of stress in a rock mass. In practice, the result of this stress is also called rock pressure. Primary rock pressure is the summary of stresses in a rock mass before influencing it, for example, by excavating underground openings. Primary rock pressure is the result of overburden, residual or tectonic stresses. Secondary rock pressure is when the primary stress field is altered by the excavation process. The secondary stress field can show considerable changes throughout the excavation process, thus indicating an unbalanced state of equilibrium.

The goal of the excavation process is to achieve a balanced state while avoiding any intermediate condition that may endanger the excavation itself, and the people and equipment working there. In practice, the stress itself does not form the critical factor, but the reactions of the rock mass caused by it.

The following characteristics of rock pressure take place in tunneling:

- Stress field is relocated resulting in the elastic deformation of the face and roof of the tunnel without fracture
- Stress release occurs by sudden rock failure ranging in intensity from spalling to rock burst
- Fracture and consecutive deformation of rock in the tunnel face and roof takes place in the rock mass with originally elastic or quasi-elastic behaviour
- Deformation and consecutive failure takes place in a rock masswith originally plastic-viscous behavior.

All the above-mentioned reactions are time dependent. The type of reaction that takes place also depends on the original state of stress and the rock mass behavior. It is also highly influenced by the mode and sequence of the excavation operations, and the size and shape of the openings.

2.6. BENCH BLASTING OPERATIONS

ASSESSMENT OF SHOTROCK FRAGMENTATION

At detonation, the bench is shattered or broken down; then thrown forward onto the quarry floor. Shotrock fragmentation is not uniform throughout the pile, but varies in accordance as to where the fragmented material originates within the bench itself.

The fragmentation of shotrock is not only due to stress-wave induced shattering at detonation. Rock fragments are also broken down by fragment collisions in the air and with the quarry floor. This is especially true for the coarser fractions from the uncharged portion of the bench.
Today, several commercial digital photo analysis systems are available for shotrock or muck pile fragmentation measurement and analysis. Shotrock fragmentation can be described by a Weibull 2 parameter (or Rosin Rammler) distribution such that:

$$P(k) = 100 \cdot \left\{ 1 - e^{-\ln 2 \cdot (k / k_{50})^n} \right\}$$

P(k)	=	cumulative passing for fragment size k	[%]
k	=	fragment size (dimension L)	[mm]
k_{50}	=	fragment size for 50% passing	[mm]
	=	volumetric mean fragment size (defined by dimension L)	
n	=	uniformity index	[-]

It is the combination of the mean fragment size k_{50} and the uniformity index n that describes the overall degree of shotrock fragmentation.

UNIFORMITY INDEX, n

The fact that *k_{50} and n are dependent parameters* leads to a major simplification of shotrock fragmentation data normalization work - since it is not necessary to find independent blast design guidelines for shotrock fragmentation based on both fragmentation parameters - only the mean fragment size k_{50}.

A simplified expression for the uniformity index is :

$$n = 1,60 \cdot (k_{50} / 270)^{0,61} \cdot f_{CL}$$

Blast parameters that show little influence (< 5 %) on the uniformity index n are :

- explosive energy and velocity of detonation, VOD
- sequential row firing
- mechanical properties of intact rock
- rock mass jointing

since these parameters influence the degree of shotrock fragmentation in bench blasting operations primarily through their influence of the mean fragment size k_{50}.

BENCH BLAST GUIDELINES FOR SHOTROCK FRAGMENTATION

Observed ranges for mean shotrock fragment sizes k_{50} from extensive field studies of various bench blasting operations in Norway are shown in the following table:

Bench Blasting Operation	*Shotrock Designation*	*Mean Fragment Size, k_{50} [mm]*	*Loading Equipment*
Aggregate Quarries	Crushing & Screenin	125 - 290 [1]	Wheel Loaders or Front Shovels
Rockfill Dam Quarries		Supporting Fill:	
	Fine Zone	160 - [2]	Wheel Loaders
	Fine Zone	200 - 250	Wheel Loaders
	Coarse Zone	250 - 320	Wheel Loaders
	Coarse Zone	- 440 [3]	Wheel Loaders + Hyd. Excavators
Open Pit Mining	Crushing & Milling	160 - 250 [4]	Rope-Shovels + Wheel Loaders
Road Construction	Sub Base	200 - 310	Hyd. Excavators

1) Targeted mean fragment sizes depend on primary crusher openings, primary crusher capacities and marketability of fines
2) Blasts with a high portion of rock for transition zones (k_{max} = 200 mm)
3) Blasts with a high portion of rock for dam slope rip-rap and crown cap. Fragment size criteria for supporting fill rock is generally $k_{max} \approx 2/3$ of placement lyer thickness
4) Blasts with the largest mean fragment sizes were observed for or ebodies with low mechanical prop erties for intact rock.

BENCH CHARGING AND BLASTING

For a given rock mass, the degree of shotrock fragmentation k_{50}, depends on the type and quantity of explosives used for blasting a cubic metre of solid rock. This is termed the specif- ic charge (or powder factor) *q* .

The specific charge, in turn, affects the amount of drilling required to achieve this degree of shotrock fragmentation since the drill pattern itself (burden x spacing) affects the mean shotrock fragment size.

Studies linking the mean shotrock fragment size k_{50} to the specific charge or powder factor q by various authors is shown in the following table:

Olsson (1952) [1]	$k_{50} = k_1 \cdot q^{-0,56}$
Lundborg (1971) [2]	$k_{50} = k_2 \cdot [S / B]^{-0,145} \cdot [Q^{0,20} / q]^{1,47}$
Kuznetsov (1973) [3]	$k_{50} = k_3 \cdot [Q^{0,21} / q]^{0,80}$
Rustan (1983) [4]	$k_{50} = k_4 \cdot [Q^{0,28} / q]^{1,64}$
Brinkmann (1985) [5]	$k_{50} = k_5 \cdot q^{-1,37}$
Lislerud (1990) [6]	$k_{50} = k_6 \cdot [Q^{1/5} \cdot \rho^{4/5} / q]^{\delta}$; $\delta = f\{Q^{-0,05}\}$

1) ditch blasting
2) bench blasting - small blastholes
3) bench blasting - large blastholes
4) laboratory model - scale blasting in m agnetite - concrete
5) underground blasting - hand held equipment
6) bench blasting - small and large blasthole operations

k_{1-6} basically rock mass and/or explosive strength constantsResults of bench blasting simil- itude modelling for a given rock mass show the following important relationships for bench blasting:

- the reduced drill pattern, $(BxS)^{1/2}$ or the square drill pattern burden, B increases in prin ciple with drill-hole diameter as $d^{4/5}$ for a constant mean fragment size, k_{50}
- the specific charge, q increases in principle with drill-hole diameter as $d^{2/5}$ for a constant mean fragment size, k_{50}

Results of bench blasting similitude modelling for a given powder factor and blasthole charge including parameters such as:
- mechanical properties of intact rock (anisotropy, strength or modulus of elasticity, density and porosity)
- rock mass jointing (joint type, frequency of occurrence and orientation)
- explosive strength (charge density, explosive energy and VOD) illustrates the following important relationships for bench blasting regarding rock mass blastability and explosive strength:

Rock Mass Blastability $k_{50} = f\ \{ I_a^{3/5} \cdot 0^{1/2} \cdot \sigma^{3/10} / (\rho^{4/5} \cdot n^{2/5})\}$

Explosive Strength $k_{50} = f\ \{CD^{1/5} / (EE^{2/5} \cdot VOD^{2/5})\}$

In other words, *rock mass blastability* decreases systematically with rock specimen an isotropy, joint spacing, rock specimen strength or modulus of elasticity but increases with rock specimen porosity.

Several geomechanical classification methods are available today for rating rock mass blastability for bench blast design - such as:

Kuznetsov (1976) *Blastability Index A*
(used in the Kuz-Ram Model; A is partly based on the Protodyakonov rock hardness value f)

Lilly (1986) *Blastability Index BI*
(used in the Kuz-Ram Model; A ~ 0.12 • BI)

Lislerud (1990) *Prediction model used* in Tamrock´s Surface Study Programme

Achieving the optimum bench blast design for a particular rock mass type, be it in quarrying or mining, can be an expensive and time - consum ing procedure. In particular obtaining a good first-estimate or starting point for the quantity of explosives required (powder factor) to achieve satisfactory shotrock fragmentation can be problematical. Estimates are usually based on the operator´s experience and/or the experience of a hired consultant.

LOADING OF SHOTROCK

The most commonly used equipment for loading shotrock in quarries are wheel loaders, front shovels and hydraulic excavators. The choice between these loaders depends on production requirements, shotrock fragmentation, loading conditions and selected haulage method.

Loaders	*Muck Pile*	*Disadvantages*	*Advantages*
Wheel loaders	low and flat	toe problems tire wear in wet conditions	high capacity flat quarry floor versatile equipment
Front shovels	topped	not so mobile coarse	loading very shotrock
Hydraulic Excavators		low capacity	clearing toe problems clearing high-walls

Accurate drilling decreases the amount of oversize and reduces the occurrence of loading problems. The drill pattern can also be increased; which in turn affects the explosives consumption. However, the max fragment size k_{max} increases disproportionately with blasthole diameter.

2.7 CRUSHABILITY

GENERAL

Rock crushability is defined as the capacity of a crusher to produce a certain product fraction. It is also the capability of a crusher to produce good product gradation and particle shape. Rock crushability is a combination of many elements. Rock tests generally measure a single mechanical rock property that gives a rough estimation of its crushability. In addition to mechanical rock properties, feed gradation also has an impact on the quantity and quality of the crushed product.

SPECIFIC GRAVITY

Crushing capacity is generally measured in tons per hour. Crusher throughput in compressive crushing depends on the volume of the material, which means that a crusher running on a constant setting provides approximately the same capacity in cubic meters per hour for all rock materials. The heavier the rock is, the higher the capacity in tons per hour. Crusher manufacturers report crusher capacities usually in tons per hour for rock material, which specific gravity is 2.65-2.7 t/m^3 or bulk density of zero based fraction is 1.6 t/m^3. If the specific gravity or bulk density differs from these values, crusher capacity in tons per hour is correted in proportion to actual and reference values.

WORK INDEX

In the 1950´s, F. C. Bond developed the Impact Work Index (W_i) that defines the theoretical power consumption of comminution in relation to the crusher product gradation.
The higher the W_i is, the coarser the product gradation is. For example, if granite and limestone have a W_i of 16 kWh/t and 10 kWh/t, then the setting of the crusher for granite must be smaller than for limestone in order to produce the same gradation. Because a smaller setting means a lower capacity, the crusher gives a lower capacity for granite than for limestone.

10-15 specimens were selected for the tests. The ideal sample is approximately 50 x 75 mm, with two natural parallel or near parallel sides of 50 to 75 mm in width. These samples can not be prepared by cutting or any other method not normally used in the crushing process. This is very important because a sample must have the exact same characteristics as in the actual crushing process. (FIGURE *2.7.-1.*)

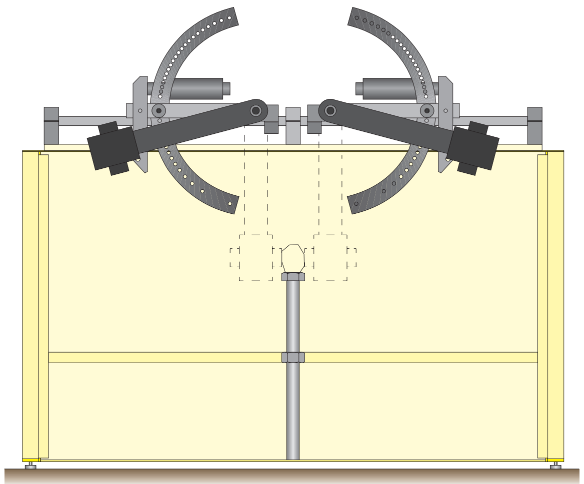


FIGURE *2.7.-1. Test device.*

The test device (FIGURE *2.7.-1.*) consists of two 17 kg hammers which strike the sample simultaneously and perpendicular to the parallel surfaces. This allows the impact force to transverse the sample between the two parallel surfaces. The dropping height of the hammers is increased until the sample breaks.Rock impact strength is calculated by the following formula:

$$A = 2 \cdot M \cdot H / C$$

where A is impact strength [kpm/cm], M is the mass of one hammer [kp], H is the drop height [m] of hammer when the sample breaks, and C is thickness of the sample [cm].
Work Index is calculated accordingly:

where A is impact strength [kpm/cm], M is the mass of one hammer [kp], H is the drop height [m] of hammer when the sample breaks, and C is thickness of the sample [cm].
Work Index is calculated accordingly:

$$W_i = 47.6 \cdot A / p$$

where W_i is work index [kWh/t], A is impact strength [kpm/cm] and *p* is specific rock gravity [t/m³].The final result is the average W_i of all the samples. The maximum value of W_is is also recorded. Average Work Index for granite is approximately 16 kWh/t.

Table 2.7.-1. shows the crushability for different W_is.

Table 2.7.-1

Crushability	***W_i [kWh/t]***
Very easy	$W_i < 10$
Easy	$10 < W_i < 14$
Average	$14 < W_i < 18$
Difficult	$18 < W_i < 22$
Very difficult	$22 < W_i$

LOS ANGELES VALUE

The Los Angeles value is correlated to the work index so it can be used to estimate crushability if the work index value is not available. ***Table 2.7.-2.*** shows the crushability estimation for different Los Angeles values.

Table 2.7.-2.

Crushability	***LosA***
Very difficult	LosA < 7
Difficult	7 < LosA < 14
Average	14 < LosA < 25
Easy	25 < LosA < 40
Very easy	40 < LosA

2.8 CONCRETE

Concrete essentially is a rock-type material best compared with conglomerate rock. It is relatively brittle and abrasive, and its mechanical properties vary significantly from case to case. Even though various concrete mixes greately differ from each other, the properties within the batch are uniform if compared to a similar amount of rock.

Concrete consists of sand and gravel, rebar (= reinforcement iron), pores and cement which bonds all the constituents together. Concrete's porosity is approximately 3% and its specific gravity is typically 2.2 - 2.5 tons/m^3, which means that it is a relatively light material.

MECHANICAL PROPERTIES

Concrete is usually classified according to its compressive strength, along with a letter, such as K, Z or B, and number, such as 30 or 300, which corresponds to its compressive strength.

The compressive strength of a typical concrete is between 20 - 40 MPa however seldom higher than 50 MPa. This makes concrete a relatively weak material. Concrete's hardness and abrasivity depends on the sand and gravel used.

The strength of concrete depends mainly on raw materials. However manufacturing methods also have a strong influence on its strength:

- The correct amount of water ensures that all cement is used to bond the ingredients and has low porosity.
- Suitable compaction (vibrations) of wet concrete reduces pore formation and segregation of large and heavy parts.
- Impurities in the raw material such as mud reduce the bonding ability of the cement.

Concrete's mechanical properties usually change in time. Under favorable conditions, concrete hardens slowly for a long time after it has been manufactured. In normal conditions, however, it usually degrades due to weathering, acid conditions etc.

REBAR

The tensile strength of concrete without reinforcement is approximately 5 - 10% of its compressive strength. In a finished product, rebar is designed to carry the tensile load. Using rebar is thus used in places where tensile stress bearing capacity is required.

The appearance of rebar depends on the intended use, dimensions and form of the construction. Reinforcement evaluation requires knowledge of local construction habits and original drawings (which usually aren´t available). Reinforcement is extensive, for example, in bridge pillars, nuclear power plants etc. This should be taken into consideration when selecting the proper destruction methods and calculating production estimates.

3.1 MECHANICS OF ROCK BREAKING

When a tool is loaded onto a rock surface, stress is built up under the contact area. The way the rock responds to this stress depends on the rock type and the type of loading, for example, the drilling method.

Rock breakage by percussive drilling can be divided into four phases:

CRUSHED ZONE

As the tool tip begins to dent the rock surface, stress grows with the increasing load and the material is elastically deformed, zone III in **FIGURE** *3.1.-1.* At the contact surface, irregularities are immediately formed and a zone of crushed rock develops beneath the indenter (the button or insert of a drill bit) **FIGURE** *3.1.-1.* The crushed zone comprises numerous microcracks that pulverize the rock into powder or extremely small particles. 70-85% of the indenter's work is consumed by the formation of the crushed zone. The crushed zone transmits the main force component into the rock.

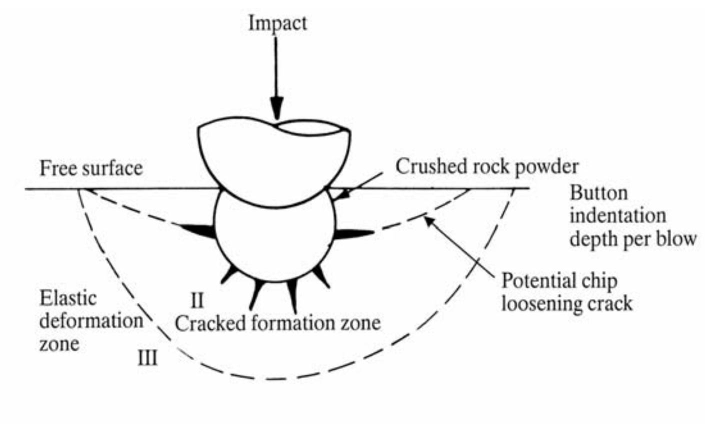


FIGURE *3.1.-1. Rock breakage in percussive drilling*

CRACK FORMATION

As the process continues, dominant cracks begin to form in the rock. This initial stage of restricted growth is described as an energy barrier to full propagation. The placement of major cracks depends on the indenter shape. Generally, the dominant placement of major cracks with blunt indenters, such as a sphere, is located just outside the contact area, pointing down and away from the surface.

CRACK PROPAGATION

After the energy barrier has been overcome, spontaneous and rapid propagation follows, zone II in **FIGURE** *3.1.-1.* At a lower depth than the contact dimension, the tensile driving force falls below that necessary to maintain growth, thus the crack again becomes stable. The crack is then said to be "well developed".

CHIPPING

When the load reaches a sufficient level, the rock breaks and one or more large chips is formed by lateral cracks propagating from beneath the tip of the indenter to the surface. This process is called surface chipping. Each time a chip is formed, the force temporarily drops and must be built up to a new, higher level to achieve chipping. Crushing and chipping creates a crater.

3.2 TOP-HAMMER DRILLING

There are four main components in a drilling system, **FIGURE** *3.2.-1.* These components are related to the utilization of energy by the system for attacking rock in the following way:
1. The piston inside the rock drill is the prime mover, converting energy from its original form (fluid, electrical, pneumatic or combustion engine drive) into mechanical energy to actuate the system. 2. The shank adapter transmits impact energy from the piston to the rod(s) or tube(s). Additionally, rotation torque is delivered via the shank adapter.
3. The rod or tube transmits impact energy and rotation torque. 4. The bit applies the energy in the system, mechanically attacking rock to achieve penetration.

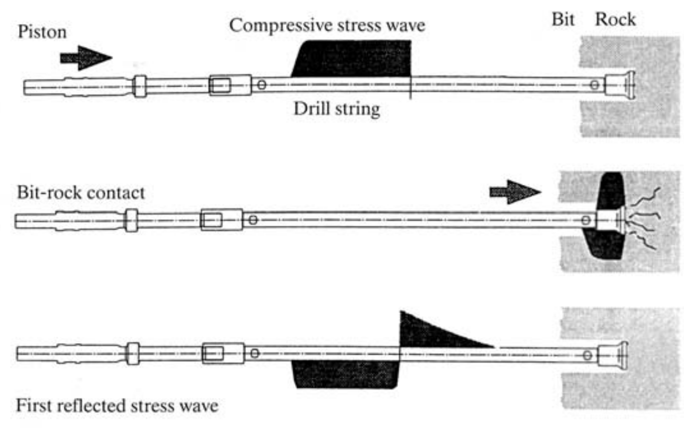


FIGURE *3.2.-1. Percussion dynamics*

In tophammer drilling, the piston inside the rock drill accelerates to the desired velocity before striking a shank adapter or drill rod. Upon impact, the drill rod particles achieve velocity, and therefore, a displacement in a direction away from the piston.
The particles transmit this motion to adjacent particles which then repeat the process, creating a stress wave pulse that travels down the rod. The shape of the stress wave is determined by piston and drill string geometry. A rectangular stress wave is formed when a piston with a uniform cross-section area hits a rod made of identical material and the same cross-section area. Hydraulic rock drills produce rectangular stress wave, **FIGURE** *3.2.-1.*

PERCUSSIVE DRILLING PARAMETERS

Percussive drilling consists of four drilling parameters which affect performance: percussion power (percussion energy and frequency), feed force, bit rotation speed and flushing. **FIGURE** *3.2.-2.*

Percussion power

Percussion output power in percussive drilling is produced by the rock drill's impact energy and frequency. Pneumatic drilling has a typical impact frequency of between 1,600 - 3,400 hits per minute; hydraulic drilling, 2,000 - 4,500 hits per minute.

Percussion output power is a function of hydraulic or pneumatic pressure and flow rates. Compared to pneumatic drills, hydraulic drills are capable of higher percussion power and faster penetration rates. The net penetration rate achieved with TAMROCK hydraulic rock drills as a function of drill hole diameter and rock drillability shown in **FIGURE** *3.2.-3.*

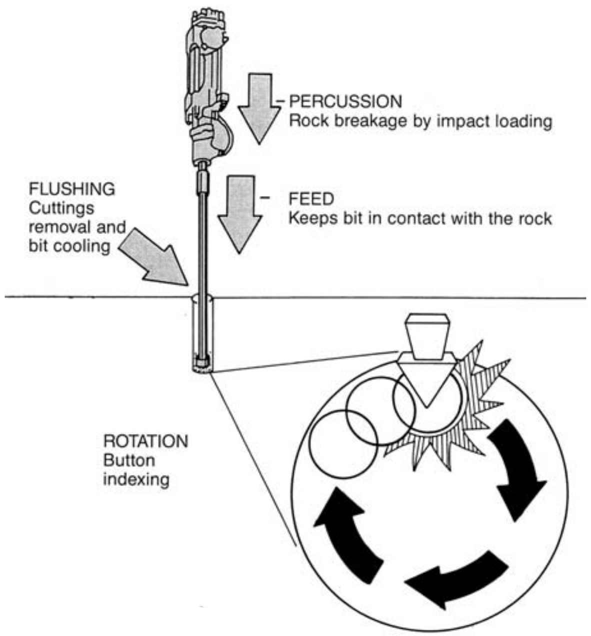


FIGURE *3.2.-2. Top-hammer drilling.*

One limitation in percussion drilling is the capacity of the drill steel to transmit energy. Only maximum kinetic energy is transmitted through a particular steel before excessive drill string deterioration occurs.

For field drilling, the optimum percussion pressure setting depends on financial aspects. Higher penetration rates are achieved through increased percussion power, however, the drill

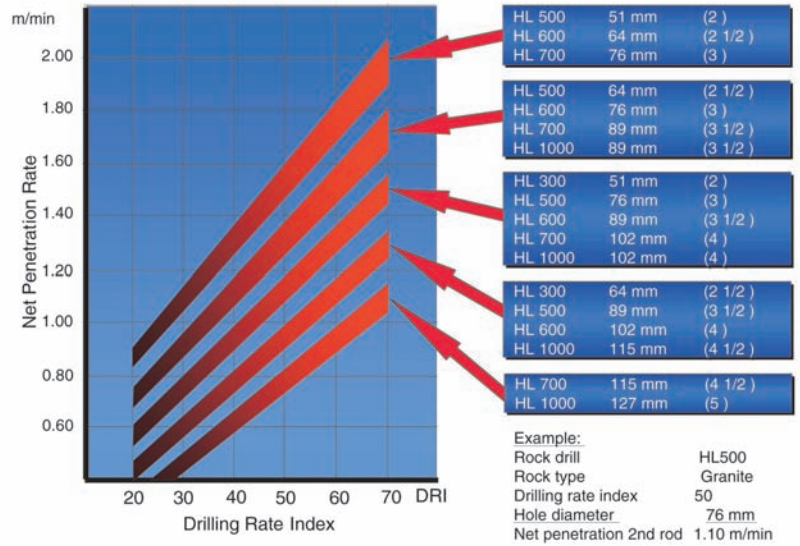


FIGURE *3.2.-3. Net penetration rates achieved with TAMROCK rock drills.*

steel's life time simultaneously decreases. Possible increased hole deviation and its impact on burden and spacing must also be taken into account.

Feed

Feed force is required to keep the shank in contact with the drill and the drill bit in contact with the rock. This ensures maximum impact energy transfer from the piston to the rock. When percussion pressure is increased, feed pressure must also be increased. Optimum feed force depends on the percussion pressure levels, rock condition, hole depth, drilling angle, and the size and type of drill steels. Broken rock should be drilled at low percussion and feed pressures.

In top-hammer drilling systems, the drill normally travels on a feed. The required feed force is transferred to the rock drill cradle by chains or cylinders. Optimal feed pressure can easily be observed by monitoring penetration, bit wear and steel thread wear. Quite often, visual monitoring of feed and rotational "smoothness" during drilling is sufficient to determine the optimum feed pressure.

Low feed force results in:

- Poor transmission of percussion energy, shank damage and increased thread wear since couplings tend to loosen
- Reduced penetration rates due to poor percussion energy transfer through the drill string
- Almost no resistance to rotation and low torque

- Increased inner bit button wear
- Overheated and rattling coupling

Very high feed force leads to:

- Unnecessary bending and drill steel and shank wear
- Flushing problems
- Rapid button wear on the bits due to increased drag against the hole bottom and because bit is forced to work in inclined position when drill string bends
- Increased hole deviation
- Uncoupling becomes difficult due to excessively tight threads
- Lower penetration rates

Bit Rotation

The main purpose of bit rotation is to index the drill bit between consecutive blows. After each blow, the drill bit must be turned to ensure there is always fresh rock under the inserts or buttons. Bit rotation speed is adjusted to the point where the penetration rate is at its maximum. The following factors affect optimum bit rotation speed:

- Rock type
- Rock drill frequency
- Drill bit diameter
- Gauge button diameter (in case of button bit) At optimum rotation speed, the size of the disintegrated chips is greatest and thus the penetration rate is maximum. **FIGURE** *3.2.-4.*

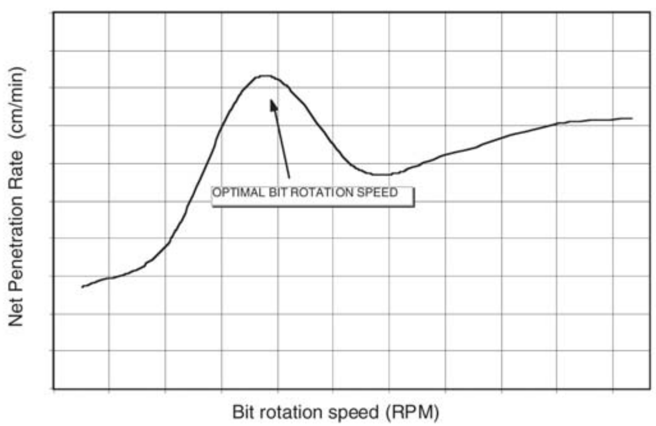


FIGURE *3.2.-4. Penetration rate in granodiorite through bit rotation speed.*

According to a rock's drilling properties, the optimum travel distance of gauge buttons between consecutive blows is the diameter of gauge button times 0,5 - 1,5. The smaller optimum travel distances applies to conditions in which the formation of chips during bit denting is poor, resulting in slower net penetration rates. Bit rotation speed can be calculated through the following equation:

$$n = \frac{S \cdot f \cdot 60}{\pi d}$$

Where S = Gauge button travel distance between consecutive blows [mm]
n = Bit rotation speed [RPM]
f = Impact frequency of the hammer [1/s]
d = Bit diameter [mm]

However, for smaller rod diameters (1 1/2" and less), the optimum penetration rate disappears after the second rod due to poor indexing, which is caused by reduced drilling string stiffness. Therefore it is necessary to:

- Decrease RPM values by 20–30% for a combination of small rod diameters and long drill holes to extend the bit regrind limit

For cross bits, the following RPM values relative to button bits are suggested:

- Diameters 35 - 51 mm cross bits, RPM 5 - 10% lower values
- Diameters 76 mm or larger cross bits, RPM 5 - 10% higher

Insufficient bit rotation speeds result in energy loss due to recutting and result in low penetration rates. However, sometimes bit rotation speed is intentionally set under its optimum value since lower RPMs

- Reduce in-hole bit deviation
- May be required in very abrasive rocks to maintain gauge button velocity under critical wear speed

Excess rotational speeds result in excessive bit wear as rock is forced to break by rotation instead of percussion. High rotational speeds also lead to excessively tight couplings, which result in uncoupling problems.

Flushing

Flushing is used to remove rock cuttings from the drillhole and to cool the drill bit. The flushing medium - air, water, mist or foam - is forced to the bottom of the drillhole through the steel's flushing hole and the holes in the drill bit. Loose rock cuttings mix with the flushing agent and are forced out of the drillhole through a space between the drill steel and the hole wall.

Insufficient flushing leads to low penetration (increased recutting), decreased drill steel life (bit wear and jamming of the steels) and high bit wear. The amount of flushing required to remove cuttings from the drillhole depends on the hole size, cuttings produced and hole length. Air flushing is typically used in surface drilling; water in underground drilling operations. If water is not allowed underground, air, airwater mist or a variety of foam flushing agents can be used.

Air flushing in a closed space requires excellent dust collection systems.Experience shows that the minimum required flushing velocity for successful cuttings removal is 15 m/s for air and 1.0 m/s for water flushing. Over flushing is also a risk. With air flushing, excess flushing pressures are used. The bit-body is eroded by the "sand-blasting" effect through a mixture of cuttings and air. High bit body wear is a problem in especially abrasive rock types. In water flushing, overly high flushing pressures decrease the penetration rate because water cushions the bit against the rock.

Required air flow for sufficient flushing can be estimated from the minimum velocity stated in ***Table 3.2.-1.***

The amount of water used in tunneling is approx. 50 liters per minute for each rock drill. In longhole drilling, drilling direction has a large impact on the required water consumption. Downholes need more flushing water compared to upholes.

Sometimes foams are used together with air flushing to bind the dust and to stabilize drill hole walls. Airwater mists are also used for these purposes. Normally the amount of water added to the flushing air varies between 2-5 l/min. Air and air-water mist flushing increases penetration rates up to 10–20% compared to water flushing.

Table 3.2.-1. *Required air volume for sufficient hole flushing.*

Steel diameter, mm	25	32	38	45	51	64	76	87
Hole size, mm	Required air volume m^3/min (air velocity = 15 m/s)							
32	0.28	*Drifting*						
38	0.58	0.30						
45		0.71	0.41					
51		1.11	0.82	*Long hole drilling*				
64		2.17	1.87	1.46				
76			3.06	2.65	2.24	1.19	*Tubes*	
89				4.17	3.76	2.70	1.52	
102					5.52	4.46	3.27	2.00
115					7.51		5.27	4.00
127								6.05

HYDRAULIC DRILLING

Hydraulic percussive drilling rigs were introduced to the market in the early 1970s. These new, high-power rock drills not only doubled drilling capacities but also improved the drilling environment. The introduction of hydraulics to rock drilling also led to improvements in drilling accuracy, mechanization and automation.

Working principle of the hydraulic rock drill

The general working principle of a hydraulic percussion rock drill is presented in **FIGURE** ***3.2.-5.***

a. Piston at front end
b. Piston moves backwards
c. Piston in rear position
d. Piston moves forwards

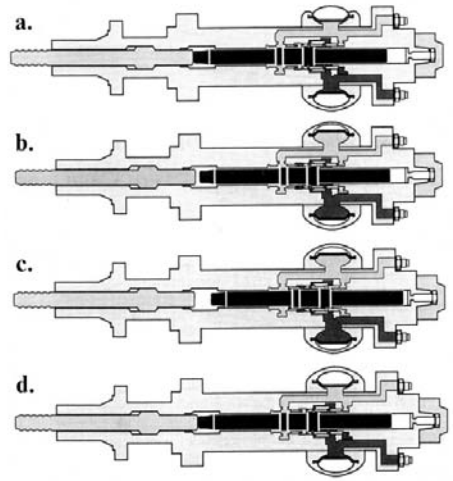


FIGURE ***3.2.-5*** *a, b, c and d. The working principle of the hydraulic rock drill.*

TAMROCK rock drills are designed with a minimum of modules and parts. The hydraulic rock drill has a reliable percussion cartridge structure, comprising a piston and a distributor. This design allows large and short flow channels, which ensures maximum flow efficiency. The compact modules for percussion, rotation and flushing minimize the number of joints (**FIGURE** *3.2.-6*).

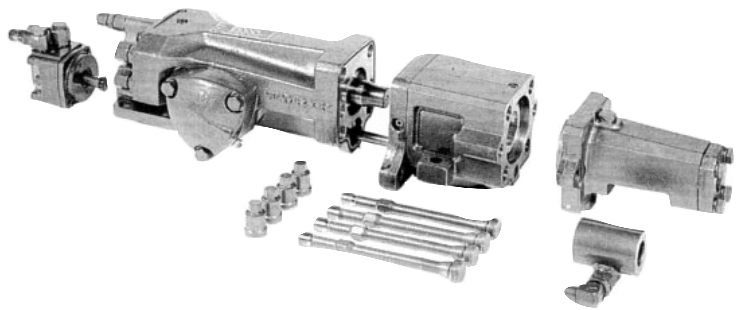


FIGURE ***3.2.-6.*** *Modular design of modern rock drill.*

PNEUMATIC DRILLING

Pneumatic rock drills and mechanized pneumatic jumbos were most popular during the 1960s and early 1970s. The power source for the pneumatic rock drill is existing compressed air lines or a portable compressor.

COMPARISON BETWEEN HYDRAULIC AND PNEUMATIC DRILLING

Table 3.2.-2. *Comparison between hydraulic and pneumatic drilling.*

Hydraulic drilling	*Pneumatic drilling*
Efficient	Fairly inefficient
50% higher drilling capacity	Low capacity
Reliable, constantly high efficiency level	Efficiency level depends on outside supply of air and compressed air pipeline configuration
Easily adjustable to changing rock and drilling conditions, smoother drilling	Fairly inflexible
Ergonomic	Non-ergonomic
Less noise, moisture, mist, no surrounding temperature fluctuation	Noisy, air-water mist, cold air flow, uncomfortable working environment
Economical	Low-economy rock drill
High capacity, independent, minimal labor versatile and user friendly	Low capacity, non-independent (air lines and compressors), more labor-intensive

3.3 PRINCIPLE OF DTH DRILLING

DTH (Down-The-Hole) drilling, also known as ITH (In-The-Hole) drilling, is a method in which the percussive hammer works in the hole during drilling, as opposed to above the hole in top-hammer drilling. DTH hammers are used in underground benching operations.

In DTH hammers, the rock drilling bit is a continuation of the shank, which the rock drill piston strikes directly. DTH machines are driven by compressed air and require a fairly large compressor to operate effectively. Since the piston is in almost direct contact with the drill bit, little energy is lost. This gives a nearly constant penetration rate regardless of hole length. Hole accuracy is also good. DTH machines are limited by their relatively low penetration rates and poor mobility, because they require a large separate compressor. Rotation is usually hydraulic. Energy consumption is also large compared to top-hammer drills.The hole sizes most commonly used for underground DTH drilling are 89 - 165 mm in diameter, and can extend up to 1,100 mm. Hole lengths in underground benching operations vary up to 60 meters. (**FIGURE** *3.3.-1.*)

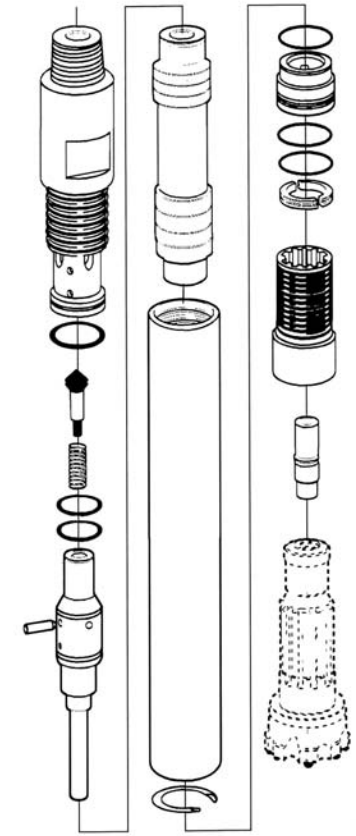


FIGURE ***3.3.-1.*** *DTH rock drill.*

3.4 ROTARY PERCUSSIVE DRILLING

Rotary percussive drilling is based on the same rock breaking principle as top-hammer drilling except that feed force, rather than percussion force, is used to dent the rock. When the bit is pressed against the rock and rotated, the cutting force promotes chip formation and rock cutting. Cuttings are removed via air or air-mist-flushing. Underground rotary percussive drilling techniques are most applicable in soft or semi-soft formations. Rotation and thrust set special requirements not only on feed and rotation systems but also on drill steels and tool design.

3.5 HYDRAULIC HAMMERS

Hydraulic hammers are attachments used on standard hydraulic excavators. The weight of hammers ranges from 80 kg - 7000 kg. The weight range for suitable carriers is from 0.8 - 100 tons.

The hammer works in two different ways: penetrative breaking and impact breaking. Penetrative breaking applications include softer materials that are demolished by a combination of stress waves (= a high stress level applied for a very short time) and a wedge effect. Used together they split the material. For typical penetrative work, a chisel or a moil-point tool is used to break concrete, asphalt or other soft materials. In penetrative work, the tool's sharpness is often essential for good productivity.

Impact breaking is accomplished by having only the stress waves break the material. Wedge penetration is undesired for this method. The stress wave is best transmitted to the material by a blunt tool. Typical applications include over-sized boulders, heavily reinforced concrete structures or steel slag.

The hammer's productivity depends on numerous factors such as impact energy and frequency, material strength and toughness, operator skill and technique, carrier condition and job-site logistics. Operating related factors include impact frequency, impact energy, breaking and demolition techniques and the amount of down pressure applied to the rock.
Proper down pressure is essential for efficient energy transfer from hammer to rock. Down pressure is applied by pressing the hammer onto the rock with the excavator. If contact is insufficient, the tool moves up and down on the rock. The consequence is poor breaking efficiency, rapid tool wear and strong vibrations on both the hammer and carrier. Too much force, however, will overload the suspension system, cause premature wear to hammer components and cause immense vibrations on excavator when the rock collapses.

Insufficient tool-to-rock contact results in dust between the tool and rock. Dust cushions occur also when the hammer is not powerful enough to cause large fractures in the rock. A dust cushion under the tool tip cushions the energy from the tool, and therefore, reduces breaking productivity. Early tool repositioning is essential for good hammer performance.

Optimum impact frequency depends on the material to be broken. If the material is soft and penetration is high, a high impact frequency is advantageous. If the material is very hard, a slightly lower impact frequency reduces tool heat build up and softening of the tool tip. This results in less tip wear.

The material is usually broken with several impacts (2-200 blows) when working with a hammer. The material behaves differently in the beginning and in the end of such a series of impacts. In penetrative breaking, the penetration rate varies considerably (4.2.2. Primary breaking with hammers). In secondary hard-rock breaking, there is barely any noticeable damage at first but eventually the rock completely fractures.

Optimum impact energy depends on the dimensions of the material to be broken. If concrete is penetrated or boulders are broken upon 2 - 5 impacts, the impact energy is too high for the application.
In this case a relatively large percentage of the hammer energy will go back into the hammer with blank firing. The impact energy should be reduced if possible. This should not significantly affect overall productivity. Otherwise, maximum impact energy can and should be used.

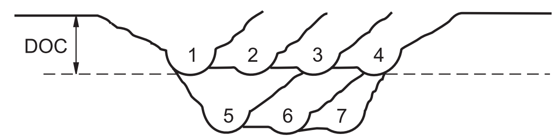
The goal of using the hammer is to make existing cracks larger or wider. Concrete and soft rock are the only other exceptions. For example, in secondary breaking of blasted rock, small cracks are always present, even though the rock seems to be compact. The best method is to locate the cracks through frequent repositioning of the tool. The operator should also try to locate weak points in the rock (=experience). This may be a challenge because in primary breaking the surface is usually covered by a layer of dirt or mud.
Breaking hard and compact rock that contains few or no cracks only results in very small chips being removed (as in percussive drilling, only somewhat larger), and is not an economical way to use a hammer.

3.6. CUTTER-CRUSHERS AND PULVERIZERS

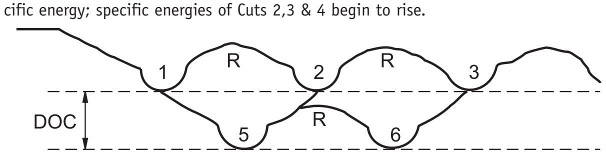
Cutter-crushers and pulverizers are hydraulic attachments used to break, cut and pulverize in demolition and recycling applications. Cutter-crushers and pulverizers are two jaws operated by a hydraulic cylinder (sometimes two cylinders). Jaw design can vary with application.

Compared to hydraulic hammers, cutter-crushers and pulverizers are non-impact or static breaking devices (i.e. no stress wave involved). The main advantage is in its simple design and extremely reliable construction. Cutter-crushers are relatively quiet and used for selective demolition in which part of the structure should remain intact.

Close kerf spacings, S/DOC<2
Results of Cut 6 and Cuts 2, 3 &4 are identical because the rock breaks straight across between preceeding cuts - leaving a flat surface. Cuts 2, 3 & 4 are relieved cuts in a flat surface. Cut 6 is a relieved cut in a precut surface.



Intermediate kerf spacings, 2<S/DOC<5
Cuts 2, 3 & 4 have relatively smaller yields because of volume of ridges R left between them. Cut 6 yield keeps increasing because ridge above it compensates for ridge left to side. Tool forces are virtually identical because the ridge above the tool does not require much force to remove. Therefore Cut 6 has lower specific energy; specific energies of Cuts 2,3 & 4 begin to rise.



Wide kerf spacings, S/DOC<6
Cuts 2& 3 are unrelieved; tool forces and yield become constant. Cut 6: tool forces and yield keep rising with increasing area of ridge removed.

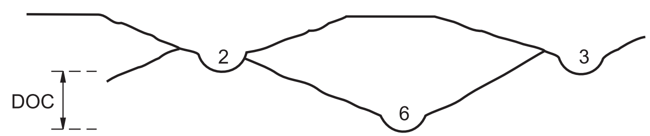


FIGURE 3.7.-1. *The effect of spacing to depth-of-cut ratio (S/DOC) for cutting with drag tools on flat and precut surfaces in sandstone.*

3.7. CUTTING

CONCENTRIC AND SEQUENTIAL CUTTING

The most effective cutting sequence made by cutting tools is a series of relieved cuts, in which each cut is made adjacent to the preceding cut at a predetermined spacing small enough to substantially reduce the tool focus compared with an isolated (i.e. unrelieved) cut made to the same depth. Although relieved cutting is the most common and desired cut type, many machines have tools arranged in such a way that more complex cut forms are made. (**FIGURE** *3.7.-1.*)

Using scrolled tool vanes or lines in concentric and sequential rock cutting / (**FIGURE** *3.7.-2.*) is an attempt to ensure relieved cutting for individual tools while maintaining a well-balanced cutterhead.

For varying tool density across the cutterhead:

TPL_{mean} = ∑ disk or carbide insert rows/∑ kerfs

SPR = $(360/\Delta AP_{\text{scrolls per line}})$

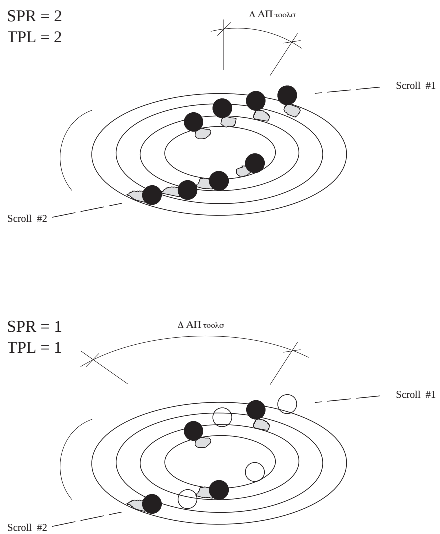


FIGURE *3.7.-2. Concentric and sequential in-line kerf cutting with two scrolled tool vanes.*

INDIVIDUAL TOOL AND CUTTERHEAD BOUNCING FOR SEQUENTIAL CUTTING

Individual tool bouncing is caused by the following mechanisms (**FIGURE** *3.7.-3*):

1. Inability of tool to cut variable rock hardness formations to a constant depth of cut, resulting in transient tool peak loading.
2. Tool hammering when re-entering the tool path after passing through a face void. Voids are created by fallouts along intersecting joints and fissures in the tunnel face (heading).
3. The recutting of chips and fallouts on the tunnel invert (especially in fractured rock) also initiates tool bouncing.

Cutterhead bounce frequency and cutterhead bounce amplitude for sequential cutting is a combination of:

- Individual tool bouncing
- Bounce amplitude for individual cutters decreases as roller cutter diameter increas es; also called the “buggy wheel” effect
- Differential angular posi tion (AP of the tracking tool in the adjacent kerf
- Cutterhead rotary speed RPM
- Adverse affect of peak or transient tool loading due to tool bouncing is enhanced in cutterheads with low hydraulic stifness

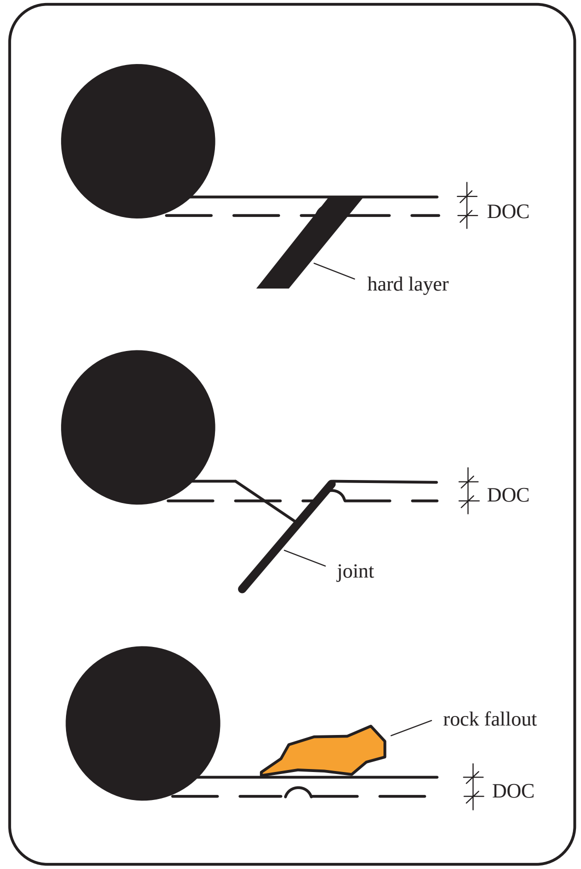


FIGURE *3.7.-3. Individual tool bounce mechanisms.*

Cutterhead bounce frequencies originate as:

- Individual tool bouncing occurs for tools a,b in line i
- Tracking tools a,b in line i + 1 bounce individually after a given time $\Delta t = (60 / RPM) \cdot (\Delta AP / 360)$ resulting in cutterhead excitation frequency $f = 1 / \Delta t$

The cutterhead bounce frequency f functions as an excitation frequency for boom or machine body vibrations. It can not be eliminated and must be designed away from the natural boom or machine frequency. There are two controllable design variables that affect cutterhead excitation bounce frequencies:

- Cutterhead RPM (advantage with variable speed-drive electrical motors).
- Differential angular position (ΔAP) of the tracking tool in the adjacent kerf.

3.8 LOADING AND HAULING

Any chosen excavation method presents challenges to the loading and hauling system. Machine size can differ greatly in tunnel sites compared to surface sites. LHD machines are mainly used underground, and although seldom used on construction sites, they can also be used on surface sites. The most common loading machines for underground work are wheel loaders. In quarries, hydraulic excavators (including front shovels) and wheel loaders are most commonly used. Selecting the right machine for the job depends on feed material, production requirements, operating conditions and following operations.

When selecting a loader, it is important to consider the following:

- Amount rock to be loaded -at quarries, the production requirement
- Loading cycle time according to the whole excavation cycle
- Size limit due to tunnel
- Turning and loading niche availability
- Bucket size (Loader bucket size varies according to operating weight. Those which have small bucket /weight ratio may have higher fill factors than normal.)

LHD MACHINES

In conventional drilling and blasting underground excavation projects, fast and effective work phases are essential. Drilling, charging, face cleaning and other related functions usually take place at different times. Tunnel advancing depends on the time spent on the critical path of each operation. This is the reason why contractors try to minimize it. Fast face cleaning increases the demand for more effective loader equipment and face cleaning methods.

LHD offers fast and effective face cleaning because it is flexible and versatile. LHD machines are specially designed and built to load, haul and dump. LHD technology provides a profitable solution whether the tunnel is large or small. LHD's loading philosophy is to clean the face and haul the blasted rock to a secondary muckpile or dump trucks. If trucks are not available, LHD dumps the material onto the secondary muckpile, which makes the cleaning effective.

The truck loading point and secondary muckpile is placed far enough away so that it does not disturb other operations near the face such as drilling, charging and roof support. If the tunnel is narrow, the secondary muckpile can be placed in turning/loading niches.(**FIGURE** *3.8.-1.*). In larger tunnels, muckpile and loading can be done in the tunnel itself with one or two LHDs. (**FIGURE** *3.8.-2.* and **FIGURE** *3.8.-3.*) This two -stage loading method effectively utilizes the fleet and reduces its size. LHD also provides long hauling distances, while still maintaining high capacity.

The features that make LHD superior in tunnel jobs include its suitable weight distribution, big bucket volumes, approximately 50% higher payloads compared to fronted loaders with the same engine size, and a long wheel base that gives better stability and allows high tramming speeds with full loads. (**FIGURE** *3.8.-4.* and **FIGURE** *3.8.-5.*)

FIGURE *3.8.-1. Loading in narrow tunnel - loading/turning niche.*

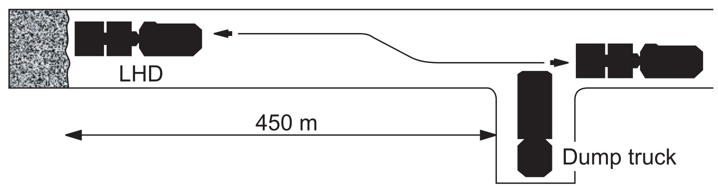


FIGURE *3.8.-2. Loading in large tunnel without loading niches with one LHD-machine.*

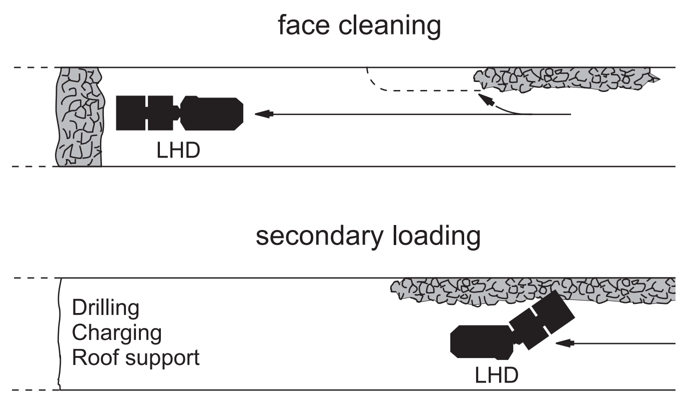


FIGURE *3.8.-3. Loading in large tunnel with two LHD-machines.*

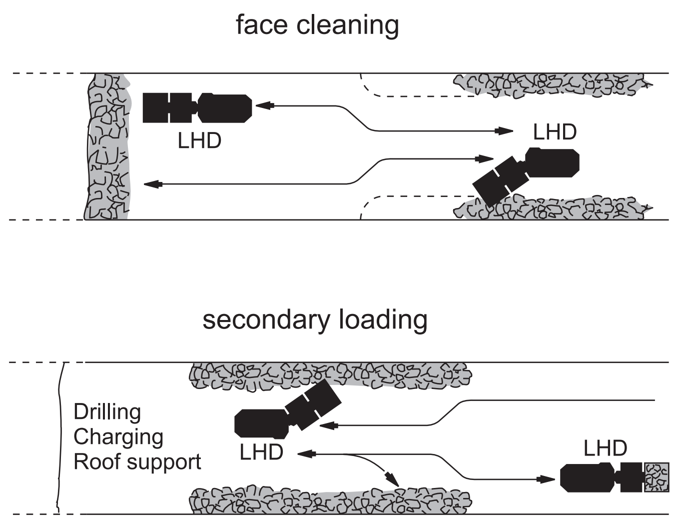
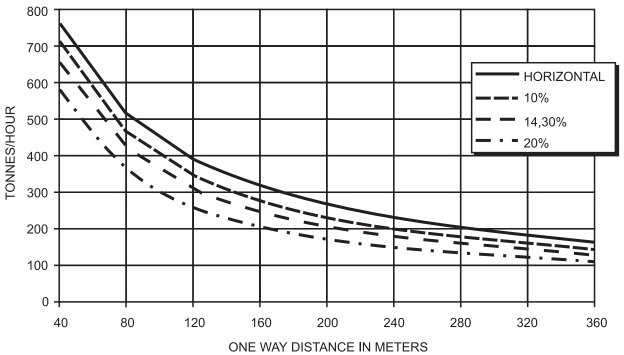


FIGURE *3.8.-4. Toro 1250 D.*



Basic data for calculations:

Payload	12 500 kg
Overall speed limit	10 km/h
Fixed cycle time	0.5 min

Fixed cycle time includes the time for filling, dumping, turning and accelerating.

Slope / Grade (%)	Horizontal/0	1:10 / 10	1:7 / 14.3	1:5 / 20
Loaded Speed (km/h)	10	7,5	5,9	4,2
Empty speed (km/h)	10	10	10	10

FIGURE *3.8.-5. Theoretical loading capacity of Toro 1250 in different inclinations.*

DUMP TRUCKS

Dump trucks are usually used at underground work sites that have steep inclines. Dump trucks are suitable in ramp driving, underground or to-surface rock transport. TAMROCK LHD and dump trucks match in each other in size, working space and capacity and are commonly used together. The greatest advantage of dump truck haulage is its flexibility, which cuts down on long-term planning. The dump truck fleet adjusts easily to production changes.

FIGURE *3.8.-6. Toro 50 D dump truck.*

Dump trucks are divided in to two categories: articulated and rigid dump trucks. Articulated dump trucks have payload capacities from 10 - 50 tons, and they are especially designed for difficult operating conditions (most climates and terrain). The ADT's low loading height does not limit the size of the loading machine. It has good maneuverability due to its narrow body width. Articulation and hydraulic steering helps operation in loading and dumping.

Rigid dump trucks still hold the dominant position in quarries. One reason is that they have no competitors in the over 50 ton class.

WHEEL LOADERS/FRONT-END LOADERS

Wheel loaders are used only for loading, therefore they have an easier and more economical design than LHDs. The advantages of the wheel loader is its mobility, versatility and high bucket capacity, which facilitate features not only for loading but also maintenance, short-distance hauling and pile preparing after blasting. The wheel loader also has some negative properties. It needs a solid working floor and well prepared heap. Break force and dump height are small compared to hydraulic excavators.

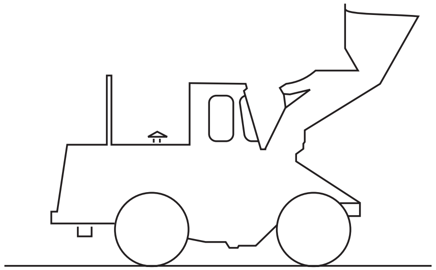


FIGURE *3.8.-7. Wheel loader.*

CONTINUOUS LOADING

Continuous loading is a method used at tunneling works.
(**FIGURE** *3.8.-8.*) The continuous loader needs either a rail or wheel mounted hauling system. Some continuous loaders have a special application, which loads on tracks, and hauls and trams on rails. One benefit of the continuous loader is that it does not require turning niches. Its disadvantages include poor mobility, low tramming speed and its inability to easily handle big boulders. Muck jams also pose problems.

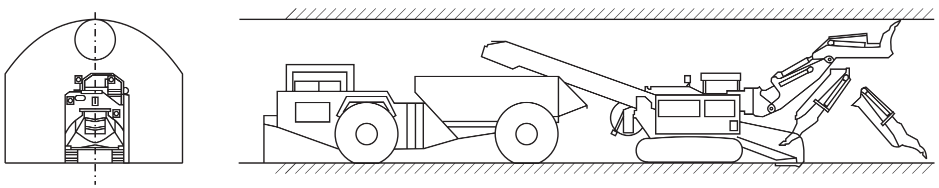


FIGURE *3.8.-8. Continuous loading to dump truck.*

HYDRAULIC EXCAVATORS

Hydraulic excavators can be divided into two different categories:

The first category consists of track (**FIGURE** *3.8.-9.*), wheel model and second category comprises the normal digging excavator and front shovel category (**FIGURE** *3.8.-10.*). Front shovels are divided further into two groups: front dump bucket and bottom dump bucket.

Excavators are mainly used in surface excavation sites. Front shovels are also suitable in large tunnels (more than 40 m^2). Some advantages for hydraulic excavators include:

- Successful attack of solid that needs blasting before being loaded by wheel loader
- Hydraulic excavators operate on rough surface
- Wheel excavators move as fast as wheel loaders

A disadvantage is that it requires a separate loader for collecting and preparing the rock pile.

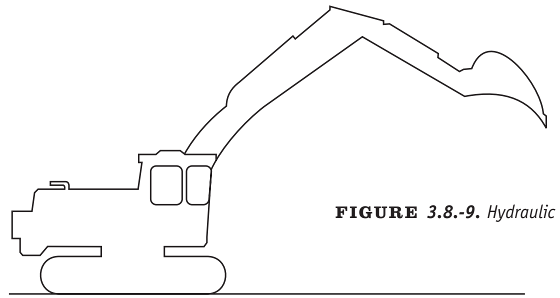


FIGURE *3.8.-9. Hydraulic excavator.*

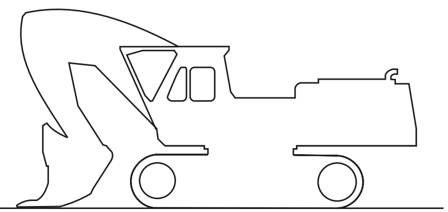


FIGURE *3.8.-10 Front shovel.*

BELT CONVEYORS

Belt conveyors are often the most economical transportation mode. Conveyors have traditionally been used for transporting overburden and other materials of limited particle size. Belt conveyors in tunneling and quarrying applications have been limited due to its maximum practical particle size of approximately 300 mm.

The development of mobile and semi-mobile crushing equipment, which enables crushing close to the face, has made it possible to switch from hauling by trucks to the more economical method of continuous transportation by belt conveyors. Belt conveyors can be adapted to any loading method.

Opposed to other transportation methods, the advantages of belt conveying include its almost unlimited capacity, low operating costs, often lower total investment costs and

environmental features. The belt conveyor is most cost competitive when handling heavy materials, high capacities and big lifting heights.
Long distances do not pose problems for the belt conveyor. Many Roxon over-land conveyors with lengths up to 15 km operate worldwide to the full satisfaction of their owners. (**FIGURE** *3.8.-11.)*

FIGURE ***3.8.-11.*** *Over-land conveyorbelt.*

3.9 CRUSHING OPERATIONS

The crushing plant offers a customer the opportunity to produce products with required capacity, distribution and sales quality from a certain feed. The crushing plant's design requires exact information concerning the feed material and end product, in addition to any other requirements. Only correct basic data results the right application for the customer. It is not possible to create just one best concept for plant design as requirements for end products vary from country to country. However, a few influencing factors for design can be given.

DRILLING AND BLASTING

The feed curve produced by drilling and blasting mainly affects the primary crushing plant design, but other factors must be taken into consideration in all crushing stages.
Before the actual selection of crushing equipment, one should study the effect of different drilling and blasting designs on the required production and end-product quality. In some cases by changing the drilling and blasting parameters, the customer can achieve benefits from down sizing the equipment fleet. By optimizing the entire process, from drilling to the crushed end product, operational savings could be achieved. The ***Table 3.9-1*** shows calculation example of the combined costs in a theoretical case in which the same capacity and

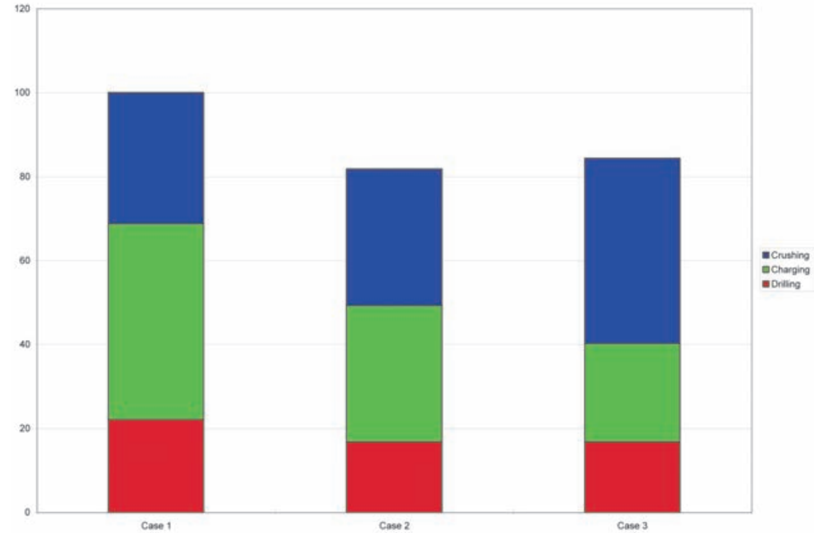


FIGURE ***3.9.-1.*** *The proportional comparison of the operations cost with same capacity and fragmentation requirement.*

Table 3.9.-1. *Case data for cost comparison for the same capacity (900 t/ph) and fragmentation requirement (0-200 mm).*

	CASE 1	*CASE 2*	*CASE 3*
Drilling			
- Machine	Tamrock CHA1100	Tamrock CHA1100	Tamrock CHA1100
- Amount	2	2	2
- Hole size (mm)	102	102	102
- Pattern (m^2)	8,8	11,9	11,5
Charging			
- Explosive type	Dynamite/Anfo	Dynamite/Anfo	Dynamite/Anfo
- Explosive price ($/kg)	2/1,1	2/1,1	2/1,1
- Specific charge (kg/ton)	0,29	0,2	0,14
Crushing			
- Vibrating feeder	Nordberg B16-56	Nordberg B16-56	Nordberg B16-56
- Jaw crusher	Nordberg C 110 B	Nordberg C 140 B	Nordberg C 200 B
- Jaw feed opening (m x m)	1,1 X 0,85	1,4 x 1,05	2,0 x 1,5
- Max. feed size (mm)	700	900	1400

Proportional. Only the investment and operational costs of each basic unit has been accounted for.

fragmentation requirements after primary crushing are obtained by three different drill patterns. The combined costs account only for the investments and direct operational costs of each basic unit. Material haulage, foundation investment etc. have been excluded. Results (**FIGURE** *3.9-1.*) shows that although the largest drill pattern (case 3) has the lowest drilling and blasting costs, the middle method (case 2) is most economical in total.

FEED MATERIAL

Various material characteristic data proves that the capacities may vary significantly with different materials.

From feed gradation and material purity, the following items must be studied to determine the influence on the process design.
- Fine material quantity may cause unnecessary wear at the crusher liners and clogging. Feeding the fine material into the crusher must be determined case by case as it has both positive and negative effects.
- If the feed material is gravel or blasted rock, the fine material quantity sets requirements for prescalping, and the round stones on liner form and crusher cavity angle.
- With coarse feed material, a higher reduction ratio is needed which decreases production capacity and increases energy consumption.
- Wide feed fractions have a higher density than narrow ones. This also easily causes clogging, especially when fine material quantity is critical.
- Crusher size is defined by maximum rock size and capacity.

Feed material characteristics can be tested by the methods
mentioned in chapter 2.7. This provides good basic data for plant design. However, test crushing gives the best and most reliable results.
- The Shatter and Los Angeles index estimate the quantity of fine material produced by crushing. The curve can be affected by changing crusher rpm, stroke or crusher cavity.
- A high work index increases power demand and decreases capacity.
- Material wearing influences on the selection of impact or pressure crushing. It also influences on what wear part is chosen.
- The end product must be free of waste feed. Waste feed must be screened off and discharged before the actual process.
- Humidity decreases the friction particularly in gravel resulting in lower crusher capacity.
- A low bulk density decreases capacity, for example, in recycling, with limestone and narrow fractions. The ores are an example of an opposite effect.
- Material structure affects the end product cubicity. Pin-like shapes disturb the production of cubic end products. Difficult materials require a smaller reduction ratio. Furthermore, the whole process requires more careful planning.

CRUSHING EQUIPMENT AND STAGES

Crusher, and especially cone/gyrator behavior and performance, depend on crusher parameters that influence:
- Speed including critical and under & overcritical speeds
- Stroke
- Feed arrangements

The plant's different crushing stages have various requirements that affect plant design. When the number of crushing stages increases, they are made more precise to better meet the set requirements. A smaller number of stages means an increased number of compromises. The crushing stages consist of primary, secondary and tertiary/fine stages.

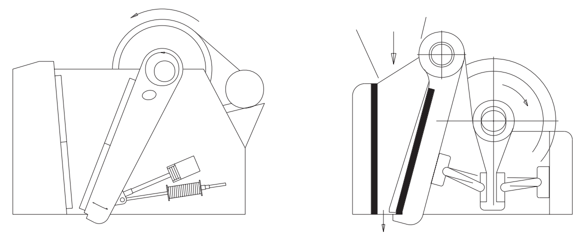


FIGURE *3.9.-2. Single and double toggle jaws.*

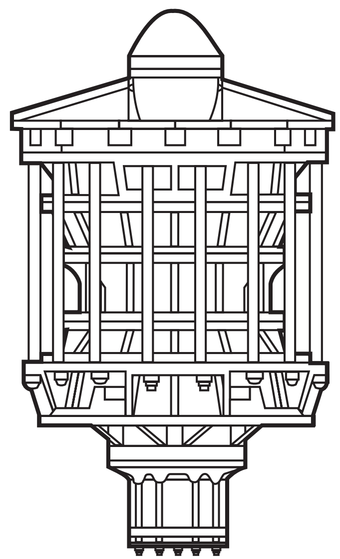


FIGURE *3.9.-3. Primary gyratory.*

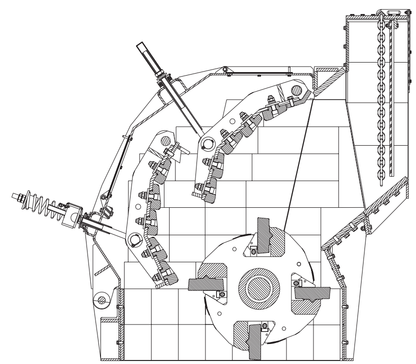


FIGURE *3.9.-4. Impactor.*

The process factors which influence on choosing the primary crusher are: maximum feed material size, capacity, end product distribution (max. size) and wear. The typical feed size for primary crusher is 500-2000 mm, and the main purpose is to crush rock for further crushing. The possible alternatives equipment for primary crushing are: impact, cone and jaw crushers.

- From jaw crusher exits two types: single and double toggle types. **FIGURE** *3.9-2*. Typical capacities up to 2000 t/ph. The single toggle jaw represents a newer design with lighter construction that offers the same capacity, through better kinematics.
- Gyrators are used for applications, in which capacity requirements are mainly over 2000 ton/h. **FIGURE** *3.9-3*. The crusher weight can be several hundreds of tons.
- The impactor, **FIGURE** *3.9-4*, is simple in design and provides a higher crushing ratio than the jaw and gyratory. Its biggest drawback is that liner wear is too high except in very soft rocks.

Products after secondary stage must fulfill quality requirements. The task is to choose a crusher which can best meet the requirements set by the feed material and end product. The typical feed size for the secondary crusher is 100-400 mm. The purpose of secondary crushers is usually to crush rock small enough so that the tertiary stage can accept the feed. In some cases such as railway ballast, secondaries also produce the final product. Gyratory, cone and impact crushers are possible alternatives for secondary stage crusher.

- **FIGURE** *3.9-5* shows a secondary stage gyratory crusher, which typically consists of a large feed opening and steep crushing cavity. Setting adjustment is done by hydraulic lifting lowering the main shaft.
- **FIGURE** *3.9-6* shows a cone-type secondary crusher. It varies from the gyratory by its lack of top bearing and setting arrangement, which is done by turning the bowl.
- What is said about primary impactors is valid also for secondary impactors.

In the tertiary/fine stage, the typical feed size is 10-100 mm. The reduction ratio is small, because the final shape of the rock product is achieved during this stage. Energy consumption per crushed ton is the highest and the end product requirements at maximum in this phase. Tertiary crushers are almost always in closed circuit and should always be kept full (choke fed). The tertiary crusher also includes gyratory, cone or impact crushers.

- In the tertiary stage gyratory-cone (**FIGURE** *3.9.-7*) the lower part of the crusher is the same as in secondary gyratories. There are several liner configurations available depending on feed size.
- The main difference from the secondary cone is that either the liners in the tertiary cone (**FIGURE** *3.9.-8*) are the so called "short-head type" or feed openings are much smaller, as in secondary types.
- What is said about primary impactors is valid for tertiary impactors. In some cases, the impactor may be of vertical shaft configuration.

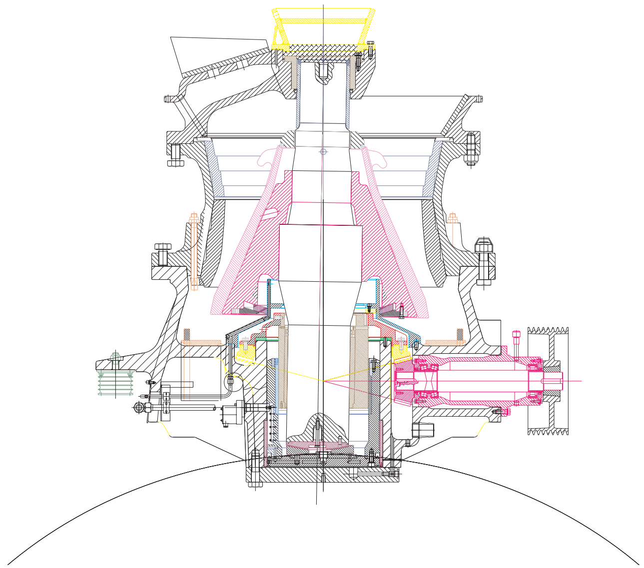


FIGURE *3.9.-5. Secondary gyratory.*

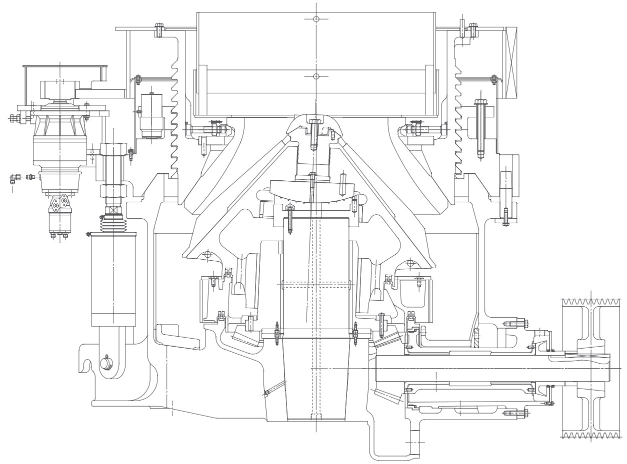


FIGURE *3.9.-6. Secondary cone.*

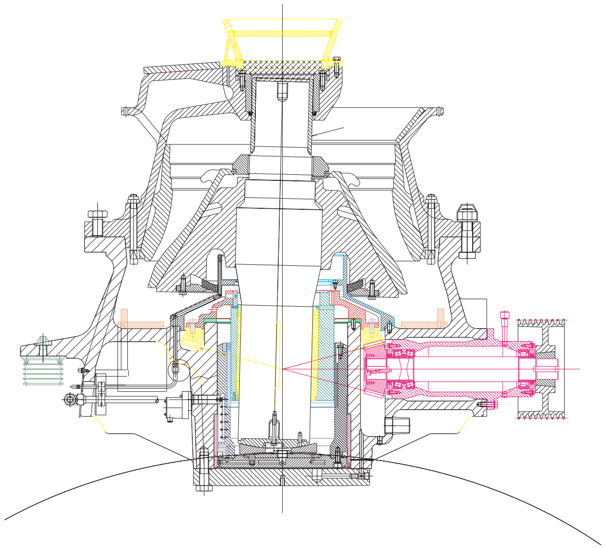


FIGURE *3.9.-7. Tertiary gyratorycone.*

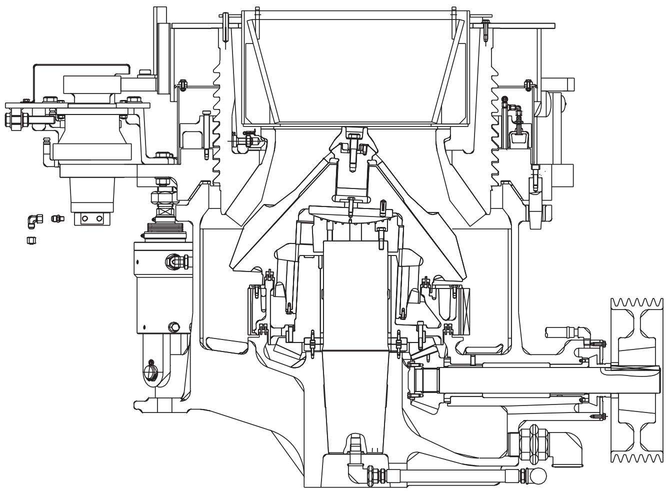


FIGURE *3.9.-8. Tertiary cone.*

CRUSHING PLANT EXAMPLE

Table 3.9.-2 and **FIGURE** ***3.9-9.*** on next page show a process example of a three-stage plant. The feed material is blasted stone and the end product is railway ballast (32/64 mm) and fractions (0/5, 5/10, 10/15 and 15/20 mm) and fraction 0/20 mm including impurities is screened off from the feed material.

CRUSHING PLANT OPTIONS

The flow sheet serves as a model to predict the performance of the plant, arranging the proper equipment into a balanced system, resulting in the desired proportions of finished material size. Crusher and screen size is selected according to capacity and product fraction requirements.

Regarding the crushing selection, the plant designer may choose to fulfill machine requirements by integrating into the design a:
- stationary (fixed) plant
- semi-mobile (skid-mounted) unit
- mobile (portable or self-propelled) unit

Each producer is looking for total crushing solutions to fit his own needs. The stationary unit is the preferred solution for a long-term plant and for high demand good-quality product. The plant is therefore customized according the required process and customer preferences. The mobile unit is the best for customers seeking business from small contracts in the neighborhood, other counties and even other countries.

Self-propelled Crushing Units

The self-propelled crushing units are those mobile units which are powered by diesel engine and have tracks as undercarriage system. For primary crushing, the unit is equipped with a vibrating feeder, either with jaw or impact crusher and a belt conveyor system. If higher quality products in several fractions are required, the primary mobile crushing unit is combined with secondary and tertiary mobile units.

The loading of the mobile crushing unit be performed with any available loader type.
In typical excavator feeding, the operator controls the unmanned crushing unit by remote control.

The mobile crushing unit applications consist or four categories:
- small crushing tasks
- recycling
- open pits and quarries
- tunnelling and underground operations

Table 3.9.-2. Process example of a three stage plant.

Crushing stage/part number	Part	Task/explanation	Remark
1	Feed material	Blasted rock	Gradation 0/750
I/2	Feeder	Adjusts primary crushing unit capacity and screens off material smaller than the crusher setting	
I/4	Jaw crusher	Crushes material down to suitable size for following crushing stages	In primary crushing unit, capacity is at maximum which requires reserve capacity
I/3	Screen	Screens off waste material from material run through grizzly	If waste material is not screened off at this stage, it enters end product
5	End product 0/20	Waste material	Can normally be sold as filling material
I/6	Inter-mediate stockpile	Separates primary stage from latter part of process	Allows different capacities at stages and operation of stages to be separated
II/8	Secondary crusher	Crushes product to a suitable size for tertiary crusher and produces railway ballast	Closed circuit improves end-product quality and gives flexibility to process
II/9	Screen	Screens off railway ballast. Oversize is carried back to secondary crusher	
10	End product	Railway ballast 32/64 mm	Standards vary by country
III/11	Screen	Screens off feed material for fine crusher	
III/12	Interm. hopper	Evens capacity and keeps fine crusher always full of material	
III/13	Fine crusher	Guarantees end product distribution and cubicity	
III/14	Screen	Carries oversize back to fine crusher and screens end product	
III/15	Screen	End product screen	
16-19	End products	Example of high-quality end products, standards vary by country	

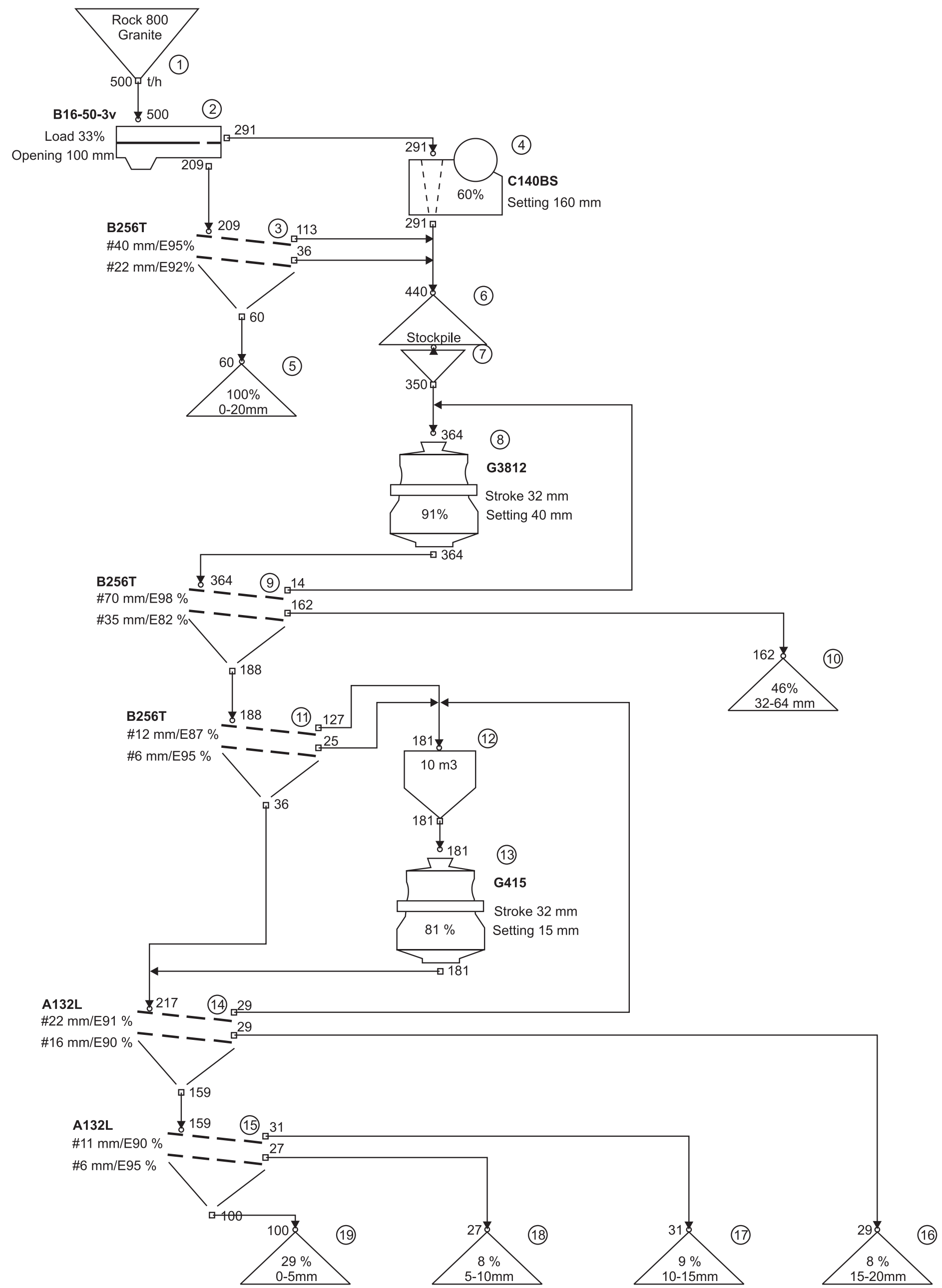


FIGURE 3.9.-9. *Crushing process example of a three stage plant.*

Small crushing tasks

The selfpropelled crushing units provide an ideal solution for exploiting small, scattered site rock, gravel and waste rock deposits. Small units are easily transported on a standard low-bed trailer on public roads to the worksite. Minimum time iswasted as the selfpropelled and selfcontained unit is within a few minutes ready to do the crushing task.

Selfpropelled crushing units are typically found on construction sites. Stone that has be blasted from the base can be rushed and used on-site. This kind of set up is also used onroad construction sites.

The main benefits of using mobile crushing units in small crushing tasks are:
- fast and easy access from site to site
- fast set up
- full on-site mobility
- easy material loading
- decreased requirement for haulage equipment

In **FIGURE** *3.9.-10* shown a self-propelled crushing unit in typical working environment

FIGURE ***3.9.-10.*** *Nordberg LT110 track-mounted primary crushing unit can be easily moved around a building site as excavation work progresses.*

Open pits and Quarries

Self-propelled crushing systems are extremely efficient crushing solutions for in-pit operations. In this area, the combination of mobile crushing units, mobile conveyors and fixed conveyor systems has very clear advantages over dump trucks to move the materials:
- haulage energy cost 50–75 % lower
- lower spare part costs
- lower labour cost
- less quarry road maintenance

Fewer dump trucks means less noise and dust, also safety is improved.

Tunneling and underground operations

The advantages achieved on surface operations by using mobile crushing units applies also to underground environment in many ways. Especially the improved working safety as the wheel-mounted equipment fleet is replaced with a conveyor system.

3.10. ROCK BLASTING

3.10.1. Blasting Products

EXPLOSIVES

Technical properties:
An explosive has three basic characteristics:
- It is a chemical compound or mixture ignited by heat, impact, friction or a combination of all three
- Upon ignition, it decomposes very rapidly in a detonation (as opposed to a deflagration, which is slower and occurs for instance with gunpowder),
- Upon detonation, heat (4500 °C) and large quantities of high-pressure gases (250,000 bar) are rapidly released. The gases expand rapidly at high force to overcome the confining forces of the surrounding rock formation.

In commercial blasting, energy released by a detonation results in four basic effects:
- Rock fragmentation
- Rock displacement
- Ground vibration
- Air blast

The technical properties of explosives used in surface rock excavations are:
- Efficiency and stability
- Easy detonation and good explosive properties

- Safe handling
- Good film characteristics
- Non-toxic
- Water resistance and good storage properties
- Environmental properties
- Resistance to freezing
- Oxygen balance
- Shelf life

Efficiency and stability

An explosive's efficiency and stability determine, how explosive should be used in a blasting project. The other characteristics are more or less common to all explosives. When comparing the efficiency of different explosives, the following technical characteristics should be taken into consideration:
- Velocity of detonation (VOD)
- Explosion propagation (sensitivity)
- Specific gas volume
- Explosion heat
- Strength per unit weight
- Charging density (density of explosive in the hole)
- Strength per unit volume

The detonation velocity of an explosive is the meter-per-second speed at which the detonation wave travels through the explosive charge. Many factors affect detonation velocity such as product type, diameter, confinement, temperature and degree of priming.

The detonation velocity of today's commercial explosives range from approximately 1500 to 7000 meters per second. Two types according to their detonation velocity exist (**FIGURE** ***3.10.-1.***):

- Low explosives (1500 - 2500 m/s)
- High explosives (2500 - 7000 m/s)

To a certain diameter and depending on the type of explosive, the diameter of the product influences the velocity. In general, the larger the diameter, the higher the velocity until hydrodynamic velocity (maximum velocity) is reached.

Every explosive also has a "critical diameter" which is the minimum diameter at which the detonation process, once initiated, becomes self-supporting in the column of the charge. With diameters smaller than the "critical diameter" the detonation is not supported and is extinguished.

Temperature changes affect the velocity depending on the explosive type. Typically, explosives that contain little or no liquid are relatively unaffected at the normal low temperature experienced in commercial blasting.

Sufficient priming ensures that explosives reach maximum velocity as quickly as possible. Insufficient priming results in the failure of the explosive to detonate, in slow build-up to final velocity and in a lower order detonation.

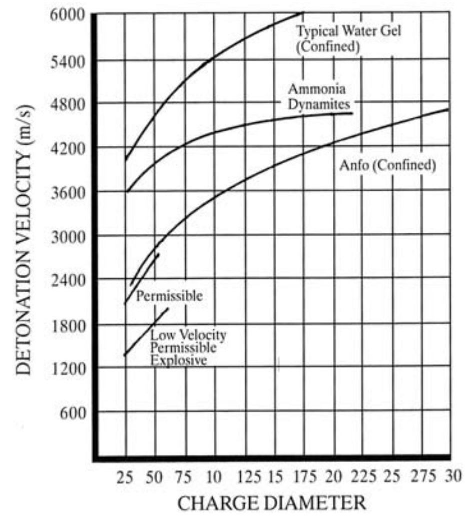


FIGURE ***3.10.-1.***
Detonation velocity as a function of diameter.

Explosion propagation is required to continue the reaction through the total charge, or the ability to ignite the next charge through a distance in free air. Propagation increases in the blasthole. Increasing diameter also increases explosion propagation.

Some types of explosives are so sensitive that they propagate between blastholes over considerable distances. This depends on the type of material to be blasted, the explosive, the size of the charge, the distance between blastholes and other factors such as the presence of water. In most situations, it is important that individual charges do not propagate, but detonate independently with a predetermined delay.

The specific gas volume is the amount of gas created by one kilogram of explosives under normal conditions (0°C and 760 mm Hg) expressed in liters/kg. The explosion heat is the amount of energy released upon detonation. It is usually expressed in kJ/kg. The effect of the gas pressure wave depends on the amount of heat and gas volume created by the explosion. In heat expansion 30 - 40% of the heat is converted into mechanical work.

The combined effect of specific gas volume and explosion heat is that the heat released actually expands the gases produced. The greater the gas volume and the hotter the gases, the more effective the explosive is.

Strength per unit weight (s) Langefors Weight Strength is a ratio that indicates the explosive's energy, which is calculated from the volume of gases and explosion heat:

$s = 5e / 6 + V / 6$ where s = Strength per unit weight
e = Energy coefficient
V = Coefficient of volume

$e = 425Q_V / 500000$ where Q_V = Explosion heat in kcal /kg

$V = V_1 / 850$ m where V_1 = Gas volume at 0°C / 1 atm

The s values of other explosives are thus expressed in relation to the s value of dynamite.

The performance of an explosive is not only determined by the total energy released by the explosion; it also depends on the rate of energy release and how effectively the energy is utilized for breaking and moving the blasted material. In short, both the explosive properties and the properties of the material being blasted influence the explosive's effectiveness. The s value also underrates the explosives used in the largest quantities in modern rock blasting, such as ANFO, heavy ANFO and emulsion explosives. Such explosives have a low flame temperature and are therefore used early in volume expansion before the reaction products enter the atmosphere. These explosives have a higher efficiency in rock blasting than higher temperature nitroglycerin explosives (compensating for the lower explosion energy which is much lower for ANFO and emulsions than for most nitroglycerin explosives).

The charging density is the amount of explosive in a certain hole volume. It is normally expressed in kg/dm^3. This is a very important when planning the blast because by knowing the explosive density, operators can design shots of any size with the proper charge per meter. The density of most commercial explosives ranges from approximately 0.8 kg / dm^3 to 1.65 kg / dm^3.

In the planning phase, it is useful to know approximately how many kilograms of explosives will load in one blasthole meter. **FIGURE** ***3.10.-2.*** shows the figures for kilograms of explosive per loaded blasthole meter. The higher the charging density, the better the crushing ability of the explosive charge. Explosives, which are packed in cartridges, are compacted in the blasthole. The degree of compaction varies according to diameter of the hole and the cartridge, hole length and inclination and the cartridge type. Typically is the degree of compaction 5-10% to dynamite and packed emulsion explosives, counted against the diameter of the cartridge. The crushing pressure, which causes the first cracks in the blasthole, depends on the pressure against the hole walls:

$P = 0.25 Dv^2$ where P = Explosion pressure (bar)
D = Charging density (kg/dm^3)
v = Explosion velocity (m/s)

The strength-per-unit volume gives the extracting efficiency of an explosive at different charging densities compared to the extracting efficiency of dynamite at the same charging densities.

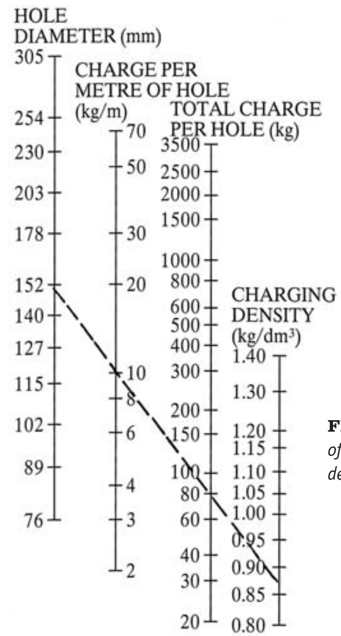


FIGURE ***3.10.-2.*** *Charge per meter of blasthole as a function of charging density and blasthole diameter.*

Easy detonation and good explosion

All commercial explosives can be ignited by a blasting cap or an electric detonator, except for ANFO, slurry and most emulsions. In some cases, ANFO explosives are ignited with a detonator, however, primers are usually used. For this purpose, the primer should consist of an explosive with high detonation velocity. Dynamite can be successfully used in igniting ANFO.

Slurries and most emulsions must always be ignited with a specially designed primer. These primers typically have a very high detonation velocity. In practice, the critical blasthole diameter for slurries and most emulsions is 51 mm or 2".

Dynamite must be sensitive enough to ensure detonation over the entire length of the charge. For this reason sensitivity is especially critical in small diameter holes, such as 25 - 38 mm or 1" - 1 $^1/_2$".

Safe handling

Modern commercial explosives must be safe to transport and charge, and risk-free to the handling or blasting crew. Therefore explosives are extensively tested before they are approved by the authorities. A basic test for commercial explosives is the stress test, in which a weight is permitted to fall onto the explosive from a certain height. This test determines the sensitivity of the explosive to impacts.

Explosives are also subjected to tests where their sensitivity to friction and shooting is determined. Safe handling often depends on the insensitivity of an explosive when it is subjected to drilling. In practice, for one reason or another, certain parts of bench blasts or even complete blasts do not detonate when fired. Additional holes must be drilled and drill bits sometimes punch the explosive charges in the rock. Although modern explosives are relatively insensitive to shocks, it is not possible to be 100% sure of the safety of drilling through charges.

Toxic fumes

Some fumes created by detonating explosives and blasting agents may contain reaction products that pose health or environmental hazards. The majority of reaction products from detonating oxygen-balanced explosives are harmless. Small amounts of toxic reaction products are, however, formed as a result of deviations from oxygen balance, incomplete reactions or secondary reactions with the atmospheric air.

The amount of non-ideal detonation products formed depends on factors such as explosive composition and homogeneity, effect of water on the explosive after being loaded into a wet drillhole (water resistance), velocity of detonation, charge diameter, loading density, initiation type, cartridge wrapper type and most importantly explosive confinement. Before and during detonation, additional reactions can also occur between the explosive and the surrounding rock such as when the rock contains sulfide or other reactive components.

Table 3.10.-1. *Threshold values for toxic fumes.*

Fume	*Finland (8 hr working day)*	*Sweden*	*USA (Threshold values, ppm)*
CO	50	25	50
NO_x		20	
NO_2	3	2	5
NO	25		25
NH_3			50
CO_2	5000		5000
SO_2	2		
H_2S	10		

Most commonly, only carbon monoxide (CO) and nitrogen oxides (NO and NO_2) reach a high enough concentration so that it is dangerous to be exposed to the fumes after a blast. Carbon monoxide (CO) is a colorless, odorless gas formed by the incomplete combustion of organic material. Its density is slightly lower (1.25 kg/m^3) than atmospheric air (1.293 kg/m^3). However, it is extremely unlikely that separation could occur in a mine or tunnel. If inhaled, the gas is extremely toxic. Carbon monoxide´s affinity for blood hemoglobin is considerably higher than that of oxygen. Asphyxiation can occur if the CO concentration is high enough.

Nitric oxide (NO) is also colorless and highly toxic if inhaled. NO is a strong irritant to skin and mucous membranes. Through oxidation, it forms nitrogen dioxide (NO_2). NO_2 is a reddish-brown gas and inhalation is fatal at high concentrations. If the fume cloud has NO_2 concentrations exceeding 30 ppm, the color is orange/red when observed against a white light source.

Table 3.10.-1. reviews threshold values of exposure concentrations established in Sweden by the Labour Safety Board [Arbetarsskylddsstyrelsen, 1974] and in the USA [ACGIM, 1970]. Investigations made by G. Persson showed that 4 - 8% of the fumes from tunnel rounds are trapped in the muck pile for 1 hour. It is possible to evaluate 25 - 50% of trapped gas.

Fumes from nitroglycerin explosives contain small amounts of nitroglycerin and/or nitroglycol which dilate blood vessels, cause headaches and lowers blood pressure. Skin contact with the explosive is perhaps the most important form of exposure. Inhalation of vapors during charging or inhalation of the vapors from dead or partially reacted explosives after the blast cause headaches.

In the USA, the IME has established the Fume Classification Standard, which is based on the volume of poisonous gas emitted by a 1 1/4 * 8-inch cartridge when detonated under standardized conditions. High explosives are classified under this standard as follows:

Class 1:	0 - 0.16 ft^3	per cartridge
Class 2:	0.16 - 0.33 ft^3	per cartridge
Class 3:	0.33 - 0.67 ft^3	per cartridge

Only those explosives meeting the requirements of Class 1 may be used in underground mines. For coal mines, the MSHA has determined that the volume of poisonous gases produced must not exceed 71 liters per 454 gm of permissible explosive (2.5 ft^3 / lb). In any event, good ventilation is necessary. Dispersing the fumes from a blasted heading before miners return to work is a vital precaution.

It is possible to considerably reduce the amount of toxic fumes formed in tunneling if the following recommendations and procedures are adhered to during the planning and charging of the blast:
- Use an oxygen-balanced explosive with good fume characteristics.
- Use alignment devices when drilling so that most holes are slanted slightly upward, therefore, preventing water from accumulating in the drillhole, contaminating the explosive and affecting its detonating performance.

- Do not use detonating cords in ANFO-loaded small-diameter holes as it might not initiate the ANFO to complete reaction, resulting in large amounts of toxic fumes. The cord itself is strongly oxygen deficient, and by itself generates approximately 3 l CO per meter length of cord.
- Leave an unloaded (or preferably stemmed) hole length equal to 10 times the hole diameter at the mouth of the hole in tunneling and drifting. Filling explosives all the way to the collar does not contribute significantly to the breakage process, but it produces a lot of toxic fumes.

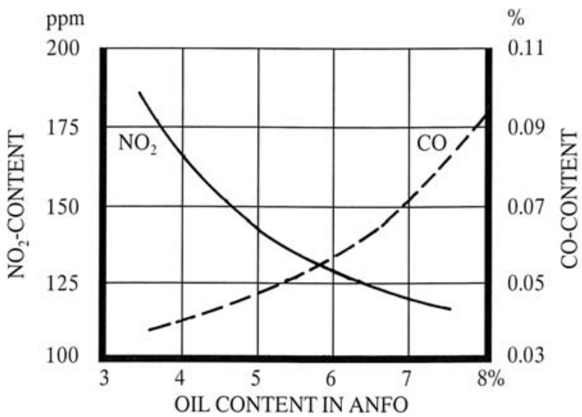


FIGURE *3.10.-3. ANFO oil content and CO and NO2 gas quantities.*

It is important to keep the oxygen balance at the correct level when ANFO is produced. Should the proportion of fuel oil in ANFO differ considerably from 5.7%, the mixture loses its power and hazardous gas production increases (**FIGURE** *3.10.-3.*).

Water resistance and storage properties

Many types of charging involve explosives that remain charged under water for long periods of time. Even in normal rock blasting, blastholes are often full of water. Plastic explosives normally have a high water resistance, and a well-packed explosive in a relatively solid hole has been shown to resist water for a considerably longer period than normally guaranteed. An explosive's water resistance is usually defined as the product's ability to withstand water penetration. It is generally expressed as the number of hours the explosive is submerged in static water and can still be reliably detonated. When water penetrates an explosive, it decreases the explosive's strength and desensitized it to such an extent that it does not detonate if it is exposed for prolonged periods of time or in severe water conditions. The explosive's water resistance depends not only on its packaging and inherent ability to withstand water, but also on water conditions. Static water at low pressures will not affect the explosive as quickly as high-pressure, fast-moving water.

The storage properties of commercial explosives define the period of time for which they can be stored without effecting safety, reliability, and performance. Although shelf life has improved, it is still important to avoid prolonged storage periods.

Plastic explosives should not be stored in or subjected to high temperatures since they can soften and the explosive's salt substance can penetrate the paper wrapping around the cartridge. If this happens, the cartridges become deformed and difficult to use. Powder-type cartridge explosives are usually more sensitive to moisture in storage. In high humidity, the explosive's salt can form deposits on the cartridge which harden. Aging does not apply in this case.

Mixed explosives can in certain cases separate, in which their characteristics change completely. Factory-manufactured explosives are normally produced in such a way that this does not occur.It is important to keep explosive stores clean and dry. Storage should be planned in such a way to ensure fast usage.

Types of explosives

An extensive range of different types and grades of explosives is made to suit all blasting requirements. A breakdown of blasting explosives is presented in ***Table 3.10.-2.***

Table 3.10.-2. *Breakdown of blasting explosives.*

Explosive base	*Explosive type*	*Features*
Nitro-glycerin	Dynamite Gelatin	A highly adaptable cartridge explosive currently widely used because of its excellent performance in smaller diameter holes.
Ammonium nitrate	ANFO	A low-cost, high-power, extremely safe, liquid explosive made from porous prilled ammonium nitrate and fuel oil. Poor water resistance.
Water Oil	Slurry Emulsion	Essentially, ANFO which becomes water resistant by adding water and forming either a gel (water gel), or creating astable oil/water emulsion and ANFO called heavy ANFO. Available in package or liquid form.

Dynamite and gelatins

Over the years, ammonium nitrate has become more important in dynamite, replacing a large portion of nitroglycerin. The same is true with the dynamite currently used in the Nordic countries.

There are three basic types of dynamite: granular, semi-gelatin and gelatin. Gelatin and semi-gelatin dynamite contain nitrocotton, a cellulose nitrate that combines with nitroglycerin to form a cohesive gel. The viscosity of this product depends on the percentage of nitrocotton. Granular dynamite, on the other hand, does not contain nitrocotton and has a grainy texture.

Usually, the higher the nitroglycerin percentage, the more water resistant the explosive becomes. Dynamite issued in bottom charges. Low-percentage dynamite is often used in column charges. These explosives are often used in small diameter blastholes at construction blasting sites. Paraffin paper covers are used for small-cartridge explosives (d (40 mm) and plastic bags for bigger cartridges. In certain smooth blasting projects, pre-splitting and generally for tasks where a very light charge is necessary, special dynamite can be used as a ready-made tube charge.

Aniitti

Aniitti is a non-nitroglycerin, ammonium-nitrate explosive, which among other things contains trotyl (TNT) and aluminum. Aniitti in cartridges is used especially in drifting. Aniitti contained in plastic bags is mostly used in large holes as a column charge and in clearing.

Original ANFO

ANFO is a mixture of ammonium nitrate and fuel oil (5.7%), in which ammonium nitrate acts as the oxidizer and the fuel oil acts as the fuel. ANFO offers great economy and safety in modern blasting applications. It is generally one-third to one-half cheaper than nitroglycerin explosives and it is considerably safer to handle because it is non-sensitive. In many types of blasting, ANFO produces better fragmentation due to its high gas producing properties.

ANFO is among the best explosive for blasting dry holes in excess of 51 mm (2”) in hole diameter, which are conductive to breakage by gaseous expansion. However, it is not so good in small-diameter blastholes and conditions that require very high detonating velocities.

The main disadvantage of ANFO is that it is not water resistant:
if it gets wet, it no longer detonates. Some ANFO products can be charged in wet holes, if the water is removed before charging. Leaving ANFO in a loaded hole for an extended period of time should therefore be avoided. Additionally, ANFO can not be detonated by a normal detonator.

Detonation velocity changes with the diameter of the blasthole and reaches the highest velocity of 4400 m / s in a 250 mm blasthole. Likewise, detonation velocity decreases with the diameter of the blasthole. 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-sized 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 / s), the chemical reaction will be incomplete.

Reduced ANFO

Different methods of mixing ANFO with inert material has been tried, but today the most commonly used material for reduction is expanded polystyrene spheres. Due to varying density, ANFO and the polystyrene spheres tend to separate. However, a new charging technique developed by Dyno Industries Norway solved this problem. The technique is similar to that used for concrete grouting. ANFO and the reduction material are stored in separate containers but then mixed together in a charging hose through which the mixture is blown into the blasthole. Two containers are needed for charging by this method. However, it is most practical and least bulky when one container is placed inside the other.

Heavy ANFO

Heavy ANFO is a mixture of ANFO and emulsion explosive. It is becoming more popular because it is often as effective as pure emulsions and considerably cheaper. Prilled ANFO consists of spheres or prills of sensitized ammonium nitrate infused with fuel oil. Maximum density occurs when the prills are in mutual contact, which leaves voids that, in heavy ANFO, are filled with a base emulsion, cold pumpable explosive with sufficient viscosity and stability. The sensitizing mechanism for emulsion explosives is provided by voids. In heavy ANFO, the prills act as voids or density adjusters and the emulsion fills the gaps. The proportions of the constituents can be varied to alter sensitivity, energy and water resistance.

With the normal range of compositions, density increases with emulsion content up to a maximum of 1.3 kg / dm^3. The charge’s sensitivity is opposite to both the density and emulsion content.

Both energy and sensitivity peak at a density of approximately 1.3. Adding microballoons to the base emulsion enhances sensitivity, energy and water resistance, however it also increases the cost. Water resistance is also dependent upon emulsion content and quality, sensitivity, degree of blending and particularly storage time during and after loading.

Slurries

Slurries are specially designed for large-hole blasting and wet conditions. Slurries are not normally cap-insensitive and therefore must be initiated with a primer. Slurries are water resistant and are either pumped straight into the hole or applied in plastic bags.

Slurry contains ammonium nitrate, and often aluminum, water and substances to keep the explosive homo-geneous. The properties of any individual composition depend on the type and proportions of various solid ingredients. Because it is denser than water

(1.4 - 1.6 kg/dm^3), the slurry sinks to the bottom of a wet blasthole. The detonation velocity of slurries ranges from 3,400 m/s to 5,500 m/s.

Emulsion explosives

Emulsion explosives are grabbing more market share due to their safe and versatile features. Emulsions are so-called two-phase products. The dispersed phase is dissimulated throughout a continuous phase. Explosive emulsions consist of a mixture of fuel and oxidant components. The oxidizers are primarily nitrates and the fuels mostly mineral or organic hydrocarbon derivations. The oxidant / fuel ratio is approximately 10:1.

When inspected with an electron microscopy, the structure of the emulsion exhibits a polyhedral shape with each droplet covered by a film of the fuel phase.

The detonation reaction occurs at the boundaries between the two phases. Emulsions provide increased explosive efficiency because both phases are liquid and the dispersed nitrate solution droplets are smaller than other conventional explosives; 0.001 mm as opposed to 0.2 mm. They are tightly packed within the fuel phase and provide increased surface contact, effectively enhancing the reaction. The strength of the reaction can therefore be altered by changing the degree of fuel and oxidant. The water content of the nitrate is also reduced by using super-saturated solutions.

Increased efficiency is reflected in the detonation velocity, 5000 - 6000 m/s for emulsions compared with approximately 3200 m/s for ANFO and 3300 m/s for slurry. This high-velocity detonation is one of the major advantages of emulsions as it provides high-shock energy, which is a significant factor in hard rocks.

Unlike other explosives, emulsions do not use chemical sensitizers. Instead, voids in the emulsion fulfills this requirement. The number of voids determines the density of the mixture.

The viscosity and density of any emulsion is determined largely by the physical characteristics of the organic fuel phase, which can vary from liquid fuel oil to viscous waxes. Unlike slurries, emulsions can not be gelled or cross-linked, their structure being characteristic of the nitrate and continuous fuel phase.

Bulk emulsion explosives are especially suited for open-pit mining because of their water resistant properties and chemical stability.

Emulsions are structurally distinct from slurries because they do not contain thickeners or gelling compounds, and require mixing at approximately 80°C. Also they do not contain structural additives, the phases must cool by 30° - 50°C before the fuel phase becomes semi-rigid.

By using different percentages of microballoons and aluminum, a wide range of emulsion explosives can be manufactured.

The cap sensitive range is intended for small and medium-diameter blastholes and is delivered in paper shells and plastic bags. The non-cap sensitive range is intended for medium and large-diameter blastholes in bench blasting, and is delivered in plastic hoses.

Pumpable bulk emulsions, which are economical alternatives to ANFO in quarrying, mining and tunneling are also very interesting. Pumpable bulk emulsifying and mixing process can take place either at the bulk emulsion station or on the mixing/loading truck. When the explosive is prepared at the station, the explosive truck acts as a simple transport and loading unit for the explosives. Alternatively, the truck can be used as a mixing and loading unit. In the latter case, explosive preparation will take place at the worksite, allowing continuous blasthole loading with precise amounts of various mixtures.

This system enables explosives to be produced economically. Loading is accurate, and bulk emulsion can easily be pumped at speeds of up to 200 kilograms per minute. A further advantage is that it can even be pumped into holes containing water. The substance can additionally remain unchanged for months even under severe conditions.

INITIATION OF EXPLOSIVES

General

Today's common surface blasting systems for initiating explosives are:

- Firing cord and blasting caps
- Detonating cord and detonating delays
- Electric blasting caps (detonators)
- Shock wave conduction to detonation caps
- Electro-magnetic firing methods
- Magnadet system
- Electronic detonators

Firing cord and blasting caps (detonators)

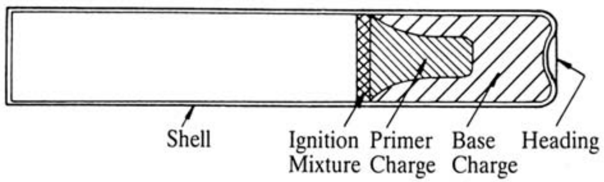


FIGURE *3.10.-4. Conventional blasting cap.*

The electric cap is the most widely used detonator. With its proper electrical energy and blast circuitry, a large number of electric blasting caps can be initiated from a single current source at a safe location from the blast area. Firing cord and blasting caps provide a non-electric method of initiating the explosive charges used in small blasting projects.

Detonating cord offers an alternative to electric detonators in smooth blasting and in possible, hazardous electric fields. The system enables the blaster to fire multihole blasts in sequence with a single firing, without cutting different length safety fuses.

The advantages of non-electric systems shock wave conduction are:

- Safety from stray electrical currents and radio frequency hazards
- Insensitivity to concussion and heat, whether confined or unconfined
- Non-electric tube will not detonate any commercial explosives, including sensitive nitroglycerin dynamite

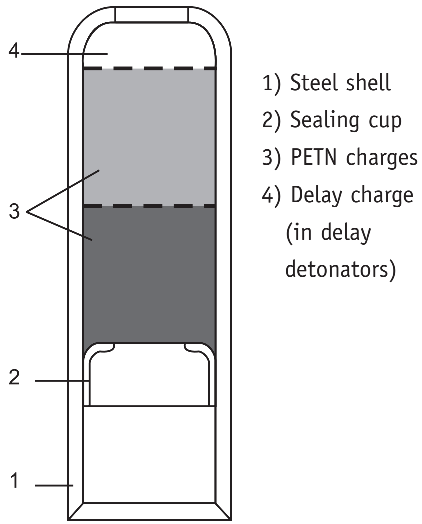


FIGURE *3.10.-5. Initiation element of new blasting cap (detonator)*

Firing cord and blasting caps (detonators)

Safety fuses and blasting caps are good for firing single and multiple shots in rotation but not for firing instantly detonating or short-delay charges because the fuse's burning rate is not sufficiently accurate.

FIGURE *3.10.-4.* shows a conventional blasting detonator. New non-primary explosive detonators (**FIGURE** *3.10.-5.*) are much less sensitive to different stimulus than conventional detonators. Blasting caps are manufactured in No. 6 and No. 8 strengths. For normal initiation with most explosives, the standard No. 6 cap is sufficiently strong. No. 8 provides additional initiation power.

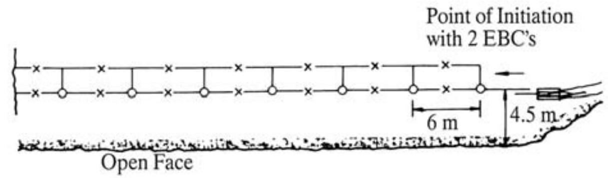


FIGURE *3.10.-6. Initiation in single- row blasting with detonation cord, detonating relays marked x. EBC = Electric blasting cap.*

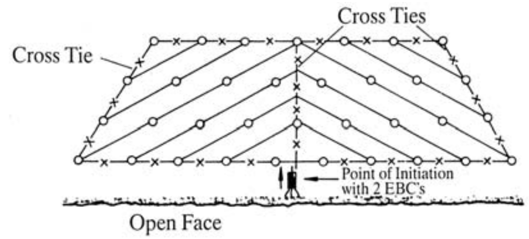


FIGURE *3.10.-7. Center initiation in multiple-row blasting with detonating cord. Detonating relays marked x. EBC = Electric blasting cap.*

The flame is conveyed through the safety fuse at a uniform rate for igniting the blasting cap. The safety fuse core consists of a black powder train, tightly wrapped with covering tape, textiles and waterproofing material such as asphalt and plastics.

In addition to its ability to initiate black powder and, together with blasting caps, initiate high explosives, firing cord provides a burning speed within reasonable tolerances of approximately 1 meter per 100 seconds (1 foot per 30 seconds).

Initiation by detonating cord

Detonating cord is a flexible tube containing a center core of high-velocity, cap-sensitive explosive that is used to

- Detonate other high explosives which it comes into contact with
- Transmit a detonation wave from detonating cord to detonating cord or to a non-electric cap

The core is covered in reinforced woven waterproof plastic. Detonation cord is not sensitive to ordinary shock or friction. The amount of high explosives in detonating cord varies from 1 to 1,300 grains of PETN per meter. The most commonly used grades contain 80 - 160 grains/m. By contrast to the 0,01 m/s burning rate for safety fuse, detonating cord detonates at approximately 6,000 - 8,000 m/s and has the effect of a blasting cap along its entire length.

Detonating cord is often used in large quarries and mines to initiate higher explosives. Today it is also common in smooth blasting. When detonating cord is used in places where there is danger of unpremeditated initiation of electric detonators, it is initiated by firing cord and blasting cap No. 8.

Due to its high detonation velocity, detonating cord together with light charges is used in smooth blasting. Multiple-charge explosives can be fired without significant delay between charges, by detonating from the top of the blasthole. In addition, the explosion's propagation direction can be selected according to the rock formation.

When blasting with detonating cord, detonating relays are used that consist of delayed units to provide convenient short-delay firing intervals (5 - 400 ms). Each relay consists of two delay detonators assembled in an aluminum sleeve which can be inserted in the detonating cord wherever a delay interval is required (**FIGURES** *3.10.-6.* and *3.10.-7.*).

Connecting a detonation cord:

- Keep each connection at a right angle. Plastic connectors are convenient and reliable
- Distance between parallel cords should be no less than 0.2 m
- Distance between relay connector and parallel cord should always be pointed in the direction of the detonating cord detonation

A simple delay arrangement is to use separate delay relays for each main line of detonating cord by connecting the individual holes in a row. This system achieves successive displacement from each row toward the free face.

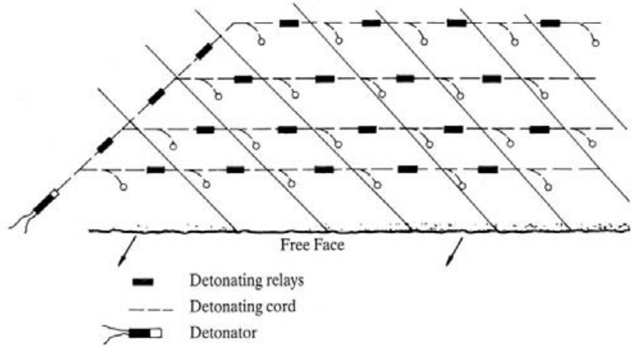


FIGURE *3.10.-8. Short-delay blasting using detonating relays*

FIGURE *3.10.-8.* shows an arrangement for a multiple-row blast via detonating relays on each hole. The holes are fired in a V shape across the cut. A maximum of four holes can be fired at any time.

The explosive's effect will be somewhat reduced when detonating with cord because the explosion does not reach maximum velocity. An explosive's duration velocity increases as the detonation advances from the initiation point. When detonating cord is used, detonation occurs at all points: in this case there is no increase in the detonation velocity. When detonating cord is used as a downline to carry the initiating impulse into a blasthole through cap-insensitive explosives such as ANFO, to a cap and primer, it burns some of the explosive it passes through, therefore diminishing the effective volume of the charge.

Detonating cord can disrupt the explosive column especially in smaller diameter blastholes, which affects the efficiency of the explosives involved. Detonating cord trunklines also contribute to noise associated with blasting in residential areas which also creates problems. One solution is to use lower-energy detonating cord. Detonating cord does have advantages; it is the safest, simplest and most reliable blast initiating system ever developed.

Because detonating cord initiates all capsensitive explosives loaded alongside the cord in the blasthole, bottom priming and delayed deck charging methods can not be used. Initiation must be either from the top of the explosive column or from spaced boosters. With ANFO and cap-insensitive slurries, energy transmitted through medium to high-strength detonating cord, for example with a 6 - 12 g/m PETN core, is sufficient to desensitize the ANFO, slurries or emulsions via a partial detonation effect, particularly in holes with diameters smaller than 102 mm (4").

Electric blasting caps

Electric blasting caps are classified by their electrical or functional characteristics. The former are based on the cap's fusehead sensitivity to electric current as follows :

- Low-current detonators with an initiation impulse of 3-5 mJ/ohm and current of 1-1.3 A (Prohibited in Finland, Sweden etc.),
- Caps with improved safety against unintentional initiation.
  These have an initiation impulse 25-35 times (120-140 mJ/ohm) greater than low-current caps, and also contain a current which is approximately three times (3-4 A) greater than low-current caps.
  Low-current caps have 3-4 times (15–20 mJ/ohm) and current approximately 1.5 times (1.5-2 A) greater than low-current detonators
- High-current electric blasting caps differ from low-current caps in their greater initiation energy requirements (2500 mJ/ohm and 35 A).

Worldwide tendency is moving toward higher currents and initiation impulses in electric blasting caps. It is a result of safety requirements in order to avoid the unintentional initiation of the detonator. Using high-current detonators requires powerful exploders with high voltages, which may limit this type of detonator. Insulation should also be carefully considered.

Classifying electric blasting caps according to their characteristics is based on detonation intervals:

- Instantaneous detonator
- Short-delay detonators
- Delay detonators

Instantaneous detonators ignite almost instantaneously when an electric current passes through the fusehead's resistance. Instantaneous detonators are used in small blasts such as secondary rock breaking. Prohibited in the same circuit with delay detonators.

Short-delay detonators are the most widely used delay caps in quarrying, openpit and construction blasting, with rows of holes breaking to free face. Typically, millisecond delay blasts will move rock farther away from the face than a slower series of delay caps due to the interaction between adjacent blastholes. The delay period (milliseconds) is 8 - 50 ms depending on the detonator manufacturer (1 ms = 1/1000 second). An adequate delay time for good fragmentation in bench blasting is 6 - 25 ms per meter burden.

Electric blasting circuitry design

In practice, two (2) types of blasting circuits exist:

- Series circuits
- Parallel-series circuits

Circuit selection depends on the number of fired electric blasting caps and operation. In general, a simple series circuit is used in small blasts. A parallel-series circuit is used when a large number of electric caps is involved.

Total resistance in a series circuit (**FIGURE** *3.10.-9.*) can be calculated using the equation:

$$R = R_1 + R_2 + R_3 + \ldots + Rm$$

where R = Total resistance of series circuit (ohms)
Rm = Resistance of an individual component in the series circuit (ohms)

Example:

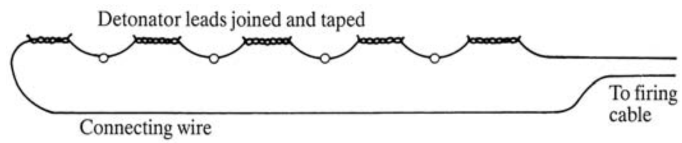


FIGURE ***3.10.-9***. *Series circuit.*

A series circuit with 40 short-delay caps, 35 meters of connecting wire and a shooting line 60 m long.

Component	*Quantity*	*Resistance each*	*Total resistance*
Cap	40	1.8 ohms	72 ohms
Connecting wire	35 m	0.05 ohms	1.75 ohms
Shooting line	60 m	0.05 ohms	3.0 ohms
Total resistance			76.75 ohms

Resistance and current calculations for matching parallel-series to the power source are:

1. Find the total resistance in one series circuit
2. Calculate total resistance in parallel-series circuit as if individual series circuits were of single resistance

The total resistance in a parallel circuit is calculated by:

$1/R = 1/R_1 + 1/R_2 + 1/R_3 + ... + 1/R_m$

where R = Total resistance in parallel circuit (ohms)
Rm = Resistance of an individual component in the series circuit (ohms)

Usually R1=R2=...Rm and Rm/n, where n is the number of parallel circuits. If individual series circuits in parallel series circuit are different, the currents in the detonators of the round are different. This may lead to unsuccesful firing of part of the detonators.

Example

A parallel-series circuit consisting of four series circuits each having 40 short-delay caps, and a connecting wire of 40 m; a 150-meter shooting wire is used.

Resistance in a single series circuit:

Component	*Quantity*	*Resistance each*	*Total resistance*
Cap	40	1.8 ohms	72 ohms
Connecting wire	40 m	0.05 ohms	2.0 ohms
Total resistance			74 ohms

Total resistance of four series circuits:
R = 74 ohms / 4 = 18.5 ohms
Total resistance of parallel-series circuit:

Component	*Quantity*	*Resistance each*	*Total resistance*
Series circuit	4	74 ohms	18.5 ohms
Shooting wire	150 m	0.05 ohms	7.5 ohms
Total resistance			26 ohms

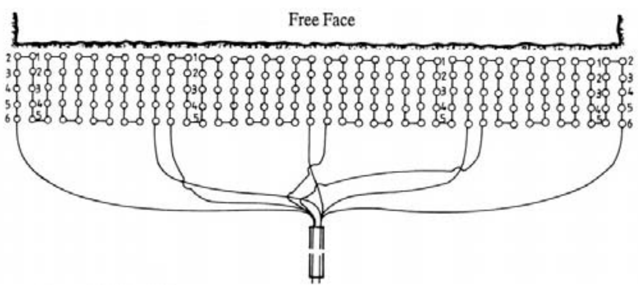


FIGURE ***3.10.-10.*** *Parallel-series circuit.*

Testing electric firing circuits

Specially designed testing instruments are required for blasting circuits and must be approved by the appropriate authorities.

The easiest instrument for testing is the Circuit Tester. It was designed to test a small amount (in Finland 4) of detonators in one series. The Circuit Tester only shows a result if the circuit is closed and no break occurs.

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

Electric blasting exploders

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

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

Exploders are devices which provide the necessary initiation impulse and current to initiate the detonator (circuits). The two basic types of exploders are:

- Generator blasting machines
- Capacitor discharge blasting machines

Generator blasting machines consist of a small, hand-driven electric generator. When activated, it produces a direct current pulse that fires the electric blasting caps. The generator connects to the blasting circuit when the blaster pushes down the handle.

Capacitor discharge blasting machines have a capacitor or a bank of capacitors that store a large quantity of electrical energy supplied by a dry cell or generator. By pushing down the handle, the blaster can discharge the stored energy into the blasting circuit in a fraction of a second through two terminal posts.

Different detonators have different energy and current requirements. The unit of initiation impulse mJ/ohms, shows that the field resistance is also significant in selecting the type of blasting machine; the type of detonator circuit - parallel or series connection - also affects the power required from the blasting machine.

Safety in electric blasting

Thorough evaluations in extraneous electricity should be made at blasting sites before any explosive is brought to the area because unwanted electrical energy may cause accidents that could seriously injure the blasting crew and damage materials.

Hazardous extraneous electricity includes:

- Stray ground currents from poorly insulated and improperly grounded electrical equipment,
- Lightning and static electricity from electrical storms
- Induced currents, present in alternating electro-magnetic fields, such as those commonly found near high-voltage transmission lines
- High radio frequency energy near transmitters
- Static electricity generated by wind-driven dust and snow storms, by moving conveyor belts, and by the pneumatic conveying of ANFO
- Galvanic currents generated by dissimilar metals touching or separated by a conductive material.

Shock wave initiation

Non-electrical blasting methods are preferred when standard electric detonators connected by wiring cause risks due to short circuits in wet conditions and when wires may damage or long lengths of vulnerable wiring are required. It is also preferred when there are special hazards from stray electric currents or static charges. The shock wave system offers the following obvious advantages:

- Safe from stray electrical currents and radio-frequency hazards
- Insensitive to concussion and heat, whether confined or unconfined
- Shock wave tube will not detonate any commercial explosives, including sensitive nitroglycerin dynamite
- Precise timing, equal to that obtained with electric firing
- Noiseless
- No “side initiation” effects on the explosive in the hole
- Waterproof under high hydraulic pressures

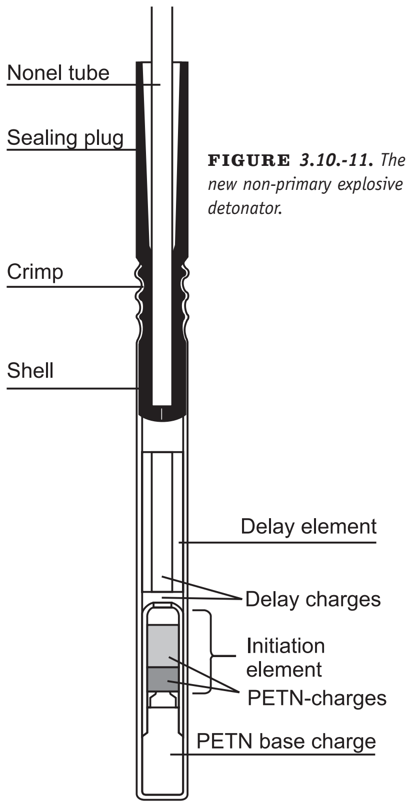


FIGURE 3.10.-11. *The new non-primary explosive detonator.*

- Ductile and not brittle, even in cold weather

The non-electric system employs plastic tube which is thinly coated internally with a reactive substance. (One example of non-electric system is NONEL-system.) The tube is 3 mm in diameter, weighs only 5.5 g/m with the explosive coating at 0.02 g/m. A shock wave introduced for example by a detonator at one end of the tube, is transmitted by the internal coating at around 2.000 m/s. This in turn initiates a delay element in a non-electric detonator connected to the far end of the tube. The speed of the transmitted signal results in a delay of approximately 0.5 ms/m of tube.

Special plastic transmitter connectors are used for joining trunk and down lines interface blasting. Special units with delays up to 200 ms can be employed to provide intervals on trunk lines, thus extending the timing range.

Connection piece

The connector, used for connecting non-electric fields, consists of a connector piece that connects at the top. The connector comes equipped with a delay element for nominal burning times of 0 to 176 ms.

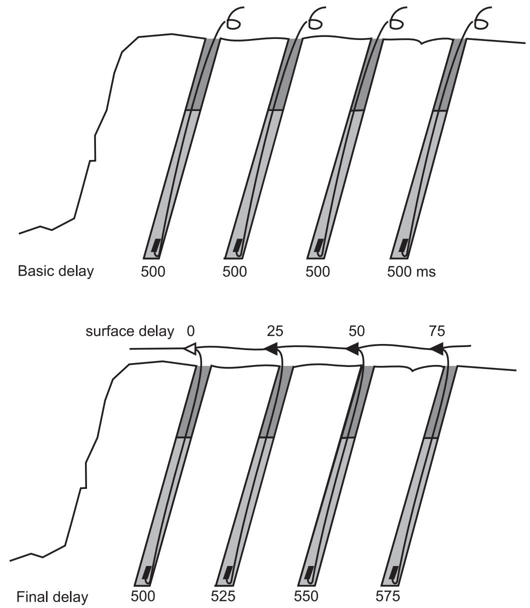


FIGURE 3.10.-12. *Principle of function of non-electric system in open pits.*

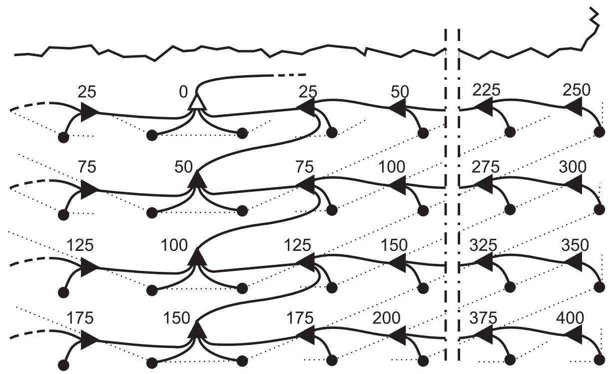


FIGURE 3.10.-13. *Bench blasting round connection.*

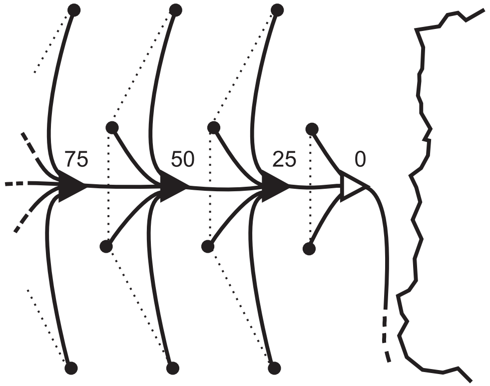


FIGURE *3.10.-14. Trench blasting round connection*

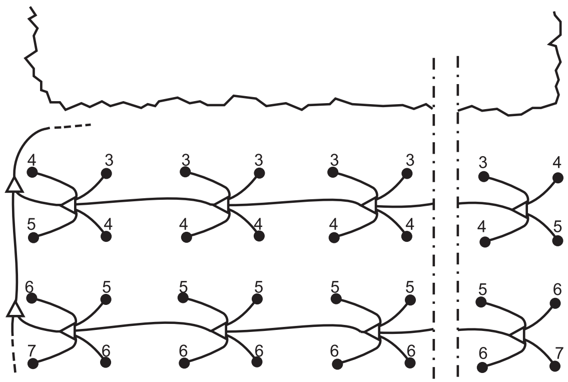


FIGURE *3.10.-15. Non-electric system connected to connecting piece in bench blasting.*

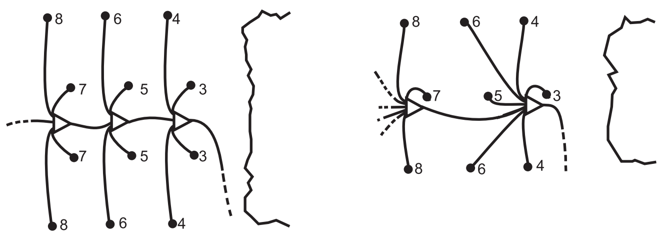


FIGURE *3.10.-16. Non-electric system connected to connecting piece in trench blasting.*

The following list describes three non-electric systems:

In an open pit, the system is built from caps that have a basic delay system and field connector pieces. (**FIGURES** *3.10.-12...3.10.14.*). Detonating cord and a connecting piece can also be used for the connecting field.

The second system is used in surface and underwater bench blasting, and can also be used for other types of blasting such as in mining.

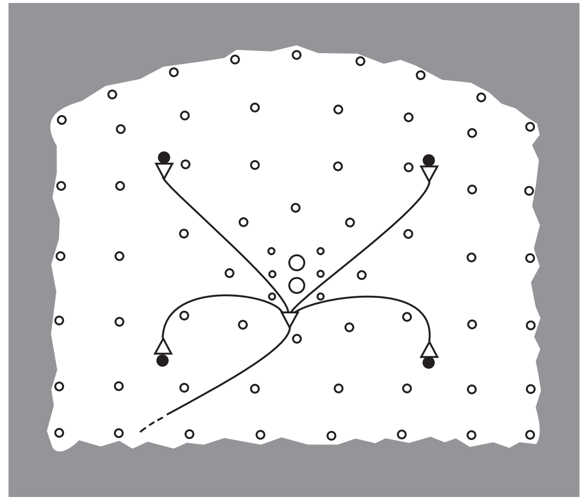


FIGURE *3.10.17. Connection of tunnel round with non-electric bunch connectors.*

The delay system is similar to the electric detonator; delay scaling is 25 ms and cap numbers 320 (75 500 ms) are available. The field is connected with the connecting piece or with a detonating cord (**FIGURES** *3.10.-15.* and *3.10.-16.*).

The third system is specially designed for tunneling. It is similar to the above mentioned systems. Delay scaling is 75–500 ms. The caps are numbered so that the number indicates the burning of the delay element time in hundred milliseconds (100 ms). The available series is numbered 060 (0–60 ms). Caps are usually connected by a bunch connector with up to 20 hoses (**FIGURE** *3.10.-17.*). Connecting can also be made with a detonating cord.

A non-electric round may be fired by safety fuse and plain detonator in locations where time precision is not necessary, such as, in underground blasting. In blasting where the moment of initiation must be precisely controlled, such as surface blasting and blasting with ground vibration control, electric firing may be used if the firing point is extended beyond the hazardous area. However, the safest and most reliable way of initiating a non-electric round is to use non-electric system blasting machines.

Electronic Detonators

Over the last few decades, the demand for precision detonators has increased due to a need for better blasting results and better vibration control. Current detonators have a relatively high delay time scatter that increases with expanding delay times.

An electric pyrotechnic delay detonator contains several elements used as the chain reaction of igniting the detonator and detonating the charge in the drillhole. Each element involves a time delay, which is not the same for all nominally equal detonators. For example, each element has a certain amount of scatter time.

Accurate timing would help to develop new blasting methods that could achieve better fragmentation control. It would also be possible to tailor the blast in such a way that would produce minimal ground vibration at a given distance. Massive blasts with multiple shot holes could also be fired since the system would allow a greater number of delay intervals.

Many explosives manufacturers are now developing their own electronic detonators and it appears likely that electronic detonators will be commercially available in the future.

Dyno Nobel's electronic detonator system has two closely interacting main components: the detonator and the blasting machine. Both are necessary for the system to function properly. An outline of a round using this system is shown in **FIGURE** *3.10.-18.*

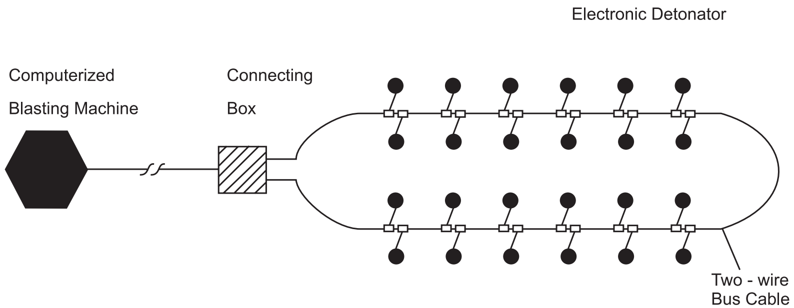


FIGURE ***3.10.-18.*** *Scheme for connecting electronic detonators.*

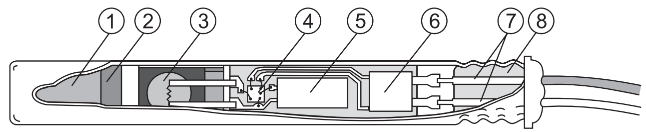
From the exterior, the electronic detonator looks exactly like a conventional electric detonator: it has the same dimensions and two wires. The detonator is marked with delay period numbers from 1 - 250.

The electronic detonator's interior design is shown in **FIGURE** *3.10.-19.* In principle, the detonator consists of an electronic delay unit used together with an instantaneous detonator.

An integrated circuit on a microchip (4) constitutes the heart of the detonator. In addition, the detonator cap also contains a capacitor (5) for energy storage, and separate safety circuits (6) on the input side (toward the leadin wires) in order to prevent electric overload. The chip itself also has internal safety circuits in the inputs. The fuse head (3) for initiating the primary charge (2) is specially designed to provide a short initiation time with a minimum of time scatter.

The most striking characteristic of the detonator is its flexibility. The period numbers, for example, do not state the delay time, but only the order in which the detonators will go off. Each detonator has its own time reference, but the final delay time is determined through interaction between the detonator and the blasting machine immediately before initiation.

Different electronic detonator systems can be selected based upon the customer's flexibility requirement. Demands for a high degree of safety against accidental initiation lead to similarities in the (various) designs produced by manufacturers. The electronic detonator consists of the following typical characteristics:



1 Base charge (PETN)	5 Capacitor
2 Primary explosive (Lead aside)	6 Over-voltage protection circuitry
3 Match head with bridge wire	7 Leadin Wires
4 Integrated circuit chip	8 Sealing plug

FIGURE ***3.10.-19.*** *Electronic detonator.*

- No initiation energy of its own
- Can not detonate without a unique activation code
- Receives initiation energy and activation code from blasting machine.
- Equipped with overvoltage protection. Low excess loads are dissipated via internal safety circuits. Higher voltages (>1000 V) are limited by means of a sparkgap. Large excess loads burn a fuse in the detonator which incapacitates it without making it detonate.
- Initiation system operates with low voltages (<50 V), which is a great advantage due to risk of current leakage from theleadin wires.

The blasting machine makes up the central unit of the initiation system. It supplies the detonators with energy and determines the allocated delay time for each period numbers. The unit is microcomputer-controlled and, therefore, its operation mode can be altered by various control programs, while it can be uniformly designed from a hardware standpoint. These factors attribute to the system's flexibility.

Detonator delay time allocation is carried out through uniquely coded signals to eliminate any possibility of error. Detonators react only to the signals from the dedicated blasting machine, which eliminates the risk of unintentional initiation by other energy sources. The blasting machine also performs automatic operation status control. The ready signal for firing is given only after the result of the check has been approved.

Typical system characteristics are:

- Shortest delay time between two adjacent period numbers (equal to shortest interval time) is 1 ms
- Longest delay time is 6.25 sec
- Detonator with lower period number can not be given a longer delay time than a detonator with a higher period number
- Detonators with different period numbers can not be closer to each other in delay time than the difference of their numbers. (For example, the interval time between No. 10 and No. 20 must be at least l0 ms)
- Maximum number of detonators connected to each blasting machine is approximately 500.

3.10.2. Rock Blasting Theory

GENERAL

The major objectives of blasting are fragmentation and rock displacement. The variables that affect blasting are:

- Rockmass properties
- Explosive properties
- Blast geometry, angles of the blastholes towards free face
- Initiation

Numerous geological factors affect blasting operations. Though they are out of the blaster´s control, he may set values on controllable variables so that the desired rock fragmentation and displacement can be safely achieved.

The selection of explosives must not be underestimated. However, the characteristics of the rock mass are more significant in controlling the breakage and vibrations than the characteristics of the explosives used.

In contrast to rock and explosive properties, blast geometry and initiation include a range of variables that can be controlled by the operator. Optimizing blasting performance requires a clear understanding of their significance.

After the round has been detonated, it is impossible to control. Therefore, the blaster must take extreme care in planning the blast. In order to successfully accomplish the task, he must use good judgement in matching the blasting requirements, methods and materials.

Rock is affected by a detonation in three principal stages. First, a shock wave released by the detonation passes through the rock mass at a (detonation) velocity of 3000 m/s - 6000 m/s, depending mainly on the rock geology. This velocity corresponds to 0.15 - 0.3 milliseconds per meter of burden. The rock is stressed by compression (**FIGURE** *3.10.-20.*). The shock wave does not break the rock, but crushes the blasthole walls and produces microscopic joints, which help to break and cut the rock in the second stage.

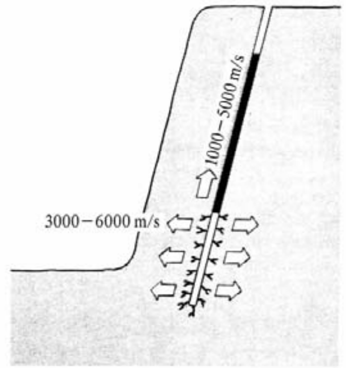


FIGURE ***3.10.-20.*** *Rock compression*

After reflecting from free faces, the shock waves expose the rock to tensile forces. Shock waves are reflected from bench faces or joints in the rock. Experiments have shown that the velocity of the shock wave after reflection is 500 m/s - 2000 m/s or 0.5 - 2.0 milliseconds per meter of burden. Tensile forces cause small primary, often radial, cracks that extend from the center of the hole (**FIGURE** *3.10.-21.*).

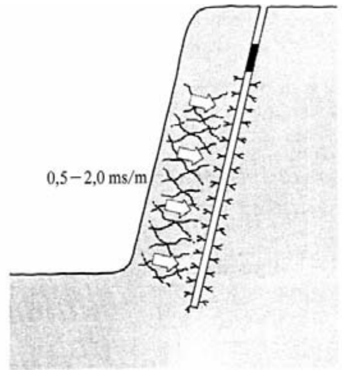


FIGURE ***3.10.-21.*** *Reflection of shock waves from free faces.*

Upon detonation, large quantities of high-pressure gases are formed. Through rapid heat release, the expanding gases spread into the primary cracks (**FIGURE** *3.10.-22.*). The cracks expand, the rock's free surface moves forward, pressure is unloaded and tension increases in the primary cracks. The primary cracks then expand to the surface which promotes the complete loosening of the rock. The burden is consequently torn off.

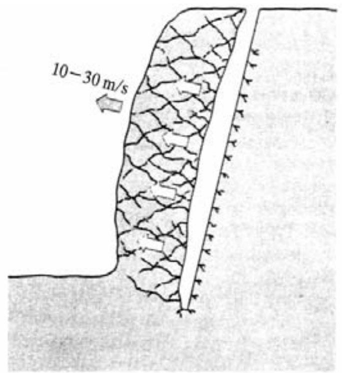


FIGURE ***3.10.-22.*** *Gas expansion.*

High-speed photographs taken during experiments show the relationship between the burden, specific charging and the speed of rock movement during blasting. Bench blasts are usually designed with the speed of 10 - 30 m/s, or in other words, 30 - 100 milli-seconds per meter of movement.

The following formula gives the movement speed of the torn-off burden as a function of rock hardness, burden distance and charging per meter of hole:

$V_0 = (K/V^{1.17})I^{0.39}$

where v_0 = speed of movement (m/s)
K = constant of rock hardness (soft rock = 15, hard rock = 33)
V = burden (m)
I = charging per meter (kg/m)

The explosive reaction in the blasthole is very fast and its power is considered completed when blasthole volume has expanded to 10 times its original volume which takes approx. 5 ms.

The graph in **FIGURE** *3.10.-23* shows how the expansion of the blasthole is related to time.

1. Shock wave initiation in rock crushing. The blasthole expands to double its original volume ($2V_0$). The blasthole remains at this volume for a relatively long time (0.1 to 0.4 ms) before radial cracks begin to open.

2. In addition to natural cracks, new cracks are formed mainly by interaction between the stress field around the blasthole and tensile stress formed by reflection of the outgoing shock wave at the free face. Reaction products expand from blasthole (volume now quadrupled) into the cracks. Fragmentation begins.

3. Gas expands further and accelerates the rock mass.

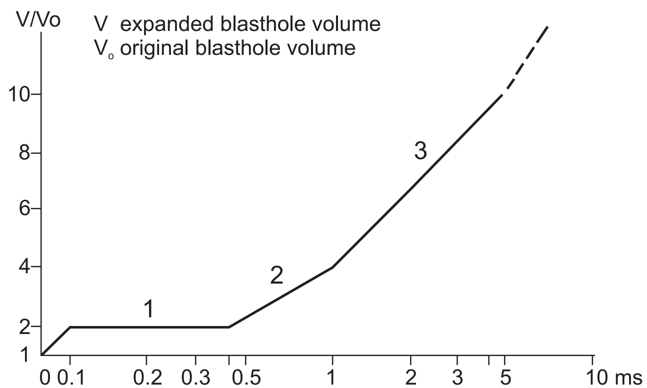


FIGURE *3.10.-23. Blasthole expansion in relation to time.*

3.10.3 Blasting and Environment

GENERAL

Three factors affect the environment in blasting (**FIGURE** *3.10.-24.*):

- Air shock waves
- Ground vibration
- Flyrock

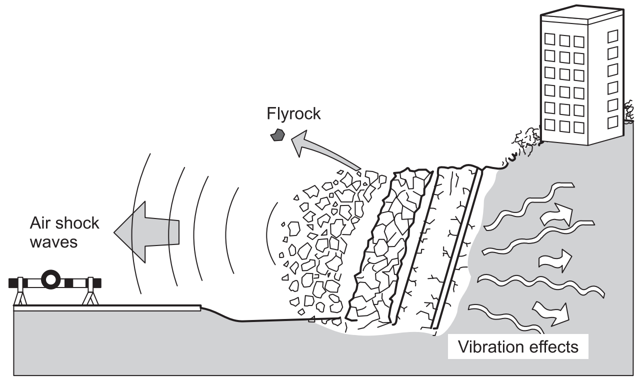


FIGURE *3.10.-24. Three factors that affect environment in blasting*

AIR SHOCK WAVES

Air shock waves are pressure waves that travel through the air. A compression wave in the air is usually produced by either the direct action of an unconfined explosive or by the indirect action of confining material subjected to explosive loading.

Air shock waves are very important because, in many cases, people near the blasting areas can not feel the vibrations, but often make complaints because the blast is so powerful that they imagine they can feel it.

Energy transmitted in acoustic waves is much the same as seismic energy. The air's elastic properties are a function of air pressure, temperature and humidity, depending on the altitude of the site, time of day and the prevailing winds at the time of the blast. Low humidity, light surface winds and high atmospheric pressure at the site can cause the air shock waves to feel more intense. Clear or partly cloudy days, warm temperatures and rapidly changing winds can disperse the acoustic waves.

Air shock waves are pressure waves which radiate in the air from a detonating charge.

Pressure intensity depends on charge size and 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 (**FIGURE** *3.10.-25.*). The maximum pressure is known as peak air over-pressure.

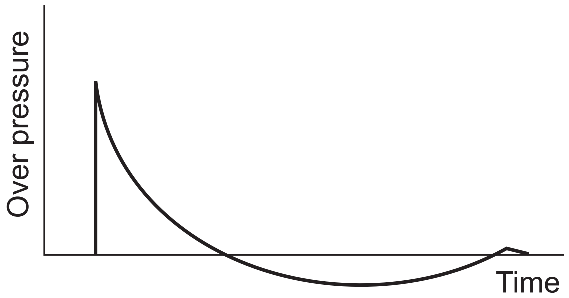


FIGURE ***3.10.-25.*** *Pressure/time curve for air shock wave*

Lower, inaudible frequencies are dampened more slowly than higher, audible frequencies and cause over-pressure over greater distances. Low frequencies can occasionally cause direct damage to structures, but usually induce higher frequency vibrations that are noticed in rattling windows, doors, crockery etc. Under such circumstances, without monitoring the blast, it is impossible to determine whether ground vibration or air shock wave is being experienced.

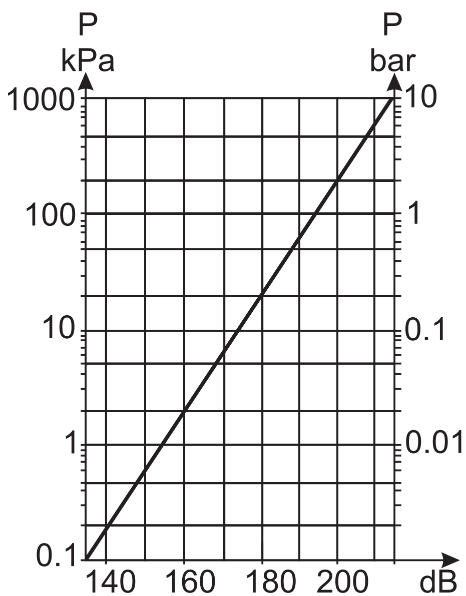


FIGURE ***3.10.-26.*** *Relation kPa/dB.*

Air over-pressure is measured as units of pressure. A pressure unit millibar (mbar) is usually used. Decibels (dB) and kilopascals (kPa) are also used as a measurement unit (**FIGURE** *3.10.-26.*). The decibel unit is expressed as:

$dB = 20 \log(P/Po)$

where P is the measured pressure and Po the reference pressure of 0.00002 Pa. Knowing the charge weight Q (kg) and the distance R (m) to the charge, over-pressure can be estimated accordingly:

$P = 700 \cdot Q^{1/3}/R$ (mbar); 1 bar = 100 kilopascal (kPa)

This applies to TNT, which means that for civil explosive-type dynamite, emulsions and ANFO, the charge weight should be reduced by 25% when used in the formula.

This ratio applies to unconfined charges. Concussion charges (mud-capping), detonating cord trunklines, welding power lines with explosives and pre-splitting with unstemmed holes etc are unconfined charges that cause problems in populated areas.

The blast effect on buildings and other structures depends on the amplitude and duration of the air pressure pulse as it hits the target and whether the exposed area of the building is facing the blast or it is parallel to the direction of the explosion.

Structures react differently to air blasts depending on the duration (frequency content) of the blast waves. For example, at intermediate distances, a glass window may withstand the peak pressure and positive momentum of wave but may fracture as a result of the negative momentum. The walls of a blast containment chamber must usually be designed to withstand the positive momentum (or the equilibrium static over-pressure), rather than the shock front pressure (positive over-pressure). Positive over-pressure is less important because it lasts only for a short time; there is not enough time for deformations under a positive over-pressure load to reach the breaking strain. As the structure continues to move with the force of the blast and maximum strain occurs, the risk of damage peaks.

Enclosing explosives in the drillhole is extremely important. Pressure increases if the uncharged section is small. Good stemming and round covering reduces pressure to a great extent. If the rock is rich in faults, pressure waves can penetrate locally through the rock which is blasted loose in stages. This happens when a particular round causes a powerful air shock wave that breaks windows. In order to calculate pressure when rock blasting, the enclosure factor must be followed.

In normal bench and pipeline trench blasting, the measurement shows that this factor is approximately 150 or more, which means that for conventional explosives, the charge values in the above-mentioned ratio should be reduced 150 times. This figure also includes reduction for normal dynamite-type explosives. The ratio for normal rock blasting is:

$P = 700 (Q/150)1/3/R$ mbar; 1 bar = 100 kilopascal (kPa)

GROUND VIBRATIONS

Seismic waves

The most common type of blasting damage is caused by ground vibrations. When an explosive detonates in a blasthole, it generates intense stress wave motion in the rock.

Three types of seismic waves are normally observed in engineering and blasting seismology: compression (P), shear (S) and Rayleigh surface waves. The first two are body waves which propagate along the surface or into the sub-surface, returning to the surface by reflection or refraction. The third propagates only along the surface and dies out rapidly the deeper it goes.

Each type propagates with a different velocity. Substitution of parameter values show that the P wave travels at the highest velocity. The S wave travels at roughly one-half of the P wave velocity.

Damage prevention criteria

When foundations are exposed to vibrations, damage can be caused by elongation, shearing and bending. The connection between these magnitudes and those of the ground vibrations is first determined in regard to the wall's elastic properties. Allowance must also be made for the fact that local irregularities and the possible static state of the tension can lower the damage limit.

Structure damage and human tolerance to the blasting vibrations have been analyzed through data collected in various studies. An international summary indicates that particle velocity is the best and most practical description for defining in potential structure damage.

As ground vibrations are approximately a sine-formed vibration, particle velocity can be estimated by the following formula (**FIGURE** *3.10.-27.*):

$$v = 2\pi fA$$

where v = Particle velocity (mm/sec)
f = Frequency (periods/sec)
A = Displacement in mm

Vibration acceleration can be calculated from the formula:

$$a = 4\pi^2 f^2 A$$

where a = Acceleration in g (9.81 m/sec^2)
A = Displacement in mm

In order to recommend realistic limiting values for buildings, blasting and vibration measurement experience is necessary. Restrictions, such as low-particle velocity limiting values can

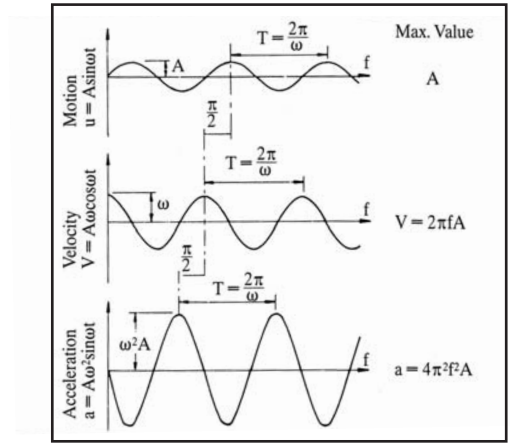


FIGURE *3.10.-27.*
Basis terms determining wave motion.

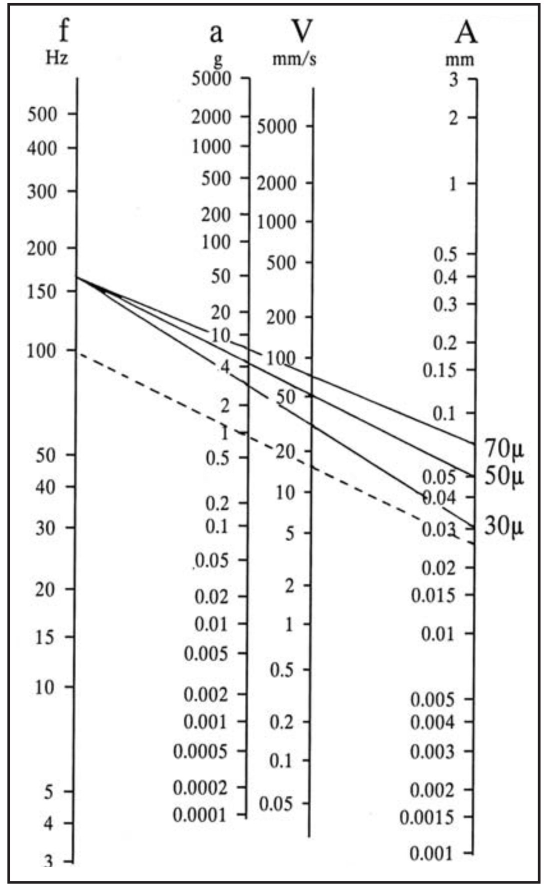


FIGURE *3.10.-28.*
Nomogram for determining the relation between frequency (f), acceleration (a), oscillation velocity (v) and amplitude (A).

considerably increase excavation costs. Therefore, before starting and during early planning stages of a blasting project, it is important to execute visual inspection and risk analysis. Based upon these results, it is possible to judge the sensitivity of the buildings and foundations to vibrations. Many factors influence the permitted vibration values. Some of the most important are:

- Vibration resistance of the building materials
- General condition of the building
- Duration and character of the vibrations
- Presence of sensitive vibration equipment
- Foundations
- Condition of foundations
- Propagation characteristics of wave in rock, earth and building material, respectively
- Replacement costs / highest likely repair cost

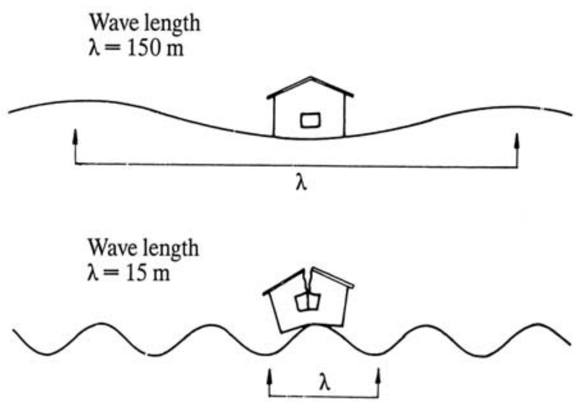


FIGURE ***3.10.-29.*** *The effect of wave length on damage risk.*

A high-velocity seismic wave front produces a longer wave length than a wave with the same frequency but traveling at lower velocity, as shown in **FIGURE** ***3.10.-29.*** If the wave length is short, the risk of damage caused by tearing is considerably larger as the end of the building is subjected to a different type of movement. Surface waves produce a similar effect.

The specific frequency of ground and rock is mainly dependent on its homogeneity, solidity and the distance between the blast and building.

As higher seismic velocities have longer wave lengths, vibration v (mm/s) in rocks with lower velocities c (m/s) (thus shorter wave lengths) may increase the risk of damage.
If buildings are built on material softer than rock, the empirical values must be divided approximately by 2 or 3 (***Table 3.10.-3b.***).

As the dominating frequency is smaller when the distance is greater, the permissible value for the peak particle velocity must be low when the distance is great.

Finnish limit values for peak particle velocity (v) are very similar to the Swedish and Norwegian standards and can be calculated accordingly:

$$v = F_k \cdot v_1$$

F_k = structural coefficient.
v_1 = peak particle velocity as a function of the distance (R) for structures and buildings that have been founded on different materials/vertical component.

Table 3.10.-3a. shows a categorization of seven classes for several structures and buildings. The structural coefficients for curing concrete are also given in ***Table 3.10.-3a.*** and are based on past experience as well as the test series. The F_k values given for electrical cables and pipelines, as well as for rock masses, are quite certain based on experience.

Table 3.10.-3b. shows the values for peak particle velocity (v_1) as a function of the distance (R) for structures and buildings that have been founded on different materials, based on the fact that the dominating frequency is lower when the distance is greater. The recommended peak particle velocices, while the low frequencies are more dangerous for structures than high frequencies.

Table 3.10.-3a. *Structural coefficients F_k ($v = F_k \bullet v_1$, where $v_1 = Fd\ (v_0)$)*

Structural categories (structures in good condition)	***Structural coefficient F_k***
1. Heavy structures like bridges, piers etc.	2.00 *
2. Concrete and steel building, rock caverns with shotcrete reinforcement	1.50 *
3. Office and commercial buildings of brick and concrete. Wood-frame houses on concrete or stone foundation	1.20 *
4. Brick and concrete residential buildings with no light concrete, limestone-sand brick etc. Rock caverns with no shotcrete reinforcement. Curing concrete > 7 days old*. Electrical cables etc.	1.00
5. Building with light concrete structures. Curing concrete 3-7 days old*.	0.75
6. Very vibration sensitive buildings, such as museums, churches, and other buildings with high vaults and great spans. Buildings of limestone-sand bricks. Curing concrete < 3 days old*. *v	0.65
7. Old historical buildings at the point of collapse such as ruins.	0.50

* Values over one (1) are permitted only when a blasting or vibration specialist is present.

Table 3.10.-3b. *Permitted peak particle velocity (vertical component) v_1 ($v_1 = F_d \bullet v_0$) as a function of distance (R) for structures and buildings founded on different materials. (The structural coefficient is $F_k = 1$).*

Distance R (m)	***Soft moraine Sand Shingle Clay***	***Moraine Slate Soft limestone Soft sandstone***	***Granite Gneiss Quartzic Hard sandstone Hard limestone Diabase***
		Wave velocities c	
	1000-1500 m/s	2000-3000 m/s	4500-6000 m/s
	Permitted peak particle velocity v_1 (mm/s)		
1	18	35	140
5	18	35	85
10	18	35	70
20	15	28	55
30	14	25	45
50	12	21	38
100	10	17	28
200	9	14	22
500	7	11	15
1000	6	9	12
2000	5	7	9

When there is a great distance between the blasting locations and the structure (approx. over 50 m-70 m) the limit v_1 values are conservative. In certain cases, more economical blasting can be achived by measuring all three components plus the time history of the vibration to carry out the frequency analysis. This allows for the use of higher limit values.

If the structure is not a residential building with a structural coefficient of 1, the permissible limit value for peak particle velocity should be calculated with Eq.

$$v = F_k \times v_2$$

F_k = structural coefficient
v_2 = particle velocity versus frequency,
F_d (v_0) (**FIGURE** *3.10.-30*)

The United States Bureau of Mines safe levels of ground vibration from blasting range from 12.5 mm/s to 50 mm/s peak particle velocity for residential type structures. Damage thres old values are a function of the vibration frequencies transmitted into the residences and the types of construction. Particularly serious cases are considered to be low frequency vibrations that exist in soft foundation materials and/or result from long blast-to-residence distances. These vibrations produce not only structural resonances (4 to 13 Hz for whole structures and 10 to 25 Hz for mid walls), but also excessive levels of displacement and strain (**FIGURE** *3.10.-31.*).

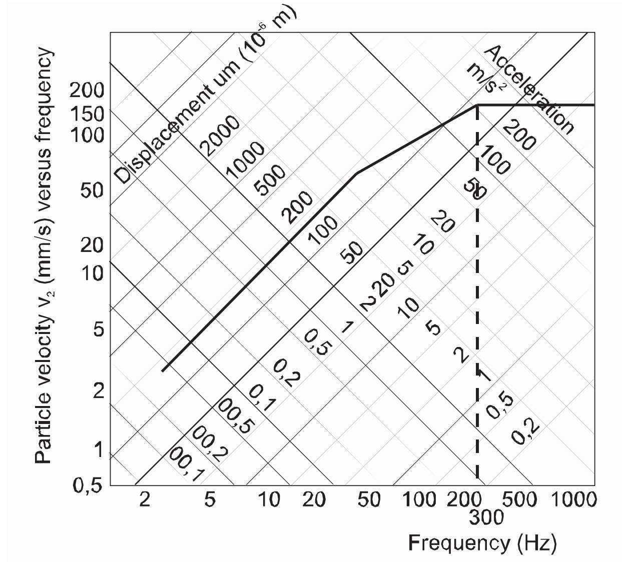


FIGURE ***3.10.-30.*** *Particle velocity v2 (mm/s) versus f (Hz) (Finnish proposal)*

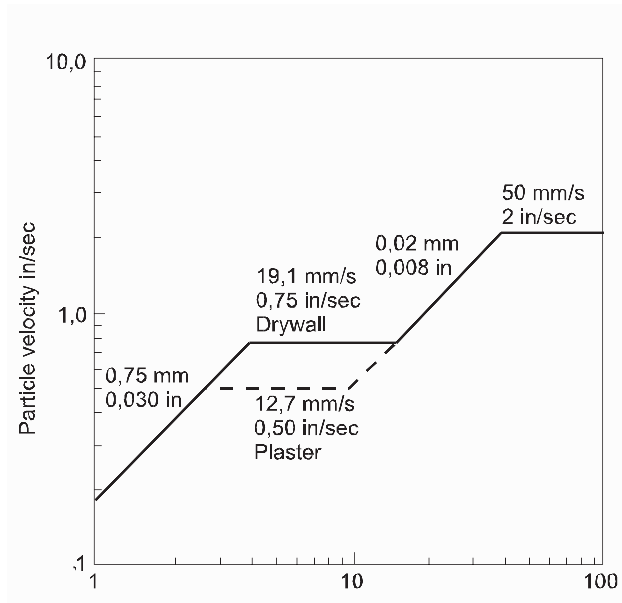


FIGURE ***3.10.-31.*** *Safe levels of blasting vibration for houses using a combination of velocity and displacement (Bureau of Mines, USA 1980)*

According to German standards (**FIGURE** *3.10.-32.* and *table 3.10.-4.*) the criteria is listed for three classes of structures and triaxial measurements for frequencies under 10 Hz, from 10 to 50 Hz and over 100 Hz. Measurements should be taken in the foundation and on the roof of the building.

Table 3.10.-4. *Criteria for particle velocity v based to effects of short term vibrations.*

Line	*Type of structure*	*Criteria for particle velocity v in mm/s*			
		Base			Top of the highest round
		Frequenses			
		< 10 Hz	10 to 50 Hz	50 to 100*) Hz	all frequences
1	Commercial and industrial structures	20	20 to 40	40 to 50	40
2	Residences (also under construction)	5	5 to 15	15 to 20	15
3	Structures which are especially vibration sensitive (for example historical buildings)	3	3 to 8	8 to 10	8

*) For frequenses over 100 Hz should at least criteria for 100 Hz be used

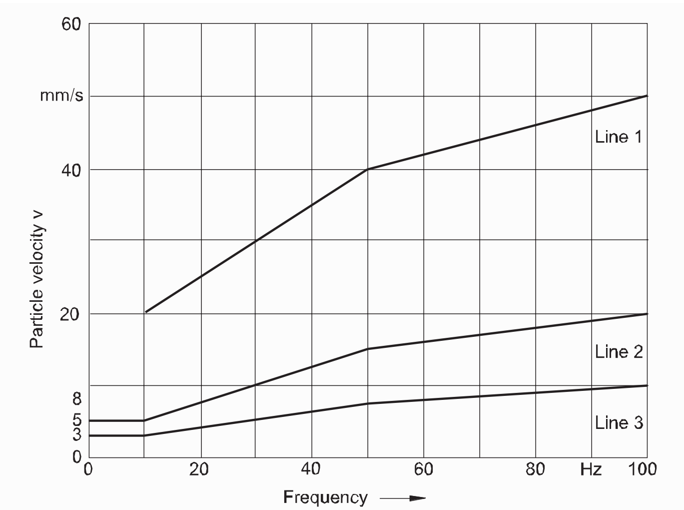


FIGURE *3.10.-32. Vibration in structures, DIN 4150, part 3/11/1986*

In the past, the same low damage threshold values, such as 0.25 g and 0.1 g, have been used for vibration sensitive equipment such as computers. These very stringent restrictions can sometimes create major problems for the contractor.

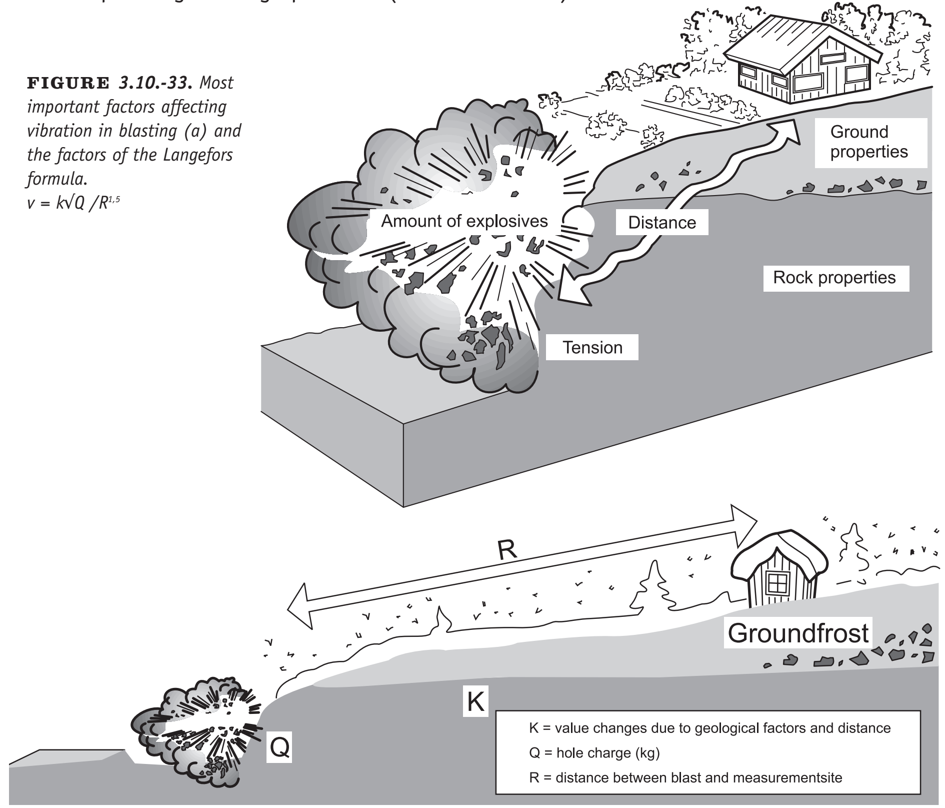
By using shock isolators it is possible to reduce shock severity by storing shock energy within the isolators, and also through the subsequent release of the energy over a longer period of time.

One method fore reducing blasting vibrations is to drill a slot in the rock, between the vibration source and the building. If the slot is drilled close to and along the foundation, the house is protected not only against vibration but also against back break and rock heaving below the foundation. By performing slot drilling in front of a vibration-sensitive building or plant, an almost 80 % reduction in vibration levels can been achieved.

Practical simple formula.

The ratio between charge/distance and ground vibration can be used to make a simple formula for planning blasting operations (**FIGURE** *3.10.-33.*).

FIGURE *3.10.-33. Most important factors affecting vibration in blasting (a) and the factors of the Langefors formula.*
$v = k\sqrt{Q} / R^{1.5}$



Vibration velocity:

$$v = K \sqrt{Q}/R^{1,5}$$

where Q = instantaneously detonating charge (kg)
R = distance (m)
v = vibration (particle) velocity (mm/s)
K = transmission factor, constant depending on the homogeneity of the rock and the presence of faults and cracks. For hard granite, this can be approx. 400 but it is normally lower. And is lower as the distance is greater.

Practical methods to reduce ground vibrations by limiting the cooperating charging weight per interval are:

- Adapt ignition pattern so that the charging level is spread over more intervals and scattering in the delay elements of the detonators is utilized
- Reduce hole number and diameter
- Use decked charges by dividing the necessary drillhole chargeevel into more ignition intervals through sand plugs
- Use decoupled charges; charge diameter smaller than hole diameter
- Divide bench into more benches. Do not blast to final depth at once.

Upon detonation, there should be as little confinement as possible. This is obtained by:

- Carefully adapting the ignition pattern, so that all holes break the burden in the easiest way
- Increasing hole inclination (drillhole)
- Avoiding too large burdens and choke blasting

For blasting at distances shorter than 50 - 100 m, the risk of interaction between various intervals is small. Cooperation risk between intervals increases with large blasts in quarries, where, for example, vibration- sensitive structures are not in the vicinity. Vibration size is then influenced by:

- Charging level
- Interval times
- Resonance frequency of the ground (which depends on the distance, depth and the character of the ground)
- Local geology

Costs of reducing vibration levels

The costs of careful blasting near residential areas increase very rapidly with decreasing permissible vibration levels. The cost increase depends primarily on the following factors:

- Drilling - smaller or greater number of drillholes.
- Charging - more detonators and higher labor costs
- Blasting - more rounds and longer standup time special charges

Planning and control work costs also increase in:

- Blasting
- Visual inspections
- Vibration measurement
- Blasting record
- Insurance administration

In addition to the above-mentioned costs, a number of problems also exist that are hard to estimate in terms of money.

Building inspections are performed to document the condition of a structure before and after blasting. Most structures have cracks which the building's occupants are usually not aware of.

To optimize blasting work, risk analysis is required to determine the acceptable vibration size and what size charge can be blasted at a certain distance without exceeding the vibration limit.

Risk analysis should be made through a careful examination of factors that affect blasting operations prior to blasting. The more information is available, the more accurate the result is. The decision should be based on as many points as possible in the list below.

Vibration measurements

Vibration measurements are performed to monitor and control the effect of vibrations on buildings, installations in buildings and to what degree the disturbance affects people. The underlying principle is to measure the vibrations where they are first transmitted to the object of concern. However, in the USA., gauges are usually attached to the ground outside the foundation, as home owners seldom allow gauges to be mounted on or inside the building.

THROW AND FLYROCK

In bench blasting, there are two types of rock movement:

- Forward movement of the entire rock mass which is mainly horizontal
- Flyrock, which is scatter from the rock surface and front of the blast

A specific charge of 0.2 kg/cu.m. does not cause forward movement of the rock; it only breaks the rock.

A normal specific charge of 0.4 kg/cu.m. moves the rock forward 20 - 30 m, which is the expected normal displacement of the rock mass (**FIGURE** *3.10.-34.*).
Insufficient 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 hazards in blasting but may cause inconvenience when miscalculated. Flyrocks are rocks that are violently ejected from the blast and travel long distances and are the main cause of onsite fatalities and damage to equipment. Flyrock is mostly caused by an improperly designed or improperly charged blast. The maximum ejection of flyrock is a function of the blasthole diameter.

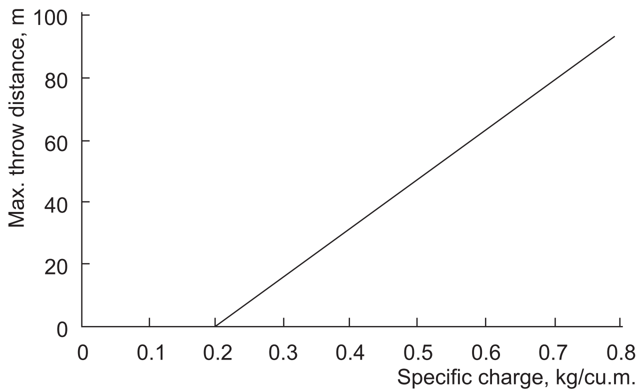


FIGURE *3.10.-34. Maximum forward movement as a specific charge function.*

$L = 260\ d^{2/3}$

which is valid for a specific charge. **FIGURE** *3.10.-35.* shows the ejection distances of flyrock as a function of the specific charge at blasthole diameters of 25 - 100 mm.
A burden of less than 30 times the diameter of the blasthole gives an excess charge, especially if the explosive is poured or pumped into the blasthole. The excessive explosive content in the blasthole may result in rocks that fly long distances.

Excess burden may cause flyrock if the explosive can not break the burden, and gases vent through the collar of the hole creating craters. An inappropriate firing pattern may cause the same effect. Blast gases tend to vent through the collar if the blasthole does not have free breakage. Deficient or excessive delay time between blastholes can also cause flyrock. Deficient delay time creates the effect shown in **FIGURE** *3.10.-36.* The traveling distance is relatively limited. A more serious hazard appears when the delay time between blastholes is too long. In a correctly designed firing pattern, the rock is held together. Rock from the front row then acts as protection when the charges in the following rows detonate. If the delay between rows or single blastholes is too long, the protective effect is not achieved.

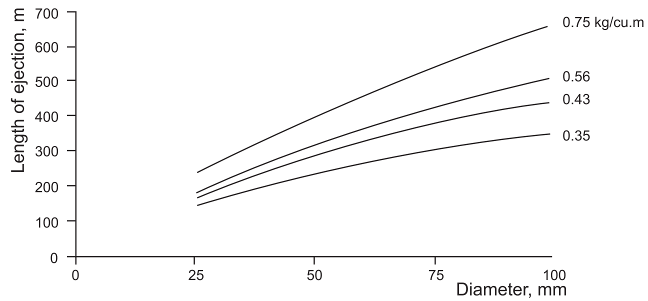


FIGURE *3.10.-35. Maximum traveling distance of flyrock as a function of blasthole diameter for different specific charges (Swedish Detonic Research*

Delay times between adjacent blastholes must not exceed 100 ms if the burden is less than 2 m; 50 ms if the burden is less than 1 m.

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

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

The flyrock is more frequent when the blasthole is top-initiated than when it is bottom-initiated (see **FIGURE** *3.10.-36.*). Detonating cord with a 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. Stemming should have a particle size of 4 - 9 mm for best confinement. The best material for stemming is crusher run.

Flyrock is often caused by incompetent rock which lets the gases easily escape due to less resistance than in the more competent parts of the rock. Necessary care must be taken particularly during the charging of the first row of the blast. Incompetent zones may be natural but they may also be caused by the previous blast, especially in the heavily charged bottom part of the hole. It is important to make sure that stemming equals or is slightly greater than the burden. Sometimes flyrock is caused by inclined holes which have been drilled too flat, resulting in too small burden of the face in the toe vicinity.

Proper planning prevents flyrock. Minimizing flyrock depends on the following:

- Careful spotting and aligning of drillholes and hole straightness
- Proper firing circuits
- Keeping dangerous areas clear of unauthorized personnel
- Optimal time to blasting

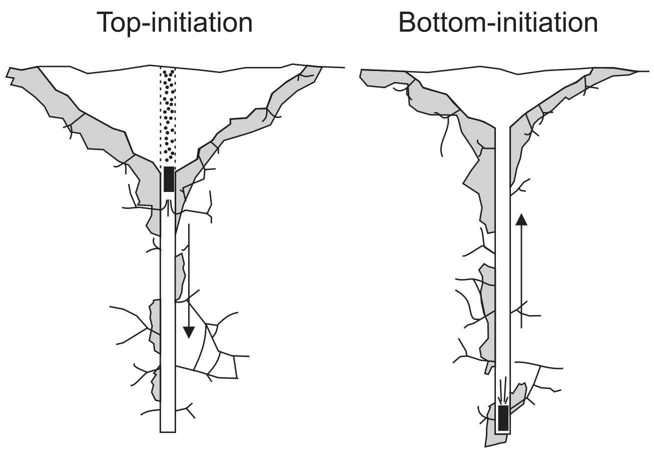


FIGURE ***3.10.-36.*** *Crater effect, vertical holes without free breakage.*

Safe blasting operations in residential areas require:

- Hole diameters less than 51 mm and cartridge diameters less than 40 mm
- Selection of burden and spacing so that specific charging does not exceed 0.3 - 0.5 kg/m^3
- For secondary blasting, the use of a suitable amount of charging - approx. 0.05 kg/m^3. Drillhole length should be 2/3 of boulder height, with the hole drilled through the boulder's center of gravity
- Charging should always be related to the burden; the blaster should always consider the effect of possible back break
- Holes should not be drilled into joints or shear zones; it is advantageous to use these zones as excavation boundaries
- Firing should be planned so that delays between adjacent holes do not exceed 50 ms. (Burden < 1m)

However, if these measures are not sufficient, the blaster must not resort to controlling flyrock by covering the field with heavy blasting mats and backfill. These are generally only used in construction blasting.

Backfilling is a method that is used to cover a blast area with soil with no stones, preferably sand, to control or prevent undesirable flyrock.

A good rule of thumb for backfilling is that the backfill is equal in depth to the stemming; however, there must be a minimum of approximately one meter of backfill. When backfilling, the blaster must exercise extreme caution not to break the cap leg wires. Backfilling is worthwhile in that it requires less equipment than blasting mats, produces better breakage and the whole blast can be shot at once. On the other hand, using backfill for blast coverage

generally requires more explosives, because backfill contains rock movement.

Blasting mats are made of netting or matting of either cable or tires. Covers efficiently protect the site surroundings from flyrock. Covers should be strong and heavy, and contain all bursting blocks within the blast. The following rules should be followed when covering blasts:

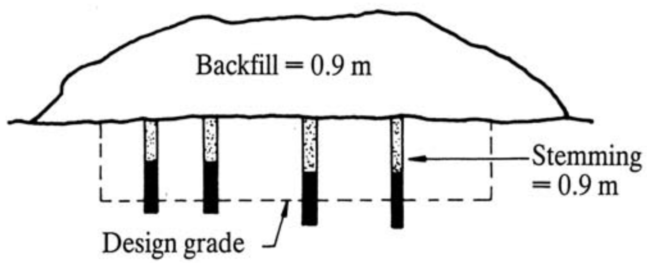


FIGURE ***3.10.-37.*** *Using backfill.*

- Blast should be covered with both rubber mats and industrialfelts; these covers should exceed the blast boundaries by at least 2 - 3 meters (6 -10 ft). If the hole is loaded close to the collar (i.e. stem decreased), at least two covers should be used,
- Rock from previous blasts should be used to cover the face of the blast; this decreases the risk of flyrock from a deficientburden in the bottom caused by hole deviation.

The main rule for blasting mats is never to blast more holes than can be safely covered with the mats. As in backfilling, great care should be taken not to break leg wires when placing the mats.

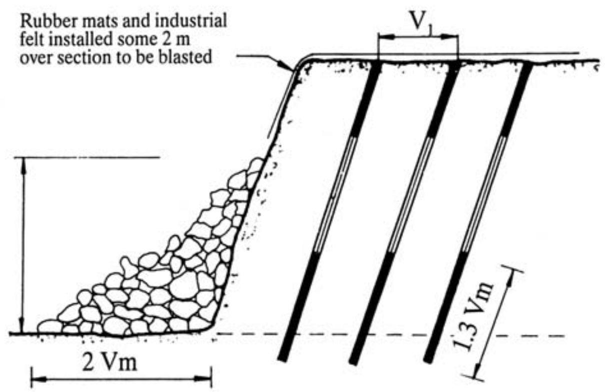


FIGURE ***3.10.-38.*** *Using blasting mats and backfill to prevent flyrock.*

4.1. General

Productivity, cost efficiency and end-product quality are key issues when solid rock is used to produce crushed rock aggregate needed for applications in the construction industry such as cement and concrete, road, railway, dam construction etc.
New environmental regulations are becoming increasingly stringent for safety, noise, dust and landscaping.

The following activities are part of quarry operations:

Laser profiling of high walls for efficient front row design.
Correct burden design provides the right amount of explosives
to be loaded into each hole to prevent flyrock.
Hole drilling for explosive and presplitting placement.
Hole surveys for hole position verification as compared to drill pattern design.
Blasting - shotrock fragmentation and throw
Blast monitoring for recording air and ground vibrations
Secondary breaking for downsizing boulders and oversized
rocks to allow free material flow through the primary crusher to minimize blockage
Loading of muckpile by wheel loaders and excavators
Hauling - shotrock transportation to crushing/stockpile by dump truck or conveyor belts.
Crushing for mechanical size reduction of feed material.
Screening for material sizing.
Final product to stockpile.

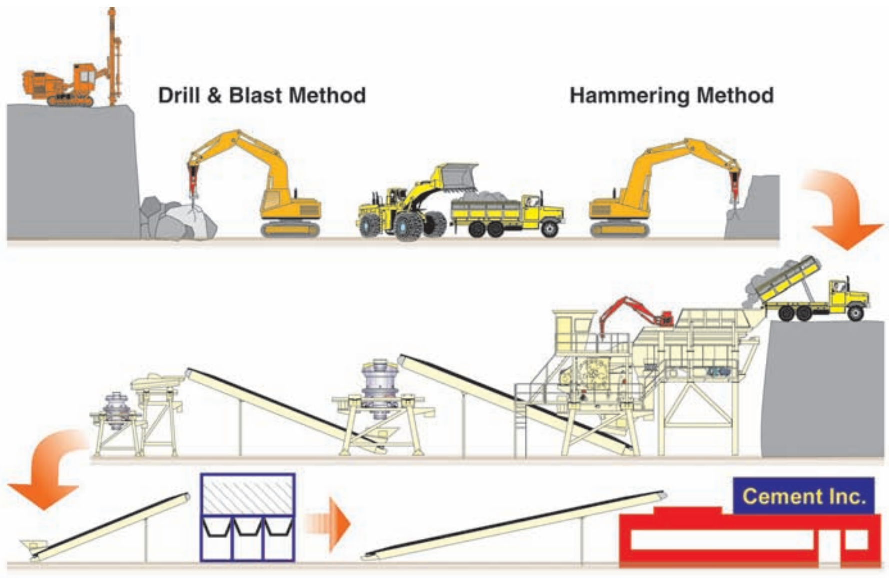


FIGURE 4.1.-1. *Quarry process.*

Shotrock fragmentation affects throughput time and cost in secondary breaking, loading, hauling, crushing and screening. The required mean/max. fragment size depends on the primary crusher opening.

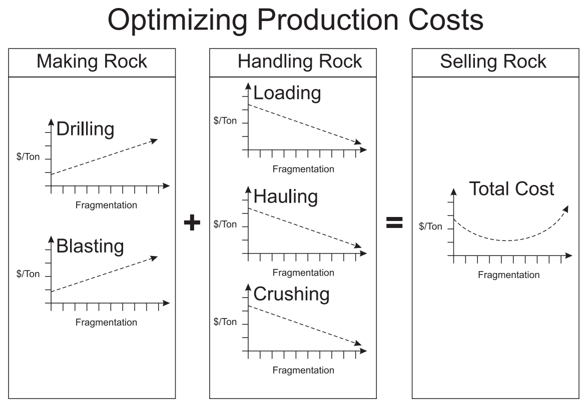
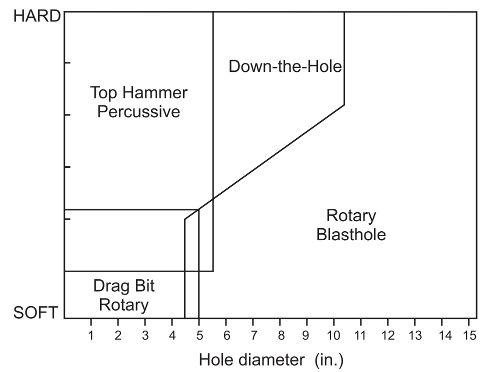


FIGURE 4.1.-2. *Optimizing production costs.*

For a given rock mass, the fragmentation degree depends on the explosive type and quantity used to blast each cubic meter of solid rock.
This, in turn, affects the amount of drilling required to achieve the degree of fragmentation since the drilling pattern, burden and spacing affect the mean shotrock fragment size.
Accurate drilling decreases the amount of oversize. The drill pattern area is also increased which, in turn, affects explosives consumption.
For a given mean fragment size, the drill pattern area increases together with the drill hole diameter. However, the max. fragment size increases disproportionately to the drill hole diameter. The required fragmentation degree depends on the hole diameter and can determine the drilling method.

In situations where explosives are difficult or impossible to use, or when the rock is ***highly fractured*** and/or of low strength, using hydraulic hammers is a viable method for primary rock breaking.

FIGURE 4.1.-3. *Drilling method selection as a function of rock hardness and hole diameter.*



4.2 METHODS

4.2.1. Drilling and blasting

BASIC DESIGN FACTORS

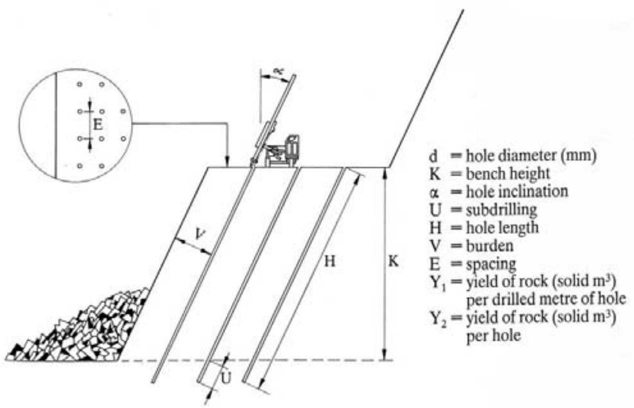


FIGURE *4.2.-1. Terminology used in drilling patterns.*

The most important terms used in bench drilling operations are shown in **FIGURE** *4.2.-1.*
In addition to rock properties, bench is influenced by:

- Hole diameter
- Bench height
- Fragmentation
- Bench stability requirements
- Terrain conditions- Environmental restrictions

Hole diameter

Selecting the drillhole diameter depends largely on the desired production rate. The bigger the hole diameter, the higher the production rates are when drilling with the same equipment. Factors restricting hole diameter are: (1) required rock fragmentation size, (2) need for low charge per hole due to danger of ground vibrations and (3) need for selective rock excavation. Rock fragmentation size tends to increase when the hole length (H) - hole diameter (d) ratio decreases below H/d = 60.

Bench height must be considered when determining the drilling equipment and hole diameter. Generally, low benches require small holes, and larger holes can be used in higher benches (**FIGURE** *4.2.-2.*).

Generally, larger hole sizes give coarser fragmentation although this can be reduced charging

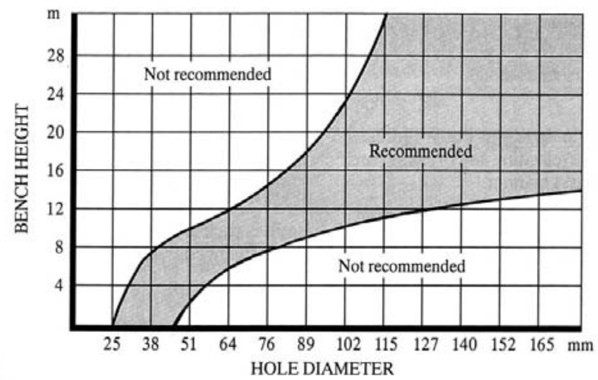


FIGURE *4.2.-2. Determining drillhole diameter for various bench heights.*

by heavier explosives. Greater specific charging can, however, result in greater rock throw. In well-fractured softer rock, smaller hole sizes and less explosives combined with denser drilling usually result in finer fragmentation.
Environmental restrictions influence working in urban areas: buildings, structures and sensitive equipment often restrict ground vibrations within specific limits. It may be necessary to limit the charge per hole, which leads to the use of smaller hole diameters. This causes an increase in the amount of drilling per cubic meter of rock (specific drilling), which in turn requires high-capacity equipment specially designed for small-hole drilling.
Certain provisional rules have been given for quittance in hole size selection. Hole diameter is closely related to bench height (**FIGURE** *4.2.-2.*) and burden, and should be between 0.5 - 1% face height:

$$d = 5...10K.$$

where d = Drillhole diameter (mm)
K = Bench height (m)

or

$$D = 0.06...0.12K$$

where D = Drillhole diameter (in)
K = Bench height (ft)

Smaller hole diameters and, therefore, smaller burdens give better fragmentation and less ground vibration and leads to lighter drilling equipment and smaller rounds. The hole diameter can be chosen to suit the loading equipment by using **FIGURE** *4.2.-3.*

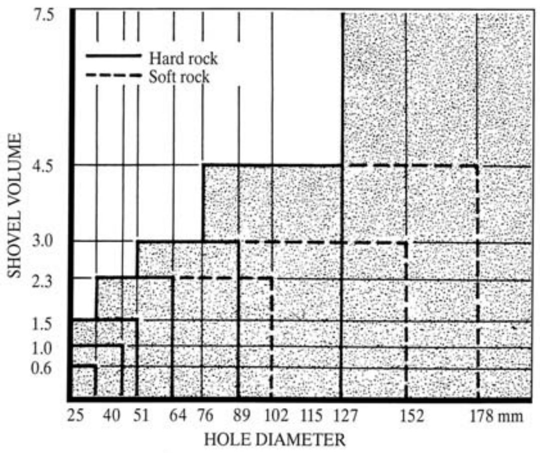


FIGURE *4.2.-3. Shovel volume vs. drillhole diameter.*

In rock strata which have open, widely spaced discontinuities (where fewer larger diameter drillholes intersect a smaller percentage of blocks), the surface of each joint reflects the strain-wave generated by the explosion. This provides better fragmentation between the hole and joints, but tends to produce boulders beyond the joint (**FIGURE** *4.2.-4.*). Therefore, blocks which do not have holes in them tend to be poorly fragmented, increasing the cost of

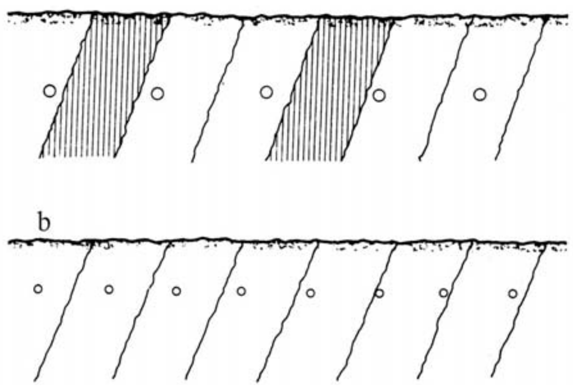


FIGURE *4.2.-4. The effect of joints on fragmentation when using large diameter (a) and small diameter (b) holes. The shadowed area shows insufficient fragmentation.*

secondary blasting and loading and crushing. Crushing tends to exceed any savings made by drilling larger diameter holes. However, if the hole diameter and drilling pattern are tool small, extra drilling costs tend to outweigh any cost reductions achieved through better frag-mentation.

The optimum drilling and blasting method, therefore, lies between these two extremes - drilling and blasting costs and fragmentation, providing minimum total production costs. Quarry and pit operators often have conflicting views about bench height; while some favor high benches up to 30 meters, others strongly feel that heights should be restricted to approximately 15 meters. Studies and experiment to investigate the effects of switching from a 30-meter face to two 15-meter faces revealed the following points (with reference to the two lower faces):

Advantages

- Greater drillhole accuracy
- Maximized burdens and spacings
- Greater penetration rates over the hole
- Greater selectivity of rock excavation

Disadvantages

- More benches to construct and maintain
- More drill downtime while shifting machine
- Subdrilling doubled
- More boulders; most big boulders come from the top of the bench, therefore, two-meter faces produce more big boulders.

Charge calculations

Bench blasting (**FIGURE** *4.2.-5.*) is the most common blasting work.

Bench height K ≥ 2 x V_{max}.

d = Diameter of blasthole in the bottom (mm)
K = Bench height (m)
V_{max} = Maximum burden (m)
U = Subdrilling (m)
H = Hole depth (m)
E = Error in drilling (m)
V = Practical burden (m)

E = Practical spacing (m)
b = Specific drilling (m/cu.m.)
I_b = Concentration of bottom charge (kg/m)
h_b = Height of bottom charge (m)
I_c = Concentration of column charge (kg/m)
h_c = Height of column charge (m)
Q_c = Weight of column charge (kg)
Q_{tot} = Total charge weight per hole (kg)
q = Specific charge (kg/cu.m.)

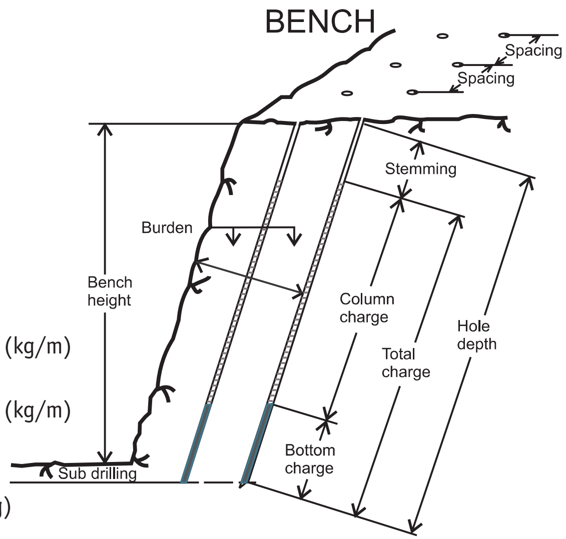


FIGURE *4.2.-5. Bench blasting*

It is defined as the blasting of vertical or close to vertical blastholes in one or several rows towards a free surface. Blastholes can have free breakage or fixed bottom (**FIGURE** *4.2.-6.*). Most blasting methods can be considered as bench blasting. Trench blasting for pipelines is also a type of bench blasting, but because the rock is more constricted, it requires a higher specific charge and closer spaced drilling. Stoping towards the cut is a type of bench blasting after the cut has been blasted in tunneling. Rock properties vary widely. Its tensile, compressive and shear strengths vary in different kinds of rock and can even vary within the same blast. As the rock´s tensile strength must be exceeded in order to break the rock, its geological properties affect its blasting potential.

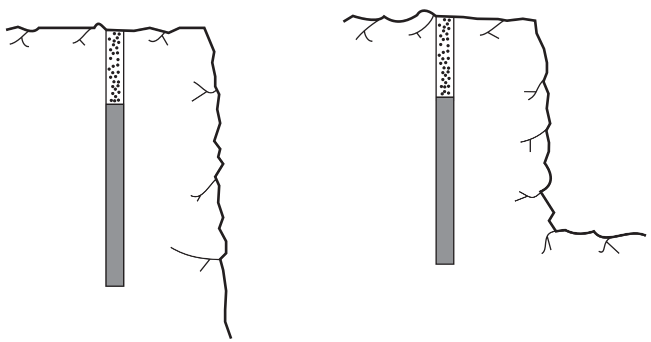


FIGURE *4.2.-6.*
Free breakage and fixed bottom

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 that has voids where gases penetrate without giving full effect, may be difficult to blast even though it has a relatively low tensile strength.

The required specific charge, (kg/cu.m.) provides a first-rate measure of the rock's blasting potential. 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 explosives in the rock is critical. A closely spaced round with small diameter blastholes provides much better rock fragmentation than a round of widely spaced large diameter blastholes, provided that the same specific charge is used. Common calculations are based on the specific charge of 0.4 kg/cu.m. of dynamite, 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 considerably less explosives are needed to break the rock. 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 up to 90% of the blasthole volume if tamping is performed after the introduction of each cartridge. If tamping is performed after every two or three cartridges, the charge concentration is considerably lower. Pneumatic charging machines give good tamping of paper cartridges with high utilization of blasthole volume.

Explosives in plastic hoses were developed for fast charging and easy handling. Dropped into the blasthole, they fill up the hole well. However, the tamping characteristics of different explosives give varying results. Emulite cartridges in plastic hoses cut along the side, fill up the hole almost completely by impact, whereas dynamite and watergel with their stiffer consistency do not fill up the hole as well, especially in the winter.

When charging wet blastholes, it is important that the holes are flushed and cleaned before charging. If the blastholes contain water, the packing of the explosive will be almost nonexistent 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 blasthole volume 100%. Dynamite, emulsions, watergels (slurries)and ANFO are explosives with differing characteristics regarding weight strength and density. Because the maximum burden, V_{max}, also depends on the fixation degree at the bottom part of the blasthole, the calculations involve drilling with an inclination of 3:1. This decreases 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 emulsion cartridge in plastic hoses and 90% for Extra-dynamite. Poured ANFO slurries and pumped emulsion fill up the hole to 100%.

For successful blasting results, the charge concentration obtained by the calculations should be achieved in practice. The formulae used in the calculations are empirical, but are based on information from thousands of blasts. The accuracy of Langefors calculation is so high, it is almost unnecessary in most blasting operations to perform trial blasts. However, local conditions may require the operator to test the theoretical calculations in the field.

The value applies to burdens between 1.0 and 10.0 m, and can be used for most rock types. The bench blasting calculations are based on the Langefors formula (section 2.6).

$$V_{max} = d/33\sqrt{(P \cdot s)/(c \cdot f \cdot E/V)}$$

where

V_{max}	= Maximum burden (m)
d	= Diameter in the bottom of blasthole (mm)
P	= Packing degree (loading density) (kg/liter)
s	= Weight strength of the explosive
c	= Rock constant (kg/cu.m.)
c	= c + 0.05 for V_{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
E/V	= Spacing to burden ratio

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

$V_{max} = 1.47 \sqrt{l_b}$ for Dynamite
$V_{max} = 1.45 \sqrt{l_b}$ for Emulsion cartridge
$V_{max} = 1.36 \sqrt{l_b}$ for ANFO

where l_b is the required charge concentration (kg/m) of the selected explosive in the bottom part of the blasthole (**FIGURE** *4.2.-7.*). Hole inclination is assumed to be 3:1 and the rock constant c is 0.4. Bench height K is ≥ 2 • V_{max}. For other values of hole inclination and rock constant correction factors R_1 and R_2 are used.

Inclination	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

C	0,3	0,4	0,5
R_2	1,15	1,00	0,90

$U = 0.3 \cdot V_{max}$
Sub-drilling = 0.3 • maximum theoretical burden

$H = K + U + 0.05\,(K + U) = 1.05\,(K + U)$
Hole depth = Bench height + Under-drilling + 5 cm/meter drillhole for hole inclination of 3:1

$F = 0.05 + 0.03 \cdot H$
Drilling error = 5 cm application error + 3 cm/meter drillhole

$V_1 = V_{max} - F$ Practical burden = Maximum burden - Drilling error

$E_1 = 1.25 \cdot V_1$ $(E_1 = V_1)$ Practical hole spacing = 1.25 (practical burden)

$l_b = cv^2$ or lb of cartridge

c = Rock constant; c = c + 0.05

$h_b = 1.3 \cdot V_{max}$
Height of bottom charge 1.3 • maximum theoretical burden
$Q_b = h_b \cdot l_b$
Weight of bottom charge = height of bottom charge • concentration of bottom charge

$I_p = 0.4$ to $0.5 \cdot I_b$
Concentration of column charge = 0.4 - 0.5 concentration of bottom charge or, when using only primer

$h_p = H - (h_b + h_o)$
Height of column charge = Hole depth - (height of bottom charge + height of stemming)

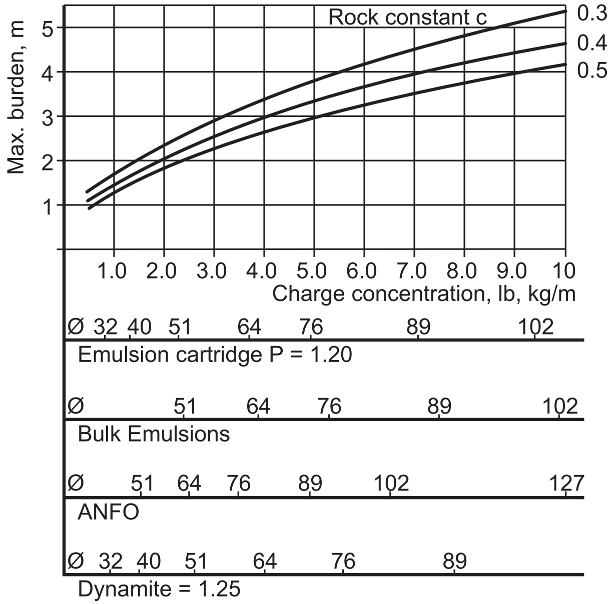


FIGURE *4.2.-7. The influence of bottom charge concentration on maximum burden, V_{max}*

Drilling and charging

The following example shows how to calculate:

Conditions:
Bench height K = 12 m
Width of round B = 20 m
Drill hole diam. d = 76 m
Rock constant c = 0.4
Bottom charge dynamite
(60mm lb = 3.9 kg/m - 4.1 kg/m

1. V_{max} = 2.9 m

2. U = 0.3 • V_{max}
   U = 0.3 •2.9 m = 0.9 m

3. H = K + U + 0.05 (K + U) = 1.05 (K + U)
   H = 1.05 (12.0 + 0.9) = 13.54 (13.6 m)

4. F = 0.05 + 0.03 • H
   F = 0.05 + 0.03 • 13.6 = 0.46 (0.5)

5. V_1 = V_{max} - F
   V_1 = 2.9 - 0.5 = 2.4 m

6. E_1 = 1.25 • V_1
   E_1 = 1.25 • 2.40 = 3.00

Number of holes in row = B / E1 = 20.0 / 3 = 6.7 (7)

E_1 = B / Number of holes in row = 20 / 7 = 2.86 m

7. l_b = (3.9) - 4.1 kg/m; l_b of cartridge in drill hole.

8. h_b = 1.3 • V_{max}
   h_b = 1.3 • 2.9 = 3.77 (3.8 m)

9. Q_b = hb • lb
   Q_b = 3.8 • 4.1 = 15.6 kg / dynamite ø 60 mm

10. l_p = 0.4 to 0.5 • l_b
    l_p = 0.4 • 4.1 = 1.64 kg / m (1.6 kg / m)
    l_p = 0.5 • 4.1 = 2.05 kg / m (2.1 kg / m) = chosen
    2.1 kg/m / anite ø 50 mm (AN / TNT -explosive)

11. h_o = V_1
    h_o = 2.40 m

12. h_p = H - (h_b + h_o)
    h_p = 13.6 - (3.8 + 2.4) = 7.4 m

13. Q_p = h_p • l_p
    Q_p = 7.4 • 2.1 = 15.5 kg

14. Q_{tot} = Q_b + Q_p
    Q_{tot} = 15.6 + 15.5 = 31.1 kg

15. q = (Holes / row • Q_{tot}) / (V1 • K • B)
    q = (8 • 31.1) / (2.40 • 12.0 • 20.0) = 0.43 kg / m^3

16. b = (Holes / row(H) / •(V1 (K (B)
    b = (8 • 13.6) / (2.40 • 12.0 • 20.0) = 0.19 drilled meters / m^3

Summary of important data

Bench height	Hole depth	Burden	Hole spacing	Bottom charge	Column charge		Specific charge	Specific drilling drilled	
m	m	m	m	kg	kg	kg/m		kg / m^3	meters/m^3
K	H	V_1	E_1	Q_b	Q_p	l_p	q	b	
12.0	13.6	2.4	2.86	15.6	15.5	2.1	0.43	0.19	

Swelling

In multiple row rounds, or in rounds where the burden is loaded with previously blasted rock, a charge larger than the limit is required. There must be a certain amount of swelling to complete the loosening. In bench blasting swelling is normally 40 - 50 % (after loading). In bench blasting, the bottom charge must also be large enough to remove the excavated rock that hampers the free face of the front blast (**FIGURE** *4.2.-8.*).

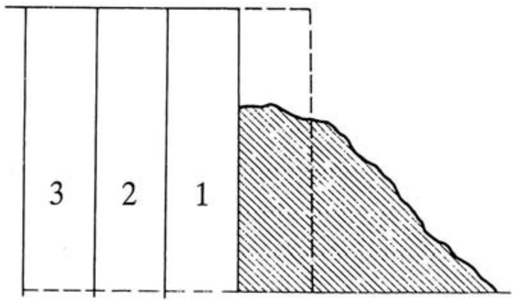


FIGURE *4.2.-8. When blasting against a muckpile, the bottom charge must be large enough to remove the muckpile.*

As the blasting continues row by row, the rock pile in front of the round will gradually increase and lie closer to the burden, which is about to be blasted. In continuous blasting without mucking, the rock's gravity center must be heaved from A to B in **FIGURE** *4.2.-9.*, requiring extra energy. This is usually attained by diminishing the burden.

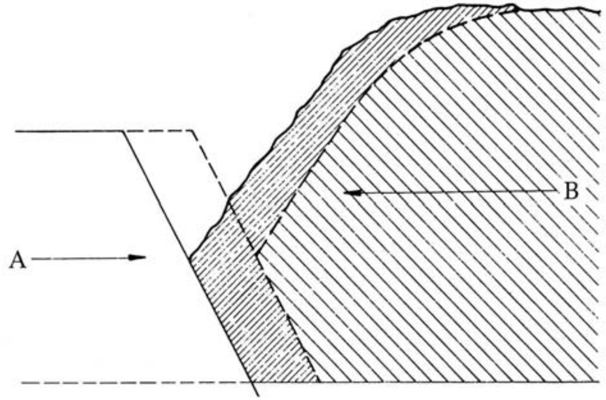


FIGURE *4.2.-9. Continuous blasting without mucking*

Blasting different rock types

The following instructions describe rock charging under varying conditions. The suggested empirical values pertain to blasting parallel to schistosity or jointing. When blasting perpendicular to these planes, normal table values may be used.

- When the rock mass is relatively solid, slightly fissured and tough, and the rock type is igneous in origin (gabbro, periotite, gneiss etc.), it may be necessary to increase the specific charging of a cut up to 30%
- In sandstone, specific charging depends on rock texture strength, when granular jointing is hard and rock jointing is almost nonexistent, charging equal to hard rock charging should be used. In loose texture and increased jointing, charging can reach the intensity granite charging
- When the rock to be blasted is highly fractured or weathered, charging should be equal to igneous rock charging
- Copper ore and limestone are generally easy to blast, and charging equal to granite charging can be used
- Rocks of volcanic origin may require heavier charging; values up to 1.0 kg/m^3 are not unusual

In fissured rocks, breaking occurs along joints and fractures rather than along irregularities; gases produced by the explosion escape through the cracks and joints, and energy is lost. Rocks containing broken zones and a large amount of loose, open joints usually require burdens and spacings up to 20% larger than normal. It should, however, be noted that a structure behavior of this kind in blasting cannot be evaluated beforehand. Attention should be therefore paid to safety.

LOADING EXPLOSIVES

Loading mainly depends on the type and packed form of the explosive used, and the diameter of the blasthole. The main loading methods are:

-Manual
-Pneumatic
-Bulk truck

Manually loading explosives

For explosives in cartridges, rod tamping is the most common manual loading method, especially for small diameter holes. The explosive cartridge is dropped into the hole and tamped with a wooden tamping rod. Charging density can be somewhat controlled by the force applied on the rod. If the degree of packing must approach the density of the explosive, only one cartridge should be inserted at a time and compressed by the tamping rod.

Pneumatic loading of explosive cartridges

A mechanical way of loading an explosive cartridge is to use a device that blows the cartridge into the hole through a metal or plastic tube with compressed air.
When mechanically loading blastholes, the operator can be sure that the volume of the hole is fully exploited and that the charge is not cut by rock spalling from the walls. At the start of loading, the primer and cap are pushed to the bottom of the hole with the loading hose. As the explosive material or cartridges flow into the hole, the hose is carefully retracted. This keeps the hole open until it is fully charged.
With pneumatic loading the yield of rock per drilled meter can be increased considerable. Burdens and spacings can be maximized, which of course reduces both drilling and blasting costs. Increased burden (due to the better utilization of hole volume by pneumatic loading) leads to increased boulder size in the column parts of the hole. This can be avoided by increasing the column charging density. Charging density, compared to normal rod tamping, is 20 - 40% higher , and devices can be successfully used to charge holes up to 30 m in underwater blasting, both with hole sizes of ø 32 - 102 mm.

ANFO loading

Depending on the size and type of surface drilling and blasting operation, the operator can choose from three ANFO loading methods:

- Manual loading: ANFO is simply brought to the bench in plastic or aluminium vessels, and poured into the hole after setting the primer or bottom charge in the hole
- Pneumatic loading by ejector
- Truck loading

Manual loading and loaders are the most feasible method for a smaller blasting project. When the blasthole diameter 102 mm(4”) and the consumption of explosives increases due to high rock production rates, truck loading is the preferred method.

FIGURE 4.2.-10. *ANFO loading vehicle for open pit mines and quarries.*

Pneumatic ANFO loading

Ejector and pressure-vessel type pneumatic loaders are specially designed to be used with ANFO; they are mainly used for small hole blasting in surface and underground rock excavations.

ANFO loading by trucks

In quarries and open pit mines where hole sizes exceed ø 102mm (4”), ANFO is generally transported to the loading site in container trucks and blown or poured into blastholes. These self-contained trucks monitor the proper amount of oil fed into the prill as it is pumped through a hose into the hole. ANFO loading systems are generally based on air or auger; auger type systems have either a side boom or overhead boom. Each type of loading system has its advantages, but maximum flexibility is achieved through the air system, in which oil is added to the airlock feeder as it receives the prills from an auger at the base of the vee-bottom truck. The loading hose can be moved to any hole within a 30-meter radius of the loading truck. The pneumatic truck method is preferred in uneven terrain, because it reduces the number of times the truck musts be repositioned. One drawback, however, is that the hose must be hauled from hole to hole.

The overhead boom auger truck has a narrower radius (about 10m). In the side boom model, the truck must practically move from hole to hole. Auger booms are normally either hydraulic or hand operated. They are also somewhat quieter in operation. Mines and quarries that use

larger quantities of ANFO often use bulk loading.
Table 4.2.-1. shows the density of ANFO when poured or blown into the blasthole. The figures are based on practical experience and are higher than the calculated density because the actual blasthole volumes are generally greater than with the drill bit. This is due to cracks and rock slabbing off the blasthole wall.

Table 4.2.-1. *Blasthole density of ANFO when poured or blown into the hole.*

Hole diameter	Poured ANFO	Blown ANFO			
mm	in	0.85 kg/dm³		0.95 kg/dm³	
		kg/m	lbs/ft	kg/m	lbs/ft
51	2	1.74	1.17	1.94	1.30
64	2 1/2	2.73	1.84	3.06	2.05
76	3	3.86	2.59	4.31	2.90
89	3 1/2	5.29	3.55	5.91	3.97
102	4	6.95	4.67	7.76	5.22
110	4 1/4	8.08	5.43	9.03	6.07
115	4 1/2	8.83	5.93	9.87	6.63
127	5	10.77	7.23	12.03	8.09
152	6	15.42	10.36	17.24	11.58
178	7	21.15	14.21	23.64	15.88
200	7 7/8	26.70	17.94	29.85	20.05
230	9	35.32	23.73	39.47	26.52
251	9 7/8	42.06	28.26	47.01	31.58
311	12 1/4	64.57	43.38	72.17	48.49

Static electricity control in pneumatic loading

Pneumatic loading of small particle bulk agents such as ANFO can generate enough static electricity to cause premature initiation of both electric caps and non-electronic detonators. There are three points in a pneumatic operation where electrical energy can be stored:
- On the operator
- On the blasting agent loader and accessory equipment
- In the blasthole, on leg wires

If a semi-conductive path is maintained between these points and ground, the energy is dissipated before it can build up to dangerous levels. In a pneumatic loading system ,a semi-conductive path for the bleed off of potentially dangerous static electricity can only be assured if a reliable semi-conductive loading hose is used. It must do two things: be conductive enough to harmlessly dissipate the static electricity that was generated during the pneumatic loading of ANFO through properly grounded equipment, and have enough resistance to prevent hazardous stray currents from reaching the initiators used to prime the ANFO.

Although both cap and fuse, and non-electric delay caps have a higher resistance to static initiation than electric blasting caps, tests show that these systems can also be initiated by static electricity. Therefore, it is recommended that semi-conductive loading systems be used even in non-electric initiating methods.

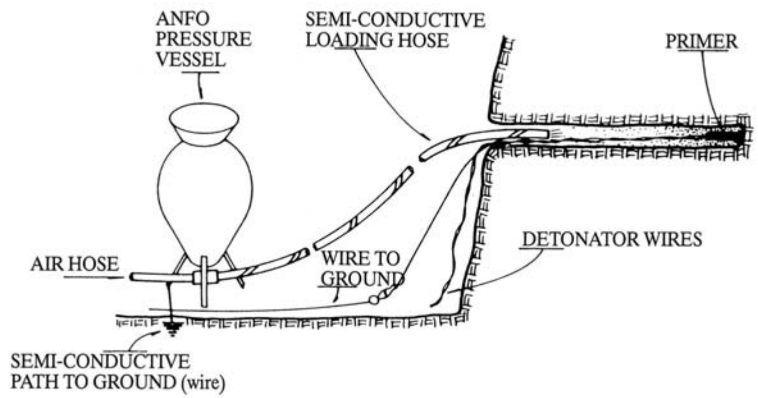


FIGURE ***4.2.-11.*** *Static energy control in pneumatic loading.*

Loading emulsion explosives

The following describes the typical manufacturing and loading procedures for Kemiitti in a Finnish quarry. The main components of Kemiitti are nitrates, fuel, oil, water and aluminum. Certain additives are used for adjusting the stiffness and density of the emulsion. Kemiitti is produced at manufacturing stations as well as in Kemiitti trucks.

Before a Kemiitti truck leaves the station for the quarry, it is loaded with the base solution, supplementary solution, fuel oil and aluminum. The truck consists of a wheel-based carrier with container silos, pumps, mixers and a control system for mixing and loading the emulsion. Carrier dimensions are mainly determined by the required mixing capacity. 9 tons is the most common capacity used on one loading. The standard loading hose length is 30 m, the inner diameter being 1”- 1.5”. The pumping flow of the Kemiitti varies depending on the blasthole diameter.

When the truck reaches the quarry, the operator selects a program that calculates the right amount and type of Kemiitti used to fill the holes. A member of the loading crew then brings the hose to the first hole and pushes it to the bottom. The Kemiitti system loads the holes as programmed. The quality of the emulsion is continuously controlled during loading. When pumping is finished, the operator checks the charge height in the hole before allowing the loading crew member to continue to the next hole.

A separate aluminum-sensitized primer is sometimes used. Adjusting the charging according to hole length, diameter and rock conditions is achieved by altering the density of Kemiitti. The most commonly used charge type has a density of 1.2 kg/dm^3 at the bottom, which gradually changes to 0.85 kg/m^3 at the top of the column. For example, the charging of a ø 200 mm hole cut 15 - 20 m in length is very fast. Up to 9 tons of Kemiitti can be loaded in two hours. The speed decreases with smaller holes.

It is most economical to construct a manufacturing station at the site when large quantities of Kemiitti are used over a long period of time at the quarry or mine. For shorter contracts, components can be transported to the site by trucks from the nearest manufacturing stations or a Kemiitti truck can be used to serve many quarries or open pits in one district.
The process of Emulite manufacturing and loading is similar to the Kemiitti System.

Priming explosive charges

In general, primer location affects:

- Magnitude and shape of stress wave in rock mass
- Movement of rock mass during blasting
- Shearing of rock mass at grade level
- Breakage of cap rock above stemming

The maximum stress normally occurs in the direction in which the explosive detonates. Moreover, the intensity and shape of the stress wave is related to the speed with which it travels through the rock and to the velocity of detonation.

Deck loading

Deck loading is a technique that is used to reduce vibrations and to increase the effectiveness of explosive charges. It uses several individually primed charges that are separated by stemming in the same blasthole. Non-electric delay primers are available that simplify this loading method by making it possible to use several primers on a single downline (**FIGURE** *4.2.-12.*)

A typical non-electric delay primer for deck loading consists of two-piece device which has a cast explosive booster in a cylindrical plastic housing and a delay. In this case, the delay is a millisecond delay, non-electric blasting cap that has a short factory-attached miniature pigtail detonating cord with an aluminum end seal.

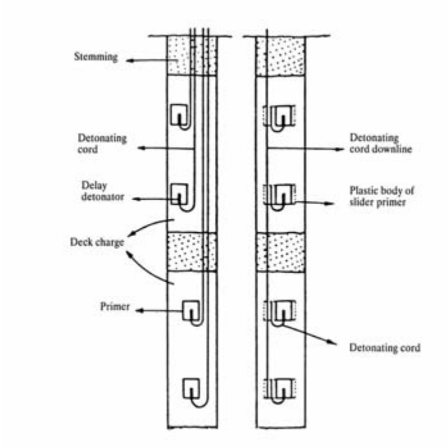


FIGURE *4.2.-12. Two typical deck loading arrangements.*

During loading, the delay is inserted in the delay channel at the bottom center of the circular plastic hauling. The pigtail is then inserted into a downline channel that runs along the edge of the housing, and is snapped into a retainer slot at the bottom of the primer. An 80 or 130 grain/m detonating cord downline is threaded through the channel and knotted at the end, and the primer is lowered into the blasthole. The charge interval is loaded with the appropriate non cap-sensitive explosive - whether in bulk, package, or slurry form - and is stemmed. The next delay primer assembly can then be threaded onto the downline and drop-loaded. The next explosive interval is added and stemmed, and the procedure is repeated.

There are several standard millisecond delays that can be used together with the delay primer. The delay primers typically weigh 0.5 kg and are designed for blastholes that are 102mm (4”) or larger in diameter. A 1 kg primer is also available. In addition to deck loading, such primers may also be used for top or full-column hole bottom initiation.

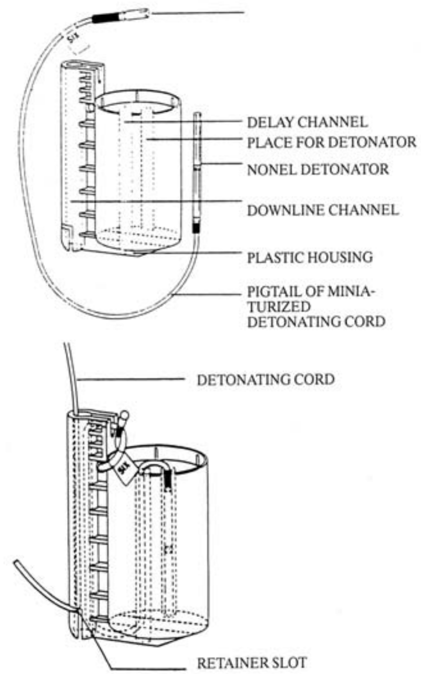


FIGURE *4.2.-13. A typical primer for deck loading.*

FIRING SYSTEM FOR BENCH BLASTING

Bench blasting is normally performed as short delay blasting. The advantages of delay blasting are obvious:

- Delay blasting makes it possible to control and reduce ground vibrations; individual hole charges or even parts of charges can be fired separately,
- Delay blasting helps control throw; the firing system’s geometry can be varied to a great extent to achieve the correct direction of throw, and type and shape of muckpile.

Delay blasting makes it possible to increase burdens and spacings, while reducing the specific charging of the cut; firing can be arranged so that each row of holes or even each hole works only for its own burden.

Single-row blasting

The simplest form of delay blasting is a row of charges that are initiated at consecutive intervals. Single-row blasting is normally applied in large hole blasting projects in quarries and open-pit mines with hole diameters larger than 89mm (31/2") in diameter.
Single row blasts with 10 - 60 ms delays between adjacent blastholes provide better fragmentation than instantaneous blasts.
The best fragmentation is achieved when each charge is given just enough time to effectively detach its burden quota from the rock mass before the next charge detonates. (**FIGURE** *4.2.-14.)*

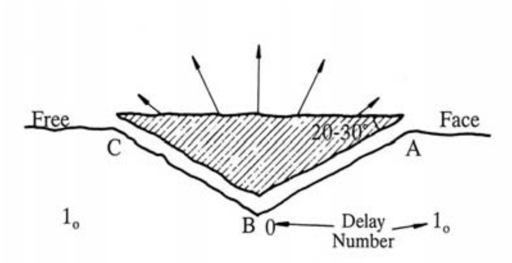


FIGURE *4.2.-14. The best fragmentation in single row blasting is achieved when each charge is given just enough time to effectively detach its burden quota.*

FIGURE *4.2.-15.* and *4.2.-16.* show firing patterns for single-row blasting. In the first example, hole spacing is 10 - 50% greater than the burden. The pattern in **FIGURE** *4.2.-16* (with burden equal to spacing) provides somewhat better fragmentation.

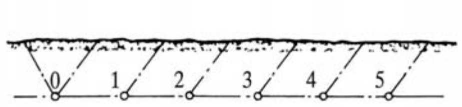


FIGURE *4.2.-15. Firing pattern for single-row blast. Hole spacing 10 - 50% greater than burden.*

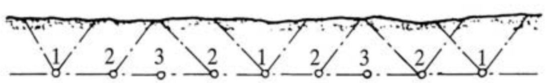


FIGURE *4.2.-16. Firing pattern for single-row blast. Burden equals spacing.*

Delay time between blastholes and rows must be long enough to create space for the blasted rock from the following rows.Delay time between rows can vary from 10ms/m burden (hard rock) to 30 ms (soft rock) but generally 15-25 ms is a good value, because it 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 delay between the rows is too short, rock from the back rows tend to take an upward direction instead of horizontal. On the other hand, prolonged delay may cause flyrock, air-blast and boulders, as protection from previously fired rows disappears due to excess rock movement between detonations. In this case, boulder increase can be compared to single-row blasting (**FIGURES** *4.2.-17.* and *4.2.-18.*).

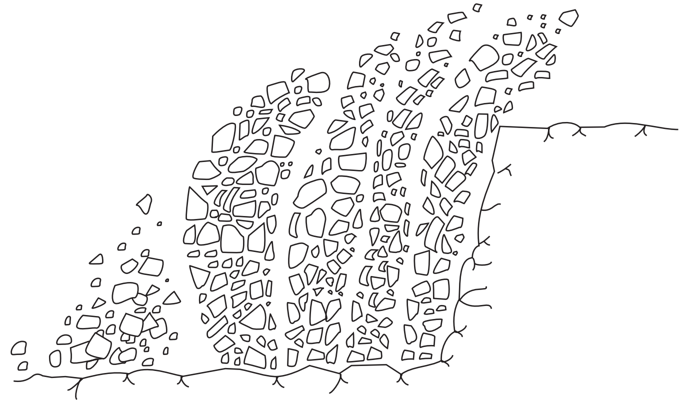


FIGURE *4.2.-17. Insufficient delay between rows.*

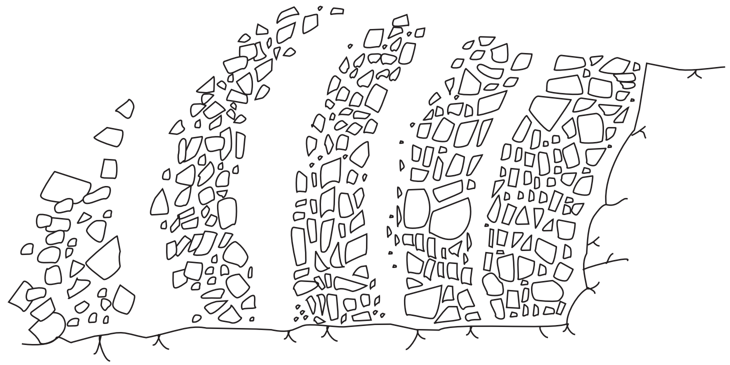


FIGURE *4.2.-18. Perfect delay between rows.*

Simple firing pattern for a lateral multiple-row round (**FIGURE** *4.2.-19.*). All holes in the row have the same delay except perimeter holes, which are delayed in interval numbers to avoid excessive overbreak beyond the excavation boundaries. The firing pattern in **FIGURE** *4.2.-20.* provides better fragmentation. The ratio between true spacing and true burden, E/V, becomes more favorable. (See wide-space drilling pattern.)

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

The firing pattern in **FIGURE** *4.2.-21.* provides separate delay times for practically all blast-holes, and gives good fragmentation and breakage in the bottom part of the round. Wide-space drilling pattern ignition is shown in **FIGURE** *4.2.-22.*

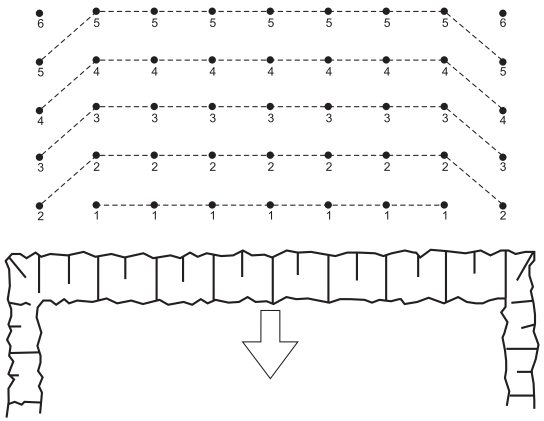


FIGURE *4.2.-19. Firing pattern, multiple-row blasting.*

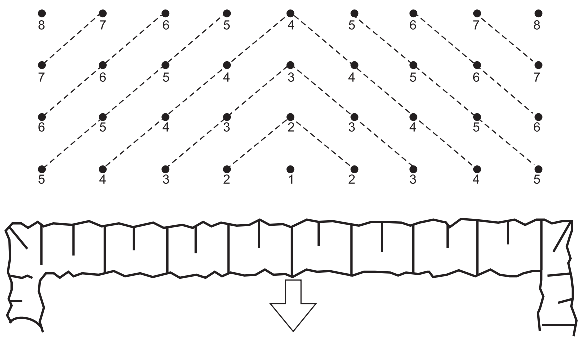


FIGURE *4.2.-20. Firing pattern/Better fragmentation.*

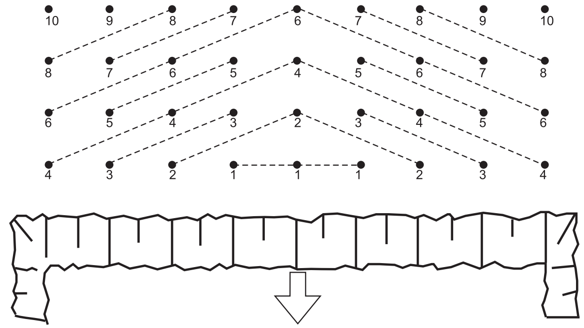


FIGURE *4.2.-21. Firing pattern/Good fragmentation and good breakage in bottom.*

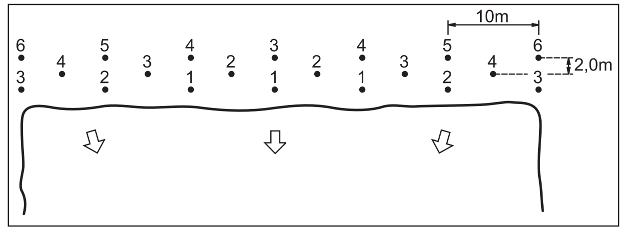


FIGURE *4.2.-22. Wide-space drilling and ignition.*

BENCHING WITH HORIZONTAL HOLES

Particularly in central Europe, horizontal holes are used successfully in limestone, sandstone and granite quarries. This technique has specific advantages in difficult rock conditions:

- Adequate and smooth rock cutting in bottom part of bench
- Lighter concentration of explosives at bottom of bench
- Reduced bench top fracturing or disturbance at lower levels

The disadvantages of horizontal drilling are:

- Increased amount of specific drilling in cuts

- Special technical features required for surface crawler drills; rig's boom must be suitable for horizontal drilling
- Increased moving of drill between the two bench levels

There are two alternatives for using horizontal holes in bench drilling and blasting. First, the same hole diameter can be employed in drilling the horizontal as well as the vertical (inclined) holes. In this case, the common method is to use the (3 1/2" - 4 1/4" (89 -110mm) hole diameter range. The second alternative is that smaller diameter holes 1 1/2" - 3" (38 - 76mm) can be utilized in horizontal drilling. The success of horizontal drilling and blasting techniques depends very much on the correct firing sequence. The following describes the design principles of horizontal hole setting (**FIGURE** *4.2.-23.)*.

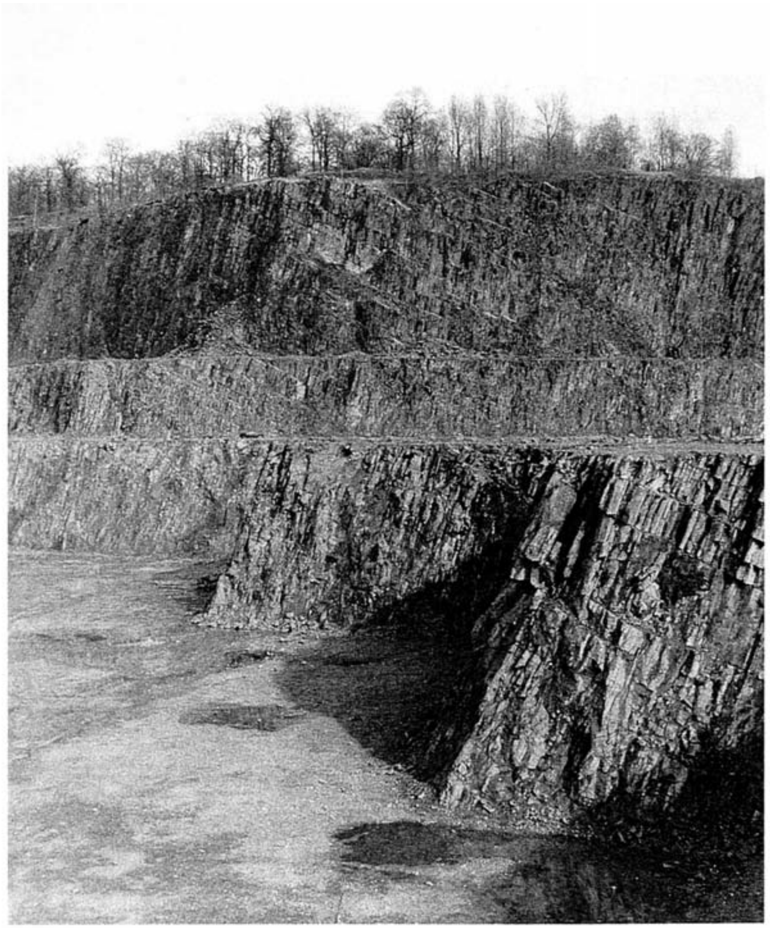


FIGURE ***4.2.-23.*** *Using horizontal holes ensures adequate and smooth rock cutting.*

The burden (V) and spacing (E) of a vertical (or inclined) drilling pattern are designed according to the theory and method described earlier in this chapter. Here the burden value (with patterns of geometry E=1.25V) depends on hole inclination (d), bench height (K), hole inclination and rock blasting properties (**FIGURE** *4.2.-24.)*.

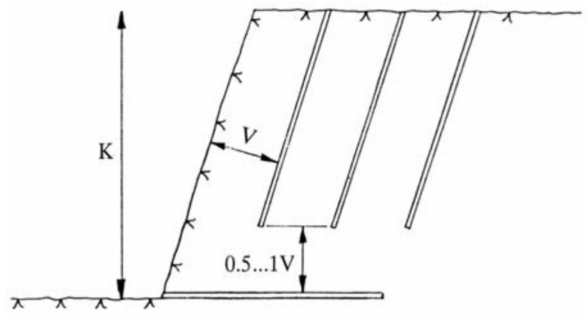


FIGURE ***4.2.-24.*** *Setting horizontal holes in bench drilling.*

The holes in a vertical pattern are usually drilled as deep as 0.5 - 1V from the bottom level of the cut. In other words, the burden V2 of the horizontal holes is as shown in **FIGURE** ***4.2.-25.***

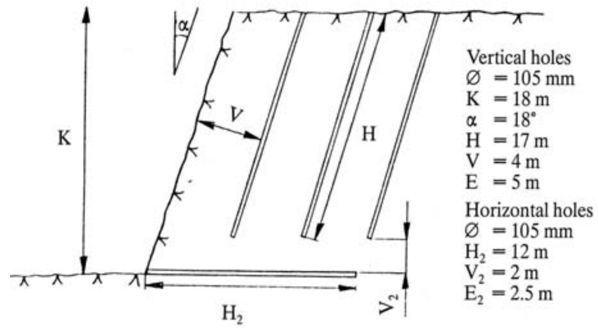


FIGURE ***4.2.-25.*** *Typical use of horizontal holes in a quartz porphyry quarry and a basalt quarry.*

$$V_2 = 0.5...1V$$

where V_2 = Burden of horizontal holes (m)
V = Burden of vertical holes (m)

Here the distance from the bottom of the cut to the bottom of the vertical holes is taken as the burden for the horizontal holes. Spacing E_2 of the horizontal holes is similarly calculated in relation to the spacing of the vertical pattern, which should be

$$E_2 = 0.5E$$

where E_2 = Spacing of horizontal holes (m)
E = Spacing of vertical pattern (m)

The length of the horizontal holes H_2 depends on the depth of the cut. The common practice is to drill the horizontal holes up to the back row of the vertical holes, so

$$H_2 = nV_1$$

where H_2 = Length of horizontal holes (m)
n = Number of rows of vertical holes
V_1 = Burden of vertical holes on the surface

Due to denser drilling at the bottom of the cut, the number of horizontal holes can be up to one third of the total drilled meters per cut. **FIGURE** *4.2.-23.* shows the typical use of horizontal holes in a quartz porphyry quarry (Belgium). Horizontal drilling has improved the quality of the quarry floors and overall rock fragmentation. The heavy concentration of explosives in the bottom parts of the bench is prevented in order to keep the vibration levels acceptable in residential areas.

ROCK FRAGMENTATION

The degree of desired fragmentation depends on the end use of the product being mined. In open-pit mining where more minerals are being extracted from a matrix, it is usually to achieve maximum fragmentation. In quarrying where rock is sized for construction use, it is often not desirable to produce a lot of small-sized material.

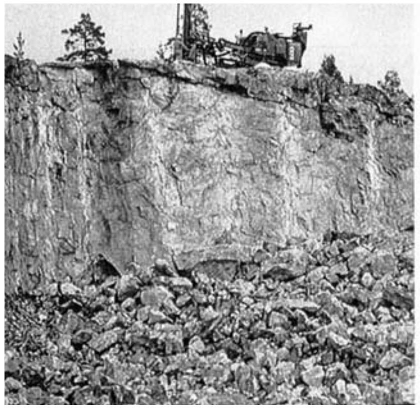


FIGURE ***4.2.-26.*** *Rock fragmentation.*

The most important factors affecting fragmentation:

- Rock characteristics
- Blasthole straightness
- Explosive properties
- Loading of blastholes
- Specific charging
- Sequential firing

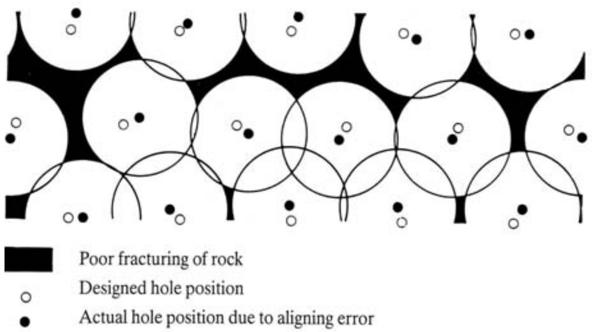


FIGURE ***4.2.-27.*** *Effect of aligning errors in drillhole positioning.*

Rock characteristics

Fragmentation is largely affected by the nature of the rock. One of the most important characteristics of rock is its variability. As a general rule, enhanced breakage is achieved by placing blastholes within solid blocks bound by these discontinuities rather transferring explosive energy across them. Blasting patterns can be designed to take advantage of rock structure, for example, by planning a free parallel rather than perpendicular to marked vertical joint planes. In rock with well developed bedding or schistosity planes, this is achieved by keeping the free face perpendicular rather than parallel to the direction of dip (**FIGURE** *4.2.-28.*). If the planes are horizontally positioned or inclined and blasting is to be performed in that direction, hole inclination should be planned to maximize explosive efficiency.
If schistosity or jointing is vertical or almost vertical, optimal fragmentation requires that the direction of movement is at a right angle to the planes. In such cases, it is difficult to obtain a smooth bottom for the cut.

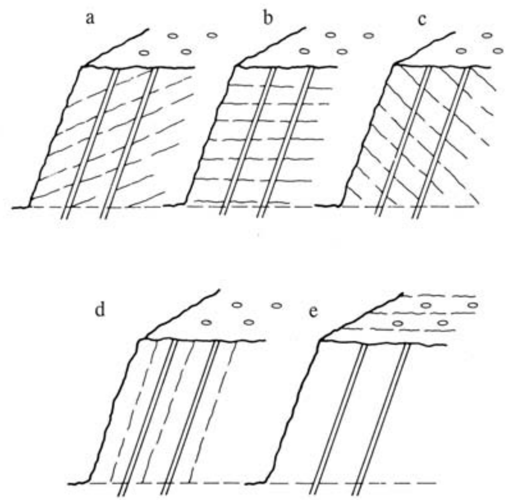


FIGURE ***4.2.-28.*** *Horizontal (b), inclined (a, c) and vertical (d, e) schistosity or joint planes.*

Straight blastholes

The reasons for not achieving blasthole straightness in a bench can be divided into two major groups:

- Incorrect use of drilling equipment, including both drill rigs and drill steels
- Structural properties of rock at the worksite

If holes are not straight, the results are poor rock fragmentation, poor toe conditions, excessive throw and rock scattering during blasting. Either the blasthole bottoms are too close to each other, which leads to local overcharging, or the bottoms are too widely spaced, which results in low specific charging. The latter may be disastrous, especially close to bottom of the bench (**FIGURE** *4.2.-29.*).

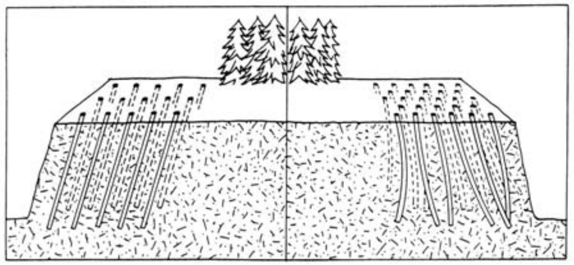


FIGURE *4.2.-29. Straight blastholes facilitate reduced specific drilling and charging in a bench blast by up to 15%.*

Hole diameter	mm	115	115
Bench height	m	20	20
Hole deviation	cm/m	2.5	1.0
Burden	m	3.3	3.6
Spacing	m	4.1	4.5
Specific drilling	drm/m³	0.078	0.065
Primer	kg	3	3
Column charge	kg	155	155
Specific charging	kg/m³	0.58	0.49

Explosive properties

The explosive properties that have the most influence on rock breakage are charging density, detonation velocity, explosion heat, gas pressure and volume. The volume of gas released upon detonation is also important in the later stages of breakage, and is critical in blasting weak or naturally fractured rock.
If an explosive's charging density can be increased, the specific charging of rock (kg/m³) also increases if all other factors remain unchanged. This naturally affects rock fragmentation in particular parts of the relevant hole.

Loading blastholes

Rock fragmentation is affected by all three parts of a continuous charge:

- Bottom charge or primer
- Column charge
- Stemming

Charging at the bottom of a blasthole should be high enough to provide adequate cutting at the bottom of the bench.

Rock pulverization occurs when the induced detonation stress exceeds the rock's compressive strength surrounding the blasthole. In hard, solid and slightly fissured rock, the extent of the crushed zone around the hole will depend on the diameter of the hole charge per meter, as shown in ***Table 4.2.-2.***

Table 4.2-2. *Depth of crushed zone around blasthole from different hole diameters.*

Blasthole mm	***diameter in***	***Charge *) per meter kg/m***	***Crushed zone radius mm***
32	1 1/4	0.95	40
38	1 1/2	1.35	50
51	2	2.45	70
76	3	5.45	90
102	4	9.80	100
127	5	15.20	110
152	6	21.75	120

*) charging density 1.2 kg/dm³

Column charges result in sufficient loosening and breakage of the bench above the bottom level. As the rock is not in the column parts of the bench, (there is a wide free face available), the column charge per meter is typically 50 - 80% of the bottom charge. With insufficient breakage in the column, the column charge can be increased - for example by increasing its charging density or simply by decreasing the burden.

Fragmentation and loosening can be reduced as a result of energy loss in the atmosphere, particularly via the stemming column. The energy wasted through prematurely ejected stemming has been captured through high-speed photography. Good stemming maintains high gas pressure in the blasthole for longer periods of time. More stemming power achieved through longer stemming columns or coarser, more efficient cuttings increases the amount of effective work performed per unit weight of charge. This reduces the cost of subsequent operations.

There are two ways of improving rock fragmentation by altering the charge configuration of a blast:

- Place smaller holes (in diameter and length terms) in the upper part of the bench in addition to the main holes in the drilling pattern (**FIGURE** *4.2.-30.*)

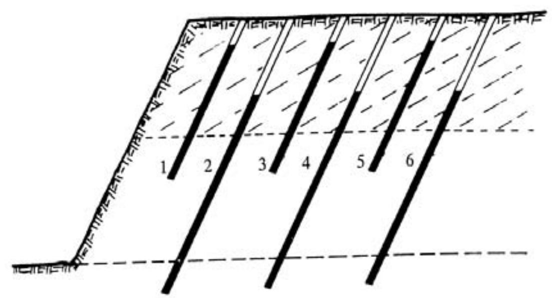


FIGURE *4.2.-30. Improving bench-top fragmentation with additional smaller holes in drilling pattern.*

- Distribute a small concentrated charge higher up in the column (**FIGURE** *4.2.-31.*).

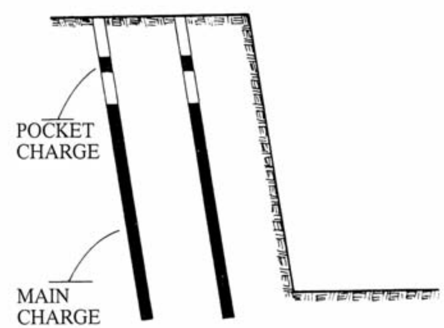


FIGURE *4.2.-31. Short pocket charges can be placed in a blasthole's stemming column to improve the top fragmentation.*

Specific Charging

Rock fragmentation depends greatly on the specific charging (powder factor) of the rock.
Specific charging tends to increase in hole diameter. Naturally, bigger quarry and open pit equipment can handle larger and coarser material, but are nevertheless designed to operate economically with larger volumes of rock, not larger sized material.
Smaller blasthole diameters give better rock fragmentation due to smaller burdens and spacings used, which provide a better charge contribution in the rock.
Rock fragmentation can be improved by increasing the hole charges in a cut, while keeping the drilling pattern constant.
The same can, of course, be achieved by decreasing burdens and spacings, but still using the same charge configuration. Hole charges can be increased by changing either charging density or length. Stemming can be reduced, however, the risk of throw increases. As specific charging grows, more material (on a percentage) is also crushed.

Firing system

A proper firing system for rock breakage is crucial. Optimum fragmentation can be achieved when each charge is given just enough time to effectively detach its quota of the burden from the rock mass before the next charge detonates.
Short-delay patterns with optimum timing lower energy levels required by expanding proven patterns, and at the same time keep fragmentation levels acceptable.

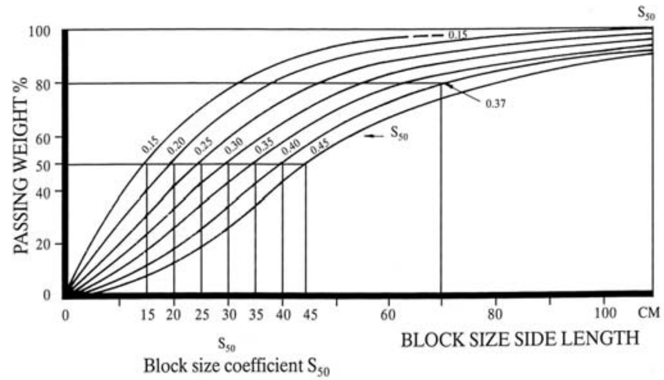
Boulder size estimation

In bench drilling and blasting, block size requirements are considered during the design stage. Different available models are typically based on block size estimation by the specific drilling and specific charging of a blast. The following calculated model is based on a formula invented by Stiftelsen Svensk Detonikforskning of Sweden.

Definitions

The coefficient of block size S50, or average breakage , is the quadratic opening of a screen, measured in meters, through which half of the extracted rock (50% of the total weight) will pass if screened in the normal manner. When the value of S_{50} is known, the total distribution curve can be drawn and the block size distribution of the blasted round determined. The distribution curve shows what percentage of the total round will be over 80 cm (block largest side), or what percentage will be under 20 cm. The accuracy of this model has been proven satisfactory in field experiments, which show that block size distribution does not differ majorly from the curves shown in **FIGURE** *4.2.-32.*

FIGURE *4.2.-32. Determining block size coefficient S50 for blasted rock as a function of largest side and passing weight.*



Rock constant c

Rock constant is the amount of charging (kg/m³) in a round that is just enough to extract the rock (not to fulfill breakage requirements). It is also called a measure of a rock material strength, and is usually expressed by the amount of dynamite in kg required to extract one cubic meter of rock. The rock constant varies between 0.3 - 0.5 in hard rocks, and is typically 0.4 for granite.

Specific charging q

Specific charging determines explosive consumption (kg/m³) in the final blast design. It can vary from 0.1 - 1.5 in surface blasting.

Specific drilling S

Specific drilling describes the amount of drilling required in a round for the extraction and breaking of one cubic meter of rock (drm/m³).

Rock structure constant

The rock structure constant takes into account the fact that even in one round the structure of rock varies widely. The following constant values have proven useful in practice:

Homogenous rock	0.40
Relatively homogenous rock	0.45
Normal rock with hairline cracks	0.50
Jointed rock	0.55
Very jointed and fissured rock	0.60

The following formula was invented by Stiftelsen Svensk Detonikforskning of Sweden showing the average breakage of rock (S_{50}) in a blast based on the above factors:

$$\ln L = \ln 0.29\, VV_{12} - \ln 1.18(q/c) - 0.82$$

where L = Length of average boulder side (m), describing the S_{50} of blasted material
V= Burden (m)
V_{12} = Burden (m) with E/V= 1
q = Specific charging (kg/m³)
c = Rock constant

By applying the definition of the block size coefficient S_{50}, and again adding the factors mentioned above, the formula can be expressed as a nomogram, as shown in **FIGURE** *4.2.-33*.

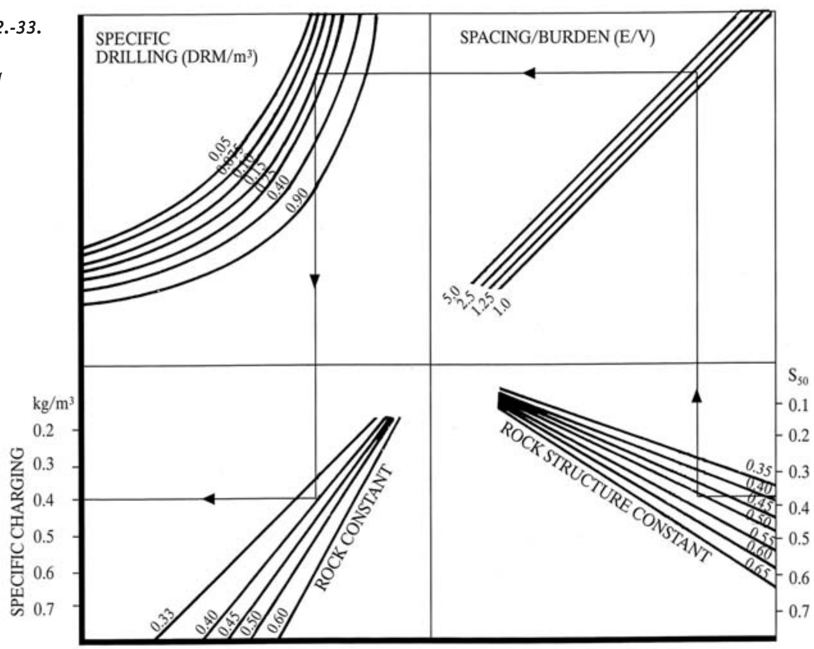


FIGURE ***4.2.-33.*** *Nomogram for determining the block size distribution in a blast.*

Large boulder size

Even rock fragmentation is one of the main goals in bench blasting. However, for certain purposes, such as building breakwaters and ports, big broken boulders are preferred. When blasting broken and jointed rock, boulder size is often determined by the rock's structural properties rather than by the specific drilling and charging used. In solid and homogenous rocks, big boulders can be produced by applying the correct drilling and blasting techniques, such as

- Low specific charging
- Spacing/burden ratios E/V < 1
- Instantaneous firing
- Single-row blasting

Specific charging in such a cut should be 0.2...0.25kg/m³ - large enough only to cut the boulders off the bench in single row blasting. If the bottom of the cut must also produce big boulders, part of the bottom charge should be replaced by column charge. This, however, increases the risk of toe problems. The spacing/burden ratio in the blast should be

$$E = 0.5\ V,$$

where E = Spacing (m)
V = Burden (m)

This type of pattern geometry causes less twisting and tearing of the rock and more splitting along the line of blastholes than spacing appreciably larger than burden.

WALL CONTROL BLASTING IN OPEN PITS AND QUARRIES

Presplit blasting

As an introduction to presplit quarrying, the figure illustrates a typical presplit layout using ø 102mm (4") presplit holes for ø 381mm (15") production holes. Presplit holes are normally drilled before main production. It is then possible to select between loading and firing

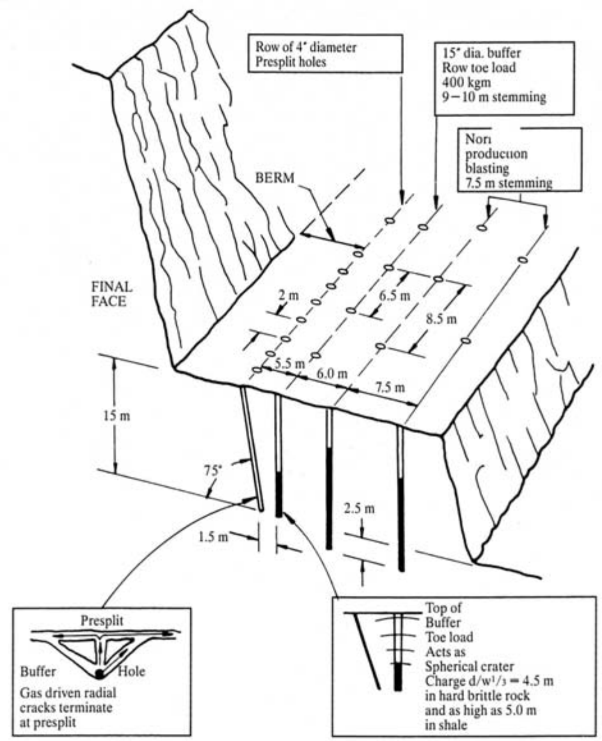


FIGURE ***4.2.-34.*** *A typical presplit blast coupled to a production blast.*

the presplit line, or infilling the main blast. In the latter case, a presplit line is fired instantaneously 100 - 150 milliseconds before the main blast. As shown in **FIGURE** *4.2.-34.*, the presplit line is formed ahead of the main blast and allows the gas, which is driven back form the buffer row through the radical cracks, to terminate at the presplit. line.

Trim blasting

Trim blasting is a wall control technique that uses large diameter blastholes for both production and final wall holes. The goal is to minimize small diameter blast hole work and associated loading difficulties, especially in severe rock conditions.
FIGURE *4.2.-35.* shows a plan of two trim blasts designed to run two benches into a final face between berms. The upper blast, called the crest trim blast, takes the upper bench to the limit. As shown in the illustration, the majority of backbreak occurs at the bench crest, and is mostly from the previous subgrade. The blast has three distinct features, which are similar to the presplit blast:

1. A trim row is used similarly to the presplit row to produce the final wall. The trim row is also decoupled in a similar way. If stemming is used, the decoupling calculation is done using 40 - 50% voids.

2. A buffer row is used as the last row of the main blast with increased stemming to prevent cratering at the surface through the trim row.

3. Normally, two other normal rows of holes are used in front of the buffer row to complete the trim blast. These two rows are at normal spacing and burden, and are loaded by the standard procedure for the appropriate material type.

All holes are of production size. For example in **FIGURE** *4.2.-35.*, the holes are ø 251mm (9 7/8").

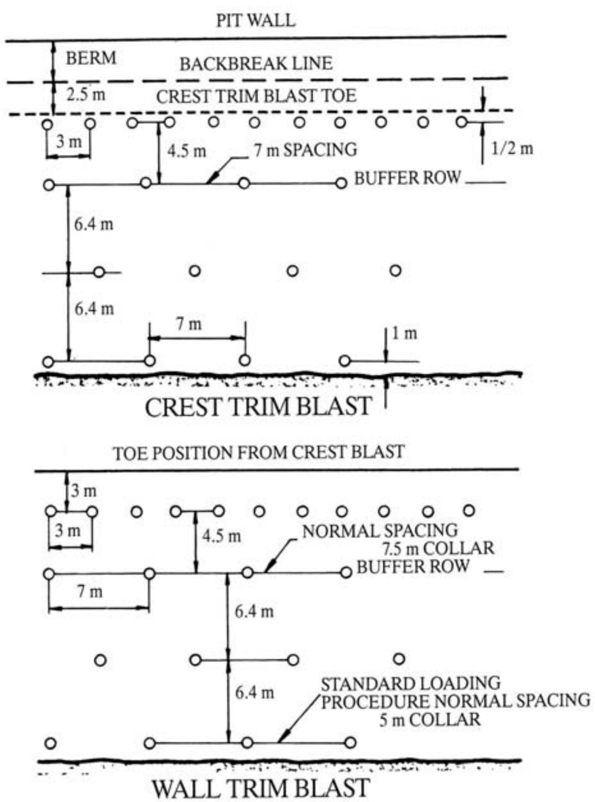


FIGURE *4.2.-35. Two trim blasts designed to run two benches into a final face between berms.*

A graph of hole spacing used in trim blasting as a function of hole diameter is shown in **FIGURE** *4.2.-36.* Spacing generally ranges between

E = 12...16d,

where E = Spacing (m)
d = Presplit hole diameter (mm)

These values correspond to the hard to moderately soft range, and are adequately decoupled charges.

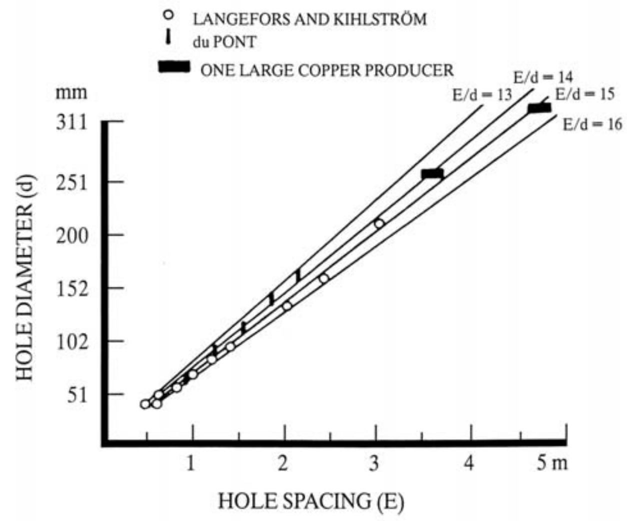


FIGURE *4.2.-36. Hole spacing used in trim as a function of hole diameter.*

SECONDARY BREAKING

Primary blasting often produces boulders too large to be handled by loading and hauling equipment, and the crushing plant. Boulders can result from the natural weathering of the bedrock, or can be caused by insufficient fragmentation during blasting. To obtain good fragmentation, a good drilling and blasting design should be used for the primary charges as secondary breaking to enable the loading equipment to handle the oversized rocks. The primary crusher is very expensive. However, secondary breaking is sometimes necessary at the site, and this is achieved by:

- Pop shooting
- Plaster shooting
- Breaking by impact hammers
- Breaking by drop ball

Conditions at the blasting site usually determine the method used in secondary breaking. Secondary blasting is the most common method in open pits and quarries where the danger of rock scatter is not excessive and the use of explosives is relatively limited. In situations where loading can not be interrupted or blasting is not possible, splitting or impact hammers are used. Large boulders are usually removed from the loading area and broken separately.

Boulders caused from blasting have been subjected to stress and very high force , and are therefore easily broken by blasting, compared to natural boulders. Blasted boulders often have through cracks that facilities follow-up treatment.

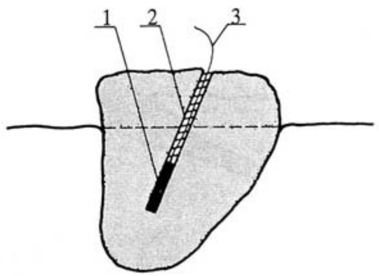
Secondary breaking by blasting

Oversized boulders can be broken for easier handling by:

- Pop shooting
- Plaster shooting

POP SHOOTING

Large boulders can usually be broken by pop shooting (**FIGURE** *4.2.-37.*) . A hole is drilled through the center of gravity of the boulder to the depth of two thirds of its height. Larger boulders may require more than one hole, as one hole should be drilled per every 1 - 1.5 of cross-section at the widest spot on the boulder.



1 Charge
2 Stemming
3 Firing cord and detonator

FIGURE *4.2.-37. Pop shooting.*

The holes are then charged with dynamite or emulsion cartridges using ***Table 4.2.-3.*** for completely visible, half-buried boulders. The explosive is then fired by a detonator and safety fuse, or an electric instantaneous cap.

If several drill holes are used in a boulder, initiation must be carried out through instantaneous detonators. When caution is critical, it is safer to use several drill holes with smaller charges. The charges should be properly stemmed with sand or drill cuttings. Thorough covering is essential in built-up areas. Outside built-up areas, (for example, when boulders are being blasted in a quarry) higher charge values than those shown in the table can be used if sufficient evacuation and supervision are observed.

Table 4.2.-3. *Determining charges for pop shooting.*

Boulder type	***Specific charging Dynamite***	***(kg/m³) Emulsion cartridge***
Completely visible	0.02-0.1	0.08-0.15
Half buried	0.1-0.15	0.15-0.2
Completely buried	0.15-0.2	0.2-0.25

Plaster shooting

Plaster shooting is a contact method of blasting whereby the explosive is placed on top of the boulder, embedded in a thin layer of mud, and covered with at least another 0.1m of mud (**FIGURE** *4.2.-38.*).

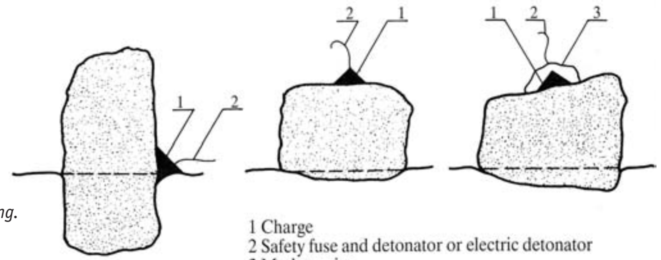


FIGURE *4.2.-38. Plaster shooting.*

1 Charge
2 Safety fuse and detonator or electric detonator
3 Mud capping

The mud reduces both the amount of explosives required and the noise. The recommended amount of charge based on the dynamite is approximately 0.7kg/m³. The charges are usually initiated with instantaneous detonators, especially when firing with bench blasts.

The advantages of this method are the small quantity of explosives required - the goal being to break the rock to the required size without undue scattering - and the large number of blocks that can be fired simultaneously. Other advantages of this method include no drilling, little fly rock is produced and the job can be quickly executed. However, it requires about four times more explosive than pop shooting and is restricted in many areas because of the noise and air blast produced.

Table 4.2.-4. *Determining charges for plaster shooting.*

Plaster weight,kg	***Largest boulder size Stemmed***	***(m³) Not stemmed***
0.3	0.4	0.6
0.5	0.8	1.0
0.8	1.3	1.6
1.0	1.5	2.0

Plaster charges can only be used far away from buildings. There have been cases where the effects of this type of shot has been felt up to 1 km away due to the air pressure wave and noise.

FIGURE *4.2.-39. A hydraulic breaker mounted on excavator boom.*

Secondary breaking by hammer

Very often the most economic way to handle oversized boulders is to break them with a hydraulic hammer, which is fast, cost efficient, safe and does not disturb other activities at the site. Existing excavators and hammers can also be used for other applications.

Stationary pedestal booms equipped with hammers are used extensively on crushers for breaking oversized boulders and clearing material for smooth and continuos feed flow. Overall quarry productivity is dependent on the proper integration of primary and secondary crusher circuits. With a stationary hammer, it takes only a matter of seconds to clear a crusher jam. Drilling and blasting or hoisting a boulder is difficult, dangerous, time consuming and expensive.

Choice of equipment

Productivity rates for breaking oversized boulders are directly related to the rock's mechanical properties, the size and power of the hammer-carrier combination, and the operator's skill and technique.

Weak and fractured rock is easy to break with a hammer. This type of rock appears in numerous mining applications and requires a small to mid-size hammer.

Construction grade material generally consists of relatively hard, tough and compact rock. The carrier's ability to handle a boulder also determines the productivity of the hammer-carrier combination in secondary breaking. Boulders are often in the worst possible position, and effective boulder breaking requires them to be turned to a suitable position.

If a carrier is selected to handle the largest boulder size, the RAMMER hammer fitting on this carrier will be powerful enough for those .

In extremely tough rock, a larger hammer than the minimum required size should be used. Usually the smallest hammer able to break a certain boulder is not the most cost-efficient, and overall productivity remains insufficient as a result.

PRO version hammers with adjustable impact energy are recommended when boulder size and hardness varies considerably. Breaking small boulders with a large hammer is effective, but can be tough on the hammer. With the PRO version, it is possible to work in a low-impact energy mode which saves the hammer. To break large boulders easily, the impact energy is adjusted to the boost mode.

Boulders should usually be broken with a blunt tool. This is due to the nature of the material to be broken; if soft and brittle, it would be quarried with a large hammer and there would be no need for secondary breaking. Hard and tough material is often abrasive, and blunt tools are designed to be abrasive resistant. Secondary breaking could be done with chisels or moils but wear is higher and productivity lower.

For extremely abrasive conditions (repeat: only for extremely abrasive conditions), a superblunt tool is recommended.

Selecting a stationary boom and hammer for a crusher depends on the size, hardness and existence of boulders. A variety of tailor-made booms for crusher plants are available from Rammer.

Working methods

There are several methods to deal with oversized boulders in a quarry, either on site, at a storage site, on a grizzly feeder or a crusher.

Usually oversized boulders do not appear in high numbers. The hammer can, therefore, be effectively used for breaking boulders and loosening muck. The hammer first works on the blasted pile and breaks all visible boulders on top, after which it spreads the blasted pile and breaks oversized boulders as they are uncovered.

This is an effective way to work: a blasted muckpile is often so compact that a wheel loader has problems removing the muck from the pile. If it is loosened with a hammer, a wheel loader can easily transport it to a nearby crusher. In small quarries this is an economic method because it requires both a smaller investment and less work force. The alternative would be an excavator and hammer to break the oversized boulders, a dump truck for transporting and another excavator for loading the dump truck (3 expensive machines + 3 operators compared to 2+2).

If there are only a few oversized boulders, they can be stocked in a separate space and be broken once a week or once a month. This releases hammer capacity for other needs and is an effective way to work rather than changing the hammer and bucket every two hours. The hammer may also be fitted on a wheel-loader for better mobility in the quarry. A suitable wheel-loader can carry the hammer easily.

Repositioning the tool

In a test run, large amounts of boulders were broken when the hammer was positioned in the center of the boulder. Breaking was most effective during the first 6 seconds and at 30 seconds, the breaking results were reduced to a almost zero. The hammer wasn't breaking the boulders, but actually drilling a hole with the same diameter as the tool.

The test was repeated with the tool in another position on boulders that remained intact for 30 seconds. The curve was similar to the first one: breaking was again most effective during the first 6 seconds. 45% of the remaining boulders were broken.

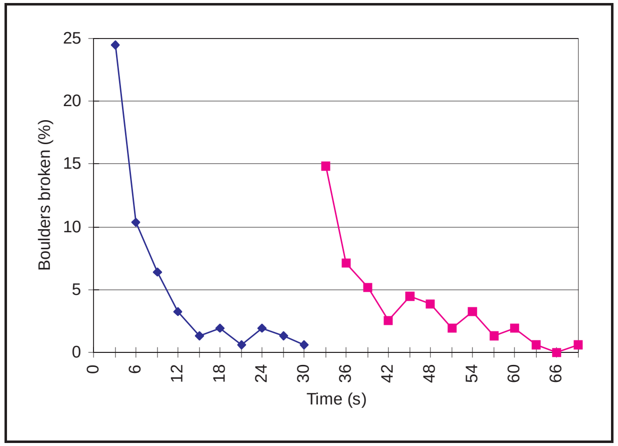


FIGURE *4.2.-40. Boulder fracture time, when tool is held in the same spot for 30 seconds. After 30 s the tool is repositioned and productivity increases dramatically.*

The most effective way to break boulders would be to use the hammer for 5 seconds, after which the tool should be repositioned.

This is a test anybody can and should verify. In secondary breaking, a hammer working for longer than 15 seconds without repositioning is a waste of time and energy. It also causes needless wear on the equipment because all energy not used to break the rock breaks the equipment (hammer and excavator) instead.

DRILLING TOOLS SELECTION

The main hole sizes in bench drilling with the Top Hammer vary from ø 51 - 127mm, which means that drilling tools also vary considerably in size to achieve optimal drilling. Main thread sizes are R32, T38, T45 and T51.

Shank adapter

To withstand impact and correctly transmit it from the rock drill, the shank adapter must be highly accurate. An inferior shank adapter can easily break and cause severe damage to the drill. Shank adapters normally have a long slim section between the hammer and thread in order to withstand stress and avoid breaking.

MF rods

The MF rod is the first choice. It eliminates the need for coupling sleeves and is specially tailored for mechanized rod handling. Energy transmission is also improved, resulting in a longer service life for the entire drill string. Regarding hole straightness, MF rods have proven to be an excellent solution for hole straightness because the connection between the rods is more rigid (**FIGURE** *4.2.-41.)*.

FIGURE *4.2.-41. MF-rods.*

Rods and couplings

Rods and couplings are an alternative to MF rods, and are less sensitive to careless handling such as drilling with loose joints because the threads are HF hardened and not carburized like MF rods.

Bits

Bits are available in various designs of cemented carbide buttons, spherical or ballistic, and various grades such as MP45 or DP65.
Spherical buttons are used in abrasive and hard rock and ballistic buttons are recommended for tough and medium abrasive rock. The MP45 is the first choice of carbide grade. The DP65 is a grade that offers higher wear resistance and can better withstand excess drilling.

Normal bit. The all-round bit with a flat face for normal rock, such as gneiss and granite.

Drop-center bit. Has a concave face and provides similar performance to the normal bit. The drop center bit has the advantage in rock with high diameter wear resulting in dome-shape wear on a flat face bit.

Heavy-duty bit. A flat-face bit recommended for very abrasive rock.

FIGURE ***4.2.-42.*** *Selection of bits.*

Retrac bit. Featuring splines to stabilise the drill bit for a straighter hole, and cutting edges at the rear to help it drill in reverse if the drill string gets stuck. Available in both flat and concave face.

Guide bit. A patent bit specially designed by Sandvik to obtain the greatest possible hole straightness. Chisel-type periphery buttons together with a long guide skirt enable the bit to produce straight holes, even in rock formations that are extremely difficult to drill.

Straight Holes

The first step in improving hole straightness is to use a ***DC Retrac bit***. This bit has a long body, the diameter of which is only slightly smaller than that of the bit head. The large diameter of the body gives the bit good guidance, while the longer body has cutting edges at the rear to help it drill in reverse if the drill string gets stuck. By using a DC Retrac bit with ballistic buttons, if permitted by the rock formation, hole deviation can be further reduced thanks to the shape of the buttons. The concave face design avoids hole deviation caused by the dome-shape of a worn flat-face bit.

Drilling precision can be further improved by using a ***guide tube*** as the first rod in a normal drill string. The guide tube has almost the same diameter as the drill bit, which results in a stiffer string that is less likely to deviate. This method offers the same straightness as DTH drilling.

To really optimize the entire drill string for long-hole precision drilling, a ***guide bit*** should be at the front of the guide tube.
The guide bit features chisel shaped buttons along the peri-phery and spherical buttons in the core area, as well as a concave face design. For optimum guidance in the hole, the guide bit also features long skirt wings along the body, separated by wide grooves to allow free and easy passage of flushing air and cuttings.

4.2.2. Primary Breaking by Hydraulic Hammer

Primary breaking with hydraulic hammers is productive and economical especially in cases where explosives are difficult or impossible to use.

Economic productivity is achieved in fractured rock. Hammer per-hour productivity does not necessarily exceed that of the traditional drill and blast process. Primary breaking by hydraulic hammer is characterized by:

- Lower investment costs
- No need for stationary boom on crusher
- Reduced need for skilled workforce
- Good job-site control
- No secondary breaking necessary
- Good job-site safety (due to lack of explosives)
- Low ground vibration levels compared to blasting
- Low noise, especially with the silenced CITY-models
- Selective excavating of valuables
- Improved size distribution of excavated aggregates (=less dust)
- Less disruption of surrounding work areas

Equipment selection

Suitable hammers generally belong to the over 2000 kg weight class, preferably over 3500 kg. Small hammers (below 1000 kg) are used in special cases.

Primary breaking does not usually require exceptional carrier-hammer agility, so carrier weight does not necessarily have to be at the top of the list.

Utilization of a large hammer in primary breaking is relatively high, at 2,000 hours and more a year. To protect both the environment and the operator from noise, silenced CITY housing is strongly recommended. Working in a silenced housing reduces operator fatigue and therefore increases overall productivity.

Tool selection is an important productivity factor in primary breaking. A wedge is used as the breaking device, so wedge sharpness is essential for achieving a good penetration rate. Sharpening the chisel is recommended in abrasive conditions. If production and tool penetration are relatively high and the rock is not too abrasive, a soft-rock chisel is the best tool. If the tool's penetration rate is low and the rock is abrasive, a hard-rock chisel is recommended. Occasionally a chisel tool will twist and cause accelerated wear to the retainer pin groove, especially in layered rock applications. Using a moil-point tool prevents this from happening.

Working methods

The usual way to use a hammer is to have the excavator located under the bench (**FIGURE** *4.2.-43.*). Bench height should be as high as can be excavated in one phase (3-5 m). Extremely high benches may collapse. Low benches (< 1 m) are difficult to work, and the root section of a low bench is often the most difficult area to excavate (**FIGURE** *4.2.-43.*). Working on the top of high benches is dangerous, with the severe risk of the bench collapsing.

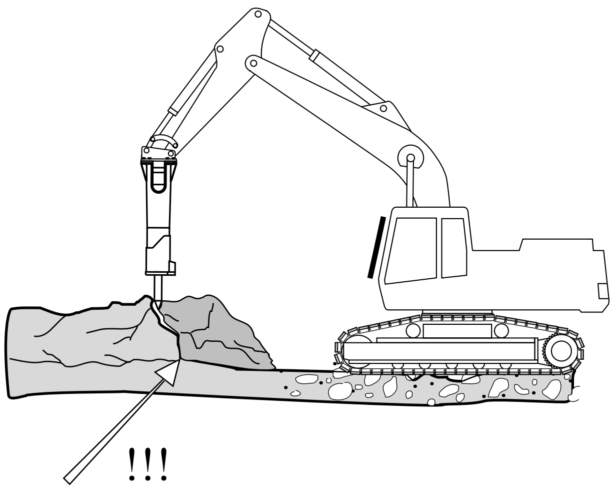


FIGURE ***4.2.-43.*** *The root of the bench (arrow) is the most difficult part of primary breaking with a hammer.*

With minor modifications, most Rammer hammers are suitable for underwater breaking. Without these modifications, the excavator's hammer and hydraulic system will be damaged. In salt water applications, special care is needed to avoid excessive corrosion to steel parts. In underwater applications, the excavator and hammer can operate from various positions such as the shore, specially constructed dam, or mounted on a pontoon or raft. The main advantage here is the elimination of explosives. Underwater hammer work is especially demanding to the operator, as he must work without being able to see what he's working on. We strongly recommend the use of biodegradable hydraulic fluids and greases for underwater applications.

Selective excavating avoids mixing overburden and valuable material during primary breaking. This benefit makes hammer utilization in primary breaking attractive for mining applications, especially where there are relatively thin seams of a rich ore.

Although large hammers are mainly used for primary breaking, smaller hammers are used in certain soft-rock applications, such as water pipe channels, sewage system excavation, ditches etc. Another advantage of the hammer is its mobility and easy hammer-to-bucket change. For small jobs in municipal areas, low productivity is less of a disadvantage than obtaining blasting permits etc.

Working with a chisel

Chisel positioning inside the bench is optimized according to tool, rock properties and hammer size. At this stage, the experience and motivation of the operator is crucial for maximum productivity.

The wedged tip of the chisel usually penetrates very fast, approximately 2 - 5 seconds. Wedge penetration rate decreases considerably upon penetration. If a large piece of rock is broken away, penetration increases again. However, this is usually not the case.
Ordinarily, the penetration rate decreases the deeper the wedge penetrates the rock and in the worst case can stop entirely. This decrease in speed is surprisingly rapid. In a typical situation, the next 30-cm of penetration after the wedge has penetrated the surface of the rock takes approximately 10 - 15 seconds. From there on, each 10 cm of penetration takes anywhere from 15 seconds or more. This can be explained by surface friction forces on the tool, and impact energy which is transferred from the sides of the tool to the ground. As a result, barely any energy is left for breaking the rock.

It is therefore up to the operator to decide when and where to reposition the tool (this is where operator experience comes in). A proper working technique for maximum efficiency seldom keeps the hammer in the same place for more than 45 seconds. The operator should avoid long hammering sequences without visible penetration.

In practice it has been shown that a tool which is shortened by 20 cm may, surprisingly enough, in some cases increase productivity. It forces the operator to reposition the tool instead of trying in vain to penetrate the last 20 cm of the tool.

FIGURE *4.2.-44. Rammer G100 hydraulic hammer at granite quarry.*

FIGURE *4.2.-45. Technobeton Spa of Pozzuoli, stationed near Naples in Italy and owns a basalt quarry in the province of Caserta. In 1987 the company decided to replace drill-and-blast by Rammer hammers at its quarry in Sulo Terme. Two S 86 Rammer hydraulic hammers were mounted on 40-ton excavators for production. The output of the two machines working 8 hours per day was sufficient to feed the crushing plant: 1500 cubic metres. The material has an average compressive strength of 250 Mpa and after crushing in an impact crusher it is used as ballast on the high-speed railway line under construction between Rome and Naples.*

FIGURE *4.3.-1. A typical quarry. (Note: The photo and following data are not interrelated.)*

4.3. QUARRY CASE

Production 3.2 million tons of hard granite from quarry.
Drilling with four CHA 1100 C hydraulic crawlers.
Mean fragment size k_{50L} = 240 mm.
Secondary breaking with a Rammer G80.
Loading with a 12 cum excavator.
Hauling with seven 50 ton dump trucks; hauling distance 2 kilometers.
Crushing in three stages with stationalry crushing plant.

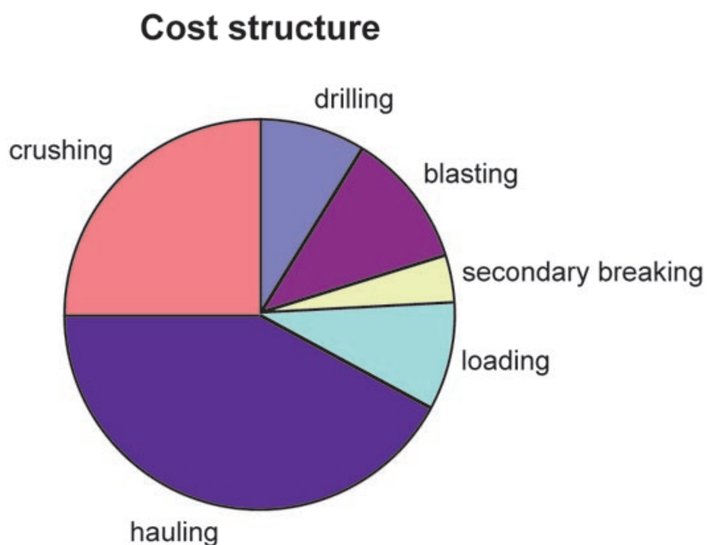


FIGURE *4.3.-2. Cost structure of above mentioned quarry case.*

5.1. GENERAL

To the construction projects such as:

- Road and highway construction
- Pipeline construction
- Rock fill dam construction
- Foundation preparation and ground leveling
- Demolition

several key basic issues apply.

Smooth and stable rock walls where presplitting or smooth blasting techniques are required for safety and landscaping. Preventing flyrock and air/ground vibrations in urban areas is best achieved through a high drilling accuracy and moderately small drill holes.
Channeling via slot drilling techniques can be used to isolate buildings from ground vibrations .

Selected drill hole sizes are typically fairly small in order to match the drill pattern area to the cut width, for example, in road, trench and pipeline construction.
Crushed rock aggregate transportation is reduced by using mobile crushers or relocating the crushing plant to minimize transportation costs.

FIGURE 5.1.-1. *A typical construction project- Tve Mun Highway expansion in Hong Kong.*

FIGURE 5.1.-2. *Foundation preparation by TAMROCK Commando 100.*

5.2. METHODS

5.2.1 Drill and Blast Excavation

GENERAL

In rock foundation bench blasting outside built-up areas (excluding construction sites), the objective is to efficiently utilize drillholes with concentrated bottom charges. If particularly favorable fragmentation is desired, the column charge can be increased considerably.

Construction site methods consist of controlled blasting with closer-spaced drilling used together with light charging. In cautious blasting within a built-up area, attention must be paid to surrounding buildings, facilities and installations and, most importantly, people in the vicinity.

Blasting must be planned so that it does not damage buildings through ground vibrations or stone throw. Air shock waves also imply risks. Flyrock is an obviously dangerous hazard as far as risk of injuries is concerned.
There are two methods of controlled blasting. Selecting of the right one depends mainly on

rock characteristics and on how feasible the technique is under the prevailing conditions. The two methods are:

- Smooth blasting
- Presplitting

Presplit blasting differs from smooth blasting in that it is performed before the primary blast is detonated; smooth blasting usually occurs after the main field has been blasted. Presplitting also involves shorter hole spacing and heavier charging. In the latter, however, the result depends to a great extent on the spacing/burden ratio.

PRESPLIT BLASTING

Presplit holes are usually 51 - 102 mm (2" to 4") in diameter, usually loaded with ø 17 - 32 mm explosive charges, and fired before any of the adjoining main excavation are blasted (**FIGURE** *5.2.-1.*). The annulus around the small diameter powder cushions the explosive shock wave, reducing the crushing and radial cracking of the rock around the drillhole (**FIGURE** *5.2.-2.*).

Where conditions require firing pre-split holes in advance of primary holes, presplit can be accomplished by delaying the primary holes to allow the preshear holes to fire first (**FIGURE** *5.2.-3.*).

As presplitting requires fairly small spacing values, the values shown in ***Table 5.2.-1.*** can be used in good circumstances, such as solid, hard rocks.

Obviously, smaller spacing values give better results. The holes nearest to the presplit line, called the buffer holes, must be drilled at a distance half the spacing of production holes. These holes are charged normally(**FIGURE** *5.2.-4.*).

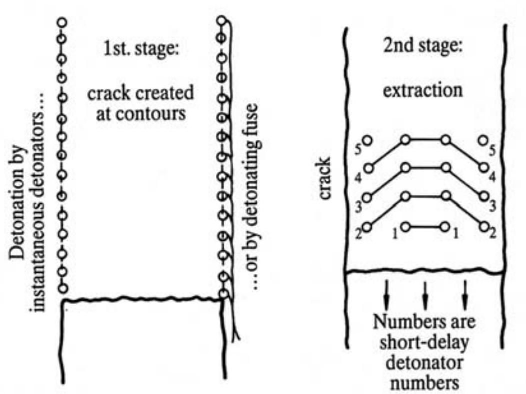


FIGURE ***5.2.-1.*** *Blasting presplit holes before bulk holes are drilled.*

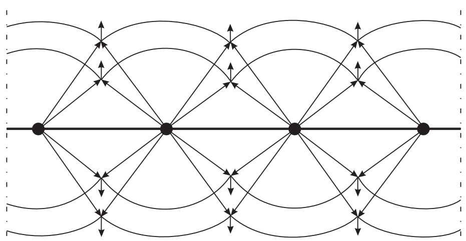


FIGURE ***5.2.-2.*** *Presplitting is based on shock waves from simultaneously detonated charges in adjoining blastholes that collide, causing tension in the rock which forms a crack in the web between the holes.*

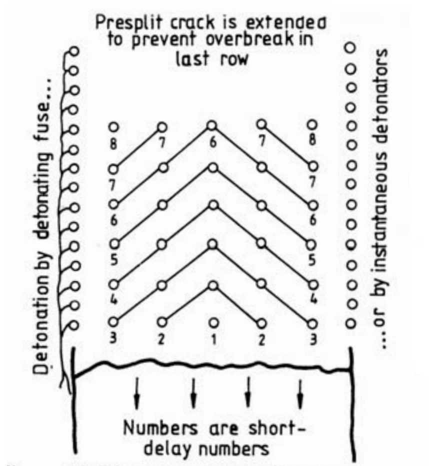


FIGURE ***5.2.-3.*** *Blasting presplit holes before bulk holes in a shot.*

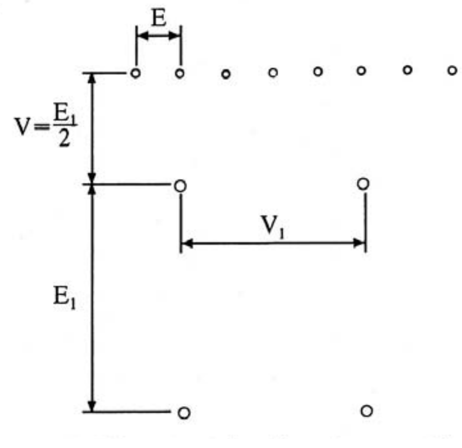


FIGURE ***5.2.-4.*** *Setting presplit line and buffer holes in presplit blasting.*

Table 5.2.-1. *Drilling and charging plans for presplit blasting.*

Hole diameter		*Charge*	*Pipe charge*		*Hole spacing*	*Specific drilling*	*Cracked zone*
mm	*in*	*kg/m dyn.*	*ø mm*	*lenght mm*	*m*	*drm/m²*	*m*
32	1 1/4	0.13	22	380	0.45-0.7	2.22-1.43	0.4
32	1 1/4	0.21	25	1140	0.45-0.7	2.22-1.43	0.4
38	1 1/2	0.21	22	380	0.45-0.7	2.22-1.43	0.4
38	1 1/2	0.21	25	1140	0.45-0.7	2.22-1.43	0.4
51	2	0.38	32	1000	0.5-0.8	2.00-1.25	0.5
51	2	0.47	32	380	0.5-0.8	2.00-1.25	0.6
64	2 1/2	0.38	32	1000	0.5-0.7	2.00-1.43	0.5
64	2 1/2	0.47	32	380	0.5-0.7	2.00-1.43	0.6
64	2 1/2	0.55	25	1000*	0.7-0.9	1.43-1.11	0.9
76	3	0.55	25	1000*	0.7-0.9	1.43-1.11	0.9
76	3	0.71	40	380	0.6-0.9	1.67-1.11	1.1
89	3 1/2	0.90	32	1000*	0.7-1.0	1.43-1.00	1.7
89	3 1/2	1.32	50	380	0.8-1.1	1.25-0.91	1.9
102	4	0.90	32	1000*	0.7-1.0	1.43-1.00	1.7
102	4	1.32	50	380	0.8-1.1	1.25-0.91	1.9

In a very soft, weathered formation, it is occasionally necessary to reduce both hole spacing and column load. In extreme conditions, spacing may be as short as 0.3m with the detonating cord forming the entire explosive charge. In weak and soft formations, results may be improved by using guide or relief holes (unloaded holes located between loaded holes) to promote fracturing along the desired plane and in the desired direction. For example, guide holes to direct cracking are recommended in corner shots (**FIGURE** *5.2.-5.*) to reduce backbreak, but do not normally permit increased spacings between loaded holes. In fact they probably decrease the distance because unloaded holes often terminate the crack.

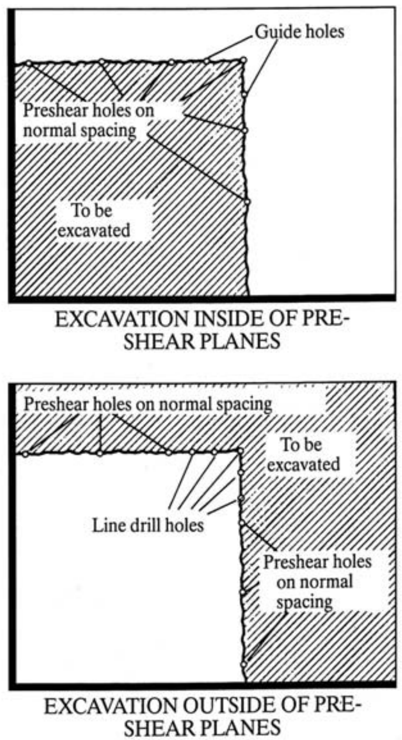


FIGURE ***5.2.-5.*** *Using guide holes in presplit blasting.*

SMOOTH BLASTING

In smooth blasting, a single row of holes is drilled along the neat excavation line, loaded with light, well-distributed charges, and fired either together with the bulk holes or after (**FIGURE** *5.2.-6.*).

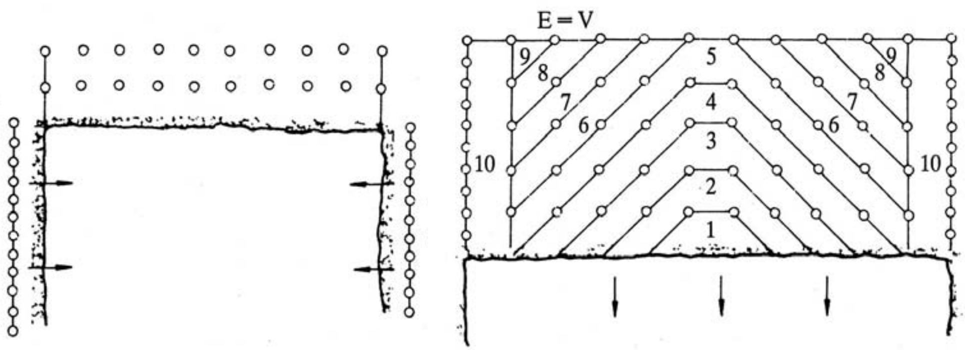


FIGURE ***5.2.-6.*** *Smooth blasting of contours (a) after and together with bulk holes.*

Successful smooth blasting requires accurately drilled contour holes. Remember that the result is mainly determined by the drilling. ***Table 5.2.-2.*** shows drilling and charging plans for smooth blasting based on experience from various countries. The spacing values in the table are maximum values for good results. Whenever hole spacing is decreased, the corresponding burden value must also be increased to keep E•V and the specific charge constant. Hole spacing in the contour line depends mainly on rock tenacity, hardness and solidity.

When the contour is blasted after the bulk holes, spacing and burden can also be calculated by using formula:

$$E = 15...20d$$

where E = Spacing (mm)

d = Hole diameter (mm) and

$$V = 1.25E$$

where V = Burden (mm)

Table 5.2.-2. *Drilling and charging plans for smooth blasting.*

Hole diameter mm	in	*Charge* kg/m dyn.	*Pipe charge* ø mm	lenght mm	*Hole spacing* m	*Burden*	*Cracked zone* m
32	1 1/4	0.21	22	380	0.6-0.8	0.9-1.1	0.4
32	1 1/4	0.21	25	1140	0.6-0.8	0.9-1.1	0.4
38	1 1/2	0.21	22	380	0.6-0.8	0.9-1.1	0.4
38	1 1/2	0.21	25	1140	0.6-0.8	0.9-1.1	0.4
51	2	0.38	32	1000	0.7-1.0	0.9-1.3	0.5
51	2	0.47	32	380	0.8-1.0	1.0-1.4	0.6
64	2 1/2	0.38	32	1000	0.7-1.0	0.9-1.3	0.5
64	2 1/2	0.47	32	380	0.8-1.0	1.0-1.4	0.6
64	2 1/2	0.55	25	1000*	1.0-1.3	1.2-1.6	0.9
76	3	0.55	25	1000*	1.0-1.3	1.2-1.6	0.9
76	3	0.71	40	380	1.0-1.3	1.3-1.7	1.1
89	3 1/2	0.90	32	1000*	1.2-1.4	1.7-1.9	1.7
89	3 1/2	1.32	50	380	1.3-1.5	1.8-2.0	1.9
102	4	0.90	32	1000*	1.2-1.4	1.7-1.9	1.7
102	4	1.32	50	380	1.3-1.5	1.8-2.0	1.9

Table 5.2.-3. illustrates the lowest charge per meter (kg/m) for different hole diameters in smooth blasting. Best results are usually obtained if the charging is brought as close as possible to the collar of the hole.

Table 5.2.-3. Minimum charge per meter of hole in smooth blasting.

Hole diameter mm	in	Minimum charge kg/m	lbs/ft
32	1 1/4	0.10	0.07
38	1 1/2	0.15	0.10
45	1 3/4	0.20	0.14
51	2	0.25	0.17
64	2 1/2	0.40	0.27
76	3	0.55	0.37
89	3 1/2	0.70	0.47
102	4	0.90	0.60

Stemming is only required when charges tend to fly out of the holes during initiation, which occasionally, happens when firing is not ins-tantaneous. Usually contour holes are blasted simultaneously, especially when blasted together with bulk holes. In order to minimize delays between adjacent holes, initiation by firing cord is often the best operation. In theory, new electrical detonators provide the best result. Short-delay electric detonators can also be successively used. Drilling and charging for smooth blasting should be planned considering the following:

1. Choose a suitable diameter for the job
2. After considering the depth of cracking allowed beyond the boundaries of excavation, choose a suitable explosive
3. Calculate burden and spacing with either tables or equations given
4. Check that selected charging is sufficient
5. Plan firing to give enough room for exploding rock if firing contour holes with bulk holes
6. Fire contour holes instantaneously with detonating cord very short delay of approximately 1 ms with electronic caps if possible, paying attention to ground vibrations.

LINE DRILLING

Line drilling involves a single row of closely spaced, unloaded, small diameter holes along the neat excavation line. This provides a weak plane that the primary blast can break. It also causes some of the shock waves created by the blast to be reflected, which reduces shattering and stressing in the finished wall. Line drill holes are generally 51 - 76mm (2" - 3") in diameter and are spaced at two to four times the hole diameter apart along the excavation line. Holes larger than 76mm are seldom used in line drilling since higher drilling costs can not be offset by increased spacing. Blast holes directly adjacent to the line drill holes are generally loaded lighter and are more closely spaced than the other holes. The distance between the line drill holes and the directly adjacent blast holes is usually 50 to 75 per cent of the normal burden.

A common practice is to reduce the spacings of the adjacent blast holes by the same amount with a 50 per cent reduction in explosive load. The explosives should be well distributed in the hole using decks and detonating cord downlines.
Line drilling is best suited to homogenous formations where bedding planes, joints and seams are at a minimum. **FIGURE** *5.2.-7* shows a typical pattern and procedure for line drilling in open work.

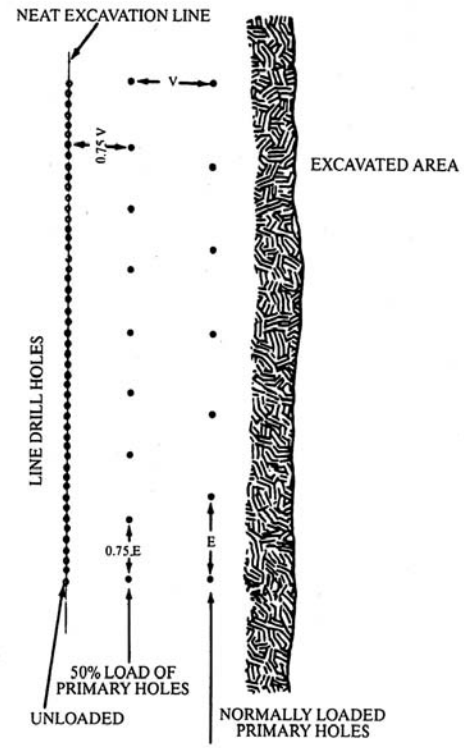


FIGURE *5.2.-7. Typical pattern and procedure for line drilling.*

Line drilling is applicable in areas where even light explosive loads associated with other controlled blasting techniques may cause damage beyond the excavation limit. When used between the loaded holes, line drilling promotes shearing and gives improved results. When ground vibrations must be minimized, initiation with short-delay detonators can be arranged so that the first five holes, for example, are initiated with cap No.1, the next with No.2 etc; or alternatively using successive cap numbers in successive holes.

In cushion blasting, a single row is drilled along the perimeter of the excavation. Drillhole size 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. (**FIGURE** *5.2.-8.*)

Stemming cushions the shock of the explosive to the remaining rock, minimizing cracking and tension. Charges in cushion blasting should be fired with no delay, or minimum delay, between the holes. Detonating cord or electronic detonators are the best means of initiation as noise or airblast do not cause any problem. The unified charge force cuts the web between the holes, forming a smooth rock suface.

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 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 at the bottom part of the excavation, and to promote shearing between the holes, the charge concentration must be increased at the bottom of the hole. Drilling precision is important. Deviation of more than 15 cm of the theoretical plane tends to give poor result.

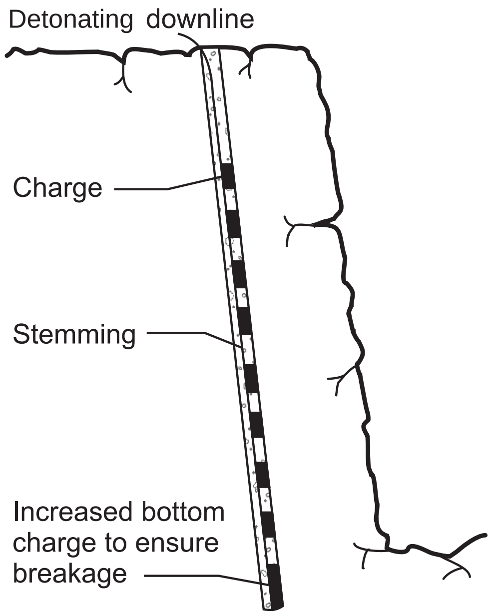


FIGURE *5.2.-8. Cushion blasting.*

When cushion blasting is used in 90° corners, it may be combined with presplitting for better result.

Table 5.2.-4 *Proposed charging and drilling pattern for cushion blasting:*

Perimeter hole diameter mm	***Charge concentration Emulite or Dynamex kg/m***	***Burden m***	***Spacing m***
50 - 64	0.12 - 0.35	1.20	0.90
75 - 89	0.20 - 0.70	1.50	1.20
102 - 114	0.35 - 1.10	1.80	1.50
127 - 140	1.10 - 1.50	2.10	1.80
152 - 165	1.50 - 2.20	2.70	2.10

Advantages:
+ Increased spacing between the holes, less drilling
+ Functions reasonably well in incompetent rock formations

Disadvantages:
- Necessary to excavate main blast before firing cushion blast
- Difficult to cut 90° corners without combining another method, such as presplitting

CHARGING EXPLOSIVES

Controlled blasting is always carried out with special explosive that faci-litate low charge concentrations in blastholes. The closer the site is situated to housing and urban areas, the more attention should be paid to correct explosive selection. The principle in controlled blasting is to create a sufficiently wide crack between the holes with the lowest possible charge concentration (kg/m) to facilitate rock cutting. Different pressure levels against the hole wall (**FIGURE** *5.2.-9. and* **FIGURE** *5.2.-10.*).

The lower the produced pressure is, the smoother and more solid the rock face will be.

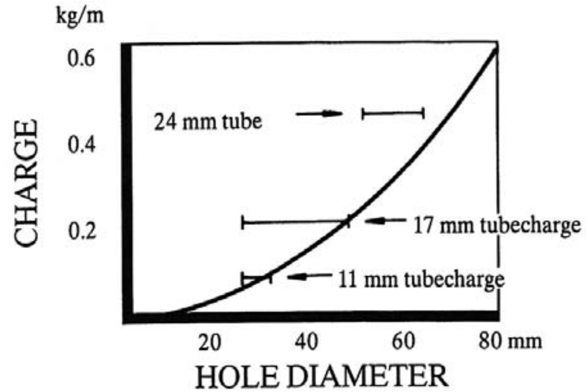


FIGURE *5.2.-10. Lowest charge per meter of blasthole for different hole diameters in controlled blasting.*

The correct level of charging is normally determined by a trial blast. It is good practice to use lighter charging in the row next to the contour row, and to adjust the design according to the result of the trial test. The design is correct once a clear crack is produced on the contour line. Stemming is recommended only if tube charges tend to fly out of the holes upon detonation. Field experience indicates that the best results are achieved when column charges reach the collar of the hole. It should be noted, however, that in urban areas this might increase the risk of flyrock and scattering of small particles.

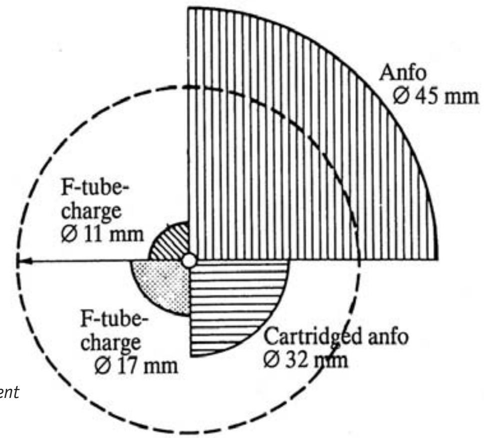


FIGURE *5.2.-9. Depth of crushed zone in rock with different explosive types of when detonated in a ø 45mm blasthole.*

The cut should be covered to avoid this.

Standard cartridge products can also be used for controlled blasting, if the cartridges are:

- Cut in two and tied to a wooden stick with suitable spacings and then lowered into the hole; detonating cord is used to ensure that all cartridges explode
- Cut in two and tied to detonating cord, then lowered intothe hole. The advantages of standard tube charges are precise charge concentration and easy hole charging.

A bottom charge should be applied to ensure that the rock is cut along the bottom line of the contour ho-les. This normally consists of explosive cartridges, and disturbs the rock more than, for example, tube charges. However, it is advantageous in bottom cutting. Without a bottom charge the spacings along the contour line would have to be reduced considerably.

In contour blasting, firing delays between adjacent holes should be as short as possible. Detonating cord can be highly recommended due to its high velocity (6500 - 7000m/s) in pre-splitting and smooth blasting, because it facilitates the simultaneous firing of different holes.

New electronic short-delay detonators (1ms) will be the best system. In urban areas, the risk of ground vibration exists, especially if a longer contour is fired with same timings using detonating cord.
When delay caps are used, slight delays between holes may exist even if the same cap numbers are used. This is due to the dispersion of initiation items as the charges explode at very small delay intervals. It is difficult to manufacture delay caps without interval dispersion. These intervals increase with the cap number.

PLANNING CONTROLLED BLASTING

Controlled blasting should be planned with the following procedures in mind:

1. Choose suitable hole diameter for job regarding available drilling equipment and explosives
2. Use ***Tables 5.2.-1. – 5.2.-4.*** to plan drilling patterns
3. Determine maximum allowable cracking depth beyond contour line, and choose correct explosive product to match it
4. Design firing system, keeping in mind that holes should be fired simultaneously as far as possible
5. Holes should be drilled into solid parts of rock, not into cracks. There should be at least two holes in each block to ensure the cutting direction is correct
6. Use proper aligning and collaring procedures to ensure that holes start from correct positions along contour line
7. Use the correct drill steels for the hole diameter to ensure that holes are straight and positioned along the intended contour face.

Hole diameters (ø 38 - 64 mm) used in surface contour blasting are normally larger than those used in tunneling. In smooth blasting, it is therefore often necessary to reduce charging in holes up to approximately 4 - 5m from the contour line.

In presplit blasting, the crack is considered to protect the rock mass beyond the presplit line. However, when the presplit hole diameter exceeds 51mm (2”) lighter charging should be used in the row next to the presplit line when blasting in jointed and fractured solid rock or broken soft rock. Controlled blasting techniques are especially advantageous in deep and narrow cuts such as shipyards, ports and canals where remaining benches are exposed to higher stress. It is also practical to use controlled blasting in finishing road walls and railway cuts where the smoothness of the excavation must withstand particular tolerances.

CAUTIOUS BLASTING NEAR BUILDINGS

When blasting close to buildings and other vibration-sensitive installations, it is not always possible to utilize blastholes in the same way as normal blasting operations. Ground vibrations (which always occur in blasting operations) depend on the maximum cooperating charge weight. Thus, for each delay it must be kept within certain limits for various distances. In big blasts and long distances, the total amount of explosives may determine the size of the vibrations.

Blast constriction is another factor that affects ground vibration size. A constricted charge gives higher vibrations than one with free breakage. The maximum cooperating charge can be determined by knowing the distance to the sensitive object. Charge weight depends on the permitted vibration velocity and the rock transmission factor.

The maximum cooperating charge can be reduced by the:

- ***Firing pattern.*** The number of holes with the same period is reduced so the maximum cooperating charge is not exceeded.
- ***Reduced drilling pattern.*** Blasthole volume is not fully utilized for explosives charge as in normal blasting. The drilling pattern is closer 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, rather divided into several lower benches. A plan of cautious blasting near building is shown in **FIGURE *5.2.-11.***

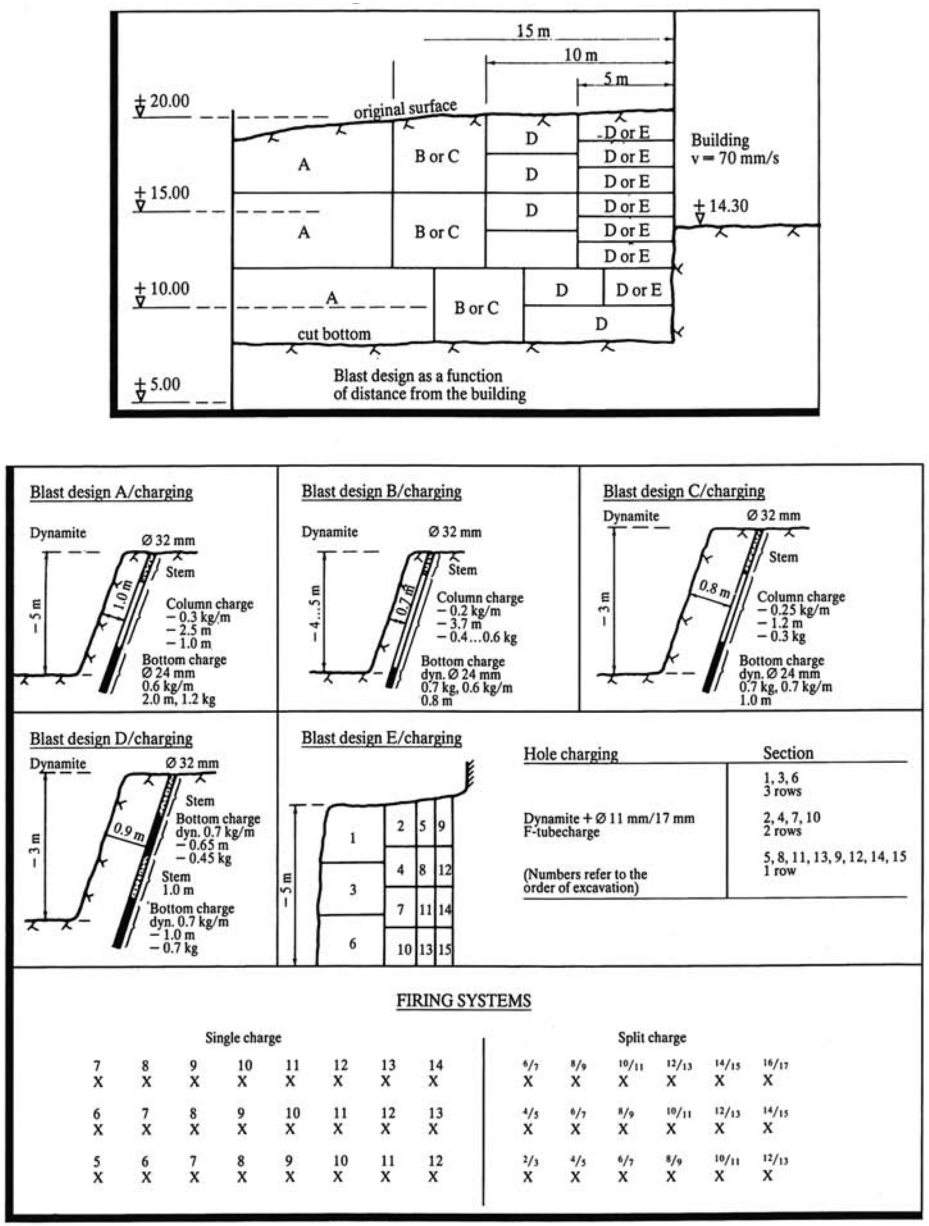


FIGURE *5.2.-11. A plan for rock excavation close to a building.*

CONTROLLED TRENCH BLASTING

Today, two different trench drilling&blasting techniques, conventional and controlled, are used. The conventional method uses staggered drilling patterns and charging in all blastholes. The controlled technique uses holes placed in rectangular patterns with center holes charged more heavily than the side holes. The relative advantages and disadvantages are as follows:

Conventional technique :

+ All holes are charged in the same way
+ Less ground vibration
- Asymmetrical drilling pattern
- Large overbreak

Controlled technique:

+ Symmetrical drilling patterns
+ Less overbreak
- Greater charging in center holes
- Higher ground vibration level

Selecting hole diameter

It is important that hole diameters are correctly chosen as they affect drilling & blasting costs, and also overall trench-making costs. Large diameters reduce specific drilling and the cost of extracting the rock, but increase overbreak while simultaneously raising infilling costs. Pipeline trenches are generally dug with high-performance top-hammer crawler drills because they are mobile and can easily move around the worksite. Trenches with a cut depth in excess of 2.4 m are generally drilled with hole diameters of 51 mm or more, whereas cuts less than 2.4 m deep are drilled with ø 32 - 45 mm holes. Smaller diameters permit close spacing which is of particular advantage in trenches less than 0.6 m wide. Restrictions on hole diameter selection must be taken into account in residential areas or parallel to existing trenches. In both cases, hole diameter selection is affected by general ground vibration restrictions. And in residential areas, the blaster must also pay attention to rock throw and flyrock. Pipeline trench construction parallel to existing line trenches is often performed with small diameter holes (ø 32 - 38mm). When trenches are blasted beyond residential areas where a certain amount of throw and ground vibration can be tolerated, larger diameter blastholes can be used together with larger spacings.

Drillhole placement

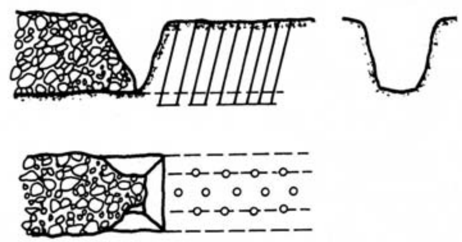


FIGURE *5.2.-12. Staggered hole placing in pipeline trenching with inclined holes.*

Holes for trenching are placed in rows that do not have sloping sides. When blasting smaller trenches, holes should have an inclination of 3:1 to 5:1 from the vertical. It is often more feasible to use vertical holes when cutting long pipeline trenches. Holes are drilled 10% over the trench depth, or at least 0.2m below the intended grade line. A good rule of thumb for subdrilling trenches is one half of the burden distance. Hole spacing and the amount of explosives per hole vary according to the type (open versus tight) and size of the trench, hole placement and initiation sequence for the excavation (**FIGURE** *5.2.-12.)*.

Charging the blasthole

Compared to normal bench blasting, trench blasting requires greater specific charging, which increases as trench depth increases and trench width decreases. However, compared to bench blasting, charging in the column part of the hole is lighter to ensure reduced overbreak beyond the design of the trench. This is nevertheless difficult to completely avoid, even when blasting solid rock formations.
The patterns are based on using dynamite as both bottom and column charge with drilling patterns for ø 32 - 38mm (1 1/4" - 1 1/2") holes.

Firing systems

When designing delay systems for charging patterns, blastholes should be delayed in order to move the rock in a particular direction, for example, away from any area where damage might occur (building or pipe). This direction may also be toward a natural free face.
In trench blasting it is important to minimize overbreak, which causes increased material removal and costly refilling work. Controlled trench blasting offers an easy way to minimize overbreak and cut the rock along the theoretical excavation lines. Controlled trench blasting techniques differ from conventional techniques in the way holes are drilled and loaded. Rectangular drilling patterns are used and the center holes are given a more powerful charging while the side holes are less charged (**FIGURE** *5.2.-13. and* **FIGURE** *5.2.-14.*).

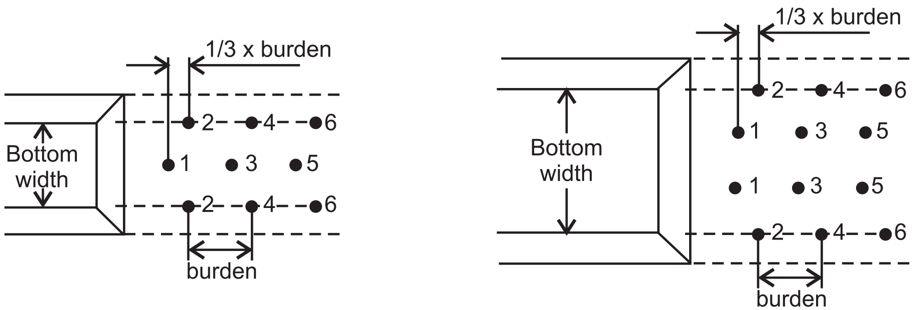


FIGURE ***5.2.-13.*** *Trench drilling, charging and firing (delay 25 ms) by conventional technique.*

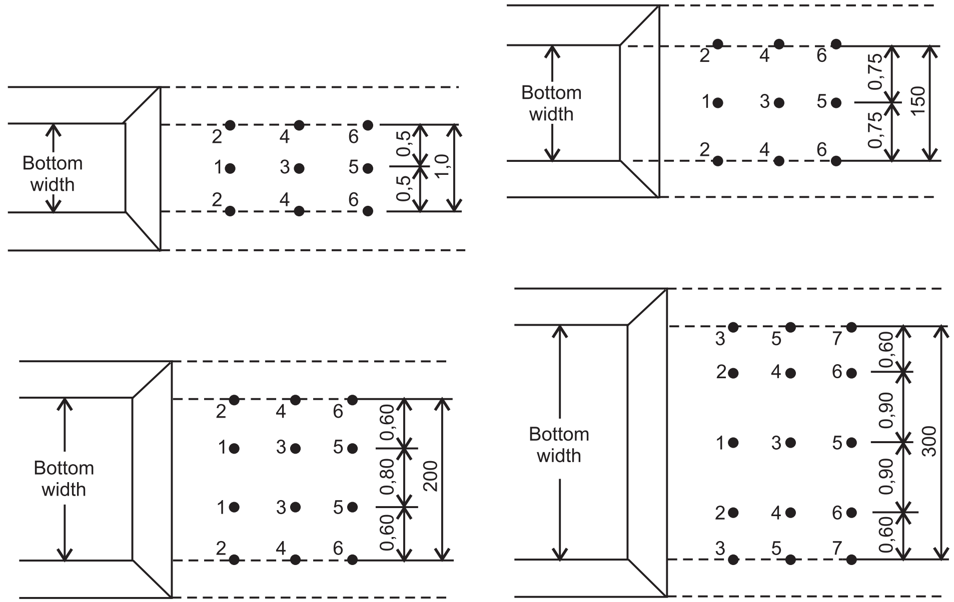


FIGURE ***5.2.-14.*** *Trench drilling, charging and firing (delay 25 ms) by controlled technique.*

UNDERWATER BLASTING

The most common methods presently used in underwater blasting are:

- Drilling and blasting through rockfill
- Drilling and blasting from platform
- Drilling and blasting with divers
- Blasting with concussion shaped charges.

If water is not too deep, it is more economical to make a rockfill over the area to be blasted and drill, and then charge through the rockfill.

Platforms are normally equipped with several percussion or rotary drilling machines, which moves on tracks in four directions for greater flexibility. The platform is supported with legs, however platforms without legs are also used.

FIGURE 5.2.-15. *The Kokkola Channel. The project was to increase the nautical depth of the access channel from 11 meters to 13 meters over a length of 18 km. This involved dredging approximately 2 million m3. Dredging activities began in September 1995 and will be completed at the end of July, 1998. Upon completion, the Kokkola Channel be-come one of Finland´s five deepest channels. The total excavation of granite was 66.500 m^3 and total area to be pre-treated by blasting is 150 000 m^2. Almost 15,000 holes between 3 - 4 meters (most holes 115 mm in diameter) were drilled. Tamrock´s rock drill and a power pack are used in this project on drill mast constructed by the contractor.*

5.2.2. Demolition and Recycling

DEMOLITION BY BLASTING

Blasting buildings

When blasting buildings, the following main principles must be taken into consideration:

- The charges must be sub-divided to provide full breakage.
- Vital supporting members must be destroyed so that the weight of the building itself does most of the work.
- Millisecond initiation is used and interval numbers located to provide the desired direction of fall or breaking up.

There are no general methods for blasting buildings. Each building is a special case, requiring thorough calculations, and specifically planned drill holes and charges. The difference between a building with brick load bearing walls and one with concrete pillars is significant.

Building blasting has advantages compared with conventional demolition work such as:

- Safer when demolishing close to traffic arteries, etc.
- Disturbs surroundings for limited time only
- Generates dust for a limited time only
- Can be controlled
- Permits the use of more rational removal work after blasting

When all the above-mentioned factors are taken into consideration, the method is also more economical. Special structures such as chimneys can be blasted, in principle, in the same way as buildings. It is usually important to adapt the blasting to ensure a certain direction of fall, as it is totally possible.

The general principle is that a portion of the chimney opposite the direction of fall is left intact. This portion should extend over 160 of arc (**FIGURE 5.2.-16.**). The entire shothole drilling should then be done in the remaining 200 of arc facing the direction of fall. After the drilling is completed, openings 76-122 cm wide and 91-152 cm high are made in the arc facing the direction of the fall, leaving the chimney supported on the 160 arc at the rear and on pillars at the front.
The charges must be covered with heavy and splinter-protective covering material.Ground vibration measurements are performed in surrounding buildings or installations. Water must be sprayed during the blasting process to keep down dust. When demolishing buildings, the surrounding area must be evacuated and supervised as carefully as in rock blasting.If surface charges are used, air shock waves must also be measured.

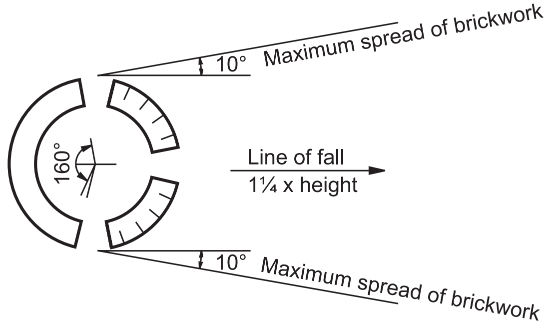


FIGURE 5.2.-16. *Position of pillars in relation to line of fall of a chimney*

Four basic building elements must be severed by the explosion in order to successfully demo-lish a structure: mass concrete, reinforced concrete walls, reinforced concrete beams and columns, and steel girders and columns. Disintegration of mass concrete follows rules designated for rock excavation, whereas the powder factor is the explosive energy per volume of material fragmented and is typically defined as the weight of explosive per volume of concrete.

FIGURE 5.2.17. *Old, 15 m in diameter, 92 meter high chimney was blasted down on February, 1998, at Imatra, Finland.*

MECHANICAL DEMOLITION

Many good methods exist for tearing down a building. The selected method often depends on the overall job scope as well as individual preferences.

Hammers (**FIGURE** *5.2.-18.*), cutter-crushers (**FIGURE** *5.2.-19.*) and pulverizers (**FIGURE** *5.2.-20*) are used when other methods, such as blasting or iron ball, are not economical or technically insufficient. Blasting is not always permitted and is increasingly unacceptable in demolition. If the structures contain substantial rebar, the job is usually finished with a hammer or cutter-crusher. The iron ball is efficient with thin brick walls but not in foundations removal, and it is also not precise enough to leave part of a structure intact.

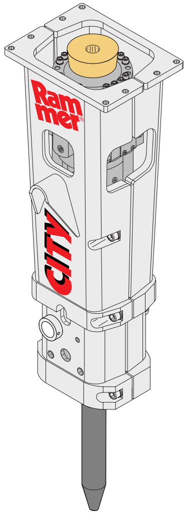


FIGURE *5.2.-18. A S 20 series silenced hammer.*

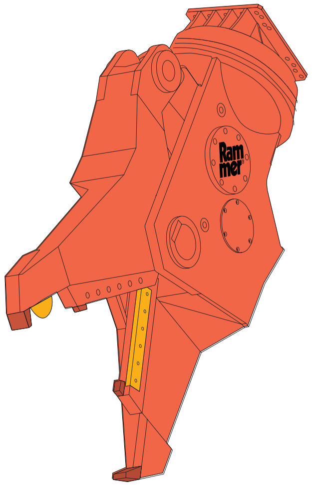


FIGURE *5.2.-19. A typical cutter-crusher*

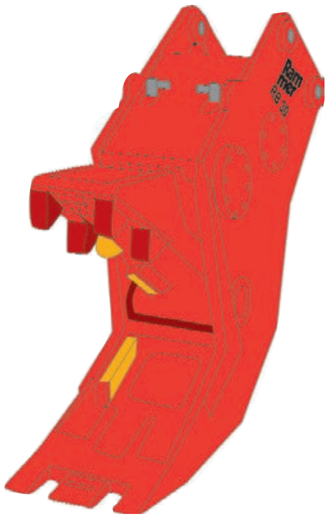


FIGURE *5.2.-20. A typical concrete pulverizer.*

In this chapter, a few influencing factors about the effectiveness of mechanical demolition are discussed.

The ideal project

The ideal project would be a never-ending job with stabile conditions performed by an excellent operator in good weather. The project would go as follows: Thick (> 50 cm), heavily reinforced concrete with a compressive strength over 40 MPa would be broken with a hammer. The best hammer has a tool length not much longer than the thickness of the structure.

Thin, moderately reinforced concrete, with both sides free and a compressive strength of 30 MPa or less would be broken with a RC-type crusher equipped with a jaw opening approximately 1.5 times the concrete thickness.

Pulverizers would be used in concrete secondary breaking, mainly to separate rebar from the concrete material.

Hammer tool selection

Tool selection depends on the preference of the operator and the amount of rebar in the concrete. Some operators feel that the chisel is the only possible choice for concrete demolition. One reason for this is that rebar can, to some extent, be cut with a chisel.

However, if the amount and diameter of rebar is large, the chisel gets twisted around its axis as it penetrates, damaging the tool's retainer pin groove. In such cases, a pyramid-point tool or moil-point tool is definitely a better choice. A pyramid-point tool also cracks the concrete, making demolition easier.

If the amount of rebar is immense (e.g. bridge pillars), a blunt tool is used together with a sufficiently powerful hammer. The mechanism of breaking is therefore not wedge penetration but a complicated mechanism with the stress wave traveling along the rebar.

A spade tool is used, if large plates are to be cut from thin walls or floors, and then trans ported.

Dust and greasing

When demolition is done by a hammer, the RAMMER WATER JET dust removal system and RAMLUBE automatic lubrication are strongly recommended. Concrete dust is a health hazard and decreases productivity due to poor visibility at the place where the tool should be posi tioned. Dry concrete dust removes the grease at the tool bushing and increases bushing and toolwear.

Carrier selection

In demolition work, the most important property of the carrier-breaker combination is agility. Demolition often requires the most awkward working positions.

Breaking power is usually not the main issue, as concrete is arelatively soft material. This means that carrier weight should be in the upper range of what is recommended for the breaker. Carrier condition also influences breaker performance. If the boom's joint pins are worn out, precise hammer positioning is difficult and progress is slow. When using longfront equipment, the choice of breaker is dictated by the stability of the boom, not by overall carrier weight.

Demolition robots, are recommended in dangerous and narrow places. As the operator is not in the immediate vicinity of the breaker, there is less risk of injuries from ceilings caving in. A demolition robot is also the obvious choice when dealing with polluted material, such as asbestos, dioxine or radioactive materials.

Small worksites and worksites with various structures

Unfortunately there are no on-going demolition sites. Instead project duration is usually relatively short. Circumstances also vary significantly even on large sites: there are extremely thick foundations, brick walls, reinforced ceilings and so forth.

The question is then, what is the most difficult part of the job, and how long do you want it take? A not too extensive foundation can be demolished with an hammer that is too small, just as a thin floor can be broken with an unsuitably large hammer. Selecting the right breaker for changing applications is challenging.

Underwater demolition

Underwater demolition is possible as long as proper precautions are taken. Be sure to contact your dealer before you start working your hammer underwater!

Underwater demolition with hydraulic breakers is interesting as the reach of the attachments depends on the reach of excavator and is not restricted to the immediate vicinity of the place to be broken. This may, in some cases, eliminate costly damming and water pumping actions, and working from platforms.

Recycling

Recycling is simply a question of money as garbage dumps are becoming overfilled and environmental regulations for ground-water, for example are becoming increasingly stringent. European governments are taking measures to have an ever increasing part of demolition debris recycled.
Demolition recycling is the separation of different materials. Rebar is iron scrap metal and can be sold if it is clean enough. However, rebar within concrete debris is often unwanted in a landfill. Unpolluted (no oil, PVC, bitumen etc.) demolition waste is easy and cheap to get rid of.

Controlled demolition

Controlled demolition is a method used on high constructions in city centers with neighboring buildings. It actually consists of breaking down the structure in the reverse order it was built (like running a movie backwards). First doors, windows, carpets, insulation material etc. are removed and recycled. Then through a hole in the roof, a small carrier-breaker combination is hoisted to the top floor. The breaker removes walls and ceilings, working its way down one floor at the time. The debris is removed from the top floors either by containers with a hoist or dropped into an elevator shaft.

Extreme care must be taken for the operators safety and that the floors are able to carry the breaker and carrier.

The advantages with controlled demolition are:

1. Minimum hazard to the surroundings from flying bricks and rocks, dust, noise etc.
2. Promotes advanced recycling, since garbage dump costs are not decrease in the future.
3. Savings through minimized handling costs of polluted material at special waste disposing plants.

Crack propagation

Sometimes it is essential to save part of the structure. In this case, thorough examination of the existing structure together with clear allocation of responsibilities are highly recommended.As a rule of thumb, damage made by a large hammer is harder to control than by a smaller hammer. Usually damage made by a cutter-crusher is easier to control than that of a hammer, however this depends on hammer and crusher size.

In critical places where expensive damage might result, such as bridges and skyscrapers, thorough examinations are recommended. The problem is in the brittle nature of concrete materials: crack propagation (direction/length) is very hard to predict. This together with unknown or unforeseen pre-existing stress in structures makes it very hard to guarantee that any part of a structure remains intact.

Structural vibrations

The propagation of structural vibrations is sometimes critical when working near sensitive electronic equipment. Here again, the vibrations by a large hammer are naturally stronger than those of a small hammer. A cutter-crusher itself does not cause vibrations like a hammer, but falling debris can. Clear and defined statements on responsibility are strongly recommended at this point also.

How structural vibrations propagate depends on material properties (density and modules of elasticity), porosity and cracks. These factors vary so much from case to case that it is impossible to give any specific information. Instead it is recommended to monitor vibrations during a test stage (this is a relatively simple procedure).

Prestressed structures

Extreme caution is suggested when dealing with any prestressed structure. Uncontrolled prestress could change a cubic meter of concrete into a projectile.

Tight schedules

There are several options for working through tight schedules. Working in shifts is not always possible due to local regulations for noise.

On large sites it is, of course, recommended that the proper equipment is used for the job, such as a heavy hammer for foundations, a crusher for walls and ceilings, and a cutter for scrap metal etc.

Another way to handle a tight schedule is to break the structures into the largest possible pieces that can be conveniently transported away. Recycling and final crushing is then handled without disturbing other functions.

Safety

Safety at a demolition site is particularly crucial because of collapsing structures, electricity, gas and fire hazards. The demolition contractor must double-check that electricity is disconnected, gas lines are not pressurized etc. Fire hazard is further reduced by using a cutter-crusher instead of acetylene to cut rebar.

Noise is an important health hazard that should not be ignored. In addition to health and municipal regulations, noise is also a productivity problem. A high noise level tires the operator out and slows the progression of the job. Every step should be taken to reduce noise levels. This includes both demolition equipment and the operator's personal protective gear. A cutter-crusher is the quietest method as are quiet City-model hammers.

Laws and regulations

Laws and regulations vary in different countries. The demolition contractor and those involved with the demolition project should be aware of what is required (demolition plan, permits etc.) because he will be obliged to pay penalties if a local law is violated.

RECYCLING CRUSHING MATERIAL

Special crushing tasks consist of crushing recycling material, such as construction waste material usually made up of concrete from demolished buildings or roads, and occasionally asphalt. The recycling material is usually located in the middle of a densely inhabited area, making it too expensive to transport it to a dump on the outskirts of town. A dump fee must also be paid.

A self-propelled crushing unit makes it possible to economically crush small amounts of material on-site. The crushed material is then used on the construction site, therefore, eliminating transportation and dump fees. The material can be used as subbase material, or it can be mixed with concrete and asphalt aggregates in small amounts.

5.3. CASES

5.3.1. Highway and railway cutting

FIGURE *5.3.-1. Typical view of railroad construction site - TAMROCK Ranger 700 working a bench drilling project in Sweden. The Arlanda Rail link connection between downtown Stockholm and the Arlanda airport will cut the duration of the trip from 35-50 minutes to 19 minutes. The rail system will be ready in autumn 1998. This work site consists of many open-bench excavation and tunneling projects. The project has a total of 8.6 km of tunnels and three stations beneath the airport. Rock type varies from mica schist to granodiorite. Hole sizes are 32-102 mm depending on the scale of excavation. Bench heights are 10-15 meters in the typical highway or railway cutting project.*

SURFACE EXCAVATION AT THE YELLOW RIVER SITE IN CHINA

The 1800 MW Xiaolangdi multipurpose dam project on the Huang He, the Yellow River in China, is one of the most demanding water conservation schemes ever attempted. Due to its hydraulic and silt conditions, complicated geology and high operational demands it poses great challenges. The rock in the area is mainly strongly layered sandstone, siltstone and claystone.

FIGURE *5.3.-2. Intake at Xiaolangdi.*

Preliminary design work for the project was completed in March, 1988. In the project, contractors use a total of 36 Tamrock rigs for underground and surface excavation. For more on underground excavation in this project, please see chapter 6.3.3.

The project was divided into three lots. The rock fill dam (Lot I) required the excavation of 30 million m^3 of rock.
In Lot II, both surface and underground excavation was carried out to build the water intake, tunnels and spillway. The powerhouse on Lot III was totally excavated underground.

FIGURE *5.3.-3. TAMROCK bench drilling rigs at work in Xiaolangdi.*

Tamrock PowerTrak CHA 660s and CHA 1100s track drills drilled 127 mm and 89 mm diameter holes. The dam filling is scheduled to be completed in 2001, after a 57-month construction period. The river was closed at the end of 1997, and the diversion tunnel is scheduled to be completed in November, 1998. Generator assembly started in January, 1998. Upon completion, the dam will be 900 m wide and 180 m high.

5.3.2. Demolition and recycling

Cutter crushers

FIGURE *5.3.-4. An English demolition contractor, CDC Demolition Ltd., from Woodbridge, Suffolk, pulled down a 14-floor apartment building in Bristol. CDC used an RC 22 cutter crusher with a digging machine for this project. The first phase included pulling down the upper five floors with two 3-ton mini-diggers that were rented from the David Meek Plant. Both machines were equipped with Rammer hammers: S 22 and S 22 CITY. Immediately after the completion of the first phase the RC 22 continued the demolition and pulled down the rest of the building.*

Pulverizers

FIGURE *5.3.-5. German contractor Bavak, Bauschutt and Holzverwertungsanlage Karlsruhe GmbH turns demolition waste, soil and road surfacing into greve and chipping for reuse. Such recycled material is being increasingly used in road building and other construction purposes. Bavak has used its Rammer RB 28 pulverizer since 1994. It is used to remove steel reinforcement from concrete debris brought to the site. The treated concrete is then fed into a crusher for recycling.*

6.1. GENERAL

SELECTING TUNNELING METHODS

In modern tunnel and underground cavern excavation, it is possible to select from many different methods. The following factors should be taken into consideration when selecting the method:

- Tunnel dimensions
- Tunnel geometry
- Length of tunnel, total volume to be excavated
- Geological and rock mechanical conditions
- Ground water level and expected water inflow
- Vibration restrictions
- Allowed ground settlements

The methods can be divided into drill & blast, and mechanical excavation. Mechanical methods can be split further to partial face (e.g. roadheaders, hammers, excavators) or full face (TBM, shield, pipe jacking, micro tunneling).

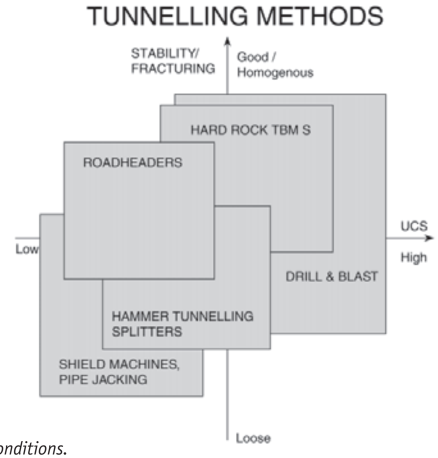


FIGURE 6.1.-1. *Tunneling methods in different rock/soil conditions.*

The drill & blast method is still the most typical method for medium to hard rock conditions. It can be applied to a wide range of rock conditions. Some of its features include versatile equipment, fast start-up and relatively low capital cost tied to the equipment. On the other hand, the cyclic nature of the drill & blast method requires good work site organization. Blast vibrations and noise also restrict the use of drill & blast in urban areas.

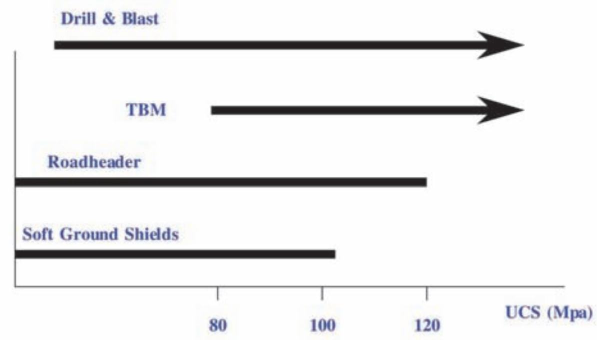


FIGURE 6.1.-2. *Range of methods compared to uniaxal compressive strength.*

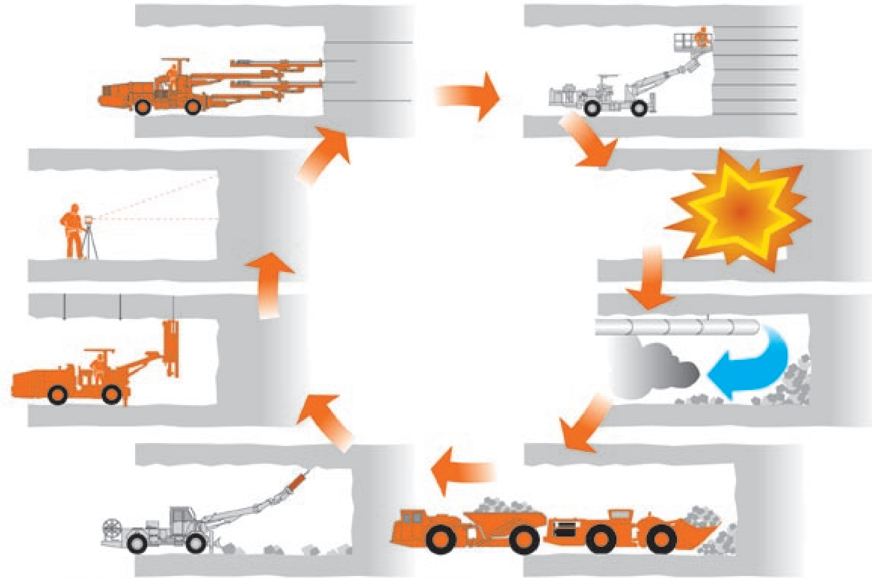


FIGURE 6.1.-3. *Drill and blast cycle.*

Hard-rock TBMs can be used in relatively soft to hard rock conditions, and best when rock fracturing & weakness zones are predictable. The TBM is most economical method for longer tunnel lengths, in which its high investment cost and timely build-up can be utilized by the high advance rate of excavation. TBM excavation produces a smooth tunnel with low rock reinforcement cost, and is optimal in terms of flow resistance in long ventilation or water tunnels.

Shielded TBMs or shield machines are used in loose soil and mixed ground, and in conditions where high water ingress is expected. The mechanical and/or pressurized shield prevents ground settlement and ground water inflow. Because of continuous ground control and no blast vibrations, this method is commonly used in urban tunneling. Pipe-jacking is a special application, in which the tunnel lining is continuously pushed by heavy hydraulic jacks as the tunnel advances. Microtunneling is a special application of pipe-jacking in no-man-entry sized tunnels.

Roadheaders can be used for tunneling in stable rock conditions of low-to-medium hardness. Where it is applied, the roadheader combines the versatility of drill & blast for producing various tunnel geometries, and the continuity of full-face mechanical excavation. As it lacks blast vibration, this method can be used in sensitive urban areas. In harder rock conditions, use of roadheaders is limited by a shorter lifetime of tools and increasing cutting tool cost.

Hammer tunneling evolved in the late 1980´ and combines a continuous method with low equipment costs. It has gained popularity mainly in the Mediterranean countries and Japan. The tunnel face geometry is unlimited, and the method is effective in rocks of low-to-medium compressive strength, when the rock mass is relatively fractured. In hard and compact ground, application is limited by low production rate.

6.2. METHODS

6.2.1 Drilling and blasting

DRILLING PATTERN DESIGN

The drilling pattern ensures the distribution of the explosive in the rock and desired blasting result. Several factors must be taken into account when designing the drilling pattern: rock drillability and blastability, the type of explosives, blast vibration restrictions and accuracy requirements of the blasted wall etc. The basic drilling & blasting factors, and drilling pattern design are discussed below. Since every mining and construction site has its own characteristics, the given drilling patterns should be considered merely as guidelines.

DRIFTING AND TUNNELING

Many mines and excavation sites still plan their drilling patterns manually, but advanced computer programs are available and widely used. Computer programs make it easier to modify the patterns and fairly accurately predict the effects of changes in drilling, charging, loading and production. Computer programs are based on the same design information used in preparing patterns manually.

Basic design factors

The tunnel of drift face can be roughly divided into four sections (**FIGURE** *6.2.-1.*).

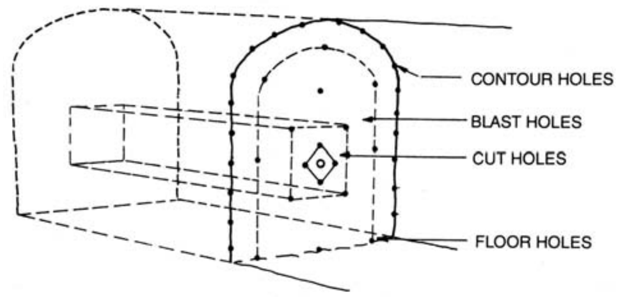


FIGURE ***6.2.-1.*** *Types of holes in a tunnel face.*

Drilling pattern design in tunneling and drifting is based on the following factors:

- Tunnel dimensions
- Tunnel geometry
- Hole size
- Final quality requirements
- Geological and rock mechanical conditions
- Explosives availability and means of detonation
- Expected water leaks
- Vibration restrictions
- Drilling equipment

Depending on site conditions, all or some of the above factors are considered important enough to determine the tunnel drilling pattern.Construction sites typically have several variations of drilling patterns to take into account the changing conditions in each tunnel. Drifting in mines is carried out with 5 to 10 drilling patterns for different tunnel sizes (production drifters, haulage drifters, drawpoints, ramps etc.) The pattern is finalized at the drilling site.Tunnel blasting differs from bench blasting in that tunnels have only one free surface available when blasting starts. This restricts round length, and the volume of rock that can be blasted at one time. Similarly, it means that specific drilling and charging increases as the tunnel face area decreases.When designing a drilling pattern in tunneling, the main goal is to ensure the optimum number of correctly placed and accurately drilled holes. This helps to ensure successful charging and blasting, as well as produce accurate and smooth tunnel walls, roof and floor. A drilling pattern optimized in this way is also the most economical and efficient for the given conditions.

Hole size

Hole sizes under 38mm in diameter are often considered small, holes between 41mm - 64mm intermediate, and those over 64mm large. Most tunneling operations today are based on hole sizes between 38 - 51mm in diameter. Only cut holes are larger than 51mm. Rock drills and mechanized drilling equipment used in tunneling and drifting are designed to give optimum performance in this hole range.Drifting rods are designed to match hole sizes and needs of horizontal drilling. Typical applications use tunneling rods and 1 1/4" and 1 1/2" drill steel sizes. Drill steels between 1" and 1 1/8" are used for hole sizes less than 38mm.

The number of holes needed per tunnel face area decreases as hole size increases. The difference is not much in small tunnels, but becomes more significant in large tunnel face areas. Small hole sizes require smaller steels, but these bend more easily, giving rise to inaccurate holes and poor blasting.

Cut types

The blasting sequence in a tunnel or drift always starts from the "cut", a pattern of holes at or close to the center of the face, designed to provide the ideal line of deformation. The placement, arrangement and drilling accuracy of the cut is crucial for successful blasting in tunneling. A wide variety of cut types have been used in mining and construction, but basically they fall into two categories: cuts based on parallel holes, and cuts that use holes drilled at certain angles.The most common types of cut today is the parallel and V cut (**FIGURE** *6.2.-2*). The V cut is the older of the two and is still widely used in construction. It is an effective type of cut for tunnels with a fairly

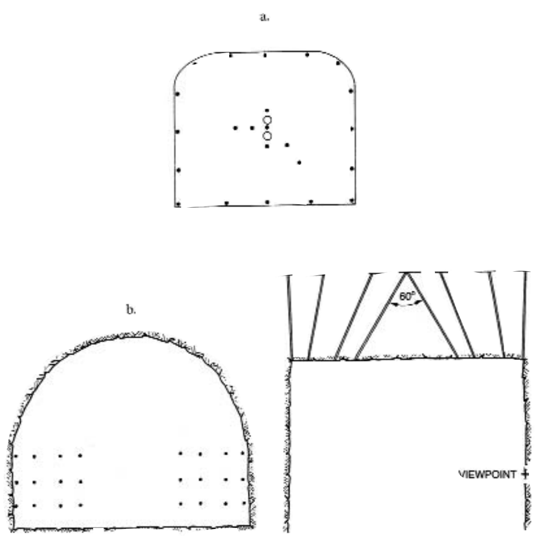


FIGURE ***6.2.-2.*** *The parallel cut (a) and V cut (b).*

large cross-section and requires fewer holes than a parallel cut. The parallel cut was introduced when the first mechanized drilling machines came on the market making accurate parallel drilling possible.

Parallel cut

The parallel cut has a large number of minor variations, however the basic layout always involves one or several uncharged holes drilled at or very near the center of the cut, providing empty space for the adjacent blasted holes to swell into. Uncharged cut holes are typically large, 76 - 127mm in diameter. A less common alternative is to use "small hole" openings (several small holes instead of one or two large holes). Small hole opening make it possible to use the same bit size throughout the whole drilling pattern. Experience proves that big hole openings give more reliable results than small hole openings.

To successfully blast a full round, the cut must be drilled, charged and blasted correctly. Cut holes are drilled very near to each other, as parallel as possible, as shown in **FIGURE** *6.2.-3.* Specific drilling in the cut section may rise above 10drm/m^3. Apart from the large cut holes, other holes in the cut are the same size as the stope (face) holes. Large cut holes are normally drilled by reaming. First, a smaller, for example, 45mm diameter hole is drilled then reamed to the final size using a pilot adapter and a reaming bit.

Drilling holes several meters long as close together as possible demands great accuracy, but the advanced boom design and automated functions of modern drill jumbos make this quite easy. The parallel cut is especially suitable for modern mechanized tunneling equipment. This cut type has also made long rounds common in small tunnels. An earlier version of the parallel cut is the "burn cut" which does not use uncharged holes, relying instead on a very strong charge to burn the rock. Today, the parallel cut has replaced the burn cut.

Purpose of cut holes

In the parallel cut, the cut holes provide enough expansion space for the remaining blasted rock around it. The face area of a typical parallel cut varies from 1.6m x 1.6m to 2.5m x 2.5m. The right size is determined according to area of the tunnel face.

Big, uncharged cut holes (76 - 127mm dia.) provide an opening for the blasted, expanding rock from the surrounding cut holes. All holes are drilled very close to each other and detonated each with its own detonation number (**FIGURE** *6.2.-3*). The main idea is for each hole to loosen the rock in the front of it, allowing it to expand and fill the available open space. Cut holes are quite heavily charged and the blasted cut becomes a square opening. Basically, only drilling errors limit the gained advance per hole length.

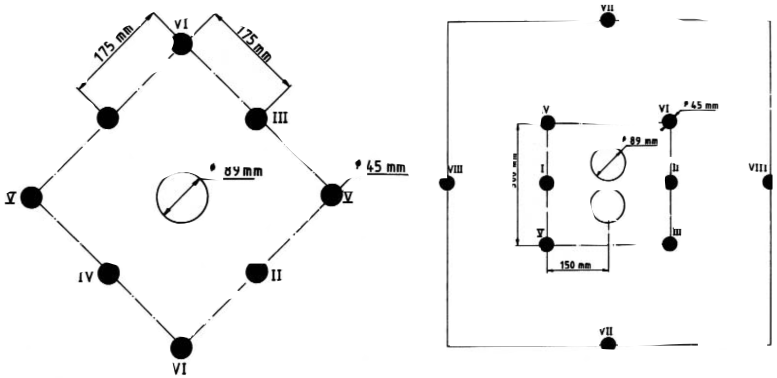


FIGURE 6.2.-3. *Typical 2m x 2m cut hole arrangement*

When designing the cut, the following parameters are important for a good result:

- Diameter of large hole
- Burden
- Charge conditions

Additionally, drilling precision is of utmost importance, especially for the blast holes closest to the large hole (holes). The slightest deviation can cause the blasthole to meet the large hole or make the burden too big. An exceedingly big burden causes breakage or plastic deformation in the cut, resulting in a short advance.A parameter for good advance of the blasted round is the diameter of the large empty hole. The larger the diameter, the deeper the round can be drilled and a greater advance expected.

One of the most common causes of short advance is an overly small hole in relation to the hole depth. An advance of approx. 90% can be expected for a hole depth of 4m and one empty hole 102mm in diameter. If several empty holes are used, a fictitious diameter must be calculated. The fictitious diameter of the opening may be calculated by the following formula:

$$D = d\sqrt{n}$$

where D = Fictitious empty large hole diameter
d = Diameter of empty large holes
n = Number of holes

In order to calculate the burden in the first square, the diameter of the large hole is used in one large hole and fictitious diameter in several large holes.

Calculation of the 1st square

The distance between the blasthole and the large empty hole should not be greater than 1.5ø for the opening to be clean blasted. If longer, there is merely breakage and if shorter, there is a great risk that the blasthole and empty hole will meet.

Therefore, the position of the blastholes in the 1st square is expressed as:

$$a = 1.5ø$$

Where a = C - C distance between large hole and blasthole
ø = Diameter of large hole

In the cases of several large holes, the relation is expressed as:

$$a = 1.5\ D$$

where a = C - C distance between the center point of the large holes and the blasthole
D = Fictitious diameter

Charging of the holes in the 1st square

The holes closest to the empty hole (s) must be charged carefully. An insufficient charge concentration in the hole may not break the rock, while an excess charge concentration may throw the rock against the opposite wall of the large hole with such high velocity that the broken rock will be re-compacted and not blown out through the large hole. In this case, full advance is not obtained.

The required charge concentration for different C - C distances between the large hole and nearest blasthole(s) can be found in **FIGURE** *6.2.-4.* The normal relation for the distance is a = 1.5 ø. An increase in the C - C distance between holes causes subsequent increment of the charge concentration.

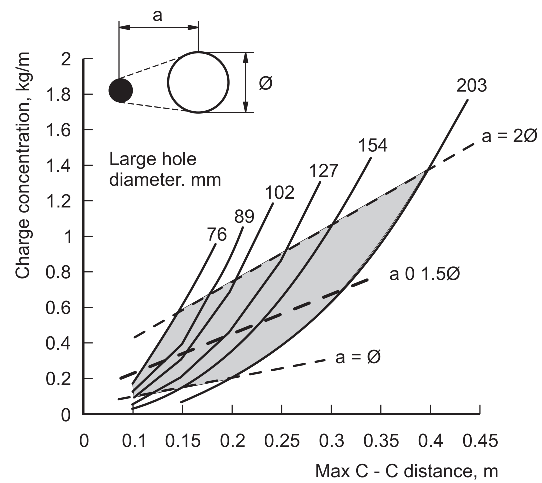


FIGURE 6.2.-4. *Minimum required charge concentration (kg/m) and maximum C - C distance (m) for different large hole diameters.*

The cut is often somewhat overcharged to compensate for drilling errors which may cause insufficient breakage angle. However, excess charge concentration causes re-compaction in the cut.

Calculating 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, but differs in that breakage is towards a rectangular opening instead of a circular opening.
As is the case of the 1st square, the breakage angle 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.

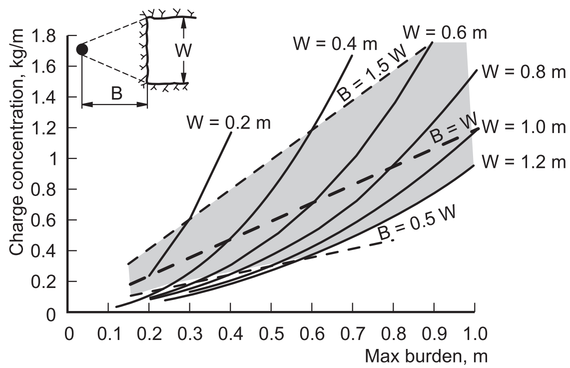


FIGURE ***6.2.-5.*** *The required minimum charge concentration (kg/m) and maximum burden (m) for different widths of the opening.*

The charge concentration obtained in **FIGURE** ***6.2.-5.*** 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.

Stoping

The holes surrounding the cut are called stopeholes. The diameter of a stopehole is typically between 41 - 51mm. Holes smaller than 41mm may require drilling an excessive number of holes to ensure successful blasting. Holes bigger than 51mm can result in excessive charging and an uncontrolled blast.

Holes are placed around the cut section in an evenly distributed pattern using a space/burden ration of 1:1.1. If hole size is between 45 - 51mm, typical spacing and burden are both between 1.0m - 1.3m. Actual rock conditions and ability to drill in the required positions are factors that can reduce or add to the number of holes needed.The design of the drilling pattern can now be carried out and the cut located in the cross section in a suitable way.

Contour holes

Floor holes have approximately the same spacing as stope holes, but the burden is somewhat smaller; from 0,7m to 1,1m. Inaccurate or incorrect drilling and charging of the floor holes can leave unblasted bumps, which are difficult to remove later. The contour holes lie in the perimeter of the drilling pattern. In smooth blasting, contour holes are drilled closer to each other and are specially charged for smooth blasting purposes. Spacing is typically from 0.5m to 0-7m and burden varies between 1 and 1.25 times the space. This type of layout makes it possible to use special smooth blasting explosives, which limits the width and depth of the fracture zone in the walls and roof caused by blasting. In special circumstances, two or more smooth blasting rows can be used.

In tunneling, however, contour holes are blasted with stope holes, but timed to detonate last. The result in smooth contour excavation mostly depends on drilling accuracy. The required amount of shotcreting and concrete casting can be significantly reduced by using smooth blasting, particularly in poor rock conditions.Smooth blasting increases the number of holes needed for the drilling pattern by roughly 10 - 15%.

Rock hardness is occasionally incorrectly considered to be the only dominant factor when optimizing the drilling pattern. The change from very hard rock to soft rock therefore causes a change in the drilling pattern. Rocks that are hard but abrasive are fairly easily blasted, where as the blastability of rocks such as some limestone, although relatively soft, is poor. However, it is beneficial to redesign and optimize the drilling pattern long before this stage is reached and, more important still, to take rock blastability into account. In a 10-km long tunnel project, each extra hole means about 11,000 unnecessary drilled and blasted meters. Diagram (**FIGURE** ***6.2.-6.*** and ***6.2.-7.***) shows specific charge and drilling for different tunnel areas.

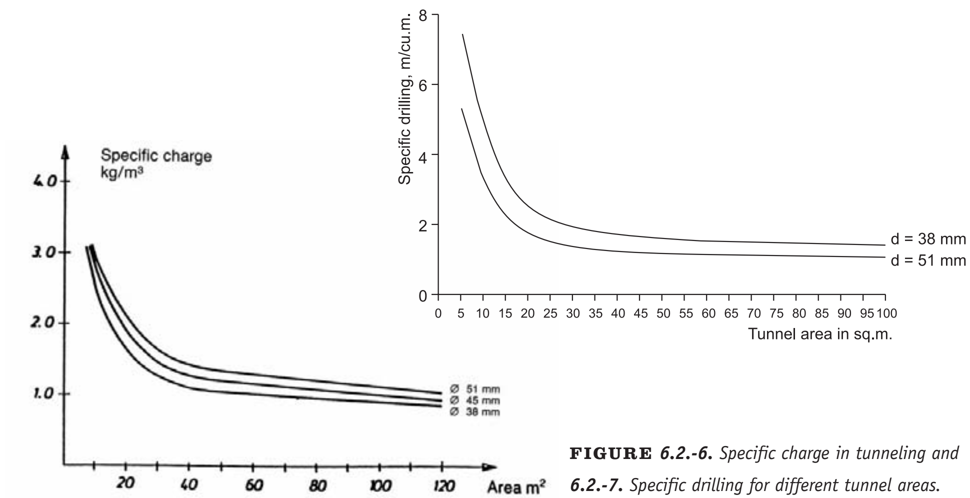


FIGURE ***6.2.-6.*** *Specific charge in tunneling and* ***6.2.-7.*** *Specific drilling for different tunnel areas.*

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° (**FIGURE** *6.2.-8.*)

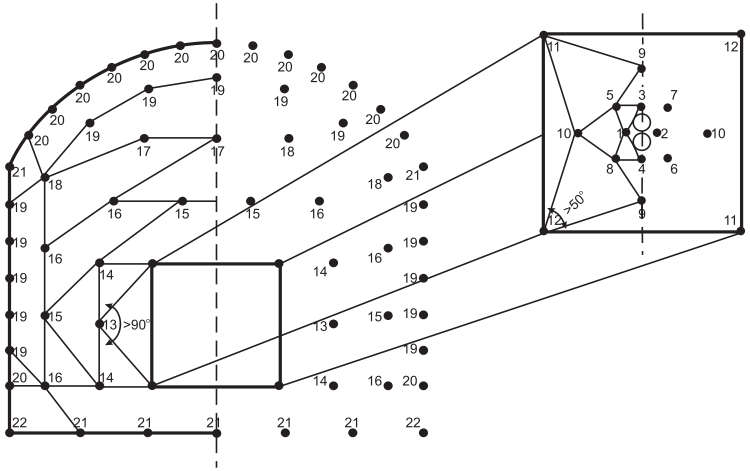


FIGURE *6.2.-8. Firing sequence for tunnel in numerical order.*

It is important in tunnel blasting to have a long enough time delay between the holes. In the cut area, it must be long enough to allow time for breakage and rock throw through the narrow empty hole. It has been proven that the rock moves with a velocity of 40 - 70 meters per second. A cut drilled to a depth of 4 - 5 m would therefore require a delay time of 60 - 100 ms to be clean blasted. Normally delay times of 75 - 100 ms are used.

In the first two squares of the cut, only one detonator for each delay should be used. In the following 2 squares, two detonators may be used. In the stoping area, the delay must be long enough for the rock movement. Normally, the delay time is 100 - 500 milliseconds.
For contour holes, the scatter in delay between the holes should be as little as possible to obtain a good smooth blasting effect. Therefore, the roof should be blasted with 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. Contour holes should be initiating with detonating cord or with electronic detonators to obtain the best smooth blasting effect.

V cut

The V cut is a traditional cut based on symmetrically drilled, angled holes. It has lost some of its popularity with the widespread adoption of the parallel cut and longer rounds. However, it is still commonly used in wide tunnels where tunnel width sets no limitations on drilling. The working principle of the V cut is similar to surface excavation applications. The V cut requires slightly fewer hole meters than the parallel cut, which gives it an advantage in large tunnels. The V cut is based on surface blasting principles in which the angle for rock expansion equals or exceeds 90 degrees. The angle at the bottom of the cut holes should not be less than 60 degrees. Maintaining the right angle is the main difficulty in V-cut drilling; and, the correct drilling angle limits round length in narrow tunnels (**FIGURE** *6.2.-9a.*).

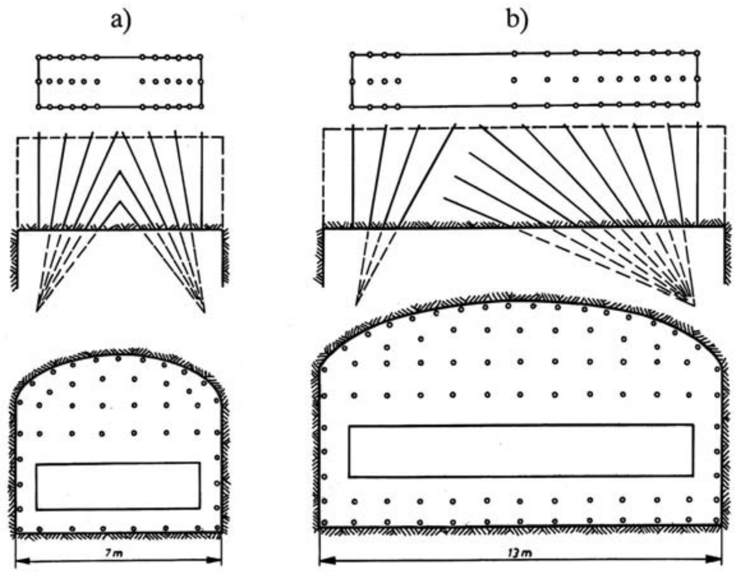


FIGURE *6.2.-9. V cut (a) and Fan cut (b).*

Tunnel width limits the use of the V cut. In narrow tunnels, the advance per round can be less the one third of the tunnel width, which increases the number of rounds and the amount of drilled meters when excavating small tunnels. V cuts are easily drilled with mechanized rigs in large tunnels where tunnel width sets no limitations. The cut normally consists of two Vs but in deeper rounds the cut may consists triple or quadruple Vs.
Each V in the cut should be fired with the same interval number by MS detonators to ensure coordination between the blastholes in regard to breakage. As each V is blasted as an entity

one after the other, the delay between the different Vs should be in the order of 50 ms to allow time for displacement and swelling.

The fan cut

The fan cut (**FIGURE** *6.2.-9b*) is an other example of angled cuts. Like the V cut , a certain tunnel width is required to accommodate the drilling equipment to attain acceptable advance per round.

The principle of the fan is to make a trench-like opening across the tunnel and the charge calculations are similar to those in opening the bench. Due to the geometrical design of the cut, the hole construction is not large, making the cut easy to blast. Hole drilling and charging is similar to that of cut holes in the V cut.

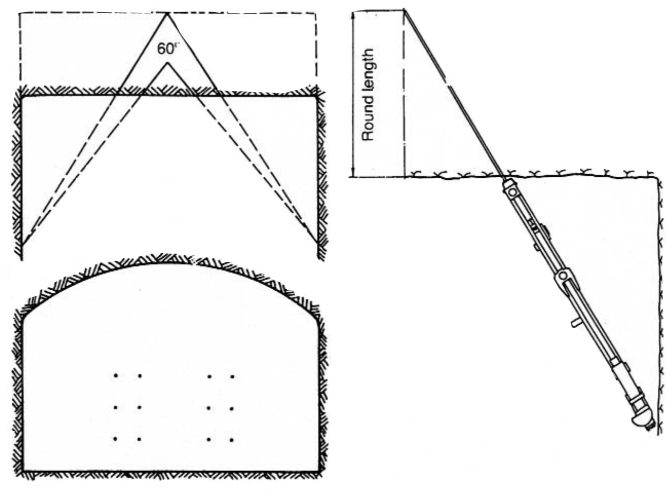


FIGURE ***6.2.-10.*** *Feed set-up and drilling limitations in V cut.*

Other design features

The design of the drilling pattern for tunnels should correlate with tunnel shape and size. The cut is normally placed vertically in the middle or side section, and horizontally on or slightly under the center line of the tunnel. The exact place is often left or right of the tunnel´s center point and varies with each round (**FIGURE** *6.2.-11.*).
Sometimes the tunnel is excavated in several sections, such as a top heading, followed by benching with lifters. The top is excavated as described above, but benching with lifters only requires stope holes since the excavated top heading acts as the “cut”. It is also possible

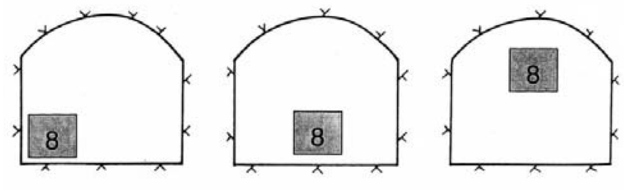


FIGURE ***6.2.-11.*** *Various cut locations*

that the opening for the blast or the cut section has been produced earlier by other means such as the full profile method (tunnel boring). In such cases, cuts are not required and the remaining excavation holes are drilled as stope holes. It is recommended that ditches and drains be excavated at the same time as the tunnel face but sometimes their design is more complicated and they must be excavated separately.

Look-out angle

The drilling pattern also includes information on the look-out angle needed at different points on the tunnel face. The look-out angle is the angle between the practical (drilled) and the theoretical tunnel profile (**FIGURE** *6.2.-12.*). If the contour holes are drilled parallel to the theoretical line of the tunnel, the tunnel face gets smaller and smaller after each round. To ensure that the correct tunnel profile is maintained, each contour hole is drilled at slight angle into the tunnel wall, the look-out angle, which of course can not be smaller than that permitted by the profile of the rock drill.
Adjusting the look-out angle by eye requires an experienced and skillful operator. Modern drilling rigs have electronic or automatic look-out angle indicators that enable correct adjustment of the look-out angle relative to standards alignment. Computerized drilling jumbos make setting, adjustment and monitoring of the look-out angle even easier. An incorrect

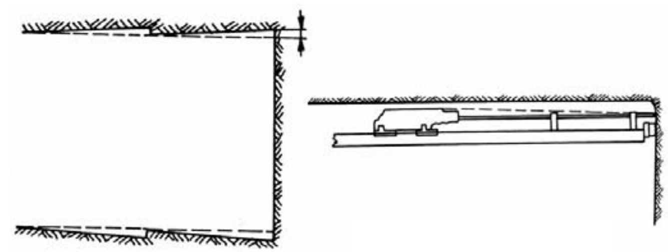


FIGURE ***6.2.-12.*** *The look-out angle.*

look-out angle produces over- or underbreak, both of which give uneconomical results. Other aspects such as curve and tunnel inclination also need to be considered when the drilling pattern is designed. Any excavation later on is both costly and time-consuming.

Tunneling accuracy

Inaccurate drill steel and rig set-up is often the result of "natural causes" or errors of human judgment. The tunnel may not provide enough room for proper alignment of the drilling unit. Cross cuts and curves can also cause similar difficulties due to the changed drilling boom alignment and coverage.

Inaccurate set-up causes misalignment of the entire tunnel and leads to over- and underbreak. Rectifying the whole tunnel alignment later is extremely expensive.
The rock face frequently has cracks, joints, bumps and holes that prevent the hole from being drilled to its optimal position. When the hole is drilled in a new position, it can become unaligned or does not end in the same plane as the other holes. Inaccurate hole length leads to blasting difficulties causing uneven tunnel walls, roof and floor. Once started, misalignment can easily become compounded, making the following rounds even more difficult to drill (**FIGURE** *6.2.-13.*). The effect of hole accuracy on costs is described in the following example:

Example
A 5 x 5m construction tunnel has overbreak of 20% as a result of hole misalignment. After tests to correct misalignment five new holes are added to the pattern. A successfully drilled 5.1m round in this tunnel size would include fifty-four 45-mm, and two 89-mm cut holes. The difference in drilled meters per round is 25.5 drm and blasted rock volume approx. 25m^3. In a 5000 meter-long tunnel, this adds up to 25,500 additional drilled and charged meters and 40,000m^3 (loose) of extra rock to muck and haul away. Mucking and hauling would require more capacity, time and money. In addition to the direct effects, misalignment creates other unfavorable factors such as increased need for rock support, such as increased bolting, shotcreting, concrete casting and unnecessary finishing works, e.g. filling or other support structures.

If overbreak is nearer 20% instead of the accepted or calculated 10%, the effect in a 5m x 5m tunnel is 12.5m^3 of extra volume needing to be filled per round (5.1m hole length).
For a 5,000 meter-long tunnel, the added volume would mean 12,500 m^3 more concrete with corresponding costs. Extra concrete laying would require more time, which would set back the whole excavation process. If overbreak added one additional bolt per tunnel meter, the 5,000 meter-long tunnel would need 5,000 extra bolts, also increasing excavation costs and consuming more time.Underbreak always requires further excavation and is for this reason even more serious than overbreak.

Accurate tunneling, and accurate drilling and charging go hand-in-hand. The following topics need to be planned in advance to ensure accurate tunnel profile:

- Known geological and rock mechanical conditions
- Planned drilling pattern/patterns, correct hole size and hole length for the planned excavation

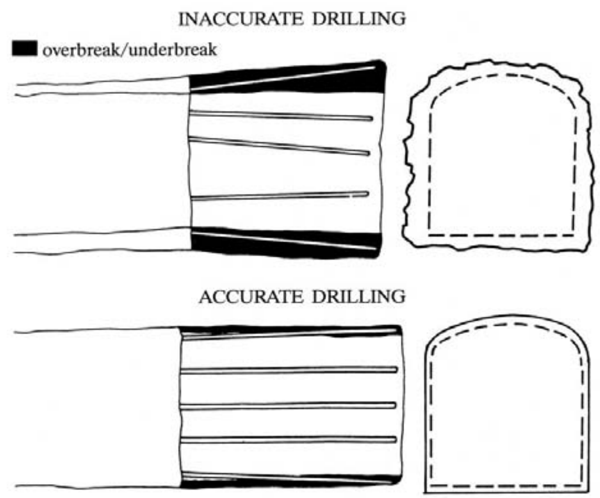


FIGURE ***6.2.-13.*** *Cumulative drilling errors in drifting and tunneling.*

- Smooth blasting (contour blasting) procedure
- Correct rig set-up
- Correct hole alignment and look-out angle, with special consideration for the walls, roof and floor
- Cut placement; inclined and curved tunnels are especially prone to under- and overbreak in the walls and roof and "bumps" in the tunnel floor
- Accurate charging, the correct detonators and drilling pattern
- Continuos follow-up procedures

Advance and yield

The parameter used to describe the advance of the excavation work in tunneling and drifting is called "pull" or advance per round, or yield per round.
In tunneling, the length to which the holes are drilled and charged is called the round length. It is one of the most important parameters when planning excavation since excavation depends on selecting the optimal round length.
The mechanization and automation of drilling equipment has led to longer rounds, typically 3 - 5 meters. Experiments have shown that round up to 8 meters long can be drilled and blasted successfully with special care and equipment (special explosives, rock conditions, special drilling equipment).

Round length must be optimized, bearing in mind several important aspects (**FIGURE** *6.2.-14.*).

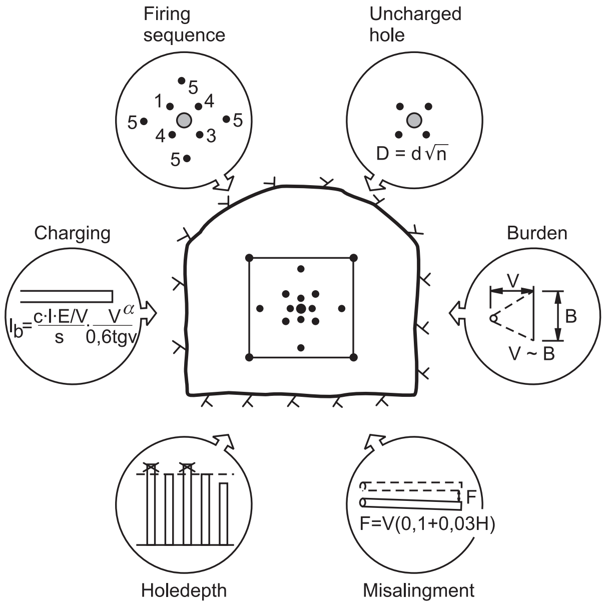


FIGURE ***6.2.-14.*** *Some aspects for optimizing the round length.*

- Stability
- Rock geological and mechanical conditions
- Drilling, charging, mucking and rock support equipment and related size, reach, maneuverability and efficiency
- Allocation of time within and between each round
- General working arrangements, work layout, distance between working places, support works needed, general regulations and legal questions (inspection needs, ground vibration restrictions etc.)
- Amount of equipment and labor, if restricted

A successfully blasted round still leaves some 20cm of unloosened drilled hole length. The excavated portion of the blasted round is called “pull” or the advance per round. Drilling accuracy, accurate hole placement and correct blasting methods are the most important factors affecting pull. In the drilling pattern layout, the choice of cut and cut hole placement also affects the final advance.

The introduction of computerized drilling equipment has greatly improved hole and profile accuracy and extended the advance per round due to 97%. Computerization has proven especially efficient when drilling long rounds where poor accuracy with conventional drilling equipment leads to uneven hole bottoms. The preplanned optimal drilling patterns are described in three dimensional form and up-loaded into the drill jumbo´s on-board computer. The pattern includes information on the starting and ending point of the holes as well as hole length and look-out angle. Even when manual changes are made to the drilling pattern during operation, the program will adjust the new hole to finish at the same hole bottom as the other holes.

The effect of pull on the final result is easily seen when excavating a 5,000 meter long tunnel. If pull of a 5.1 meter long round is 90% instead of 95%, due to poor drilling or blasting accuracy, a total of 59 extra rounds must be drilled, blasted and mucked to complete the job. The cost of these extra rounds will depend on tunnel size, labor, equipment, time penalties and other site-related factors.

UNDERGROUND CHAMBER

As for rock blasting techniques, the construction of underground chambers does not differ from that of tunnels of the same magnitude. The width of underground chambers can not be too great due to the inability of the rock to support the roof with its own strength. For oil storage chambers and machine halls for hydro-electric power-plants, widths of 20 - 24 m have been constructed with no required heavy reinforcement. The height of the chambers may be up to 40m.

The construction of underground chambers is based on qualitative sound rock. Some economic aspects must be considered. If the chamber is located at too shallow a level, the cost of reinforcing the rock may be high because the quality of surface rock is normally poorer than rock at deeper levels. However, a deep location results in long access roads, which may cause problem during construction and when the chambers come into use.

Small underground chambers, with a height of less than 8 m are blasted as tunnels. In larger chambers, the operation is divided into several drilling and blasting stages (**FIGURE** *6.2.-15.*) in which different methods are used:

- Pilot tunnel with side stoping
- Horizontal benching
- Vertical benching

Example of a chamber excavation procedure plan.
Blasting a 31.5 m high x 21.1 wide rock cavern can be divided into three or four stages.

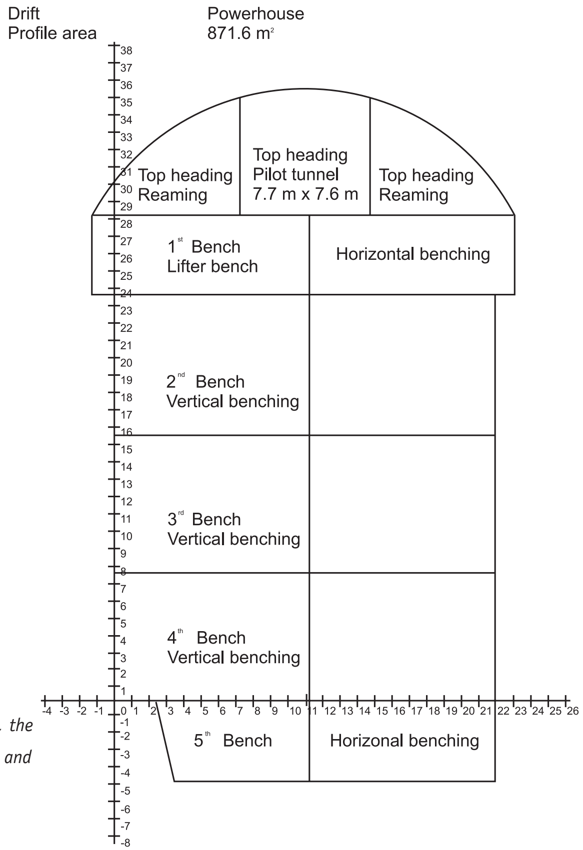


FIGURE *6.2.-15. In larger champers, the operation is devided into several drilling and blasting stages.*

First, the gallery with a height of 7.5m is drilled and blasted along the entire length of the cavern. A pilot heading is driven in the middle of the gallery (**FIGURE** *6.2.-16.*) one or two rounds before the reaming of sides take place to enlarge the gallery to full width and length. The smooth blasting system is described in **FIGURE** *6.2.-16* **and FIGURE** *6.2.-17.*

x-holes: Smooth blasting
ø 17 mm or ø 22 mm
•-holes: Anit ø 24 mm - 28 mm
o-holes and other holes: ANFO if hole diameter d< 45 mm

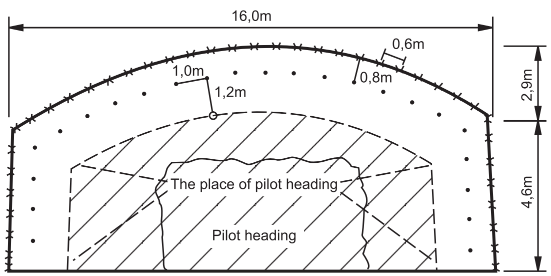
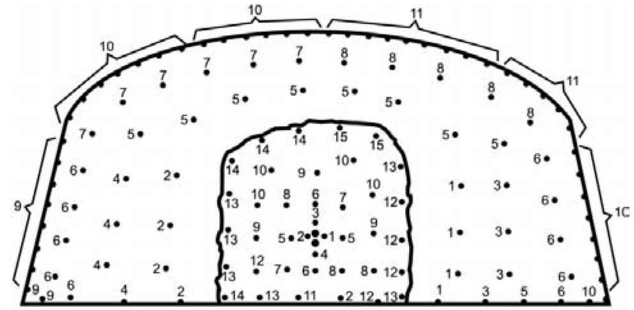


FIGURE *6.2.-16. Smooth blasting system is in gallery. Peak particle velocity on roof and walls v about 1000mm/s.*



PILOT TUNNEL:	NO. OF HOLES 41 ø 45mm + 2 ø 102mm
	HOLE DEPTH 4.6m ADVANCE ≈ 4.0m
TOP HEADING:	NO. OF HOLES 90 ≈ 45mm
	HOLE DEPTH 4,6 m, ADVANCE ≈ 4.5 m
CHARGING:	PROFILE HOLES, SILOSEX + PRIMER
	DETONATING CORD
	2ND ROW, ANITE + PRIMER
	FIELD HOLES, ANFO + PRIMER
IGNITION:	SEE CAP NUMBERS
CONTOUR HOLE DISTANCE 0.6m	

FIGURE *6.2.-17. Firing patterns in gallery.*

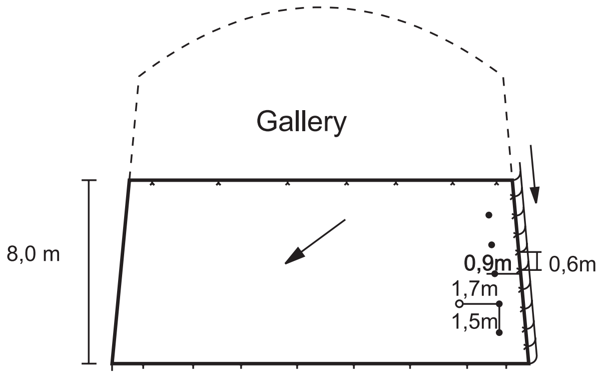
The transient strain in the rock due to blasting depends upon the liner charge concentration per drill hole length, explosive strength and distance from the charge. For example, granite may fail in dynamic tension at a stress of approx. 30 Mpa or around a peak particle velocity of 1000 - 2000 mm/s depending on the wave type. Assuming that damage would occur around v = 1000mm/s, it is possible establish a proper blast pattern. Attention is paid also to rows adjacent to the perimeter row in order to minimize unwanted fracturing. In the stoping stage, an 8m-high bench was removed by horizontal stoping (**FIGURE** *6.2.-18.*) and finally a vertical bench (**FIGURE** *6.2.-19.*) or possibly two 8m horizontal benches are excavated.

The reinforcement methods are bolting and shotcrete lining. Systematic roofbolting is later carried out with a bolt density of 1 bolt/4 m^2. Bolt lengths in the arch part of the caverns range typically from 2.0 - 4.0 m, which is 0.15 - 0.30 times the width of the span. In walls, the corresponding lengths vary from 2.4 m - 6.0 m. The need for grouting has been limited.

When calculating the largest instantaneous charge permitted for different distances from buildings, the formula below is currently very commonly used when blasting large rock caverns.

$$v = k\,(R/\sqrt{Q})^{-n}$$

where (v) is the maximum particle velocity (mm/s); Q the cooperating charge and R the distance. The constants k and n vary with foundation conditions, blasting geometry and type of explosives.



x-holes: smooth blasting ø 17 mm or ø 22 mm
+ detonating cord
•-holes: Anit ø 32 mm
o-holes: ANFO

FIGURE ***6.2.-18.*** *Presplitting system in horizontal stoping.*

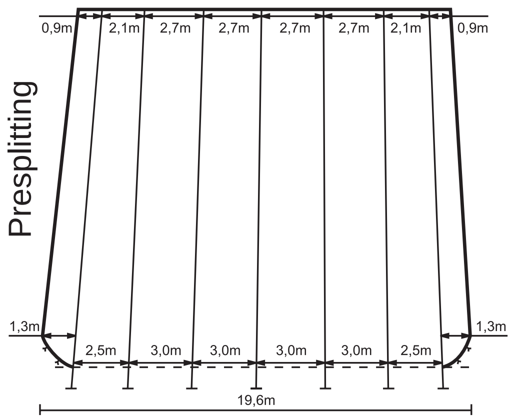


FIGURE ***6.2.-19.*** *Drilling, charge calculations.*

ENVIRONMENTAL ASPECTS

Cautious Tunnel Blasting in this connection refers to tunnel blasting with reduced risk of ground vibrations, air shock waves or throw of stones (Chapter 3.10.). Ground vibrations constitute the primary problem when a tunnel passes under buildings or other sensitive installations or facilities. Air shock waves and stone throw occur when tunnels are being opened and, in many cases, this occurs in exposed places.

Often, mining problems with limited rock coverage and the need of reinforcement usually appear during initial blasting work on tunnels. The following measures are advised when carrying out initial blasting work on tunnels in built-up areas:

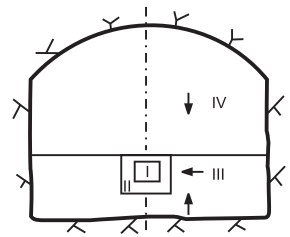
- Cautious blasting with limited hole depth, charges and holes per round
- Millisecond firing
- Suspended covering material.
- Ground vibration and air shock wave measurement (Chapter 3.10.).

In the opening up of tunnels, large hole cuts, preferably with two large holes, function well. Drilling is performed with a limited hole depth between 1.0 - 2.0 m depending on the location of the blasting site and the technical conditions of the rock. The first round consists of one cut hole, after which normally two cut holes per round are fired. In due course, the number of "cut spreader" holes and stoping holes per round increase depending on the weight of the covering and its capacity to remain in position during blasting. It is not advised to increase the number of drill holes per round to a great extent because in sensitive locations just a few too many holes can lift the covering material. Care is exercised even after the first advance so that the covering material used is able to block throw and reduce air shock waves.

Millisecond firing is the safest method to use. When using half-second firing, there is risk of the first delay lifting the covering material resulting in throw. Covering material should be used for each round until the tunnel has extended so far that air shock waves no longer have an influence. In straight tunnels, this can imply considerable distances. If vibration and air shock wave measurements are performed, blasting can be adapted to the values obtained. Since the air shock wave causes vibrations in the surrounding buildings, the horizontal shock wave component can be of the greatest interest in blasting of this type.

Air shock wave magnitude can be theoretically calculated based on charge amounts, delay sub-divisions and distance. The most difficult estimate is the charge enclosure factor which must be included when the explosive is charged in a drill hole. As more and more measuring material becomes available from air shock wave measurements, the accuracy of this type of calculation can be improved.

FIGURE ***6.2.-20.*** shows the principle for opening a tunnel within a built-up area where buildings are located very close by.



I = 1–2 holes / delay
II = 3–4 holes / delay
III = 4–7 holes / delay
IV = 5–10 holes / delay

FIGURE ***6.2.-20.*** *Holes are closely located so that the charge in each drill hole can be limited.*

In ground vibration problems, it is often necessary to drill closely spaced holes and limit advance in order to reduce the instantaneously detonating charge.
Based on permissible simultaneously detonating charge, hole location, drilling depth, charge per hole and firing pattern are adapted so that ground vibrations fulfill requirements.

Normally in tunnel blasting, the drilling pattern can be adapted so that the cooperating charge is not larger than the charge in an individual drill hole. A spread of the delay numbers distributes the ground vibrations throughout the surrounding rock.

Certain cuts, for example plough cuts, are unsuitable in ground vibration problems when there is a risk of coordinating a large number of holes in the cut. “Burn” cuts of various types are also unsuitable. The fan cut can be used in this connection primarily in wide tunnels. In narrower tunnels where it is difficult to drill holes at an angle, large hole cuts can very well be used. It is preferable to drill at least two large holes when carrying out particularly cautious blasting. This reduces constriction and the risk of unsuccessful breakage. There is also the possibility to reduce the charge per meter.

VARIOUS CHARGING METHODS

Charging and blasting in tunnels and drifts

In tunnel excavation, blasting works outward from the first hole around the uncharged holes in the cut. Each blast provides more space for the following ring of blast holes. Successful blasting of the cut section is critical to the success of the whole round. Because the cut holes initially have only one direction in which to expand, the specific charge in the cut is considerable higher than in the rest of the tunnel and can even exceed 10kg/m^3.
Most stoping holes (especially in large tunnels) have a large expansion area. These holes are considered close to surface blasting holes for charging calculations. The same explosive, normally ANFO, is used for stope hole charging as in the cut area. Development of explosives has moved in the direction of products with better fumes such as emulsion explosives. Lightened explosives or special smooth blasting explosives are used for smooth blasting.

Initiating systems like NONEL decrease charging time and add further safety to the blasting operation because it is insensitive to electrical hazards.

Contour holes should be blasted almost last with detonating cord or with the same detonating number. It is important to blast each smooth blasting section (walls or roof) simultaneously to achieve a smooth and even surface.

Electronic detonators will perhaps become the detonators of future in tunneling, too, due to increased timing precision.

Bottom holes are blasted last right before the bottom corner holes. This lifts the loosened rock pile a little, which makes mucking easier. The specific charge and specific drilling can become quite high in small tunnels due to the restricted free space available (**FIGURE** *6.2.-6.* and *6.2.-7.*).

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. 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.

Charging with pneumatic machines

Principally two types of pneumatic charging machines are available:

- Semi-automatic charging machines for cartridge explosives
- Pressure-ejector vessels for ANFO.

Semi-automatic charging machines are useful for upward holes, underwater blasting and fissured rocks where cartridges tend to jam but where a semi-ridged plastic hose could be introduced to the bottom of the hole.

Pressure-ejector vessels for ANFO are mostly used in tunneling. Free flowing ANFO is normally poured into blastholes which are vertical or close to vertical.

For horizontal and upward blastholes, the principal method of charging is via pneumatic charging devices. Such devices are also used for the charging downward blastholes where higher charging density is required. The principle of the charging machine is that the ANFO is transported from the 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 use high pressure in the container. The ANFO is pumped 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 charging 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 via a charger. As ANFO is highly corrosive, all machine parts that come in contact with ANFO are made of stainless steel. ANOL is manufactured in sizes of 100, 150, 300 and 500 liters. The charging machine is a combined pressure/ejector unit for the charging of prilled ANFO in upward blastholes with diameters between 32 - 51 mm and a depth of up to 45 m. The ANFO is transported by the ejector at 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. The charging hose is anti-static as the ANFO is transported through the hose at high velocity causing a risk of static electricity accumulation. Due to this risk, all ANFO charging units must be grounded during charging operations.

Charging with pump trucks

In tunnel blasting operations, the explosive or blasting agent may be charged into the hole by a pump truck. An explosive or blasting agent, such as emulsion, can be manufactured at an on-site plant and pumped directly from the plant into the pump truck.

Care must be taken when charging holes containing water. The charging hose must be introduced below water level to the bottom and lifted at the same pace as the hole is filled to avoid separation of the explosive column by water pockets.

EQUIPMENT FOR DRILLING

Underground drilling

Today in underground drill&blast excavation, drilling is mostly performed with multi-boom, hydraulic drill jumbos. Pneumatic jumbos, and hand-held drilling is being replaced by modern hydraulic units which offer efficiency, lower overall cost and occupational health & safety factors.

The equipment used in construction projects must typically be able to perform multiple duties in addition to face drilling. It must be compatible with other machines and systems at the site, in maintenance as well as service arrangements.

The payback time for most equipment is quite short, so the selection process is demanding. Detailed calculations and comparisons are necessary to determine which equipment is the most economical, efficient and technically suitable for each project.

Equipment selection

Drilling is governed by numerous rules and regulations. All drilling units must therefore conform to global and local requirements which in turn affect the construction and manufacturing methods, manuals and labels on the units.

The equipment itself must be able to efficiently execute the drilling tasks, and adapt to different and often changing conditions, such as different face areas, rock conditions and hole lengths. In most cases, drilling equipment must perform several different tasks during each project, especially during the unit´s effective life time and during different projects.
Conditions can change, for example:

- Changing face areas and geometries of tunnels
- Tunnel curvature and cross-cuts
- Design and scheduling of the work cycle
- Different rock conditions
- Conditions of the terrain
- Gradient of the tunnel
- Length of tunnel and tramming length to the face
- Different hole size and hole length
- Drilling long holes for exploration or grouting purposes
- Drilling bolt holes
- Electric supply network

Machine and component selection has a fundamental effect on performance in different conditions.

Carrier selection

In mechanized drilling units, the carrier's task is to move the unit around the worksite and provide a mounting frame for necessary components on the machine. Its main characteristics are typically tramming speed, tramming capability on various terrain and slopes and stability of the unit.

Three basic carrier models are available: rail-mounted, crawler-mounted and wheel-mounted. Each model can be used in drilling units for underground excavation.
Rail-mounted carriers are the traditional in face drilling units. Today they are used less frequently because all other equipment must also operate on rails or within the limitations set by the rails in the tunnel.

Rail-mounted equipment can be justified in long, horizontal tunnels because they can be built small in dimensions. In greater tunnel lengths, the drilling unit can be quickly transported to and from the face with the locomotive used in rock hauling (**FIGURE** *6.2.-21.)*.

Portal drilling rigs have a passage through the machine frame for letting trucks, loaders and other traffic through. They are mostly built on rail carriers and are used in large tunnels.

The disadvantages of rail carries are the necessity to build the rail system, poor ability to tram up and down slopes, and poor ability to operate in tight curves, cross-cuts and access tunnels.

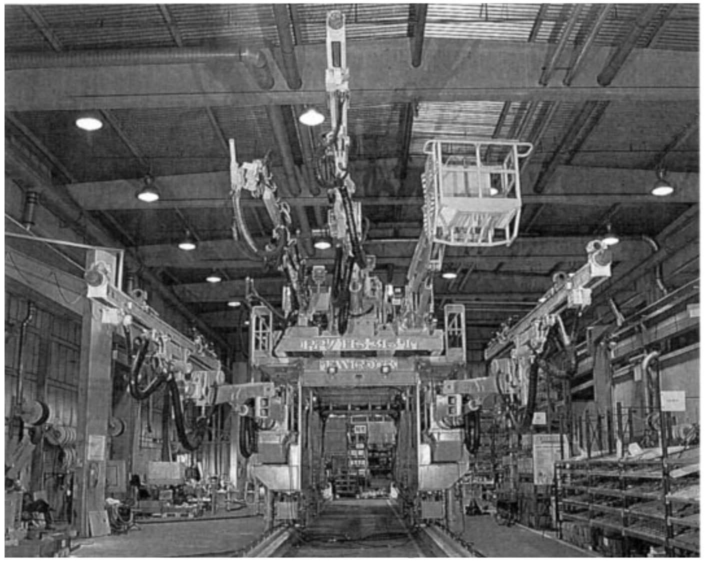


FIGURE 6.2.-21. *Rail-mounted portal-type drilling rig*

The crawler-mounted carrier is built on crawler tracks, typical in surface excavation. Due to its massive size (needed to provide sufficient stability for multi-boom jumbos) and slow tramming speed, it has been mostly replaced by wheel-mounted units.

The crawler carrier works best on rough pavement and steep tunnels, and is handy when tramming speed is not an essential criteria.

Wheel-mounted carriers are presently preferred in tunneling and are suitable in many situations from horizontal tunnels, up to 20 degrees slopes. The biggest advantages are mobility and versatility in most tunneling conditions (**FIGURE 6.2.-22.**).

The carrier can be dimensioned to give adequate stability for the machine according to its number of booms and total weight. Typically, carriers are center-articulated or rear wheel-steered.

FIGURE 6.2.-22. *Wheel-mounted drilling rig.*

Selecting booms

Earlier tunneling booms were specially designed for face drilling. It was not possible to change the boom angle for vertical drilling during operation. Requirements for modern drilling units include multiple-task performance, fast and accurate boom movements and automatic parallel holding in all directions. This has led to development of so-called “universal” or roll-over booms.

The roll-over boom’s rotation unit is located at front end of the boom arm, as the boom arm can be moved in vertical and horizontal directions. This boom type provides optimum-shaped drilling coverage, which enables the unit to drill curves, bolt hole rings, benches and cross-cuts as well as ordinary face holes.

Boom size depends on the required coverage of the drilling unit, number of booms on the machine and mounting distance and height of the booms on the carrier (**FIGURE 6.2.-23.**).

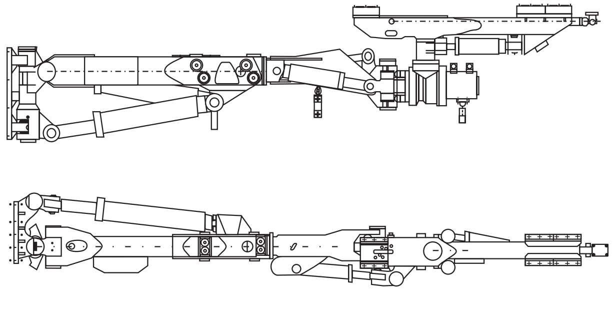


FIGURE ***6.2.-23.*** *Tamrock Roll-over boom TB60.*

Number of booms

In theory, more booms proportionately increase the drilling capacity, but in practice this depends also on utilizing all booms during drilling.

Most hydraulic drilling units have up to three mounted drilling booms that can drill simultaneously. Large portal-type rigs can have even more booms on the unit.

Drilling with a multi-boom jumbo normally involves some over-lapping or a waiting period (few minutes), that can be reduced by the boom coverage of the rig, which should be appropriate for the tunnel face area, and the experience of the operators. Computerized rigs can minimize the waiting period because they use a pre-programmed drilling pattern. Boom movements and drilling functions are also automatic and can be programmed to give optimum performance.

Selecting the feed

The feeding system keeps the shank in contact with the rock drill and the drill bit in contact with the rock during drilling. The optimum feeding system is balanced with the percussion dynamics of the rock drill and drill string, and meets the requirements for various drilling applications.

Typical feeding systems used in mechanized drilling units are operated, for example, by a feeding screw, hydraulic motor and chain or hydraulic cylinder and steel wire.
In modern hydraulic units, the cylinder feed is mostly used because it provides a constant, stable feed force to the rock drill during drilling.
Feed length, which determines the maximum length of the hole, and the round, is mostly determined by geological factors and vibration restrictions. It is typically defined as being the length of the drill rods. Typical rod length in tunneling varies between 12 - 20 feet, allowing a net drilled length of hole from 3.4 - 5.8 meters.

Bolt-hole drilling in small section tunnels sometimes requires a telescopic feed that allows the same feed to handle longer drill rods for face drilling, and when retracted shorter rods for bolt hole drilling.

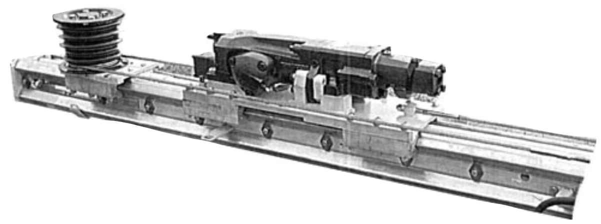


FIGURE ***6.2.-24.*** *Feed for tunneling jumbos.*

Rock drills

Correctly matched rock drills are critical components for ensuring drilling performance, a long life time for drilling accessories and good overall drilling economy. The rock drill performs the toughest job so it should be reliable and easy to maintain. A reliable rock drill ensures trouble-free drilling rig operation.

In various rock conditions, the rock drill requires adequate adjustments for the highest drilling efficiency. The percussion method can handle a wide range of drilling tasks from fairly soft ground to very hard rock, and from poor to good rock mechanical conditions. Most changes in geological conditions do not require special modifications to the drilling system. When rock conditions vary, it is usually sufficient to monitor the drilling and adjust it according to the basic parameters: percussion and feed pressures, rotation speed or flushing pressure or bit type change. For cases in which poor rock conditions occur frequently or drill steels get stuck, alternative flushing methods such as air-water mist (occasionally with foam or other chemical additives) may provide a solution.

Tunneling accuracy

The demand for quality is continuously increasing in underground excavation work. It is one of the most important factors for overall economy, and it also greatly affects safety and the environment.

Face drilling is just one of the many stages in tunneling, but it has a strong effect on the quality and cost of the total excavation process. The main purpose of instrumentation in face drilling is to improve drilling accuracy and allow tools to optimize the drill and blast cycle. The instrumentation available on modern drilling units can be defined as three different technological levels: Angle indicators, angle and position indicators and fully computerized systems.

Angle indicators are simple instrumentation tools that show the look-out angle of the drill feed. Simple versions show the direction in reference to machine direction and gravity field. Sophisticated versions can be navigated to the direction of the round, providing the hole's true look-out angles in the round. The system shows the horizontal and vertical feed angle either numerically or graphically. Other basic drilling information is also provided by the measuring system such as hole depth, drilling speed etc.

Computer Aided Drilling System can show graphically both the angle and position of the feed rails. The drilling pattern is preprogrammed, and the operator can use the display as an aid to accurately spot the holes. Because of the pre-programmed drilling pattern and navigation to the tunnel reference line, no marking-up of the face is needed before drilling the round. The instrumentation also includes features for data logging, drilled round data capture, such as actual position and angle of the holes, amount of drill meters, drilling time, drilling parameters etc. This information is useful for optimizing the drill & blast design, work control and estimating rock conditions.

FIGURE *6.2.-25. Hole position and drill angle instrument user interface (TCAD).*

Fully computerized jumbos are entirely automatic, according to the preprogrammed drilling pattern. The automatics handle the entire drilling cycle including all drilling functions, moving from one hole to another, adjustments etc. Data control allows one operator to supervise simultaneous drilling with three booms.
The role of the operator is to supervise the drilling, and make adjustments when and if necessary.
In fully computerized units, the position and depth of each hole as well as the drilling sequence of each boom must be planned and programmed into the drilling rig. For the drilling pattern designer, this offers an excellent opportunity to optimize the drilling operation, together with possibility to exactly plan and implement the charging and blasting of each round.

FIGURE *6.2.-26. A fully computerized drilling jumbo.*

SELECTING DRILLING TOOLS IN DRIFTER DRILLING

The most important factors in drifter drilling are:

- Collaring accuracy
- Straight holes
- High productivity
- Long service life and grinding interval
- High penetration rate

Together they give the customer minimum over/underbreak, smooth tunnel profile and high rate of excavated tunnel per hour.

Formula 1

Formula 1 is the system that meets all requirements on the above list. Formula 1 is a unique and patented system which offers substantially lower cost per excavated tunnel meter. The features that enable Formula 1 to deliver the benefits described above are:

- A super rigid 39 mm-round rod section
- FF (Female/Female) rod threads
- A male-threaded drill bit
- Straight transition from rod section to bit head shoulder (i.e. no "gooseneck" on the rod)
- Patented impact energy path into the drill bit's peripheral buttons
- Higher flushing velocity

Combined they offer straight holes, high collaring accuracy and a high penetration rate.

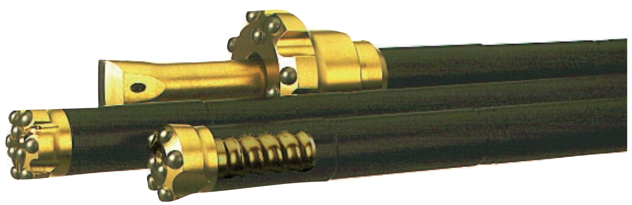


FIGURE ***6.2.-27.*** *Drilling tools.*

Formula 1 was primarily developed for demanding customers with high precision and productivity requirements. These customers use powerful hammers such as the HL500/550 and modern drill rigs. Formula 1 is the best choice when using data-controlled jumbos.

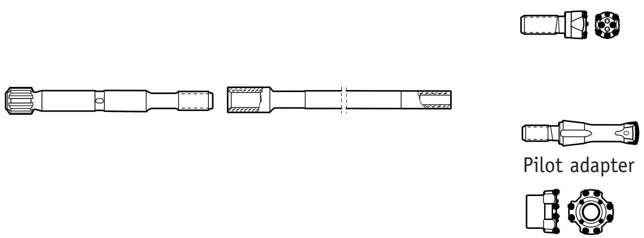


FIGURE ***6.2.-28.*** *Formula 1 drilling tools.*

The system consists of a T38 shank adapter, T38/R32 drifter rod, and a female threaded button bit, R32 pilot adapter and reaming bit. The standard rod comes in 3.7, 4.3, 4.7, 5.1, 5.6 and 6.1 m lengths. Bit sizes are 48 and 51 mm. A wide selection of drill bit designs, carbide grades and button shapes are available depending on rock conditions.

R38/T38 drifter drilling

When requirements for precision and productivity are lower, or a less powerful hammer is used, a standard system might be sufficient. The standard system consists of a shank adapter, drifter rod with lose coupling sleeves or MF rods, insert or button bit, pilot adapter and reaming bit. Two thread sizes are available as standard: R38 and T38 with rod dimension H32 and H35. Larger dimensions are suitable for hammers with up to 21 kW (HL 550) output and hole dimensions between 45 and 51 mm. The larger rod dimension is also recommended when drilling holes deeper than 3.7 m.

MF rods are more expensive, but will give straighter holes and a 10% higher penetration rate. They are only recommended in good rock conditions. In poor rock conditions, rods with coupling sleeves are recommended.

As with the Formula 1, a wide selection of bits is available. To achieve optimum life and grinding interval as well as penetration rate, bit design, carbide grade and buttom shape must be selected depending on rock conditions.

The standard drifter system is schematically described below.

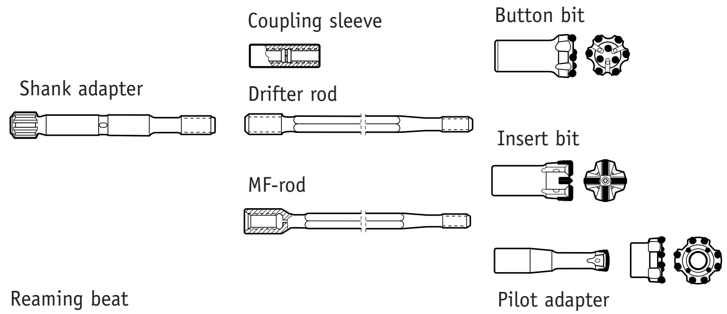


FIGURE ***6.2.-29.*** *The standard drifter system schematically described.*

Sandvik Coromant Rock Drilling Tools for rock bolting with Tamrock HE 300, HL 300S and HL 500F

H25 Integral drill steels

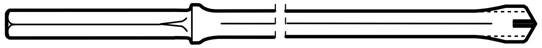


FIGURE 6.2.-30. *Integral drill steel.*

H25 integrals are manufactured in 3 standard chisel bit dimensions: ø 32, 35 and 38 mm on request to suit actual bolt lengths.

Rod and bit.

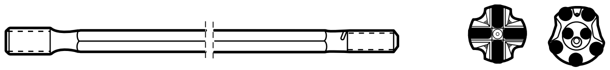


FIGURE 6.2.-31. *Rod and bit.*

Various rod alternatives are possible depending on which hole size is requested. All rods must carry a R32 shank end thread to fit into the shank adapter.

The R25 bit end thread makes it possible to drill ø 35-38 mm holes with button or cross bits.

An ø 33 mm bit can be used on a special rod with R23 bit end thread.

The R28 bit end thread makes it possible to drill ≥ 38mm holes.
As rod lengths depend on bolt dimensions, there may be cases where standard lengths can not be used. Different rod lengths are therefore manufactured upon request.

Threaded integral drill steels

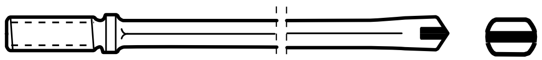


FIGURE 6.2.-32. *Threaded integral drill steel.*

An interesting alternative is the threaded integral which makes it possible to drill ø 32 mm holes suitable for resin or grouted bolts.
The rod section is Hex25 and lengths are available up to 4.5 m, suitable for 4.0 m bolts.

SCALING

The purpose of scaling is to clear loose rock from walls and surfaces after blasting. Manually done it is hard work involving many safe hazards such as falling rock and dust, and requiring awkward working positions. Scaling is often very time consuming when done manually. Today, modern mechanized scaling equipment is used whenever possible.

Barring

Barring is a scaling method that uses a hydraulically powered tooth. It is frequently used with sedimentary rocks for scaling large roof surfaces without unduly disturbing the rock layers above. This is a hydraulic, mechanized form of the manual method in which the tip of a scaling bar is placed in a joint and twisted. This method uses a hydraulic tooth instead of an iron bar.

Scraping

It is difficult to find joints using a hydraulic tooth so barring is often replaced by a scraping action. Loose rock is scraped off the rock surface either with special pointed tools or the teeth of a loading bucket. When the teeth catch on loose rock they pull it away. This method is most effective in the initial scaling phase and for removing loose rock from surfaces. Scraping is especially used for wheel-loaders when securing the face.

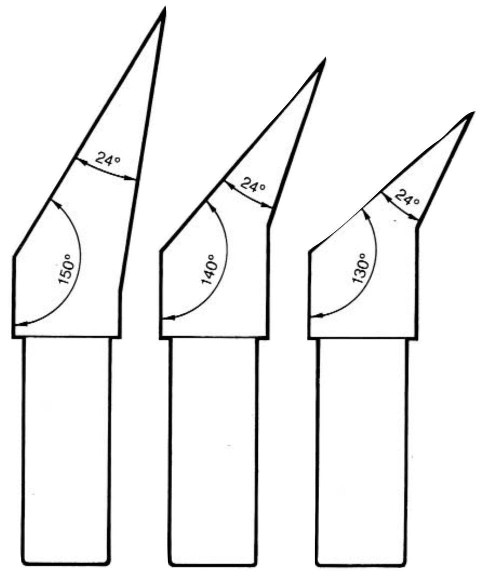


FIGURE 6.2.-33. *Typical tools for scraping and barring.*

Hammering

Impact hammers are often used for scaling hard rocks by striking the places in the rock face which are suspected of being loose. Power is adjusted to match the toughness of the rock so that excessive rock is not loosened. This method is very reliable when scaling hard rocks.

Cutting- Drag tooled cutterhead

Rotary hydraulic cutterhead tooled with conical picks are also for scaling drill and blast surfaces when the rock is from moderately soft to moderately hard as well as heavily jointed. This method is mainly used for roof scaling to reduce overbreak. Hydraulics come to the cutterhead from a backhoe excavator.

Scaling devices based on hammering

A diesel-hydraulic unit is generally chosen, since it gives the greatest independence for mechanized scaling equipment. It can constantly move freely and does not require external power cables. Modern scaling equipment has a safe and comfortable cabin to protect the operator from falling rock and dust, and a dozerblade to push aside fallen rock. Dust is also suppressed with water. Scalers that are designed for very large construction sites may have a charging basket for utility works.

VENTILATION

General

In tunnel excavation, a ventilation system is required to provide an acceptable working environment for the people in the tunnel. The environment is affected by the concentration of impurities in the tunnel air. The impurities are mostly created by blasting and traffic in the tunnel. Limit values for gas and particle concentrations are set by the authorities, thus the design and dimension of the ventilation system must achieve the defined limit values. On the other hand, ventilation system efficiency has a considerable effect on the performance of the whole excavation cycle.

Harmful concentrations

The concentration of harmful substances in air is defined as:

- For gases as parts per million:
  1 ppm = 1 cm^3 gas per 1 m^3 air

- For dust particles as amount in mg / m^3 air

Gases:

NO_2, nitrogen dioxide, is a very toxic gas. It is created during blasting and by diesel engines. Part of NO (nitrogen oxide) becomes NO_2 in the tunnel environment. Health risks from NO_2 start with very low concentrations; a typical limit value for NO_2 is two ppm. As much as 2 - 5 ppm may cause chronic bronchitis, and even short exposure to high concentrations may cause breathing difficulty or death. NO_2 has a reddish brown color and carries a distinctive smell. It is water-soluble, and therefore water-spraying the muckpile is very important after blasting before beginning other work at face.

NO, nitrogen oxide, is a colorless toxic gas, and is not soluble to water. The typical concentration limit is 25 ppm. When exposed to oxygen, it slowly transforms into NO_2. Simultaneous concentrations of NO and CO (carbon monoxide) can cause health risks, but in general, NO is not considered to be among the most dangerous gases.

Aldehydes give off a distinctive smell of diesel fumes. An 0.5 ppm limit value is typically given to formaldehyde (HCHO). Concentrations above 1 ppm cause eye irritation and respiration difficulties.

CO, carbon monoxide, is created by both blasting and diesel engines. It may be especially dangerous in closed or inadequately ventilated tunnel areas. CO is more easily absorbed into blood hemoglobin than O_2, resulting in reduced oxygen access to the blood. CO concentrations above 35 ppm may cause symptoms such as weariness, headaches, chest aches and, in the worst case, death.

CO_2, carbon dioxide, is found in exhaust fumes. Alone it is not highly toxic, but a high CO_2 concentration reduces the oxygen content in the air.

NH_3, ammonia, is a corrosive gas. It can be the result of a chemical reaction between ammonium nitrate and basic components of cement. Ammonia is easily water soluble, and therefore it is important to carefully spray the muckpile with water. The normal limit concentration for NH_3 is 25 ppm.

O_2, oxygen. Air is normally made up of 21% oxygen. Too little O_2 content causes respiration difficulties, brain damage and death. In underground projects, the O_2 content should not be lower than 19%.

Other air impurities

Dust is solid particles contained in the air. Health risks pertaining to dust depend on the chemical composition of particles, particle size and the concentration mg/m^3. Long-time exposure to dust causes lung disease. The most dangerous dust particles are, for example, quartz (silicoses) and fiber-formed particles (asbestos).Dust created during concrete spraying, especially with the dry-mix method, is also harmful. Administrative norms usually give maximum total concentrations of dust particles as a function of quartz content in dust.

Ventilation principles - Explosion gases

An air/toxic blast fume combination is created when blasting a round. This gas has a high NO_2 and CO, content so that even a short stay in the area is dangerous.

The toxic gases concentration depends on the type of explosives used and on how charging is performed. Carbon monoxide (CO) and nitrogen oxide (NOX) content may increase as a result of poor cartridge tamping, water in the blast hole and poor ignition. When using ANFO (ammonium nitrate mixed with oil), the oil content affects the creation of CO in blast fumes. If ANFO is exposed to cement or concrete, it creates also ammonia (NH_3).

Ventilation of explosion gases can be divided into two main categories: Blowing ventilation and two-way ventilation. The main purpose is to dilute the explosion gas plug so that toxic gas concentration is acceptable, and get the next stages in the drill & blast cycle started.

Blowing ventilation:

This is the easiest and most used method in tunneling. Fresh air from the outside is blown through a duct into the tunnel, relatively close to the face.

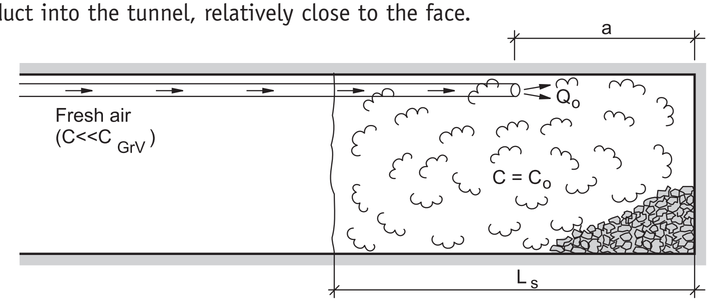


FIGURE 6.2.-34. *Blowing ventilation.*

The fresh air dilutes the gas plug and starts to move it backwards out of the tunnel. Other works such as loading and hauling can start at the face when toxic concentrations in the gas plug have been brought down to an acceptable level. Further ventilation can be dimensioned according to the loading and hauling equipment, and further impurities from the muckpile.

Two-way ventilation:

Especially in longer tunnels with larger cross-section areas, blowing ventilation is not adequate, or requires too long a ventilation time before the cycle can continue. Therefore, two-way ventilation is becoming a common method in tunnels that are longer than 1000 m. Two-way ventilation removes the explosion gas plug from the tunnel fast, providing an improved working environment in the tunnel.

In two-directional ventilation, the explosion gases are sucked from the tunnel through a duct to the outside of the tunnel. Substitutive air is led to the tunnel through a blowing duct (two-duct system), or through the tunnel (one-duct system).

The two-duct system is practical in long tunnels (> 4 km). The system removes explosion gases fast and effectively. After the explosion gases are removed, both ducts can be used for blowing ventilation to get even more fresh air into the tunnel during loading and transportation. However, the system requires space for two ducts. An out-going blowing tube must have a relatively high flow velocity, and leakage must be very small to prevent explosion gases into the tunnel through the back (**FIGURE** *6.2.-35.)*.

The one-duct system is practical in tunnels up to 4-5 km in length. With this system there is only one ventilation duct in the tunnel. At the tunnel face end, there is a two-fan system

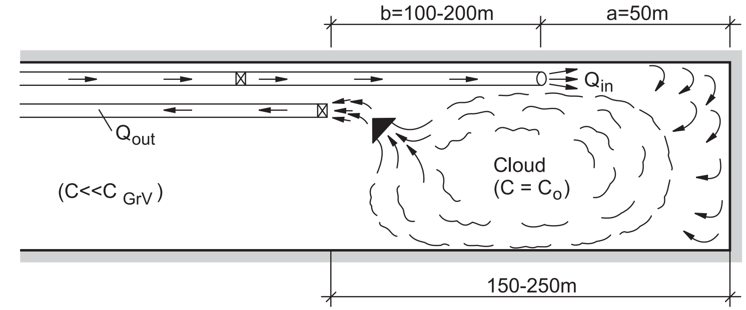


FIGURE 6.2.-35. *Two-way ventilation using two ducts.*

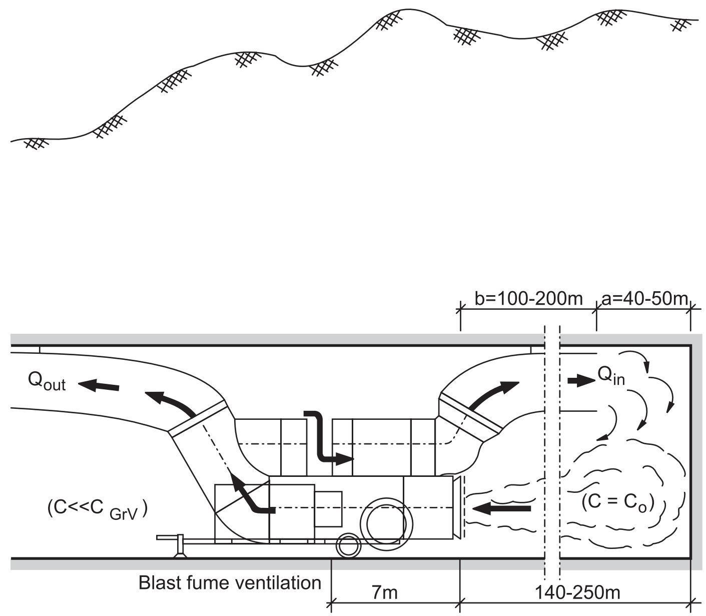


FIGURE 6.2.-36. *Two-way ventilation by single duct.*

which controls the ventilation according to the stage in the drill & blast cycle. During drilling, charging, loading & hauling, the system is used for conventional blowing ventilation. After blasting, a transverse fan is used to remove explosion gases through the duct while the other fan blows fresh air towards the face to ensure that all explosive gases are mixed and removed. The one-duct system removes explosion gases fast and effectively, and is more cost-effective than the two-duct system. The one-duct system also requires good duct quality and tightness to prevent impurities from leaking back into the tunnel (**FIGURE** *6.2.-36.)*.

Diesel engine exhaust gas

In the ventilation system and required fresh air flow in the tunnel, loading and transportation diesel equipment is usually the determining factor. Exhaust gas from diesel engines contains N_2, CO_2, H_2O, O_2 and some harmful solid particles. In most countries, engines must be approved for underground use, and the engine manufacturer is required to provide documents for approved concentrations of toxic gas or impurities in exhaust gas. However, the most important factor affecting these harmful contents is the service and maintenance of mobile equipment. Engine adjustments are important as well as the condition of the exhaust purifier. The most typical purifiers are catalytic, water scrubbers, exhaust gas ejectors and solid particle filters.

Ventilation requirements for diesel exhaust are usually estimated as the amount of fresh air per kW engine power or per kg of diesel fuel that is used. Typical values are approximately 3 - 3.5 m^3/min per engine kW, or 1.400 - 1.600 m^3 per kg used diesel fuel. Ventilation requirements also depend on road quality (tramming speed, creation of dust, rolling resistance) and tunnel inclination.

Some explosive gas is bound in the muckpile after blasting so adequate ventilation and water spraying during loading work is important and should be stressed. Released gases during loading work must be diluted and removed. The NO_2 concentration is removed and dust is minimized by spraying water onto the muckpile.

6.2.2. Mechanical tunneling

A) PART FACE

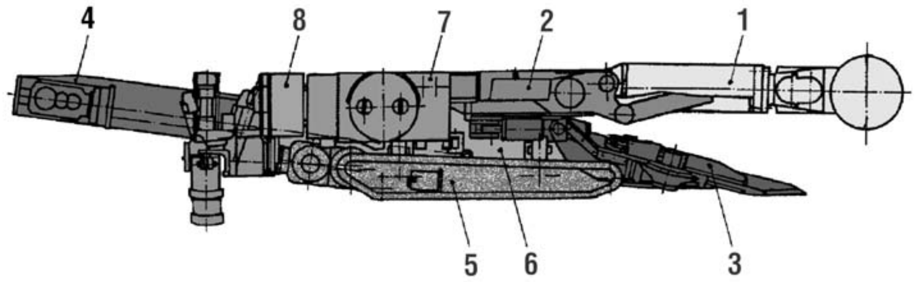
ROADHEADERS

The first roadheaders were used for tunneling in the 1960s. By the early 1970s, approximately 150-200 roadheaders were used for underground civil construction. It was during this early phase that boom-type cutting equipment in shields or on other hauling structures such as excavators also became popular.

Basic design and operating features

The standard roadheader features the following functions:

- Rock excavation (rock cutting)
- Gathering of excavated muck
- Muck transfer to secondary conveying equipment
- Machine transfer

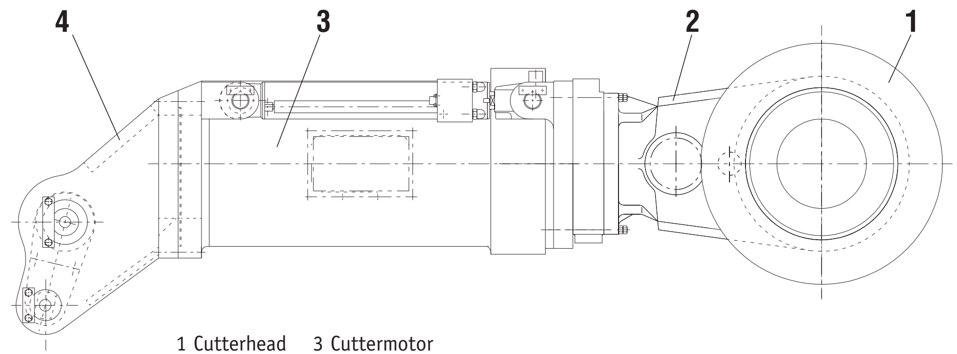


1 Cutter Boom 2 Turret 3 Loading Assembly 4 Chain Conveyor
5 Track drive 6 Frame 7 Electric Equipment 8 Hydraulic Equipment

FIGURE *6.2.-37. Roadheader main assembly groups.*

Cutter boom

The cutter boom comprises the roadheader's actual rock disintegration tool. (**FIGURE** *6.2.-38.*) The cutter boom has the following components: its base, motor, coupling between the motor and gear, and the head.



1 Cutterhead 3 Cuttermotor
2 Cutterhead 4 Cutterboom socket

FIGURE *6.2.-38. Cutter boom components.*

Cutter heads

Two main design principles are applied:

- Longitudinal or milling type cutter heads rotating parallel to the cutter boom axis
- Transversal or milling type cutter heads with rotation perpendicular to boom axis.

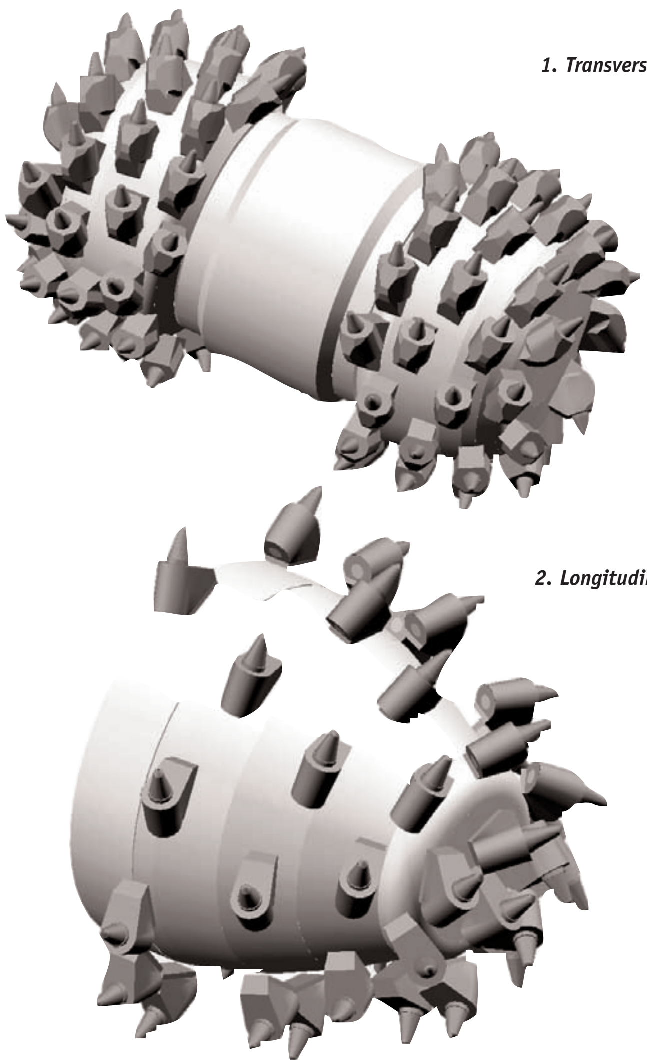


FIGURE ***6.2.-39.*** *Types of cutter heads*

Both cutter heads have several advantages and disadvantages.

Some main features important to tunneling are mentioned here:

- Transversal cutter heads cut in the direction of the face. Therefore, they are more stable than roadheaders with longitudinal heads of comparable weight and cutter head power.

- At transversal heads majority of reactive force resulting from the cutting process is directed towards the main body of the machine.

- On longitudinal cutter heads, pick array is easier because both cutting and slewing motions go in the same direction.

- Roadheaders with transversal-type cutter heads are less affected by changing rock conditions and harder rock portions. The cutting process can make better use of parting planes especially in bedded sedimentary rock.

- If the cutter boom's turning point is located more or less in the axis of the tunnel, a cutter head on longitudinal booms can be adapted to cut with minimum overbreak. For example, cutter booms in shields where the demand can be perfectly met are often equipped with the same type of cutter head. Transverse cutter heads always cause a certain overbreak regardless of machine position.

- Most longitudinal heads show lower figures for pick consumption, which is primarily a result of lower cutting speed.

- The transverse cutter head offers greater versatility, and with the proper layout and tool selection, has a wider range of applications. Its performance is not substantially reduced in rock that presents difficult cutting (for example, due to the high strength or ductile behaviour).

- Additionally, the reserves inherent in the concept offer more opportunities for tailoring the equipment to existing rock conditions.

Cutter picks

Since its first application on a roadheader cutter boom in 1972, the conical pick equipped with tungsten-carbide tips (also called point-attack picks) has become more important and is today the most commonly used pick. (**FIGURE** ***6.2.-40.***).

1 WC-inset
2 Cone
3 Shaft
4 Retainer ring
5 Pick box
C Cutting depth
F_C Cutting force
F_N Normal force
F_D Driving force due to friction on rock
F_R Frictional resistance between pick and pick box

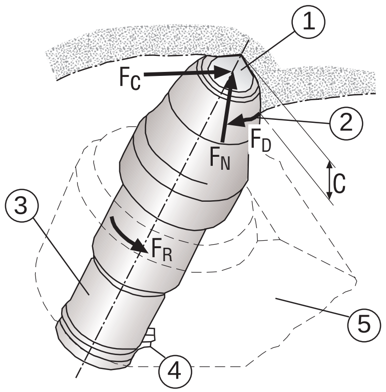


FIGURE *6.2.-40. Point-attack-pick, design and indentation parameters*

Process tuning

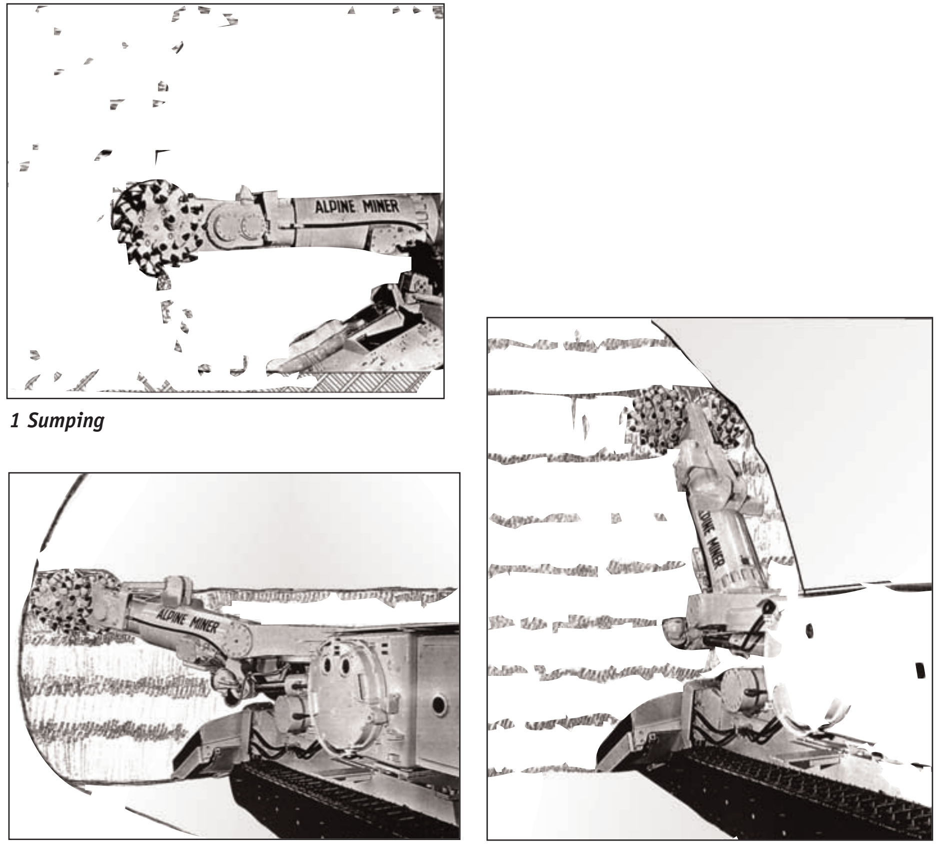
Tuning the cutter head's cutting process to the existing rock conditions is crucial to achieve optimal cutting.
Theoretically, the highest possible spacing of cutters results in the optimal interaction between the cutter head and rock:

- Relative pick track per unit volume of excavated rock is reduced
- Reduced pick-track length also results in better energy utilization and, therefore, a faster cutting rate
- Less dust generated
- Reduced wear (picks/bank m^3)

Excavation sequence

Contrary to TBMs, which simultaneously attack the entire face with a fixed tool configuration, the operation of a roadheader comprises different steps of the excavation process. (**FIGURE** *6.2.-41.*)

The first step, the sumping of the cutter head into the face, is performed by the forward movement of the entire roadheader via its crawler tracks or alternatively through special cutter boom design. Telescopic or articulated cutter boom design can also perform this task. (**FIGURE** *6.2.-42.*) Because the sumping process requires the most power in the cutting sequence, there is less impact to the floor because the sumping is performed without engaging the crawler tracks. Further excavation of the face is primarily performed by the horizontal swiveling of the cutter boom with vertical offset of the boom when reaching the tunnel outline. All horizontal and vertical movements of the boom are performed by the turret. The turret itself serves for the horizontal movement of the boom. Closed rack and pinion drives or external hydraulic cylinders are used for this task.



1 Sumping

2 Cutting of face

3 Profiling (if required)

FIGURE *6.2.-41. Steps of excavation by a roadheader*

Vertical movement is performed by various swivel cylinders; the reactive forces are again transferred into the turret. If necessary, an extra profiling step minimizes the tunnel´s ribs and brings it closer to its theoretical shape. This excavation process is fundamental to roadheader versatility regarding the shape and size of the tunnel section.

A roadheader can, within its design dependent geometrical limits (defining minimum and maximum cross sections etc.) cut practically any required shape and size. It can also follow all necessary transitions and alterations and is highly adaptive to differing excavation processes. By using cutter booms with telescopic or special design, this important feature can be enhanced even further.

main motion


auxiliary motion

FIGURE 6.2.-42. *Examples of telescopic and articulated cutter booms*

The excavation of short roof sections and consecutive benching from one machine position can be effected, making properly equipped roadheaders the perfect tool for coping with the demands of the NATM in ground conditions with poor stability.

Loading and transferring muck:

Mucking can be performed during the excavation process.
Relevant loading and hauling devices are an integral part of the roadheader.
A loading apron in front of the machine´s main body consists of:

- Gathering arms, which are considered best suited to handlecoarse, blocky muck. This application is also well suited for tunnel operations.
- Wear-resistant spinner loaders that can handle high muck volumes when used for mineral production, such as in coal mines.
- Swinging loading beams which form a very simple and rugged solution, but offer a some what restricted loading capacity.

Various loading devices can be used. The most common are shown in **FIGURE** ***6.2.-43.***

Tramming facilities

Roadheader weight, together with the high loads and vibrations of the cutting process, makes crawler tracks the only reliable solution.
In tunneling, roller-type crawler tracks are considered generally advantageous because they offer better maneuverability and higher tramming speeds (up to 35 m/min.). Nevertheless, sledge type crawler tracks offer superior resistance against shockloads and are used in hard rock applications.

Classification of roadheaders, performance

Main classification features:

Two interrelated features form the main figures for classification:

- Machine weight
- Power of cutter motor

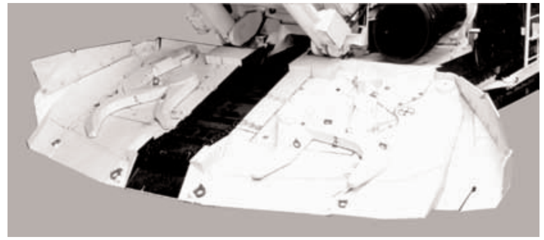
Machines with two types of cutting range are featured in the table: Machines with standard and extended cutting range. ***Table 6.2.-1.*** indicates the range of the defined classes with regard to their main features and limit of operation.
The max. section in the table represents the position max. area which the roadheader can cut according to its design parameters.

Capacity and performance:

Technical capacity, such as the highest cuttable rock strength, is also shown in ***Table 6.2.-1.*** This represents the highest strengths that can be handled according to the weight and power of a machine equipped with certain features.
The above-mentioned limitations regarding rock strength must be seen as the first indication of capacity and performance. In practice operating limits and performance are influenced by other rock parameters and also depend on the actual layout of the machine and actual site conditions (**FIGURE** ***6.2.-44.***).

FIGURE ***6.2.-45.*** shows the involved parameters. It also outlines the practical way to determine the most important parameters of roadheader operation:

- Cutting rate
- Pick consumption


1 Gathering arms


2 Spinner loader


3 Two lateral loading beams

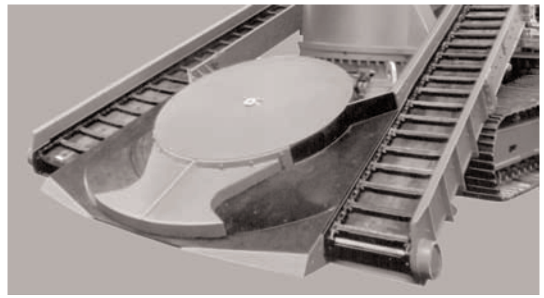

4 One central loading beam

FIGURE ***6.2.-43.*** *Main types of loading assemblies on roadheaders.*

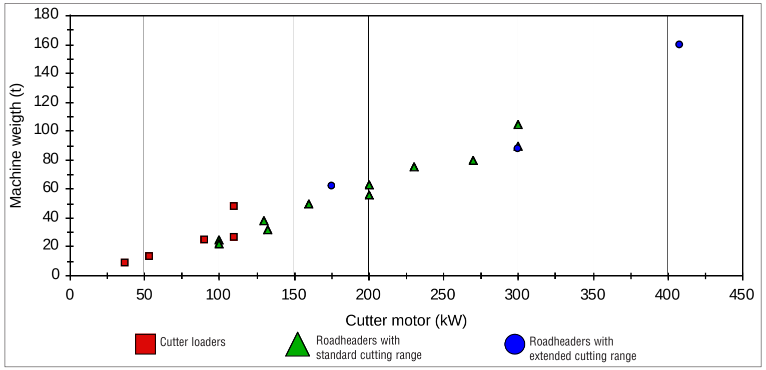


FIGURE 6.2.-44. *Relation between weight and cutter head power of roadheaders.This table also provides an introduction to machine selection for various project conditions.*

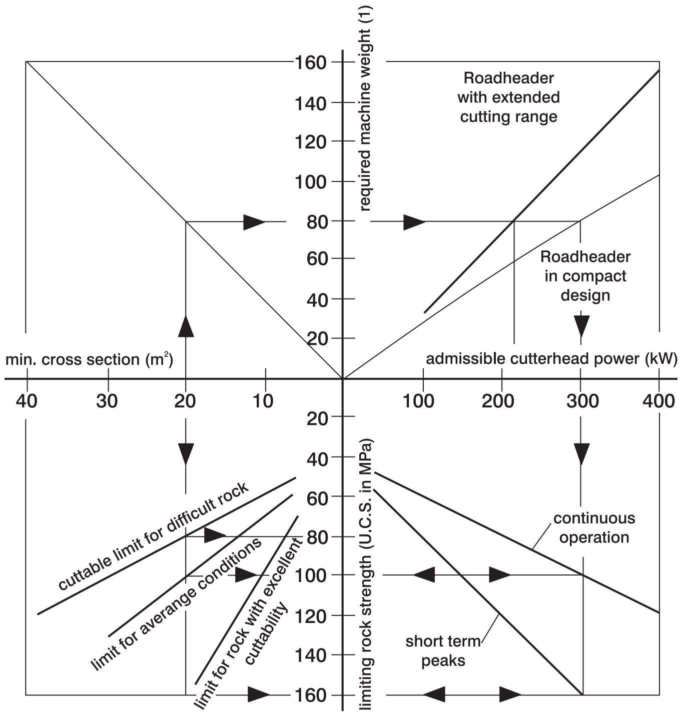


FIGURE 6.2.-44B. *Indiative diagram for roadheader selection. This diagram can be used for first selection of an appropriate machine for certain project conditions. It indicates the maximum weight installed power to be used on this machine and maximum rock strength, which can be tackled. Smaller machines with lower powering can be also used, if they cope with the demandes of rock and project.*

Table 6.2.-1. *Classification of roadheaders.*

Roadheader class	*Range of weight (to)*	*Range of cutter head power (kW)*	*Range of operation* Roadheaders with standard cutting range		*Roadheaders with extended cutting range*	
			max. section (m^2)	*max. u.c.s. (MPa)*	*max. section (m^2)*	*max. u.c.s. (MPa)*
Light	8 -40	50 - 170	~ 25	60 - 80	~ 40	20 - 40
Medium	40 - 70	160 - 230	~ 30	80 - 100	~ 60	40 - 60
Heavy	70 - 110	250 - 300	~ 40	100- 120	~ 70	50 - 70
Extra heavy	> 100	350 - 400	~ 45	120 - 140	~ 80	80 - 110

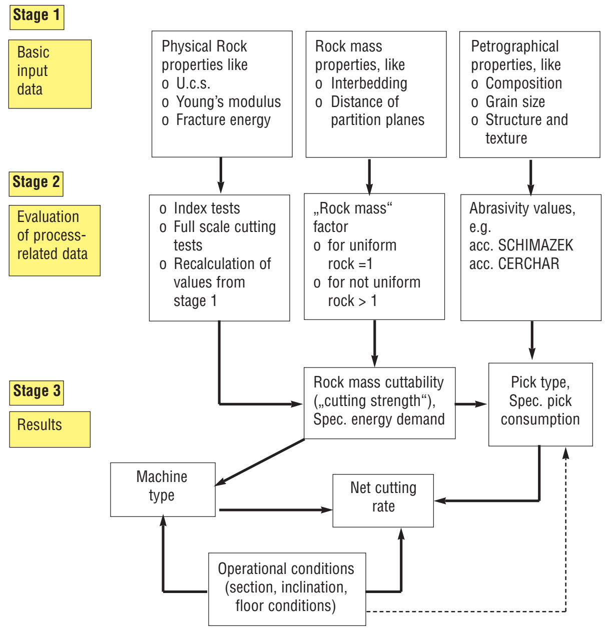


FIGURE 6.2.-45. *Assessment of cutting rate and pick consumption for roadheader operation.*

This diagram is valid for standard operating conditions. Additionally, it is based on results achieved by skilled personal and is not influenced by rock mass properties. It also provides a good picture of roadheader potential in various operating conditions.

Recent developments that improve performance and cost effectiveness:

Since 1990, advanced-design roadheaders have operated in rock formations that are considered not only difficult but also where the economic application of roadheaders is not attainable.
One important limiting factor was insufficient power; the machine's inability to transfer the installed power into the face.
Two important developments, which were effected or became efficacious in tunneling during this period shall be presented here:

- Roadheaders with switch gear, which allows the application of the fully installed cutter head power at a reduced cutting speed.
- Improved pick technology pertaining to the quality of tungsten carbide and support offered by high-pressure flushing systems.

Switch gears on roadheaders were added during the development of the ALPINE MINER AM 105. Via the switch gears, the advantages of a variable cutting speed, previously achieved only through pole changing motors and thus only at reduced power available at a lower speed, can be now utilized without drop of available power.

Special advantages of roadheaders for tunneling applications

General:

Primarily, roadheaders offer the same advantages as other equipment for mechanized hard rock tunneling. The fact that roadheaders are limited in regard to rock strength and abrasion at lower values compared to the TBM has been mentioned earlier in the text.
A decrease in performance in higher rock strength is also more pronounced than in machines equipped with roller cutters.

Within its range of application, the roadheader offers advantages that are exclusive to this type of equipment:

Versatility and mobility:

While a TBM is practically fixed to a circular section and a certain diameter, roadheaders can handle a great variety of sections within their layout parameters.
The face remains accessible. By retracting a roadheader from the face, all required measures for rock protection can be performed without space restrictions up to the face without significant slowdown.
Therefore, it is no problem for the roadheader to adapt to changing rock mass conditions.
Larger sections can be subdivided and excavated in progressive steps enabling the excavation of large sections that require perfect tuning of the excavation sequence.
Free space and accessibility are also key if the need for auxiliary measures, such as draining or advanced grouting, are necessary.

Low investment:

Compared to the TBM, the similar size of cross section investment costs for a roadheader amounts to approx. 0.15 (large sections) to 0.3 (small sections).
Roadheaders are also commonly rented.
Therefore, roadheader application is also attractive in short projects if the conditions fit.

Quick and easy mobility:

Comprehensive assembly equipment and chambers are not required. Roadheaders can be operated immediately upon arrival.
Although they are not sold off the shelf, roadheaders require much shorter mobilization periods. Depending on the site location, a new machine can be delivered and be ready for operation in 3 - 6 months.
Delivery time is often considerably less for used and refurbished machines.

HAMMER TUNNELING

Hammer tunneling has proven to be economic mainly in the Mediterranean countries and Asia. Hammer tunneling is successful compared to drilling and blasting when the fractured rock structure makes controlled blasting hard to achieve. Additionally, hammer tunneling involves only a few work phases and there is less need for skilled work force than in drilling and blasting.

Compared to a TBM (Tunnel Boring Machine), hammer tunneling investment costs are much lower and tunnel profile is not restricted to a particular shape. Hammer tunneling economics are governed by many factors including rock type, tunnel area, tunnel length, tunnel location, schedules, and availability of equipment and skilled work force. Usually a suitable hammer would be in the weight class of over 2000 kg; preferably over 3500 kg. However, even significantly smaller hammers are used in special cases. The soft-rock chisel tool is usually recommended for tunneling.

In a typical hammer tunneling case, the main advantages over other methods are lower investment costs, lower work force costs, safer job-site conditions (because explosives are not used) and little or no over-excavation with costly refills.

Rock types

For hammer tunneling to be economic, a reasonable productivity rate is required. This can be achieved in different rock types. Rock to be excavated has relatively incoherent structure. Distance between cracks, joints and other discontinuities should not be more than 30 - 50 cm. The rock to be excavated is compact but soft enough to allow a reasonable productivity rate by tool penetration (best case: an excavator bucket is barely insufficient).

Rock strength, abrasion level and general toughness also influence productivity to some extent. Rock is seldom homogenous in long tunnels. If extremely compact rock is encountered, auxiliary blasting is recommended. It is often sufficient to fracture the rock, enabling further excavation with a hammer. Auxiliary blasting is applied at the lower middle part of the tunnel where excavation normally would start. This way hammer excavation is best enhanced and the negative effects of blasting (such as overbreak) are minimized.

Ground vibrations

Considerably less ground vibration is associated with hammer excavating than with the drilling and blasting method. The vibration level caused by hammer excavation is 5 - 10% the level of blasting. This can be a decisive factor when excavating rock in the vicinity of structures that require vibration limitations.

Working methods

The working method is dictated by the section area and length of the tunnel.

Areas 30 - 70 m^2:
Hammer tunneling is suitable for tunnels with a cross-section greater than 30 m^2. With smaller areas, an excavator suitable to carry a 2000 kg hammer will have difficulties fitting or operating properly.

In a small and narrow width (less then 8m) tunnel profile, only one excavator-hammer combination can work at the front of the tunnel. This divides work into 5 phases:

- Excavating
- Transportation of muck
- Scaling
- Transportation of scaling muck
- Reinforcement and support of tunnel walls

In an 8-hour shift, excavating and transporting muck takes about 2 hours each. Scaling and transportation of scaling muck takes approximately an hour, and the rest of the time is used for reinforcement of the walls.

Area more than 70 m^2:
A larger tunnel profile allows hammer excavating and muck transportation to be done simultaneously. This reduces the actual amount of work phases into two:

- Excavating (and scaling) + transportation of muck.
- Reinforcement and support of tunnel walls.

Broken rock can be removed during excavation of a 70 m^2 tunnel face, which can accommodate an excavator equipped with a hydraulic hammer, and a loader and truck.
The excavating and transportation work phases actually complement one another. When material has been excavated from one side and instantly taken away, the hammer can immediately be transferred to the opposite side. Immediate muck removal also improves visibility to the material to be broken.

Tunnel height more than 7 m:
When tunnel height becomes too high, the reach of the hammer is insufficient for excavation in one stage. Excavation is then done in two stages (**FIGURE** *6.2.-46*):

- Tunnel excavation with suitable height for hammer and excavator.
- Another excavator-hammer combination starts approx. 100 - 150m behind the initial tunnel front to deepen the existing tunnel with the trenching method (**FIGURE** *6.2.-46.*)

When a hydraulic hammer is used, the work force requirement becomes smaller in comparison to traditional drilling and blasting excavation. This is largely because the drilling and blasting method calls for more highly trained personnel. Drilling and blasting operations also mean regular interruptions and disturbances to the tunneling process as a whole, while hammer excavating is a continuous process.

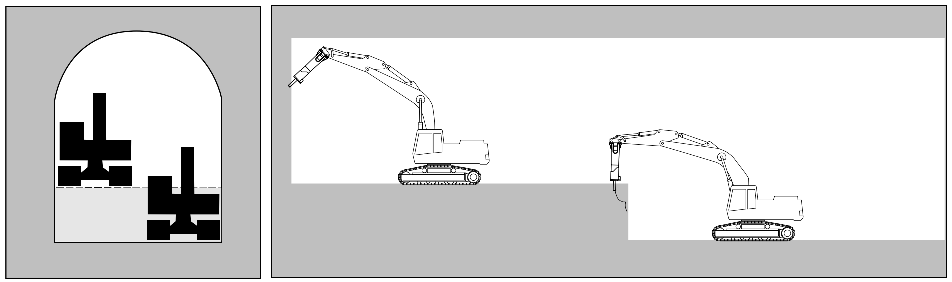


FIGURE ***6.2.-46.*** *Two-phase tunneling with two hammers.*

Long tunnels

If the tunnel is sufficiently long, it is advantageous to start at both ends and in the middle to cope with tight schedules.
Starting in the middle improves equipment and operator availability. The hammer excavates at one side, while the other side is reinforced. When each working phase is completed, the excavation group and the reinforcement group trade places.
When starting in the middle of the tunnel, hammer and wheel-loader trade places with the stabilization team, as support is erected and concrete spraying completed.

The hammer work cycle

Excavation starts at the center of the tunnel at a height of 1.0 - 1.5m. A hole with the depth of 1.5 - 2.0m is excavated. Tunneling then continues from the sides of the hole as close as possible to the final sides of the tunnel. Once this stage is reached, work continues in the same way from the floor up until the roof of the tunnel has been formed (**FIGURE** *6.2.-47*).

If the rock is jointed, excavating follows the shear planes in the normal manner from floor to roof, using the rock's natural weak points and planes to maximum the effect (**FIGURE** *6.2.-48*).

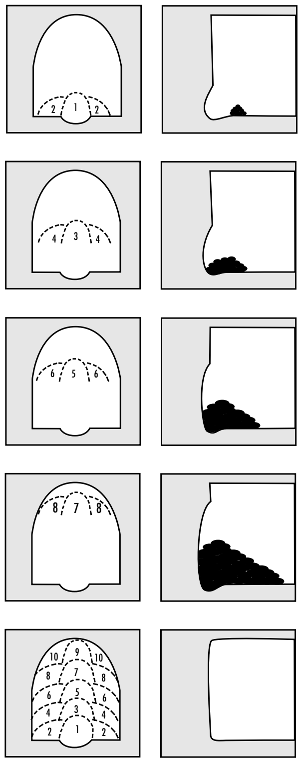


FIGURE ***6.2.-47.*** *Hammer working sequence from floor to roof.*

Technical considerations

Tunnel work is among the toughest jobs a hammer can do. During tunneling, hammer availability is extremely high (60 - 80% of excavator time compared to 30 - 50% in primary breaking). The contact force applied by the excavator to the tool is much higher in a horizontal position than in a vertical position. Due to extreme circumstances, frequent preventative and regular maintenance is essential in effective and productive hammer tunneling. This is best handled with service contracts.

When uninterrupted production is critical, a system utilizing two hammers and one on stand-by is the perfect solution.

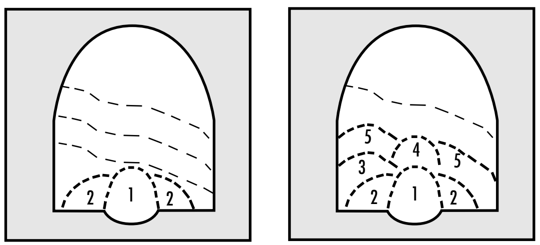


FIGURE ***6.2.-48.*** *Hammer working sequence when rock layers are inclined.*

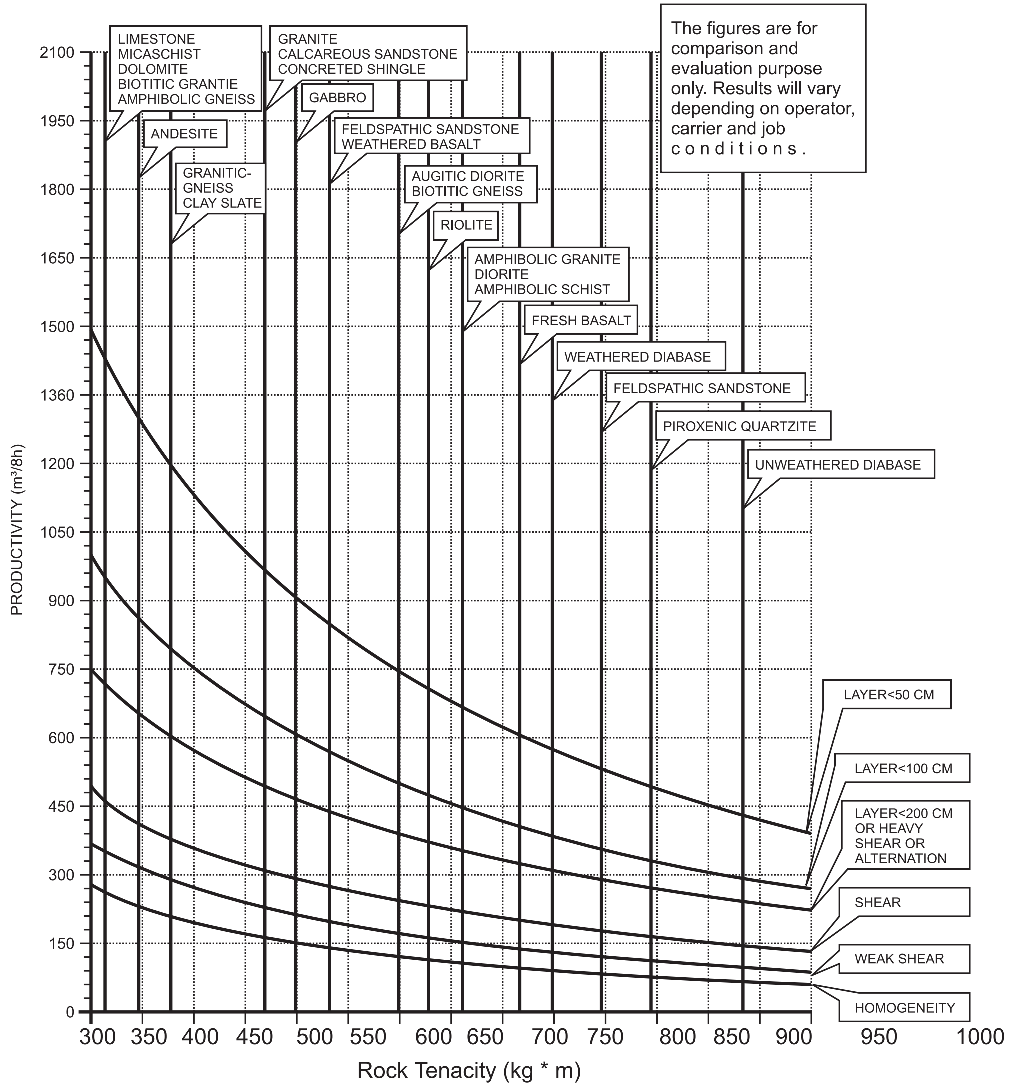


FIGURE ***6.2.-49.*** *G 100 hydraulic hammers productivity, in open pit quarrying.*

Equipment selection

- Choose the biggest possible hammer type
- Choose the CITY model for lower noise and dust protection.
- Choose the Water Jet version for optimum dust prevention and good visibility
- Choose Ramlube automatic lubrication for maximum tool and bushing life

If the tunnel job is extensive (over 1000 m), use extension carriage or front shovel boom to carry the hammer.
In tunneling, the best productivity is achieved with long chisel tools, as excavating frequently must be done near a wall. One should, however, be aware that bending stress on the chisel is hard to avoid in a tunnel. This makes it hard for an inexperienced operator to avoid tool failure. If tool failure becomes a serious problem, using shorter tools is a solution.

EXCAVATORS IN TUNNEL EXCAVATION

Cross-section excavators have generally been used for loading due to their high capacity. However, these rigs are gaining more popularity as

- Carriers for rock breakers
- Carriers for cutterbooms
- Excavators with shovel, special kinematics for tunnel excavation

When using an excavator as a carrier for a cutterboom, the following issue must be taken into consideration:

- Can the excavator withstand the loads from the cutting process, taking into account its stability as well as design?
The following approx. operating weights are necessary to apply cutterbooms on excavators:

Cutter motor power (kW)	Min. operational weight of excavator (t)
100	35-40
200	55-60
300	80-90

- The cutting process requires higher swivel forces than the loading process. As a consequence, it is highly recommended to use hydraulic jacks linked to the excavator's undercarriage to assist the swivel motion of the excavator.
Main application of excavators with cutterbooms are large-section tunnels (for example, for traffic purposes) of comparably short length and softer rock conditions, scaling in existing tunnels (extension of section, removal of destroyed lining) and drill and blast tunneling for achieving a smooth contour and minimize rock fracture close to the tunnel's roof and wall **FIGURE** ***6.2.-50*** shows an excavator with a Tamrock VAB 100 kW cutterboom.

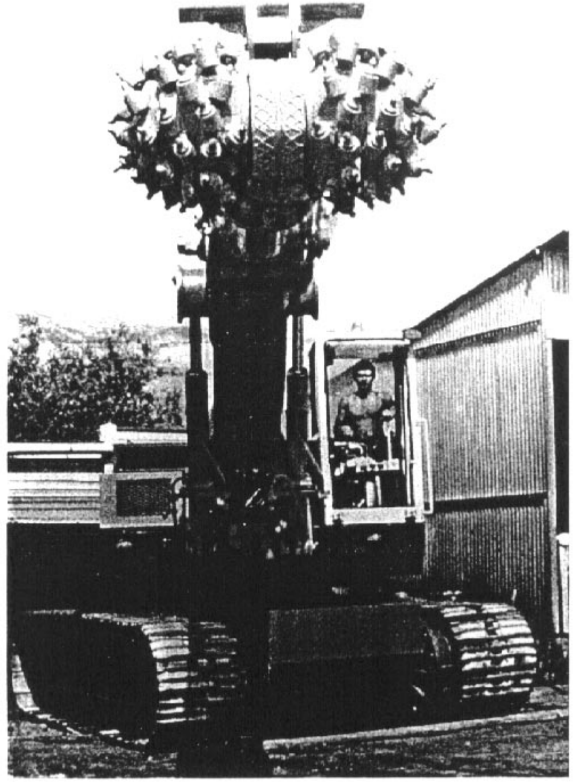


FIGURE ***6.2.-50.*** *Excavator with 100 kW cutterboom.*

While A.M. applications are based on standard excavator types, special tunnel excavators were developed in the eighties mainly for operation in soil-like ground conditions.
Through special digging kinematics design options and short-advanced roof sections, comparably deep invert arches can be excavated and loaded, making these excavators well suited for urban tunneling where fast closure of the lining ring is required. Additionally, shovels with roll-over kinematics allow the perfect contour shaping of the opening.
The success of this relatively cheap and versatile equipment has led to the development of equipment with even more flexibility.

Increasing structural strength and quick interchangeable tooling is a common feature that covers:

- Various shovel types
- Cutterbooms
- Hammers

Drill rigs have also been mounted on-board for blasthole drilling and bolting. Consequently, such tunnel excavators are frequently used in small sections where standard-sized equipment is too big or can not get through the restricted available space.

B) FULL FACE

HARD ROCK

Contrary to soft-ground tunneling where the main objective is to control and support the ground, the goal of hard-rock tunneling is to cut the rock as fast as possible. Daily advance rates of 170 m (diameter 3,4m) have been reported. The application range is extensive and compressive strengths up to 300 MPa can be handled. The diameter range of available TBMs extend from 1.6m - 12m.

The tunnel length should take into consideration the investment costs including as to whether a new or refurbished TBM should be used. A TBM's life time (including some overhauls) is up to 25 km. Full depreciation of the investment on one project is an exception.

Long and small tunnels can be driven effectively by Tunnel Boring Machines, TBMs; short and large tunnels (such as highway tunnels) often are more suited to D&B, where permitted.

The cutting tool used on TBMs are important. Starting with relatively small discs (< 14" dia)

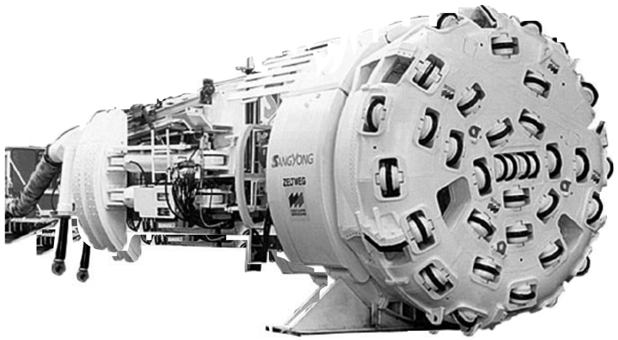


FIGURE *6.2.-51. Example open hard-rock TBM.*

it required more and more power and one solution was to increase the cutter discs diameter. Large disc diameters require higher loads to achieve reasonable penetration rates and levels off at approx. 19". There is a tendency today to focus on 17" high performance discs which provide sufficient service life and which keep the design of the TBM in feasible limits.

Basic operation of a TBM

The cutting process is performed by disc cutters. These cutters
 - generally steel rings - are pressed against the face. The contact pressure between the disc and rock pulverizes the rock on contact and induces lateral cracking towards the neighboring kerf - and rock chipping. To achieve the best performance, kerf spacing (distance between two adjacent tracking cutters) and cutterload must have suitable values for each rock type. Average values of 80 - 110 mm spacing and 250 kN cutter load for 17" (= 430 mm) discs are sufficient in most cases.

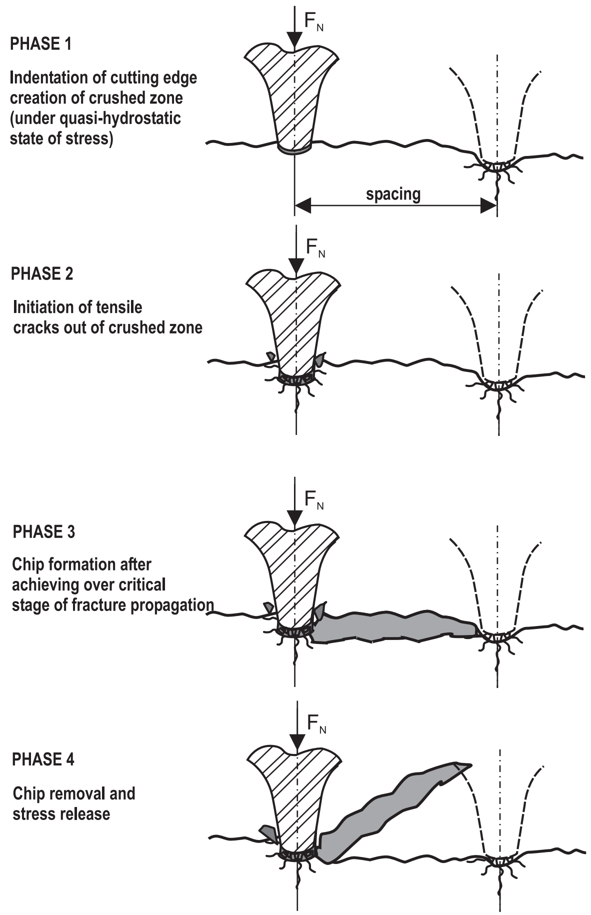


FIGURE *6.2.-52. Cutting process TBM.*

TBM design

Two basic TBM design principles are available:

- Single-gripper machines
- Double-gripper machines

Both principles have advantages and disadvantages, single-gripper machines are used more frequently in standard tunneling projects.

The single-gripper TBM

The basic concept comprises a main frame with a main drive, a floating support at the front end and a gripper at the rear end, required for transfering induced forces into the tunnel wall. A rotating cutterhead is attached to the main gear and rotates at approx. 2.5 m/s peripheral speed. The cutterhead is thrust forward by advance jacks. After a stroke of 1.5 - 1.8, the machine must regrip for a new stroke. The front support is provided by a dust shield, which is a steel structure with expendable plates in the upper area and a rigid support in the lower area. It seals off the working area and makes dust collection easier. This front support is kept in frictional contact with the tunnel wall and overthrusted by the installed thrust force.

The machine is steered by adjusting the rear end of the frame and turning the machine around the front support. A single-gripper machine can be steered continuously during the boring operation which results a smooth surface in the tunnel. Careful steering only while the head is rotating is essential so as to avoid gage cutter and main bearing damage.
The curve radius of the TBM is approximately >150 m, and < 100 m in special designs.
A belt conveyor handles muck discharge. It is installed in the main frame and loaded by buckets on the cutterhead via a hopper in the center of the cutterhead. For maintenance reasons and cutter change, the belt can be retracted to give access to the rear inside area of the cutterhead. The belt discharges into the main conveyor which leads through the back up and discharges into the muck train on the back-up.
The operator's cabin can be placed on the TBM or the back-up, depending on tunnel requirements, which influence the back-up design (**FIGURE** *6.2.-53.*).

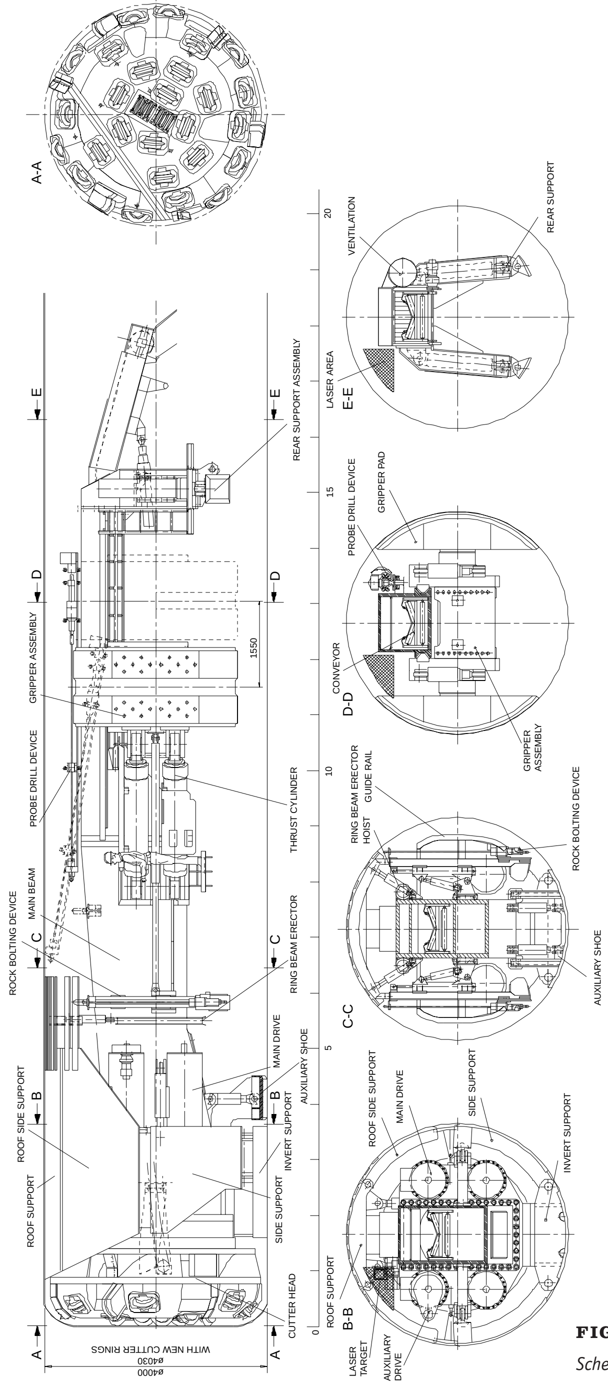


FIGURE *6.2.-53.*
Scheme single-gripper TBM .

Double-gripper TBMs

Contrary to the single-gripper machine, the double-gripper TBM is supported by two sets of grippers that perform the whole guiding function of the TBM. The front dust shield only seals off the dust from the tunnel and cleans the invert.

The main frame, which is stabilized by the grippers, does not move. To advance the cutter-head, a sliding inner frame is used. Steering during boring is almost impossible; and therefore double-gripper TBM´s bore a polygonal tunnel line.
Muck discharge is also done by a belt conveyor from the top of the frame to the end of the TBM.

Double grippers have the advantage of better distributing the gripper forces to the tunnel wall in weak ground. However a disadvantage is taking up free work space for passage and consolidation projects at least in smaller diameters.
Furthermore the skewed process stresses gage cutters and main cutterhead bearing.
(**FIGURE** *6.2.-54.*).

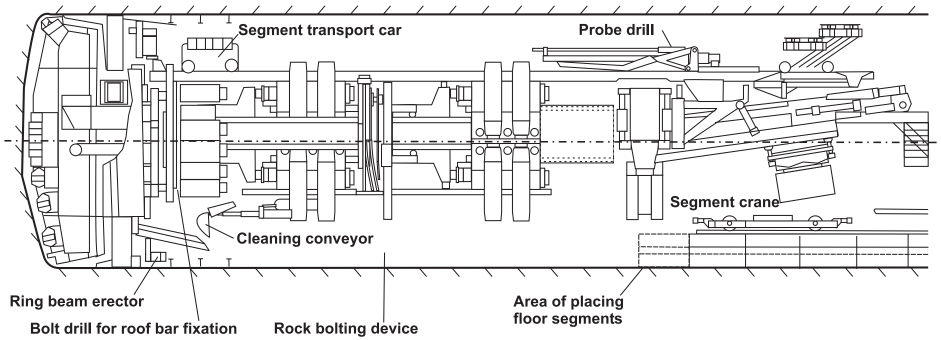


FIGURE *6.2.-54. Scheme Double-gripper TBM.*

Main TBM assembly groups

Cutterhead:
The cutterhead is a rigid steel structure that supports the cutters and loads the muck onto a belt conveyor. Depending on machine size and site conditions, the cutterhead can be one piece or of sectional design. For sectionally designed cutterheads bolted versions are used. Replacement of worn discs on the cutterhead is performed by replacing the cutters held in special saddles by bolts or a wedge lock system. Particularly 3.5 m double-gripper machines usually have front loading systems, which means cutter change can only be performed from the front. In bad ground conditions, this procedure can be dangerous to people performing this job. Bigger machines have back-loading systems that allow cutter change from inside the head.

FIGURE *6.2.-55. ATB 50 HA Back loading Cutterhead.*

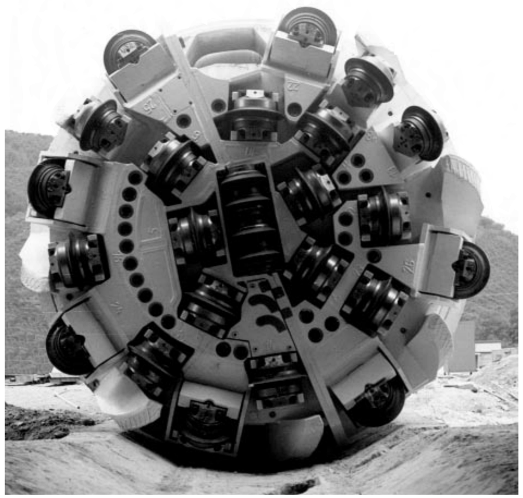


FIGURE *6.2.-56. ATB 35 HA Front loading Cutterhead.*

The buckets load the muck from the invert and discharge it into the hopper. It is very important to keep the bucket lips in good condition and as close as possible to the cut wall to reduce gauge wear on the cutters and on the cutterhead. Specially designed backloading buckets reduce the remaining fines in the tunnel invert.

Disc Cutters:
Disc cutters have an important role in tunnel boring including the layout of cutters on the cutterhead (kerf spacing) and the shape of the cutterhead itself. In special cases and if the diameter must be kept constant for as long as possible, button cutters with tungsten carbide inserts are recommended. Button cutters are commonly used on micro TBMs where access to the cutterhead is not possible.

The steel disc rings are mounted on a hub assembly which comprises the bearing and seal arrangement. The most common type of bearing is a pre-stressed pair of case-hardened conical roller bearings.

Cutter life varies extensively from approximately 30 bcm to 3000 bcm depending on the rock type and especially on its quartz content. The most popular disc shape today is the “constant section” ring, which means the disc footprint does not change significantly with wear.

Main Drive:
The main drive is integrated in the structure of the front dust shield (single gripper system). It comprises the main bearing, generally a three axis roller bearing; double conical roller bearing, the main seal arrangement and planetary drives for the main motor in smaller machines. Most of the machines are electric, with single and double speed run on pole-changing motors or frequency controlled drives in difficult geological conditions. There is a multiple-disc clutch located between the main motor and the planetary gear that protects the main drive against overload and for start-up if stalled. For cutter change and maintenance, an auxiliary drive allows the cutterhead to turn in slow motion.

Installed power is approx. 250 kW/m of diameter (only a rough indicative value depending on the cutter size and geological situation) which means a 3.5 m TBM has approx. 1000 kW installed power on the cutterhead.

Rear Gripper (single-gripper TBM):
The gripper is thrust against the tunnel wall and the TBM is propelled forward by hydraulic cylinders connected to the grippers. Gripper force is distributed via the grippers to the rock. Depending on the rock, the contact pressure is limited to approx. 350 N/cm≤. Studs in the gripper help in slippery conditions.

The gripper cylinder is carried in a frame which allows vertical and horizontal steering. The frame is guided by a specially designed guide along the main frame.

Consolidation, Probe drills

Consolidation Drills:
Provision for dealing with weak rock conditions provisions can be made by installing a pair of roof bolters just behind the dust shield. This allows a primary roof support in poor ground conditions. Due to restricted available space, systematic bolting should be done from the back-up.

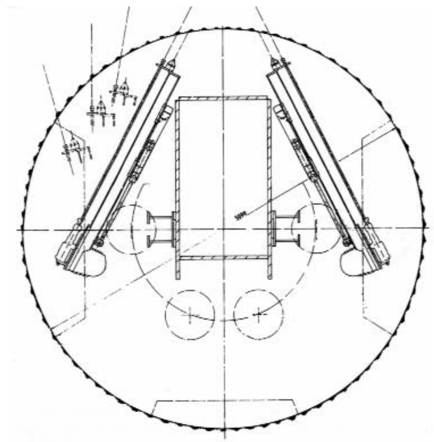


FIGURE *6.2.-57. Front Roof Bolting.*

Probe Drills:
Most TBMs provide probe drill equipment, which are generally hydraulic percussion hammers that allow drilling up to 50 m ahead. Special rock sampling rigs are provided on request.
It is highly recommended to perform probe drilling outside the tunnel diameter because if the probe hole jams the drill rod, advancing of the TBM is not hindered. If drilling in the tunnel cross-section and the drill rod gets stuck, it is hard to get the drill rod out because the cutterhead is not able to cut the steel, and this ultimately damages the cutters.

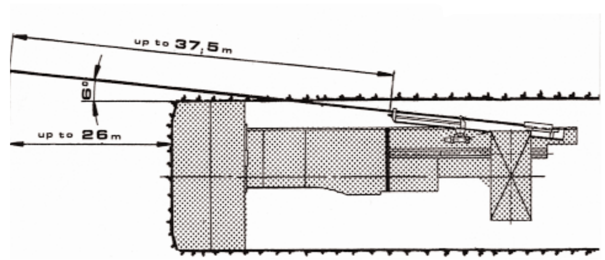


FIGURE *6.2.-58. Probe drill arrangement.*

Operator location

The operator’s location, whether on the TBM or on the back-up, is convenient because it comprises all necessary instrumentation to operate the TBM including the back-up system. Video cameras monitor the important points of the process such as the loading point, train change and areas of danger. All operating data are shown on a PLC display. Data recording of the most relevant data is a standard today. The operator's location is sound-proofed and air-conditioned.

Shielded TBMs

For special conditions in which core sampling is impossible, or the rock is known to be very weak and fractured and if the contract also specifies partial or continuous concrete lining, the shielded TBM is the right equipment for the job.
The shielded TBM, as the name says, looks similar to a shield, but the working process is different.

A shielded TBM is a hard-rock TBM enclosed by a shielded body. The rear end of the machine has a pair of integrated grippers to stabilize the TBM in the tunnel. The front end with the cutterhead is pushed forward out of the telescopic shield via advance jacks. A ring of segments can be simultaneously erected under the shield tail cover. After completing the cutting stroke and segment lining erection simultaneously, the TBM´s rearpart with the grippers is reset into the next position.

If the rock is too weak to give enough resistance for advancing the TBM, the shield mode can be used. In this case, the rear thrust jacks will push against the segment ring and advance the machine. In this event, a parallel operation is not possible which slows the advance rate.

Segment systems used together with shielded TBMs usually serve as a primary lining; and is not watertight. Honeycomb or normal lining can be used. Developments of watertight lining systems have been developed, but provide at the time only sufficient tightness of low pressure conditions. Daily advance rates of up to more than 100 m can be achieved but require excellent logistics from the jobsite organization.
The backup system performs similar requirements as for open TBMs in addition with segment handling and grouting logistics.

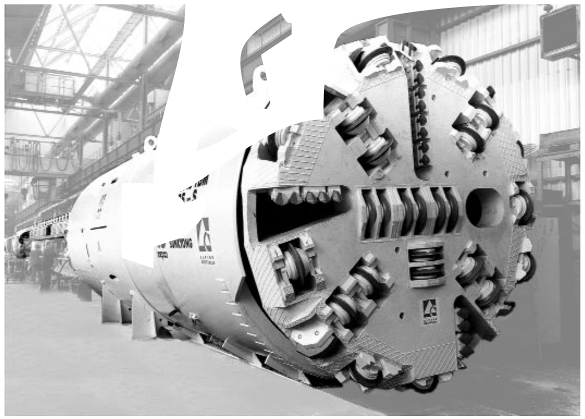


FIGURE ***6.2.-59.*** *Example of a shielded TBM.*

6.2.3. Shaft excavation

Vertical or steeply inclined tunnels called raises or shafts are usually required for ventilation, access and hoisting in hydropower projects and penstock tunnels etc. A deeply inclined or vertical tunnel is usually called a shaft. A raise is an opening underground that goes from one level to another.

Shaft and raise excavation has always been considered one of the most difficult tasks in construction. Today, however, modern equipment offers efficient and safe methods for this type of excavation.

RAISE EXCAVATION USING THE DRILL AND BLAST METHOD

Four different methods are generally used for raise excavation
by the drill and blast method. Method selection depends mainly on the length of the raise:

- Raise building
- Long-hole method
- Alimak method
- Inclined tunnel method

Raise building

Raise building is the oldest method of raise excavation.
Excavation progresses upward from a platform that must be built and dismantled before and after each blast. Drilling is performed with hand-held jackleg drills.
Excavation advancement is slow and working under a blasted roof on a high platform is hazardous. Therefore, the raise building method has been mostly replaced by more advanced methods.

Long-hole method

The long-hole method is suitable for raises with more than a 45-degree inclination (sufficient for rock removal). Maximum raise length, normally from 10 to 60 meters, depends on drilling accuracy, hole alignment and geology. For successful blasting, maximum hole deviation should not exceed 0.25 meters (10").

Excavation via the long-hole method starts by drilling all the holes in the drilling pattern through to the next level. After drilling, each hole's accurate position is recorded to determine the right detonating sequence for the holes. This must be repeated after each blast, because the positions can vary in each blasting section due to hole deviations. (**FIGURE** ***6.2.-60.***).

Blasting starts from the bottom up with the center part always some rounds ahead. The last few meters can be blasted at one go.

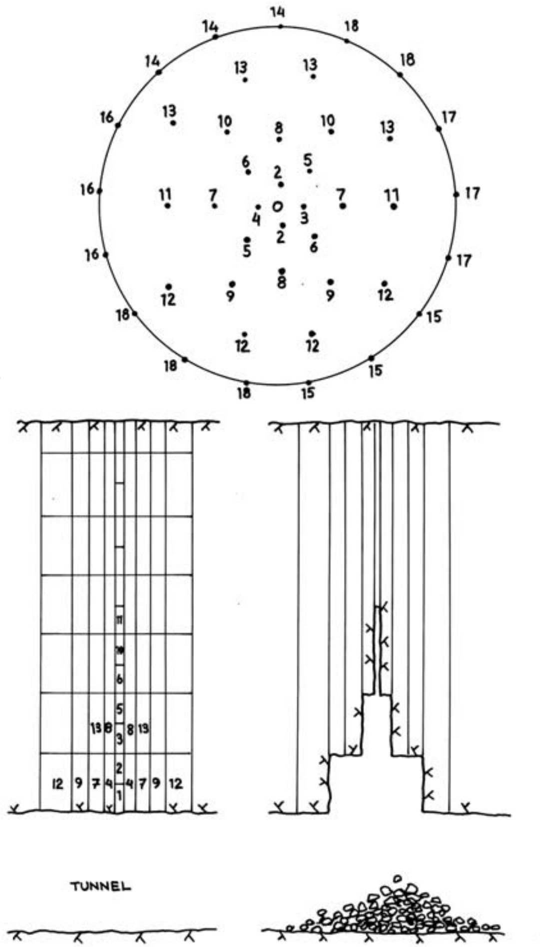


FIGURE *6.2.-60. Drilling pattern and blasting order of long-hole method in raise excavation.*

Alimak method

The Alimak method is based on a lift-type climber, which has a platform, safety canopy, lift basket and motor. The climber travels on rails that are fixed onto the rock wall and is driven by air, or an electric or diesel motor. The water and air lines are attached to the rail.
The Alimak method represents the first mechanized form of raise building. It is more efficient than traditional raise building and is much safer as the work is always performed under a protective canopy.

The Alimak method is a relatively inexpensive alternative for construction sites that have a few variable length raises.

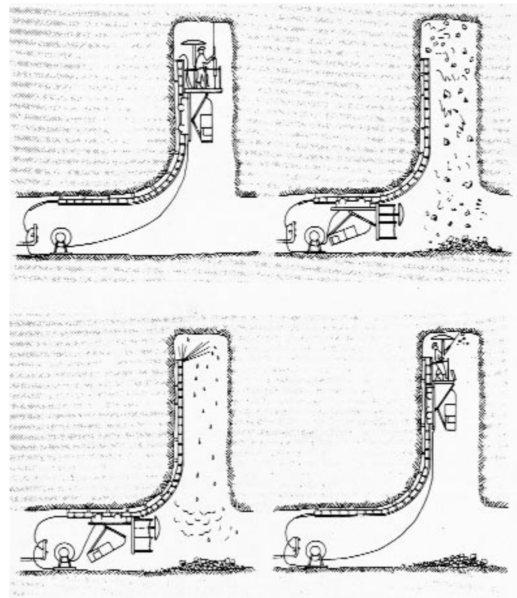


FIGURE *6.2.-61. Alimak method*

The Jora method is similar to the Alimak method with the exception that the lift is operated by a winch. A pilot hole through to the upper level is required for operating the winch.

Inclined tunnel method

One application of mechanized raise excavation is the inclined tunnel method. It can be utilized on steeply inclined raises. Excavation progresses from top down similar to tunneling. Specially made drilling units, such as rail-mounted jumbos lowered by a winch, are used.

RAISE BORING

In the past, all shafts and raises were made by drilling and blasting (methods described previously.) However, during the last decades, full-face raise boring methods have by and large surpassed drill and blast methods for making raises both in mining and civil contracting (**FIGURE** *6.2.-62.)*. In full-face raise boring, the entire cross section is bored to its final diameter. Explosives are not used. There are various alternative methods to bore the full face holes:

Boxhole boring is a special method in which the raise is made in advance from the lower level up. This must be ready when tunneling reaches the area. The rig is on the level beneath the hole and bores up. Boring is performed either by using pre-drilled pilot holes or boring straight with the final diameter boring head.

Blind (down) boring, is another type of boring where the hole is bored downwards. The name “blind boring” comes from the early use of boring down to the final diameter in one pass. Down boring via a pre-drilled pilot hole was developed from blind boring. In small diameter holes, a normal pilot drilling diameter 9 to 13-3/4 is used; in bigger holes the pilot hole is reamed with raise boring to 3 to 8 ft in diameter.

Raise Boring is the most established full-face excavation method of shafts and raises. This method consists of first drilling a pilot hole and then reaming it to the final diameter. The pilot hole diameter is somewhat larger than the drill rods. Reaming is performed in the opposite direction (back reaming) (**FIGURE** *6.2.-63.)*.

In normal raise boring, pilot drilling is performed from the upper level vertically down or inclined to the lower level.
Sometimes the pilot hole is drilled up and back reaming is done downward.

FIGURE ***6.2.-63.*** *Cutter used in Raise Boring.*

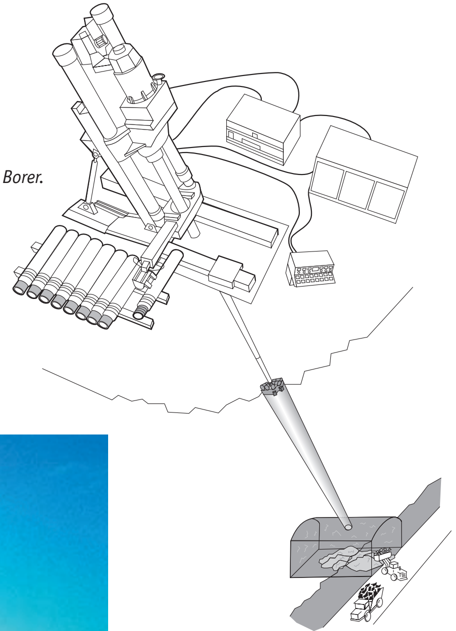


FIGURE ***6.2.-64.*** *Shaft excavation by Raise Borer.*

FIGURE ***6.2.-62.*** *Rhino 1200 raise bohrer.*

Main uses of raise boring in civil construction

The main uses of raise boring in civil construction are:

- Ventilation holes for road and railway tunnels
- Various holes and raises for hydropower stations and underground storage halls (**FIGURE** *6.2.-64*).
- Holes used as pilot holes for big diameter shaft sinking
- Raises in areas where environmental restrictions (noise, vibrations etc.) limit use of other methods. For example, urban areas, nuclear power plant or nuclear waste storage vicinities etc.

Main benefits of raise boring

The main benefits of the raise boring methods are:

Safety:
- Always working in a safe area; no working under newly blasted roof
- Clean environment: no dust, blasting fumes, exhaust gases or oil mist
- Low noise level and minimum vibration (compared to blasting)

Speed, efficiency:
- Raise boring can be typically 2 to 3 times faster than older methods
- Only one operator is required in a modern raise boring machine

Quality:
- Round cross section and smooth walls are optimal in terms of flow characteristics (ventilation, water flow) and require a minimum amount of additional support
- A regular, round cross-section makes it easy to assemble any pre-fabricated equipment in the hole

Adaptability to various rock conditions:
- The raise boring method can also be used when rock conditions are so difficult that conventional drill & blast methods are not possible.
- It does not cause any fractured zones or cracking to surrounding rock
- Optimal shape of the raise is strong against rock pressure

SHAFT SINKING

Shaft sinking is a method where a vertical or steeply inclined tunnel is excavated from the surface.

Shaft dimensions are determined by shaft purpose, geological and rock mechanical conditions. Most shafts have a diameter of 5 - 8 meters, with only a few reaching 10 meters in diameter. Shafts are usually round in shape.

After exploring of geology and groundwater conditions, overburden is removed. If the overburden requires stabilizing, it is typically lined with concrete rings. Once the rock surface has been exposed, it is reinforced and grouted. The collar for the headframe is installed after excavating has progressed a short distance. The head frame includes the hoisting system for the shaft sinking equipment. At this point, the actual shaft sinking begins.

Manual shaft sinking

Manual shaft sinking requires several men operating hand-held rock drills and shoveling the rock manually into small buckets. All equipment must be transported up and down in buckets. The work is time consuming and progresses slowly.

The number of workers and amount of effort required for manual shaft sinking makes it impossible to excavate very long and large shafts. Shaft dimensions, restricted by the excavation method, limit hoisting capacity or any plans to expand.

Mechanized methods

When mechanization started to gain ground in tunneling, it was gradually applied to shaft sinking. Pneumatic shaft sinking jumbos were first introduced, later the hydraulic versions. Using hydraulics made it possible to build more complex, multipurpose shaft sinking platforms, which in turn meant that bigger shafts could be excavated with greater accuracy and efficiency. (**FIGURE** *6.2.-65*).

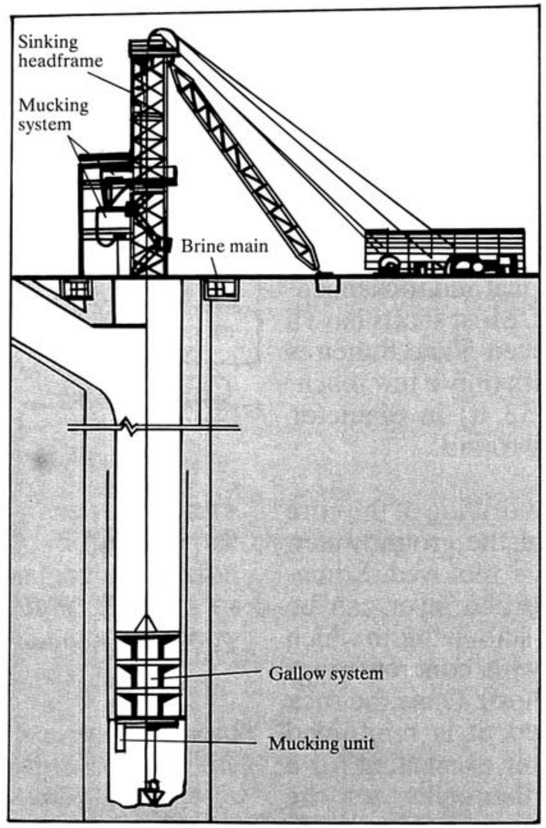


FIGURE *6.2.-65. Mechanised shaft sinking equipment.*

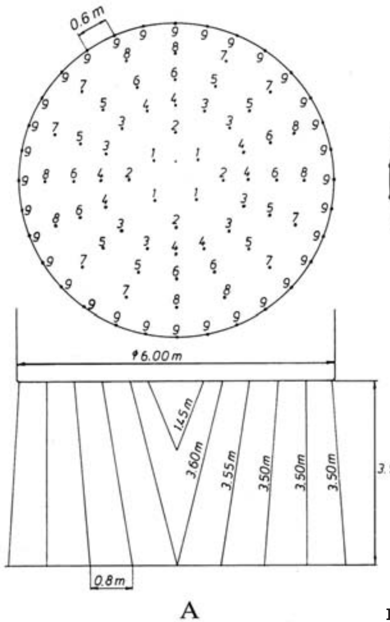
Full-bottom method

In modern shaft sinking, the drilling rig is a two, three or four-boom drilling jumbo designed specifically for the dimensions of the shaft and sinking platform. To build the rig, the manufacturer requires exact shaft dimensions, sinking platform requirements and any restrictions (power arrangements, through driving dimensions etc.).

In the full-bottom method, the V cut was most commonly used. Limitations, such as available space and the feed/hole length, are the same as in tunneling. Thus the parallel cut with large cut holes is replacing the V cut in shaft sinking. This makes it possible to blast long rounds. Today holes up to 5.0 meters long are being successfully used, rock conditions permitting. The extra round length increases the speed of sinking and enhances working arrangements and use of the sinking platform.

The drilling pattern design for both the V cut and the parallel cut is similar to round tunneling with contour smooth blasting (**FIGURE** *6.2.-66.*). Special care must be taken with charging, considering blasting direction and in case of any water problems.

Mucking is done with clam shell buckets. The skips lift the blasted rock to the surface. Even if most of the work is performed from the multi-level headframe, a certain amount of manual work is required at all stages. The headframe is lowered when the working units can no longer reach the bottom, usually after two or three rounds.



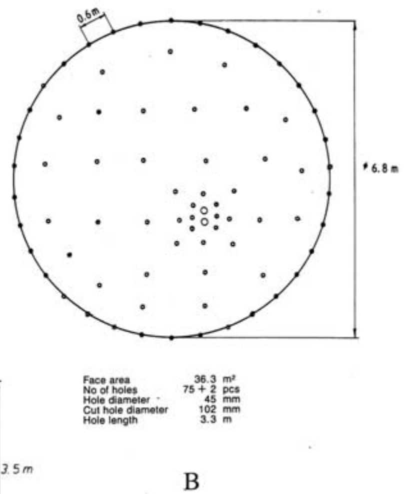


FIGURE *6.2.-66. Shaft drilling patterns a) V cut and b) parallel cut*

Depending on rock conditions and final shaft use, rock bolting and shotcreting, or steel or concrete lining can be used as rock reinforcement When shaft sinking gets down to the lower levels, all operations become slower. Using a multi-level platform makes the operation, service and maintenance easier and faster than in older methods.

Benching

Benching can be used as an alternative to full-bottom shaft sinking when rock conditions do not allow full-face excavation. Benching is an older method that is suitable for square-shaped shafts. Benching is done in halves. While one half of the cross section is being drilled and blasted, the lower half serves as a water sump and spoil dump. Work continues downward in alternately lowering benches (**FIGURE** *6.2.-67.*)

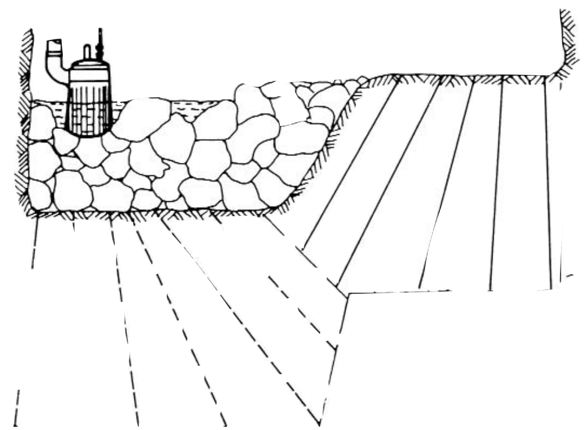


FIGURE *6.2.-67. Benching method.*

Spiral method

Spiraling is a variation of the benching method. Excavation spirals downward. This method is suitable for fairly large round or oval shaped shafts, or when the full bottom is not otherwise possible. Drilling and blasting progresses with half of the face at a time. The holes in each half have the same length.

Benching and shaft-sinking rigs can be used for both the spiral and the benching methods. The drawback with any partially mechanized method is mucking and transportation difficulties. Using these methods only in small shafts increases the workload and slows progress.

6.2.4. Rock reinforcement

Rock support for tunnels and underground cavern design is a demanding and very complex task. In principle, the problem can be approached from two directions: The first way is to define the relationship between geo-mechanical properties of the rock mass and the support methods used. This is mostly based on the utilization of statistical and empirical data gathered in similar conditions. The second way is to estimate the deformation characteristics of the rock structure, and then the related effect on supporting structures. This method typically requires very good rock property and rock mass property data.

The most important factors affecting rock reinforcement method and design are:

- Geological factors, such as rock properties and rock mass structure
- Dimensions and geometry of excavated space
- Location and direction of caverns in the rock mass
- Excavation method
- Use and expected lifetime of space

Common support methods in underground construction work are:

- Bolting
- Sprayed concrete
- Steel arches
- Concrete lining
- Grouting

BOLTING

Rock bolting is one of the most common methods of rock reinforcement. The main principle of bolting is to reinforce loose rock or fractured in-situ rock to prevent caving or spalling, and to assist the rock mass to form its own self-supporting structure.

Bolt types

Bolts can be divided into three categories according to the way they behave in the rock, for example, grouted bolts, mechanically anchored bolts and friction bolts.

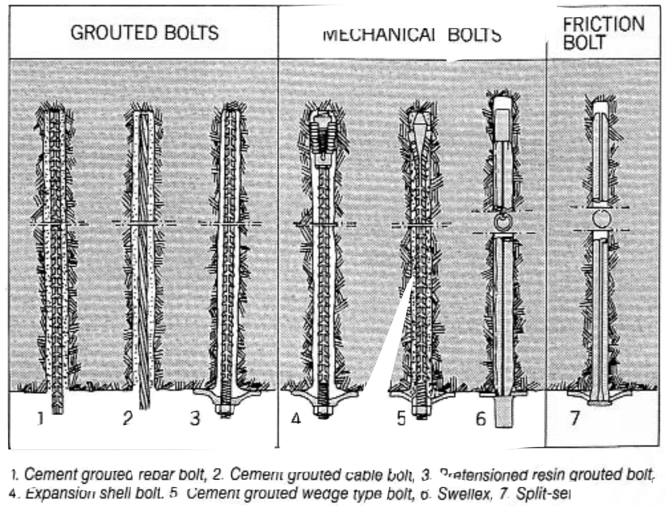


FIGURE *6.2.-68. Different bolt types.*

Cement-grouted bolts

Cement-grouted rebar is still the most inexpensive and widely used rock bolt, because it is simple and quick to install and can be used with or without mechanized equipment. Correctly installed, a cement-grouted bolt gives rock support for years.
The grout cement provides protection from corrosion. Special galvanized and/or epoxy coated bolts can be used in extremely severe conditions.
The major disadvantage of the cement-grouted bolt is its relatively long hardening period. The grout takes between 15 - 25 hours to harden, therefore it does not provide immediate support. When immediate support and/or pre-tensioning is needed, a grouted wedge-type or expansion-shell bolt can be used. Mixing additives in the grout can reduce the hardening time, but it also increases bolting cost.

The water/cement ratio considerably affects the quality of installed bolts. The best water/cement ration is 0.3 (w/c).
This grout density can be easily used and maintained when using mechanized bolting equipment (**FIGURE** *6.2.-69.*).

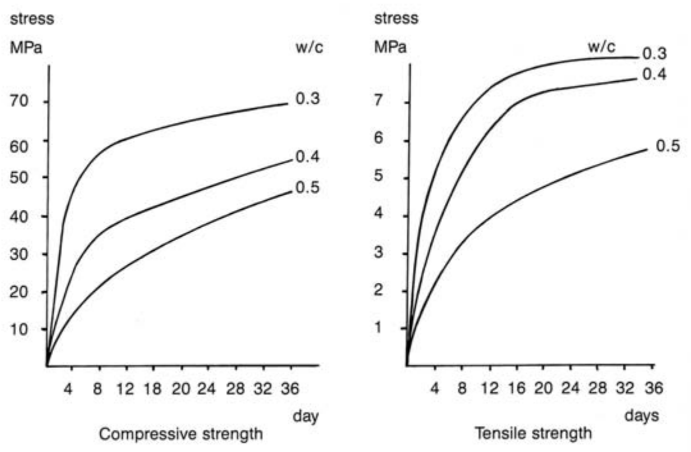


FIGURE 6.2.-69. *Effect of water/cement ration on grout quality.*

Resin-grouted bolts

Resin-grouted bolts give the required support relatively quickly due to a short hardening time. When correctly installed with full-length grouting, the resin-grouted bolt is considered to give permanent support with a life span of 20 to 30 years.

By using resins with two different hardening times, with one faster at the bottom of the hole and another that is slower at the stem, the bolts can be pre-tensioned. The same can be done for short-time support by only bottom-grouting the bolt.

Cable bolts

Cable or steel strand bolts are used to bind and secure large volumes of rock around large caverns. Cable bolts can be used both before and after excavation, and are also used for preventing rock slides in mountain slopes and quarries.
The anchor itself is a steel strand, typically two strands of 15.2 mm in diameter, with typical bolt length being between 10 - 25 meters.

Today, with mechanized equipment, the installation and grouting of cable bolts of any length is fast and efficient, and the cable bolt's bearing capacity clearly exceeds capacity of rebar steel bolts. Its lack of efficient protection against corrosion limits its extensive use in permanent rock support.

Mechanically anchored bolts

Mechanically anchored bolts are usually wedge or expansion-shell bolts that are point-anchored at the bottom of the hole.

The bolt has an expanding anchor at its end. After insertion, the bolt is either rotated or pressed/hammered against the bottom of the hole. This expands the wedged end and anchors the bolt firmly to the sides of the hole. To install anchored bolts successfully, the hole size must be accurate and the rock must be relatively solid.

Wedge or expansion-shell bolts are typically meant for temporary rock support. Together with cement grouting, it provides both immediate and long-term support.

Friction-type bolts

Typical examples of friction-type bolts are the Split-set and Swellex bolts. Both are quick and easy to install and give instantaneous support. They can not, however, be used for long-term reinforcement.

The Split-set bolt is hammered into the hole, which has a slightly smaller diameter than the bolt. Using the correct hole size for a specific bolt diameter is essential for successful installation. Split-set bolts are very suitable for layered formations. The Split-set bolt provides immediate support but only for a fairly short period of time. A disadvantage is that the Split-set bolt can not be effectively protected against corrosion. The life span can somewhat be extended by using cement grouting. The Swellex bolt has a longer life span than the Split-set. It is installed by applying high-pressure water to the bolt after inserting it to the hole. The high pressure expands the bolt to its final dimensions in the hole, therefore enabling it to utilize the roughness and fractures in the bolt hole surface. As with the Split-set bolt, poor corrosion protection limits this bolt.

Equipment for bolt installation

Development of mechanized equipment began as early as the 1970s. Today there is a wide selection of fully mechanized equipment, and a wide variety of different methods for bolt installation. The main factors affecting the choice of method are usually tunnel size, amount of bolts to be installed and work cycle arrangement at the site.

Manual operation, the hand-held drilling and installation of bolts, is typically used in small drifts and tunnels where drilling is also performed by hand-held equipment, and there is a limited amount of bolting work.

Semi-mechanized installation is still typical at tunneling work sites. The drilling jumbo is used for drilling bolt holes, and bolt installation is performed from the jumbo's basket boom or from a separate utility carrier or truck.

Fully mechanized bolting

With today's fully mechanized equipment, one operator can handle the entire bolting process from drilling to grouting & bolt installation. The operator is positioned away from the unbolted area under a safety canopy that protects him from falling rock.

FIGURE ***6.2.-70.*** *Tamrock Robolt fully mechanized bolting unit.*

Although safety is a major reason for the development of mechanized bolting equipment, the superior installation technique of mechanized bolting rigs also produces consistently higher bolting quality. Thanks to powerful cement mixers, pumps and effective grouting methods, the bolts are securely fixed and grouted to their full length, providing a sound reinforcement structure, even with long bolts.

Robolt

The first fully mechanized bolting unit, called Robolt, was introduced by Tamrock in 1979. Mechanization initially involved cement grouted rebar bolts, but extended quickly to other bolt types. Today all most commonly used bolt types can be installed mechanized with the Robolt.
Mechanized bolting with the Robolt follows the pattern:

- Mixing the cement grout (if cement grout used)
- Stabilizing the bolting head to the desired spot
- Drilling the bolt hole
- Pushing in the grouting hose and grouting the hole, starting from the bottom, or shoot ing the resin cartridges to the bottom of the hole
- Inserting the bolt from the magazine into the hole
- Mixing of the resin/tightening or pre-stressing the bolt as required

The progression from drilling to grouting and installation stages is performed by accurately indexing the bolting head to the right position.

When grouting is started from the bottom of the hole, the hole is completely filled, eliminating all air pockets. Mechanized equipment also allows the use of best possible water/cement ratio in the cement grout.

Cabolt

Manual installation of cable bolts is time-consuming, difficult and labor intensive. Grouting manually installed bolts is normally done after bolt installation, and often leads to unsatisfactory bolt quality.

The Tamrock Cabolt is a fully mechanized cable bolting unit that handles the complete bolting process including hole drilling, feeding the cement grout and inserting the cables. Bolt length can be freely selected and all the work is performed by one operator controlling the machine.

FIGURE ***6.2.-71.*** *Tamrock Cabolt. A fully mechanized cable bolting unit.*

SCREENING

Screening, which is the installation of wire mesh, is most typically used in underground mining, but also at construction sites together with bolting and/or sprayed concrete.Screening is primarily performed manually by applying the wire mesh together with bolting of the tunnel. It can also be done by mechanized equipment, such as by having a screen manipulator on the bolting or shotcreting unit, or on a dedicated screening machine.

FIGURE *6.2.-72. Robolt 320 with screen manipulator.*

SPRAYED CONCRETE

Sprayed concrete, otherwise called shotcreting, is a widely used support method in construction. It is used for temporary or long-term support, lining and backfilling. Usually shotcrete is used together with bolting to obtain the best support or reinforcement. Shotcrete can be reinforced by adding steel fiber to the concrete.

The most common forms of shotcreting are the dry-mix and wet-mix methods.
In the dry-mix method, the aggregate, cement and accelerators are mixed together and propelled by compressed air. Water is added last through a control valve on spray nozzle. The dry method is suitable for manual shotcreting because the required equipment is usually inexpensive and small. On the other hand, the dry method can pose health hazards as it creates considerably more dust and rebound than the wet method. The quality also depends heavily on the shotcreting crew, and may vary widely.

In the wet mix method, the aggregate, cement, additives and water are measured and mixed before transport. Today, wet mix is more widely used because it is easy to mechanize and the capacity can easily out-do the dry method. Rebound rate is low and the quality produced is even.

Critical factors in shotcreting are:

- Water/cement ratio
- Grain size distribution of aggregate
- Rebound ratio, affected by
- Grain size distribution
- Mix design
- Nozzle design
- Nozzle distance and angle
- Layer thickness

Manual shotcreting has been largely replaced by mechanized shotcreting machines. With mechanized equipment, multiple capacities per hour can be reached, together with consistent and even quality of the concrete layer. Safety, ergonomic and environmental conditions are other important aspects of shotcreting. These factors are efficiently improved with mechanized shotcreting units.

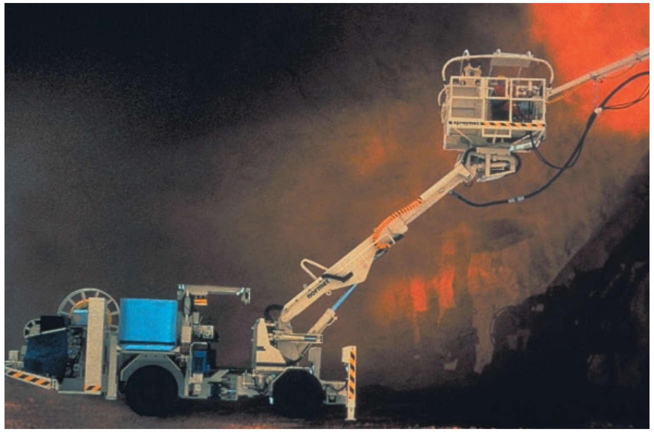


FIGURE *6.2.-73. Mechanized shotcreting unit.*

STEEL ARCHES

Steel arches are a common permanent support method for weak rock formations. The preformed steel arches are usually installed in the tunnel immediately after each round, at the same time as rock bolting. Steel arches are also commonly installed during shotcreting to give temporary support before final concrete lining of e.g. traffic tunnels.

GROUTING

Grouting is the method in which a solidifying liquid is pressure-injected into the rock mass. The main purpose of grouting is to prevent ground water leakage into the tunnel, and to increase overall of rock mass strength.

In grouting, a chemical agent or cement mass is pressure-pumped into the drillhole to penetrate fractures and fill cavities.

In drill and blast tunneling, grouting is typically performed before (pre-grouting) or after (post-grouting) excavation.

Pre-Grouting

Pre-grouting means that the rock mass is grouted before excavation begins. Usually, pre-grouting is done from the tunnel, but in situations with low overburden it is also possible to do it from the surface.

Probe holes are drilled to map possible fractures and register water flow. This helps to analyze the need for grouting. Later, grout holes are drilled in a conical-fan shape in front of the tunnel face. Typical grouting fan length is 15 - 25 meters.

After drilling, the grouting agent is pumped into the hole until leakage has reached an acceptable level. Tunnel excavation can begin once the grouting mass has settled.

Grouting fans overlap each other so that in 15-meter-long grout holes, grouting is performed every second or every third round depending on the round length.

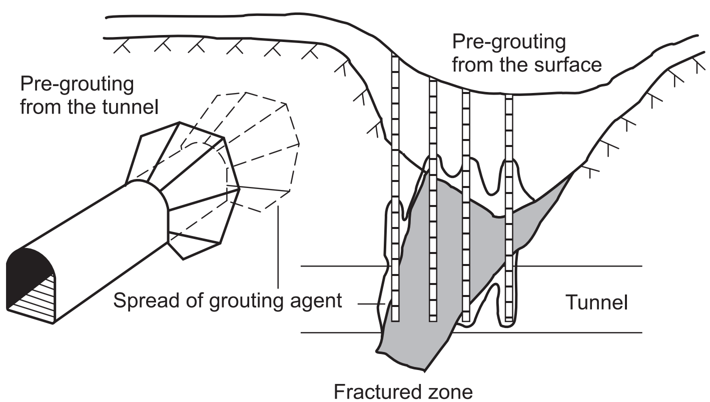


FIGURE 6.2.-74. *Pre-grouting.*

Grouting after excavation (Post-grouting)

When grouting is done after excavation, grouting holes are drilled from the tunnel in a radial form. In good rock conditions with small water leakage, post-grouting is often adequate. Post-grouting enables better rock mass structure evaluation. On the other hand, water leakage blockage is more difficult because the water flow tends to flush away the grouting agent before it hardens.

Grouting agents

The grouting agents can be divided into two categories: Suspension and chemical.

Cement water or bentonite water suspension is the most typical in rock grouting because both are cost-effective and environmentally safe. The drawback is, however, a relatively large maximum grain size, which leads to poor penetration in small cracks. Penetration characteristics can, however, be improved by adding additives. Silicate-based chemicals are also used to speed up the hardening time.

Chemical agents are silicate-based, resin polymers, polyurethane-based or lignin-based chemicals that typically penetrate very small cracks and have adjustable hardening times.

6.3. CASES

6.3.1. Railway tunnel

Tamrock Data units (DataSuper, DataMaxi and DataTitan) were used by Lemminkäinen Construction Ltd. in a project in Norrala, which began in October 1996. The Norrala tunnel is a part of the Hälsingekusten project consisting of building a railway and road, E4, from Söderhamn to Enånger in Sweden.

The 3,850 m-tunnel construction schedule held. Accurate planning of the extraction of 300,000 m^3 solid rock, mainly granite and gneiss, was made possible by the precision of the Data equipment. Additionally, 200,000 m^3 of earth was moved and 40,000 m^3 rock excavated from the surface.

The tunnel is 7.9 meters wide and 8.9 meters high. Its profile from north to south declines 35 meters and has a cross-section of 68,2 m^2. There are 113 holes in each face. The advance in each round is 5.3 - 5.4 m with 20 feet rods. The average drilling time per round was 3 - 4 hours and the average advance per week (120 hours) was 120 m. The record advance per one week was 138 m with two jumbos and with 8 - 9 men per shift.

Both system and random bolting was used for a total of 12,000-13,000 bolts. When needed, a total of 10.000-11.000m3 concrete was sprayed. 19 grouting holes, 21 meters long, were drilled every 3 rounds to provide water tight access to the tunnel.

A maximum of six faces developed at the same time. Safety was a major priority: emergency tunnels (33m²) in three different places, totalling 700 meters. The railway tunnel was made sufficiently wide to accommodate rescue vehicles. Evacuation tunnels will also help to equalize the air pressure due to the train´s high speed.

Lemminkäinen discovered how to best utilize Datamaxi´s properties. A 3 dimensional V cut was developed for the demanding conditions. There was not a single hole drilled parallel to the tunnel´s direction; all holes are inclined.

The 3-D V cut offered considerable savings because drilling big reaming holes was not required. Pull-out was achieved even with 50 - 60 m less drilled meters per round.

FIGURE *6.3.-1. TAMROCK Data Titan in the Norrala tunnel.*

6.3.2. Oil and gas storage

Oil and gas storage underground presents a popular and economical alternative to surface facilities. Underground construction offers better environmental protection, and deep rock caverns are ideal for pressurized storage tanks when general rock conditions are suitable.

Underground oil and gas storage facilities are among the largest underground excavation tasks and come in various shapes and sizes, depending on what is being stored. Some oil storage facilities are several hundred meters long, and one storage plant can contain several storage halls. Excavation work can be done through only horizontal benching, or both horizontal and vertical benching (**FIGURE** *6.3.-2.)*.

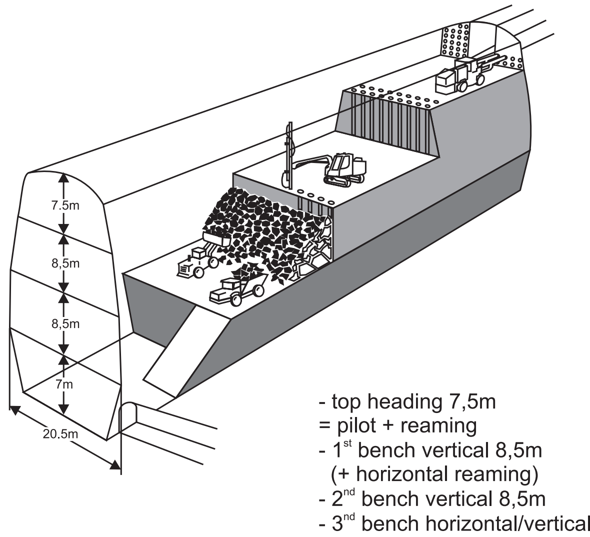


FIGURE *6.3.-2. Drifting stages in underground oil storage.*

6.3.3. Hydropower stations and waterworks

UNDERGROUND EXCAVATION AT YELLOW RIVER, CHINA

The Xiaolangdi project surface work was described earlier in chapter 5.3.2.

This project was designed to trap sediment at a point where it reaches a balance between the sediment´s outflow and inflow. A total of sixteen Tamrock rigs were delivered to the site: Four Maximatic HS 305 T units, six Paramatic HS 205 T PowerClass jumbos, four PowerTrak CHA 660 and two Commando track drills. The underground projects on Lot II consisted of three diversions; three free-flow and three sediment tunnels, resulting in the excavation of 1.4 million m³ of rock. The free-flow tunnels ranged from 450 - 700 m All diversion and sediment tunnels were approximately 1,100 m long. The largest tunnels were 18.5 m in diameter. All-in-all, the project consisted of 16 tunnels.

The civil jobs of Lot III consisted of an underground power house (120 m long, 23 m wide and 22 m high), a transformer chamber, a draft tube gate chamber, six power tunnels, six bus tunnels, a penstock, six draft tubes, three tailrace tunnels and an access tunnel. Three ventilation shafts, an elevator shaft, drainage tunnels and high-voltage tunnels were also excavated.

Lot III's underground excavation volume is 1.2 million m^3. Tamrock Maximatic HS 305 T, Minimatic HS 205 D PowerClass jumbos and Commando 100 second-hand track drill rigs operate underground. A significant amount of rock bolts were used in the tunnels for rock support.

FIGURE *6.3.-3. Underground excavations in Xiaolangdi.*

6.3.4. Hammer tunneling

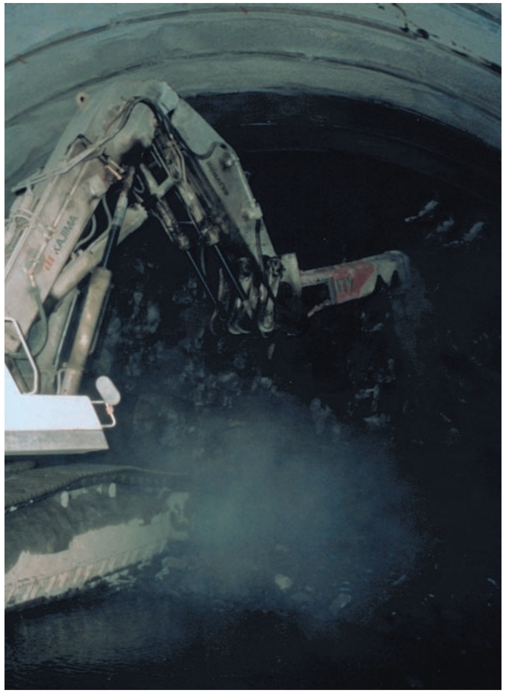


FIGURE *6.3.-4. Hammer tunneling. (Note: The photo and text are not interrelated.)*

Treno Alta Velocita (TAV) is a railway network construction project for high-speed trains in Italy. The project includes several tunnels, such as the Briccelle Tunnel near the town of Capua, which is located between Rome and Naples. Tunneling began in December 1995, and upon completion the total length of the tunnel will be 1033m. The tunnel is 12m high and 13m wide and a total of 135 m^2 was excavated. The project was executed by Condotte D´Acqua..

The project starts from the upper part of the tunnel, which is 9m high and has a cross-section of 100 m^2. This section was excavated with an S 86 installed on a Fiat-Hitachi 400, and an S 84 installed on Fiat Hitachi 330 machine. The lower part of the tunnel is 3 meter high with a cross-section of 35 m^2. A Rammer E 68 CITY hammer equipped with an automatic lubrication system and installed on a PMI 834 machine was used in this section.

6.3.5. Roadheader tunneling

NEW SOUTHERN RAILWAY TUNNEL IN SYDNEY, AUSTRALIA

The New Southern Railway (NSR) will form an additional rail link between the center of Sydney and the East Hills Railway Line, which will be met shortly west of the Sydney airport. The tunnel will provide a direct rail connection from downtown Sydney to the airport, and will be ready for operation for the Summer Olympic Games in Sydney in the year 2000.

The northern part (approximately 2.2 km) of the 10 km tunnel is excavated by roadheader - ALPINE MINER AM 105 (**FIGURE** *6.3.-5.)*.

The tunnel extends from Prince Alfred Park to the TBM exit access shaft south of Green Square, Alexandria. This section is set mostly in Hawkesbury sandstone (10 - 75 MPa, 20 - 40 MPa in average with 60 - 80 % quartz content) with some Ashfield shale and stiff clay interbedding.

Work will be performed from each end simultaneously, at the tunnel heading and bench from Prince Alfred Park and a similar operation from the TBM exit access shaft. Excavation will be carried out by the joint venture company Transfield-Bouygues. The ALPINE MINER AM 105, a powerful boom-type roadheader of the 100-ton class, has proven its unique transverse cutting technology in hard rock applications worldwide. It has an extended field of operation for mechanized roadway development in hard and abrasive rock formation.
Fine tuning and optimization of the AM 105 for the New Southern Railway Tunnel has been performed during the first period of excavation and resulted in a project-specific customized cutterhead as well as tailored operating procedures.

The machine achieved a maximum instantaneous cutting rate of 121.4 bcm/ch; a pick consumption of only 0.02 to 0.03 picks/bcm.

FIGURE 6.3.-5. *Alpine Miner AM 105.*

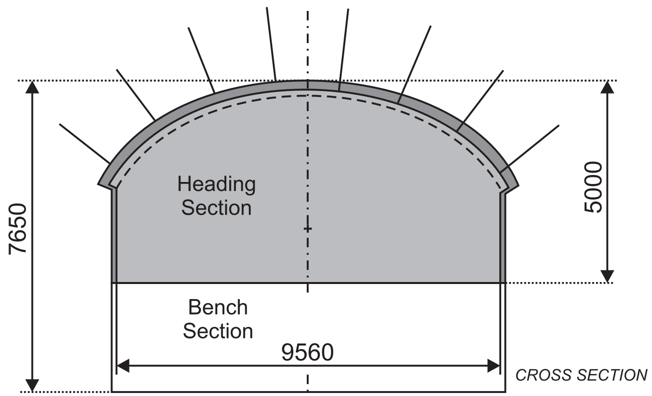


FIGURE 6.3.-6. *Excavated tunnel profile in railway in Sydney, Australia.*

7.1. BUILDING STONE MATERIAL

When choosing a particular type of stone as a building material, the most important features are its technical properties, appearance and price compared to competitive materials. Both experience in this type of rock and architectural trends are also influencing factors. Soft and easily produced stone such as marble, limestone and sandstone were the most important materials until the 1970s, after which the quarrying and working methods of igneous rocks which contain hard minerals became more efficient. Increased strength and weather resistance make hard rock a competitive floor material solution.

In the dimensional stone business, rock types are divided into two categories: soft and hard. The former consists of carbonaceous rocks, clastic sedimentary rocks and other rocks containing soft minerals. The latter consists of igneous rocks and siliceous metamorphic rocks. A third group is slate, which has different production and usage methods. ***Table 7.1-1.*** shows the most important dimensional stone rock types and their petrographic composition.

FIGURE *7.1.-1. Block.*

FIGURE *7.1.-2. Processing of blocks.*

FIGURE *7.1.-3. Use of building stone material.*

Table 7.1.-1. *Classification of dimensional stone rock types*

TYPES	PETROGRAPHIC COMPOSITION SILICEOUS				CARBONACEOUS
DEFINITION	Genesis	Sialic		Femic	
MAGMATIC	Intrusive	Granite Syenite Diorite		Peridotite	
	Sill-rock		Porphyry	Gabbro	
	Effusive		Trachyte	Basalt	
METAMORFIC	Orto	Gneiss		Serpentinite	
	Para	Quartzite			Marble
SEDIMENTARY	Clastic	Sandstone Pudding-stone		Ophicalcite	
	Chemical				Alabaster Travertine
	Organogenic				Ammonitic Fire-Marble

Usage of a stone deposit primarily depends on its structure. With the exception of slate, the jointing density of the rock should be as low as possible. Perpendicular joints are also an advantage. The folding and orientation of various minerals give a special appearance to many multicolored stones. However, this often negatively affects the rock strength and waste ratio in quarrying. Stratification of sedimentary rocks influences its strength. Veins are usually considered flaws, as are color variations because a rock's color should be even. Weathered stone is not usually used as dimensional stone.

The rock's technical characteristics influence quarrying and production. Technical characteristics are mainly affected by the mineral mix and mineral grain texture. Hardness, color, tensile strength, thermal expansion coefficient, density, porosity and chemical stability are a rock's most important technical features.

Stone color depends not only on the main minerals but also on accessory minerals that cause stains. Therefore, marble comes in various shades although it always contains pure white calcite.

A rock's tensile strength, thermal expansion coefficient, density, porosity and weather resistance are studied in the laboratory in standard tests, which can vary from country to country. In Europe, the German DIN and Pan-European CEN standards are used.

7.2. METHODS OF ROCK EXTRACTION

7.2.1. General

The object of rock extraction and cutting at a quarry is to achieve the right size of stone block for the production process. The majority of building stones are cut into slabs by big frame saws. To ensure the efficient and economical production of frame sawing, blocks that are similar in size and have a right-angled prism-shape are required. Blocks of different sizes and shapes are possible to saw but more expensive.

Typical block sizes in production are:

260 cm x 130 cm x open
290 cm x 130 cm x "
290 cm x 160 cm x "
320 cm x 160 cm x "

Block weight should not exceed 30 tons for transportation reasons. Measuring is made in net measurement, therefore irregular corners reduce the sales volume. Scratches and fissures from blasting reduce the sales price. A number, quality code and sawing directions are painted on the side of the block during measurement.

7.2.2. Extraction and Cutting of Hard Rock

GENERAL

The process of dimensional stone extraction is divided into the following segments: panel extraction, cutting the panel into blocks and finishing the block. In small quarries, it is possible to quarry suitable-sized blocks directly from the open face. This is typical for black rock such as gabro. In larger production, however, this is not considered to be an economical method.

Panels may be several meters high, tens of meters long and usually several meters wide. The panel should move ahead 5-20 cm during the explosion. If the displacement is too great, the panel can break into irregular pieces. The panel is then cut into smaller pieces in its place. When the height of the panel is right, the boulders are tilted horizontally before the final cutting. During extraction, and the cutting and finishing phases, different methods are used to complement the drill and blast method: an open channel is made on one or more sides by using slot drilling, jet burning or a diamond-wire saw. Wedging and wire sawing are used in conjunction with blasting for cutting and finishing the boulders.

BLASTING

When the drilling & blasting method is used to extract building stone, it is based on the following instructions:

FIGURE *7.2.-1. Charging.*

FIGURE *7.2.-2. Blasting*

The strength of the explosive should be weak enough so that it causes as little micro-cracking as possible. Micro-cracking depends on the degree of the explosive's packing and also on the force of the detonation. An explosive's relative strength must be considered. Explosives also influence micro-cracking depending on the brittleness of the rock.

Drill holes are drilled close to each other and blasted at the same time so that the compressive wave causes major tensile stress to the hole line, which causes the crack.

The explosion moves the panel 5 - 20 cm forward. The direction of the extraction and shape of the panel is designed according to the joints of the rock. Joints can also be used as vertical or horizontal open channels. Rock protrusion must be correct so that the block can be easily extracted.

The most important stage in the process is drilling because if hole deviation is too great it

can not be fixed by charging. Mechanized drill rig equipment is used in big sites to achieve parallel and similar length holes. Hole errors are caused not only from collaring deviation but also from the use of too much feed. Schistosity and joints also influence hole deviation. Rods are integral steel up to 8 m. Tapered cross-bit crowns of approx. 30 mm in diameter are the most commonly used drill bits. The bits are sharpened according to the quartz content. Spacing of vertical holes is 0.15 - 0.4 m and horizontal holes 0.2 - 0.5 m.

In small quarries, hand-held drilling is still used. For dimensional stone quarries, small drill rigs are suitable. The drill rig can be set on a track or on a tractor. There is also special equipment for dimensional stone drilling. Horizontal drilling is possible by pusher-leg drilling. A more common way to accomplish this is to use special rigs that are manufactured for this purpose. These rigs can also be used in vertical drilling. Hydraulic hammer drills are gaining popularity among pneumatic hammers because they are more effective and less noisy.

Black powder, which used to be a common explosive, has now been replaced by linear-explosive charges and detonating cord. Detonating cord is used as a detonation line of the round. It also ensures that the explosives in the

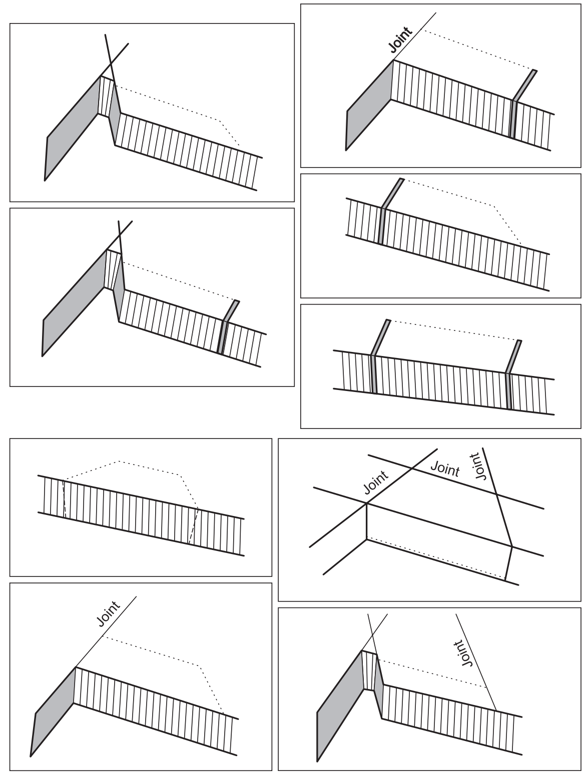


FIGURE *7.2.-3. Different methods of rock extraction*

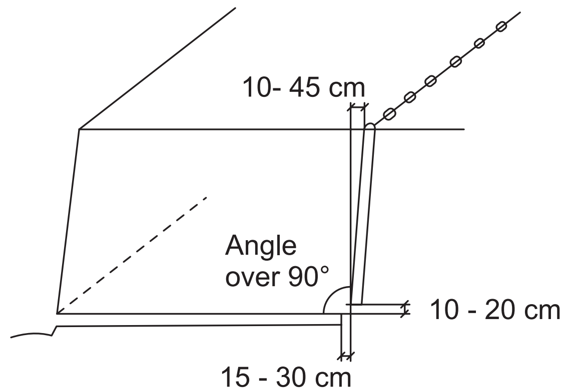


FIGURE *7.2.-4. Backside and bottom drilling* drill holes detonate simultaneously. Detonating cord can also be used as an explosive. Depending on size of the block and rock type, 0.06 - 0.15 kg/m^3 of explosives are needed. This means that the holes are not fully charged but, for example, are charged one meter from the bottom and one meter from the top. Detonating cord is detonated either by an electric detonator or safety fuse and plain detonator.

Table 7.2.-1. *Characteristics of various explosives.*

Explosive	*density, kg/l*	*velocity of detonation m/s*	*degree of packing kg/m*	*strength s ***	*micro-jointing-zone m **
Dynamite d=24 mm	1.5	6 000	0.6	1	1.2
K- pipe-explosive	0.95	1 900	0.22	0.3	0.06
Black powder	0.95	500		0.45	
Detonating cord	6 500	0.02	1.25		

* Calculated from degree of packing (theoretical).
** Calculated by Langefors method

After the extraction of the panel, solid parts are chosen according to color and joint. They can be cut with the drill & blast method. Spacing is 15 - 30 cm. The specific charge should be approximately 0.05 - 0.1 kg/m^3 . Wedging is also possible. When the block attains its final size and shape, it is tilted over on a gravel bedding. Tilting can be performed by an excavator or special hydraulic boom. Gravel bedding is used to prevent the block from breaking. In cutting and finishing, detonation cord can be used as an explosive.

WEDGING

The purpose of wedging is to split the rock boulder along the surface of the drill holes. Depending on the rock type, holes are drilled either almost entirely through the boulder or short holes are drilled. Short holes are practical, for example, in

FIGURE ***7.2.-5.*** *Wedging.*

gray small or middle-grain granite. Wedging is successful when it is done in the direction of the natural joints. Wedges can be manual or hydraulic. The most important fact is to increase pressure smoothly and slowly so that the tension has time to reach the drill line.

SLOT DRILLING

In slot drilling, overlapping holes are drilled to form an open channel. It usually begins with a 64-mm bit and a hole spacing of 114 mm. The rock, which is between two holes, is then reamed with a 64 or 76-mm bit, using a guide rod in adjacent holes. Open-channel drilling has superseded the jet burning method because it is as environmentally friendly as normal drilling and does not damage the rock. It can also be used on all stone types.

FIGURE ***7.2.-6.*** *Slot drilling*

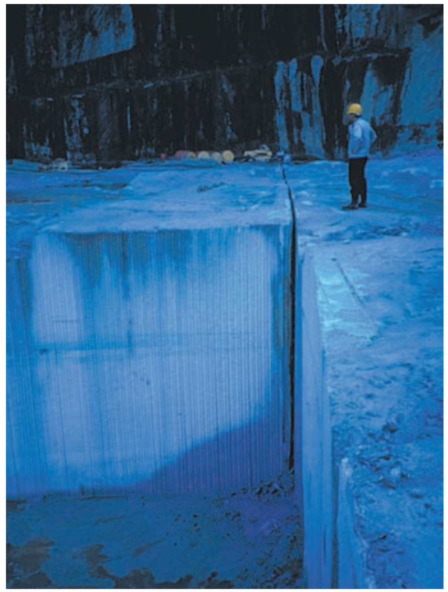


FIGURE ***7.2.-7.*** *Result in slot drilling.*

JET BURNING

Jet burning is used to make a slot for the sidelines before the blast. This method is similar to slot drilling. The rock is heated with a blow torch burning a mixture of pressurized air and fuel oil. The nozzle is at the end of a long arm and the operator moves it slowly along the slot. The rock heats up, the crystal water evaporates, and the rock, which weathers into small grains, flies away with the blow. This works best in rocks with a high quartz content.

Noise and dust emission from jet burning prevents others from working nearby and because the noise spreads to surrounding areas, it may not be permitted. Another disadvantage of jet burning is micro-cracking that occurs when the rock is heated. Micro-cracking can be up to tens of centimeters long.

DIAMOND-WIRE SAW

It is possible to make a narrow slot in hard rock by using thediamond-wire saw. This method is further described in chapter 7.2.3. The diamond wire for hard rock, called granite wire, is constructed differently than the one used for sawing marble.
The use of the diamond-wire saw is similar to slot drilling and jet burning.

SOFT ROCK EXTRACTION AND SPLITTING

Soft sedimentary rocks and some metamorphic rocks, such as marble and soapstone, were crafted in prehistoric times with primitive hand tools. Historical monuments, made by sawing, wedging, hammering and drilling, prove the long tradition of rock manipulation. Modern methods are based on the use of diamond tools and hard metal. In soft-rock extraction, drilling and blasting techniques are used in addition to other methods.

DIAMOND-WIRE SAWING

A new type of wire, which uses diamond segments mounted around the wire that act as abrasives, was developed in the 1960s. A more suitable wire type for harder rocks was developed from the diamond wire (marble wire), which was originally developed for soft rock extraction. Diamond-wire sawing is based on two drill holes that intersect at the corner of the sawing level through which the wire is threaded. Drilling is performed with the help of a plumb-line, theodolith or a laser guiding device. The wire is then pulled through the hole with a string or a hooked rod. After drilling the holes, the saw, which moves on a track, is fitted at the bottom or top of the bench so that the support wheel is on the same level as the drill holes. After attaching the wire-ends, the wire is tightened around the support wheel by moving the saw backward along the track. When the wire and rock make contact, water is flushed in (1-3m3/h) to cool down the wire and to remove the cuttings. The direction of sawing is performed away from the cut. Diamond wire consists of approximately 5 mm diameter wire and diamond coated bits of approximately 10-11 mm in diameter and 30-32 pcs/m. Wire durability depends not only on the rock, but also on the bending of the wire. Therefore at the beginning and end of sawing, the draught must be reduced. To ensure a longer wire lifetime, the pending radius must be big; for example, economical bench height is 8-9 meters.

Chain sawing

Just like wood, people have sawed rock since primitive times. For example, soapstone used to be extracted from rock by hand. In the 1960s, a saw more practical for dimensional stone extraction was developed based on the saws used in coal mines. Chain sawing with a steel-chained rock saw is performed on tracks. At first, hard metal was used as the cutting pieces. Saws with diamond segments were also later taken into use. A slot made by the chain saw is limited by the size of the device, unlike diamond-wire saws. Chain saws are generally used to extract block-sized stones and in underground extraction. Chain saws can also be economically used on rocks that are not harder than 2-3 on the Mohs scale.

7.3. QUARRY PLANNING

GENERAL

Local factors, such as topography, rock structure and excavation location influence on planning a quarry.:

- Production location is determined by total, monthly, and weekly production
- Rock waste
- The quarry should be continuously ready to produce stone for selling.
- The pursuit of “best place first” may lead to production breaks
- Even in the early stages of quarry start-up, sellable stone should be available as soon as possible
- The bigger the investment in machines and devices, the more important it is to ensure a high machine production and utilization rate

Bench leveling of the excavated area should be the proper production method. Opening the benches in the desired direction should be possible. Mapping natural cracks, color and quality variations and related follow-up during quarrying provides a solid basis for planning.

In addition to the quarry itself, space for activities essential to production must be planned:

- Geo-technical problems concerning overburden excavation must be identified.
- Waste rock sorting and storing and storage area should be optimized for cost efficiency and environmental reasons.

Quarry expansion and increased waste rock must be prepared for.

- Block finishing area, moving to the storage and storage area
- Access roads, office, other buildings and maintenance

Water and electricity requirements must also be identified as well as groundwater, rain and melt water pumping requirements when planning the working methods and sequence of subsurface quarries.

QUARRYING METHODS

Finishing big boulders is one of the earliest forms of dimensional stone excavation. It consists of breaking and shaping big loose blocks into smaller ones. Manual drilling, blasting and wedging are the methods used. Because it requires only a small investment, this method is mostly used in developing countries. Productivity is low.

An especially popular method for marble excavation is hillside or hilltop excavation. A steep hillside or hilltop is gradually leveled and the waste rock is pushed aside. The advantages of this method over bench quarrying include waste rock handling and no water problems. Expensive transportation costs and safety risks are the downsides of this method. Wire sawing and chain sawing are commonly used methods in marble extraction. Extraction by drilling and blasting with powerful explosives leads to much waste rock and hard-to-predict production. Cautious blasting explosives provides similar results to systematic bench quarrying.

In the United States, for example, a small area shaft that uses a crane for moving rock and equipment is a typical bench quarrying method. In some deposits, the amount of waste rock can be reduced, however the reach and capacity of the crane limit the possibility of production and excavation expansion. Moving blocks, waste rocks, machines and workers with only a crane reduces the effective working time and, ultimately, profitability.

Bench quarrying with wheel loaders and dumpers has become common worldwide. Excavation fans out like an amphitheater.
The machines use ramps to move from level to level. As it consists of several working sites, the size of the excavation does not limit production growth. If one part of the quarry can not produce good blocks, it can be compensated by another part. This type of excavation best fits gentle terrain (**FIGURE** *7.3.-1.)*.

FIGURE *7.3.-1. Bench Quarrying.*

7.4. FINANCIAL RESULT OF QUARRYING

7.4.1. Costs

Costs are reduced through the professional design of the quarry. However, stone is a natural material that has its own features which can affect productivity. Rock type has a great effect on productivity. For example, rock strength considerably increases the drilling costs. To attain optimal results, it is important to identify the right quarrying method.

Quarrying costs consist of capital, operating and general costs. Capital costs include machines, devices and buildings. Investments required for drill rigs and haulage equipment in a modern quarry are high. Therefore it is important pay considerable attention to productivity and minimum equipment downtime. Operating costs include labor, rock drill spares, charging, fuel, tires and maintenance. General costs include management, sales and marketing. Operating costs and rock drilling costs are further analyzed in **FIGURE** *7.4.-1.*

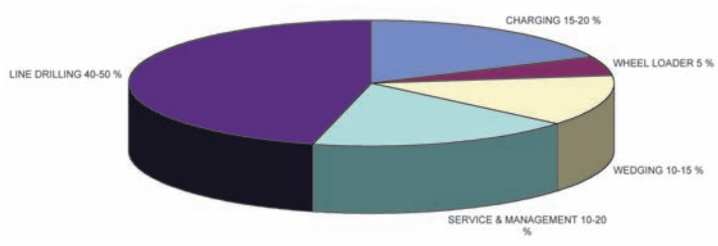


FIGURE *7.4.-1. Operating and drilling costs*

Competition is also affected by the quarry location. The distance from the quarry to harbor or production plant should be as short as possible. Building new roads considerably increases costs. The best economical result can be identified by comparing different transportation modes.

If the dimensional stone deposit is covered by thick overburden, it increases costs. Quarries are commonly located in mountainous terrain. Additionally, if the quarry is located in a residential area, it may be a problem to get the required permissions.
Only a small part of excavated stone is ultimately sold. Depending on the quarry, 50-95% breaks into too small pieces or has unacceptable features. Optimizing the use of waste rock would considerably increase the profitability of the quarry. Waste rock can be reduced by choosing the right technique for rock extraction, exact drilling and choosing the right type of explosive for blasting. Color variations, veins and openings in the rock reduce the supply. (**FIGURE** *7.4.-2.)*.

Typical amounts of waste rock for various granites:

- Gray granite 60 – 75%
- Red granite 75 – 85%
- Black granite 95 – 99%

If the rock is overly fragmented, it can result in too small blocks. Likewise, a small deposit results in unprofitable quarrying.

A horizontal stress field is found all over the world. Because of the non-fractured structure of dimensional stone deposits, the stress field is often rather high. Rock stress can increase production costs. It appears, for example, in diamond-wire sawing, when the open channel gets blocked. Costs are increased by difficult weather conditions, such as snow, below-zero temperatures and heavy rains. In winter, unprofitable downtime is caused by snow removal and problems associated with freezing. In warmer climates, downtime comes from road maintenance and pumping water out of the quarry. Additional costs may also come from unprofessional workers.
Professional drillers and charges can reduce overall costs through their working efficiency. Choosing the right type of drilling technique also reduces production costs. Today, for example, the annual cost of hand-held drilling may multiply as opposed to drilling with advanced drilling rigs.

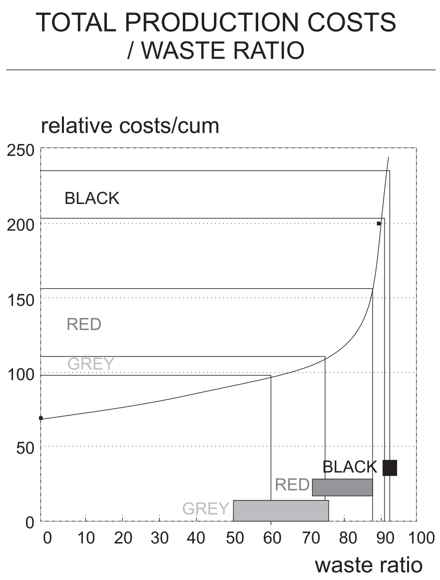


FIGURE *7.4.-2. Relative costs according to waste ratio.*

Differences between rock extraction and supply can be identified by comparing the drilled drill hole meters with obtained volumes of blocks.:

- Gray granite	< 20
- Red granite	20 - 50
- Brown granite	20 - 50
- Black granite (gabro)	50 – 100 drilled m/m³

A single quarry does not effect the world-market price of dimensional stone. Based on demand, prices of dimensional stone are calculated accordingly:

- Gray granite	20 - 40
- Red granite	15 - 40
- Brown granite	15 - 40
- Black granite	50 – 100 price index

The most economical way for getting a block out of hard rock is to use the drilling and blasting technique. Yet, if the open channel with slot drilling or diamond-wire saw sufficiently improves the supply compared to drilling and blasting technique, it may be better to change the technique (**FIGURE** *7.4.-3.)*.

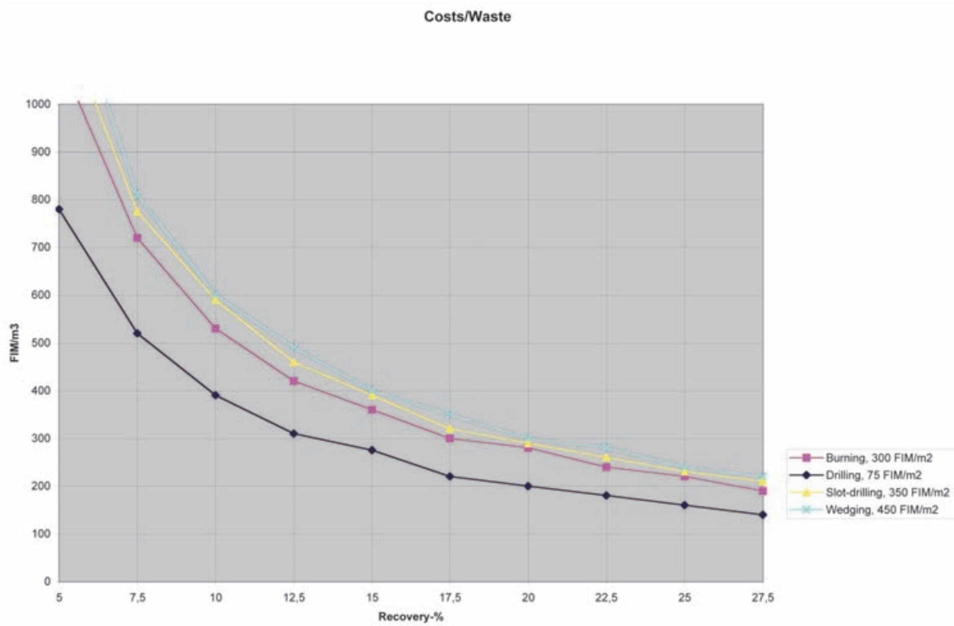


FIGURE *7.4.-3. Costs of different methods/ waste ratio*

7.4.2. Methods of succesful quarrying

The decision to buy dimensional stone from a particular quarry is made by the buyer and his designer. The decision is usually based on the stone's

- Appearance
- Technical features
- Price

A professional stone designer takes more things into consideration. As a professional, he pays attention to problems relating to the quarry. Depending on the project, one must pay attention to the following:

- General uniform color and appearance of stone
- Size and shape of block, which affects production costs
- Production capacity, according to rock type, quarry design and production equipment
- Delivery reliability
- Professional skills of the operation organization

In a competitive situation, when all above-mentioned issues have been considered, the customer bases his choice on the overall sales process. He may also base his selection on the type of customer service he received.

7.5. FOUNDING A QUARRY

When starting a new quarry, one must pay attention to:

- Sufficient demand for produced dimensional stone
- Production cost for loosening blocks
- Amount of raw material that can be taken into production

Common methods of geology mapping are used for locating a particular type of stone. Geologic, terrain and surface maps of rock deposits and possible geophysical research methods are used. Core drilling and test drilling are used for researching the features of a particular deposit. It is important to ensure that the rock has an even quality and has as few fragments as possible.

Production requirements, such as transportation, technical requirements for economical operation and obtaining the correct environmental permissions should be carefully considered when selecting the location for the quarry.

Test samples are loosened from rock for technical researching. The rock's technical features are tested from the stone plate and results are used when determining the proper application for the stone. To determine commercial demand for a particular type of stone, the test plates are shown to designers and at production factories. It is usually impossible to sell before delivery at the quarry is secured. Before actual test marketing can begin, a block must be turned into polished plates, which also enables a clearer view of the jointing and color balance of the rock.

Starting a quarry and securing its production depends directly on sufficient demand and on building up a clientele, although it is unprofitable to produce blocks for storage before sufficient demand is confirmed.

7.6. DRILLING TOOLS SELECTION FOR DIMENSIONAL STONE INDUSTRY

Within DSI, hole sizes are generally quite small (ø 25-34 mm) and are therefore too small to accommodate threaded equipment. The main equipment are integral drill steels and tapered equipment.

FIGURE *7.5.-1. Integral drill steels.*

H19 Integral drill steels

Suitable for the smaller hole range, ø 25-27 mm with an H19 x 108 mm shank and lengths up to 3.2 meter. Smallest available size is ø 23 mm.

H22 Integral drill steels

The H22 Integral drill steels, H22 x 108 mm shank, are the most common type within the DSI. The product range includes ø 26-40 mm and lengths from 0.4-7.2 meters.

Tapered equipment

Tapered equipment is available in different taper sizes 12°, 11°, 7° and 4°46´. Different taper sizes have different characteristics when it comes to the "knock-off" index, for example, a smaller angle will prevent the bit from spinning. The tapered bits have either ballistic buttons (ø 31-38mm) or inserts (ø 28-45mm).

R23 Extension equipment

If larger hole diameters (33 mm) are used, R23 equipment is a good solution. It contains both conventional rods and couplings as well as MF-rods.

Plug-hole drill steel

Plug-hole drill steels are used for splitting and squaring the blocks. It demands a stone easy to split straight, even with short holes. The plug-hole drill steel range includes both H19 x 82.5 mm and H19 x 108 mm shanks with bit sizes ø 17-22 mm.

8.1. GENERAL

Water is necessary for sustaining life on earth. All life depends on it, not only human beings. About 97% of the earth's fresh water supply is located underground (glaciers and ice caps excluded).

Most ground water that we use comes from rain and snow. Some water that falls on the earth's surface penetrates the soil and becomes subsurface water bearing formations (**FIGURE** *8.1.-1*). Useable ground water also includes water trapped in sediments and water that vaporizes from molten rock as it cools below earth's surface.

Subsurface water with formations that are capable of yielding sufficient water for wells are called aquifers, which can be found in gaps and pores between particles and grains in alluvial or unconsolidated formation or in cracks, and joints in consolidated formation. Although ground water exists almost everywhere, it is not always accessible to tap it. Layers containing water may be impermeable, thus preventing water from flowing into the well.

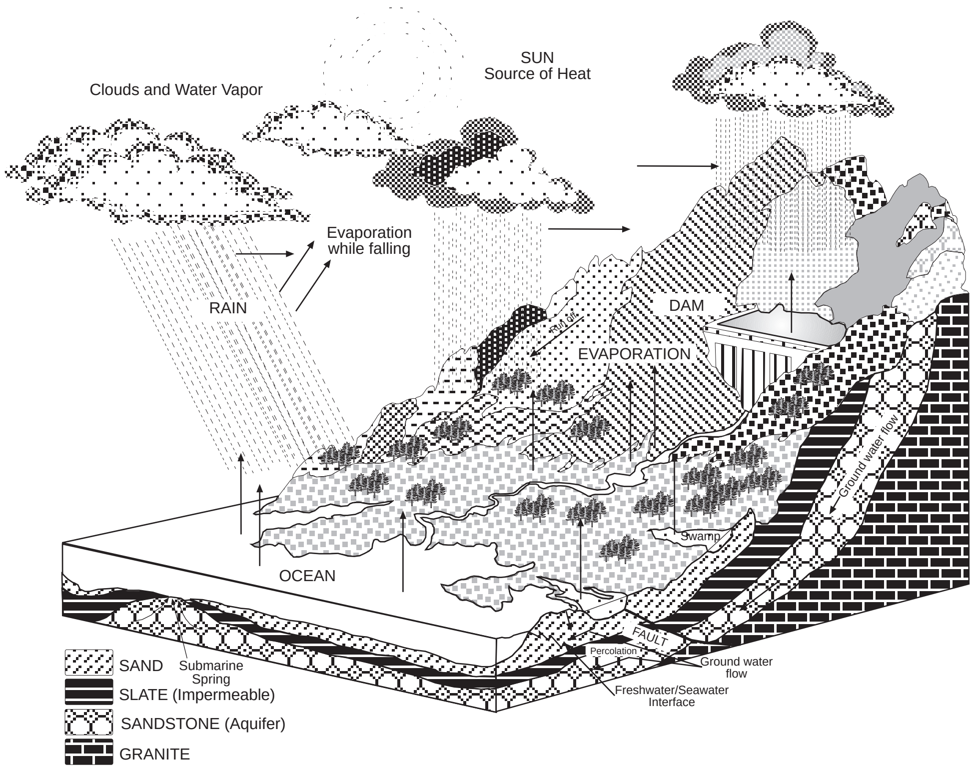


FIGURE *8.1.-1. Hydrologic cycle.*

The importance of fresh water has inspired people throughout history to develop various methods of tapping these valuable ground water resources. Many of these methods are still used today because different types of aquifers and environments (physical, financial or political) require different drilling techniques.

8.2. METHODS

CABLE TOOL DRILLING

Cable tool drilling is the oldest percussive drilling method and was developed by the Chinese about 4,000 years ago. The technique is still used today, but the original bamboo tools have been replaced by steel tools, and human and animal-generated power has been replaced by gasoline and diesel engines.

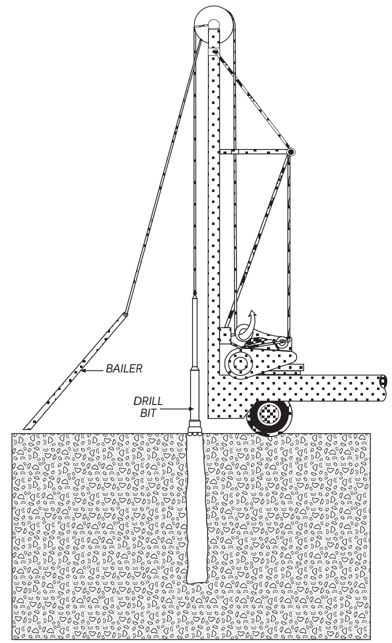


FIGURE *8.2.-1. Cabletool rig.*

Cabletool rigs (also called spudder rigs) drill by repeatedly dropping and lifting a string of heavy tools into a hole. The reciprocating motion required for lifting and dropping is imparted by the vertical movement of a spudding beam (**FIGURE** *8.2.-1.*). The loosened material or rock cuttings are mixed with water and removed from the hole periodically by a bailer or sand pump.

The bailer is a section of pipe with a check valve at the bottom. The cuttings open the valve when the bailer is lowered to the bottom. The valve closes automatically when the bailer is lifted. The sand pump is a bailer equipped with suction pipe and plunger. The plunger creates a vacuum that opens the check valve at the bottom and sucks sand and cuttings into the bailer.

The drill string for cable-tool drilling consists of 5 components: drill bit, drill stem, drilling jars, swivel socket and cable (**FIGURE** *8.2.-2.*). The drill bit (chisel) crushes the rock and mixes the cuttings. The drill stem is a heavy section of pipe that adds weight to the string and guides the bit in the hole. Drilling jars consist of two steel bars that are linked together. The free sliding jars are used to loosen a stuck drill bit on the upward stroke.
A swivel socket connects drilling tools to the cable and transmits the rotation of the cable to the drill string so that the drill bit crushes fresh rock in every downstroke. The cable carries and rotates the drilling tools by twisting the swivel socket on every upstroke.

Although cabletool drilling is a slow method and can´t compete with modern rigs, it has maintained its popularity in certain applications and geographical locations. Cabletool machines are simple, mechanical machines that can be repaired in almost any workshop. Initial investment costs are low and most spare parts can be locally produced. Cabletool rigs also have the advantage when drilling in alluvial formation that contains boulders because the casing is set during the drilling process. The casing stabilizes hole walls and prevents boulders from falling onto the bit.

BORING

Well boring is the non-percussive drilling of shallow wells in soil or unconsolidated formation with mechanical hole cleaning. Two main types of boring are commonly used in water well drilling: bucket drilling and auger drilling.

In bucket drilling, the excavated material is collected in a cylindrical bucket that has auger-type cutting blades at the bottom. The bucket is connected to a kelly bar (square cross-section tubing), which is rotated by a truck-mounted ring gear. Once the bucket is full, it is raised by a winch to the surface and dumped beside the rig. This procedure continues until the desired depth has been reached. The well diameters drilled with a bucket drill vary from 18" - 48" (457 mm - 1219 mm) and typical depths are less than 150´ (45.7 m).

In auger drilling, the drill string consists of one or several sections of tubing equipped with welded spiral flanges, called flights. When auger pipes are rotated, a special bit or cutter head, which is attached to the leading auger pipe, cuts the hole. The flights then carry the soil or cuttings to the surface (**FIGURE** *8.2.-3.*). The cutter head is usually 2" (51 mm) larger in diameter than the flights. Augers can be divided into two types: solid stem and hollow stem. The latter is mainly used in exploration drilling, but is becoming more popular in well drilling due to its ability to support the hole walls while screen, filter packing and permanent casing are set through the center hole. The common auger outside diameter in water well drilling application is 6" - 14" (152 mm - 356 mm) and the depth range is 40´ - 120´ (12.2 m - 36.6 m). Drilling rigs for auger drilling are usually tophead driverotary drills with adequate torque capacity.
Large diameter augers require drilling rigs that are similar to those used in bucket drilling.

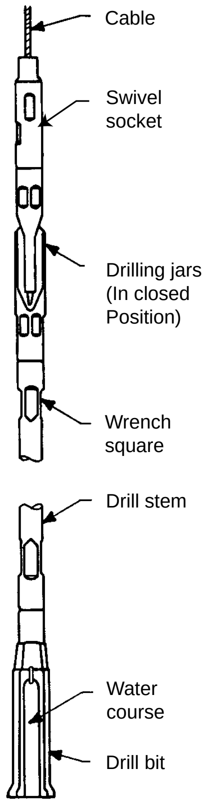


FIGURE *8.2.-2. Cable tool rig drill string.*

ROTARY DRILLING

Rotary drilling is the method in which a rotating bit crushes or cuts rock under heavy down pressure. This method was developed to increase drilling speed and depth in most formations. The drill string, which has an attached bit at the bottom, is rotated by rotary table or hydraulic top head. The weight of the drill string provides the required feed force for the bit. In tophead drive rigs, the weight can also be applied by a hydraulic feed system at the top of the hole. As the drill bit penetrates the hole, drilling fluid is forced down to the bit where it flushes the cuttings out of the hole. Depending on the drilling fluid and channel used to force it down to the bit, drilling can be divided into the following main types: air drilling, reverse air drilling, air hammer drilling (DTH), direct mud circulation drilling and reverse mud circulation drilling.

Air Drilling

Air drilling is a rotary drilling method that uses air, air/water or an air/water/foaming agent mixture as the hole cleaning fluid. Air drilling has the advantage of reducing drilling costs by increased penetration rates, longer bit life and reduced well development (cleaning) time. Air for hole cleaning is produced by a compressor that is either on the drill rig or a portable unit. Air is directed down the drill pipe where it cleans the face of the bit and flushes the cuttings out of the hole through the annulus between the drill pipe and hole wall. The pressure requirement for air drilling depends on the water head in the hole, but is typically 100 - 150 psi (7 - 10.5 bar). The air volume needed for hole cleaning depends on the hole and drill pipe diameter calculated by the

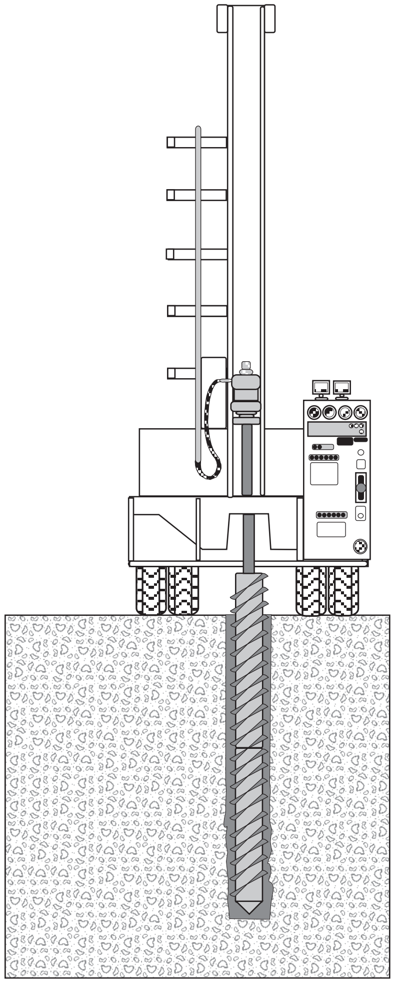


FIGURE *8.2.-3. Auger drilling*

following formula:

$$BV = \frac{183.3 \cdot CFM}{D^2 - d^2}$$

where BV = bailing velocity in fpm
CFM = compressor output volume in cfm
D = hole diameter in inches
d = drill pipe diameter in inches

In normal drilling conditions, bailing velocity for well drilling application should be 3000 - 7000 fpm (16 - 35 m/s). Bailing velocity can also be determined from **FIGURE** *8.2.-4.*.

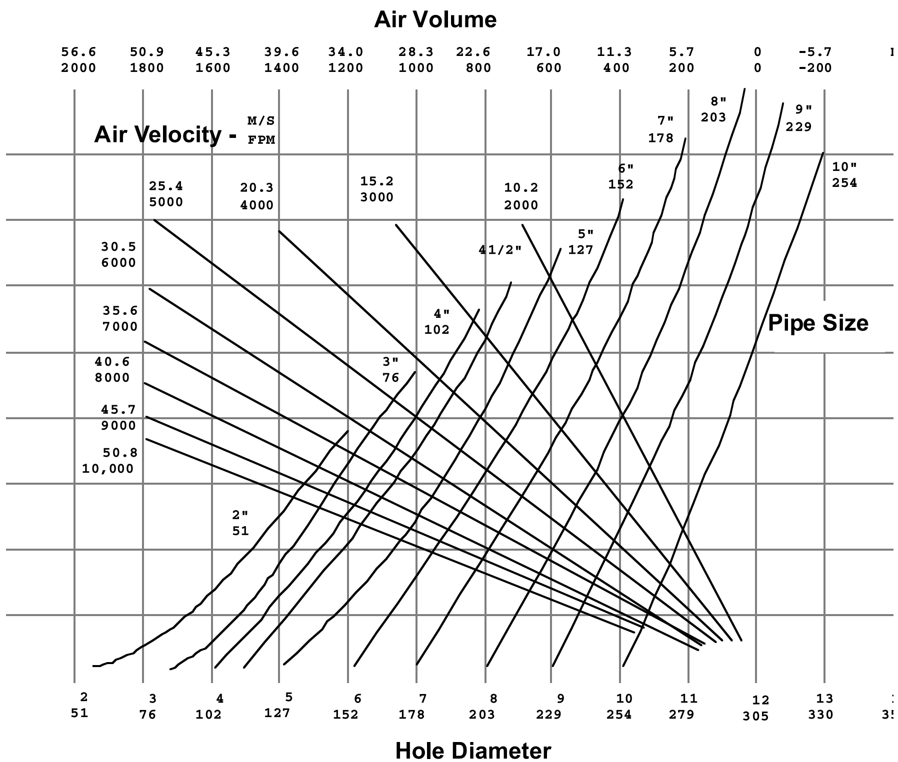


FIGURE *8.2.-4. Calculating bailing velocity when hole, pipe size and compressor volume are known.*

The formula above shows that the available compressor limits the maximum hole size. Bailing velocity can also be adjusted by the drill pipe, but a larger pipe diameter weighs down the drill string, thus limiting the maximum drilling depth. Hole cleaning is enhanced by adding small amounts of water to the flushing air. Water also suppresses dust from the hole and makes the entire drilling process more environmentally friendly.

Air lifting capacity can also be improved by adding a foaming agent to the air/water mist. The air/water/foam method carries larger cuttings, therefore improving the penetration rate in larger holes where the bailing velocity is insufficient to clean the hole properly. Bailing velocity in the hole annulus can be as low as 150 fpm (0.76 m/s) if foam is used.

Air drilling in water well applications is not especially common because it is limited to semi-consolidated or consolidated materials and it can not compete with DTH hammer drilling in medium hard or hard formations.

A typical air drilling rig comes equipped with a mast, holdback/ feed system, rotation unit, air compressor, prime mover (which may be a truck engine or a separate deck engine), hydraulic system or mechanical drive system and truck (**FIGURE** *8.2.-5.*). The holdback/feed system is needed to move the drill string up and down, and to hold it during the drilling process. In deep holes, the weight of the drill string can be too heavy for the drill bit and must be reduced by the holdback system. The holdback/ feed system in top-drive machines is operated by a hydraulic cylinder and chain or cable. Rotary-table rigs use a winch to carry the drill pipe load which means that feed force other than the drill string weight can not be applied.

FIGURE *8.2.-5. Hydraulic top-drive air drilling rig.*

A rotation unit can be comprised of either a hydraulic top driveor table drive, which is often a mechanical system. The top drive rotates the drill string from the top. The table drive is located at deck level on the drill rig (**FIGURE** *8.2.-6.*). Table drive requires a special drill steel joint (also called a kelly) to rotate the drill pipe. A kelly is the first section of the drill steel after the swivel and must be removed each time a drill pipe is added. The kelly's cross section is usually square, but can also be hexagonal or round with lengthwise grooves.

Usually, the prime mover in an air drilling rig is a deck-mounted diesel engine that powers all drilling functions through a hydraulic or mechanical drive system. Mud drilling rigs and small air drilling rigs that may require separate, portable air compressors, are often run off a truck engine.

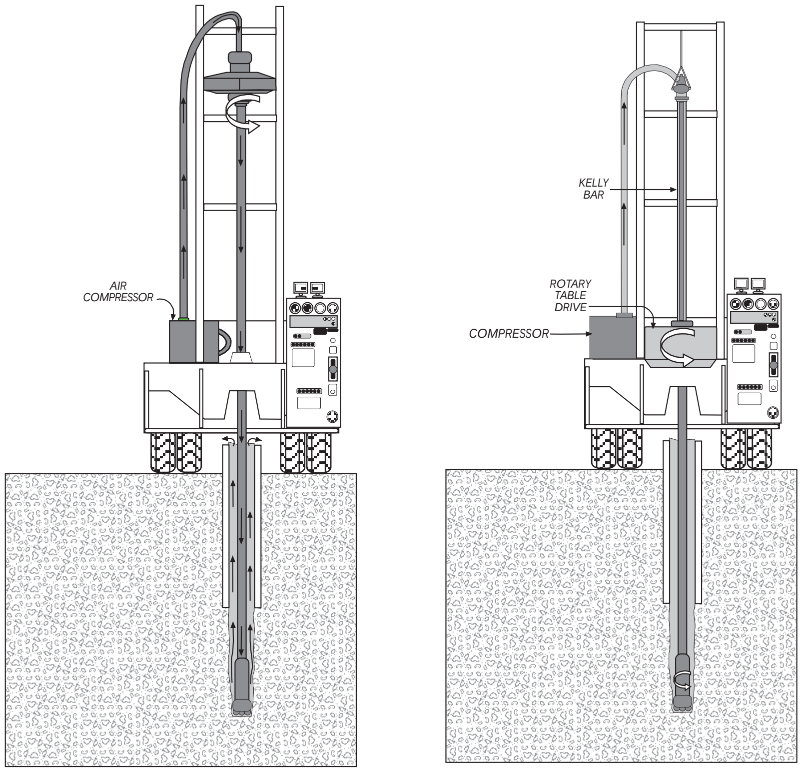


FIGURE *8.2.-6. Hydraulic top-drive and mechanical table-drive rigs.*

The drill string in rotary drilling consists of several components, each with its own important function: top sub, drill pipe, cross-over sub, drill collar, stabilizer, bit sub and bit (**FIGURE** *8.2.-7.*).

The top sub is an adapter between the top head or swivel and the drill pipe (threads may vary). The drill pipe conveys the rotation torque and drilling fluid to the bit. Drill collars add weight to the bottom of the drill string and keep it and the hole straight. Drill collars also play a very important role at the top of the hole when drilling with table-drive machines because they don't have any feed force. A cross-over sub is an adapter that is needed when the thread of the drill collar and the stabilizer is not the same. The stabilizer, usually attached next to the bit and has a diameter of 1/4″ smaller than the bit diameter, is used to guide the drill bit so the hole stays straight. The bit sub is another adapter between the stabilizer and the drill bit. In DTH drilling, the hammer is located next to the bit at the bottom; stabilizers and drill collars are not normally used. Reverse circulation air drilling is yet another type of air drilling and is further explained in chapter 9.2.2.

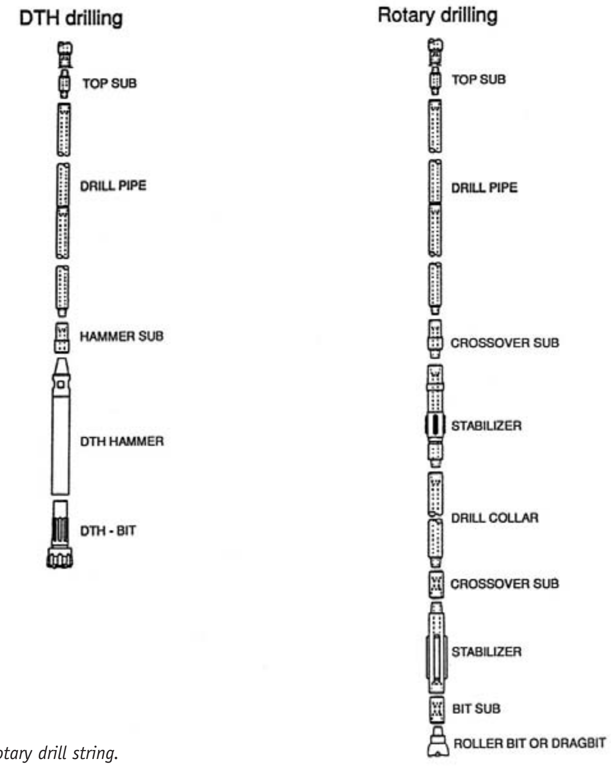


FIGURE *8.2.-7. Rotary drill string.*

DOWN-THE-HOLE (DTH) HAMMER DRILLING

DTH drilling is actually a percussive method, but is usually classified as a rotary drilling method because rotary air drilling rigs are used.

A pneumatic hammer and bit, attached to the lower end of the drill string, hit the rock at the bottom of the hole at a high frequency. Air with high pressure, which is directed down the drill pipe, operates the DTH hammer and cleans the face of the bit so that it always hits a clean surface. The drill string, hammer and bit are rotated slowly by the top head. The rig's feed system keeps the bit firmly on the rock. Since DTH drilling is a percussive method, it requires neither high torque or bit load.

Today, the optimum operating pressure for DTH hammers is 350 psi (24.6 bar), which sets the requirement for air compressors. Necessary air volume depends not only on hole size, but also on the size of the DTH hammer. The bigger the hammer is the more air it requires to hold 350 psi (24.6 bar) operating pressure (**FIGURE** *8.2.-8.*). The DTH hammer's penetration speed is proportional to the operating pressure.

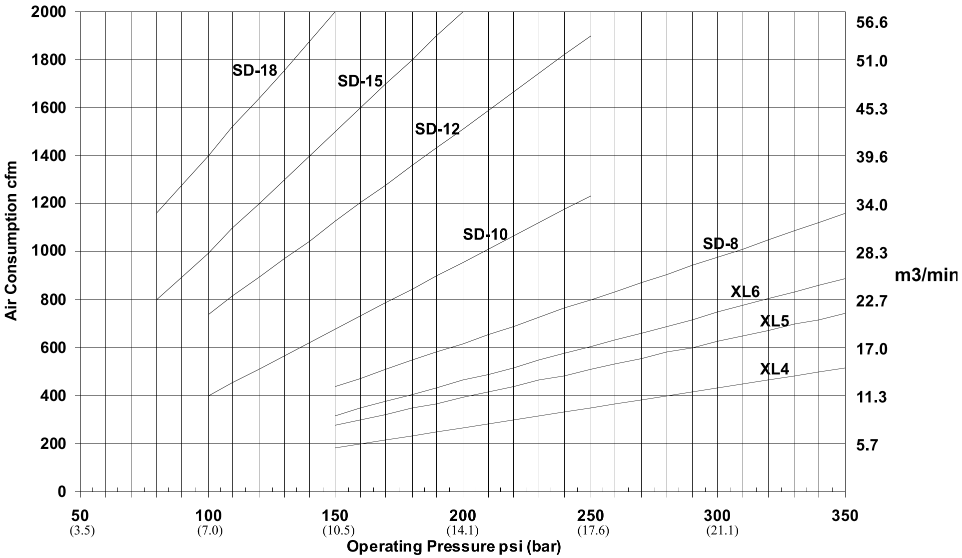


FIGURE *8.2.-8. Down-the-hole hammer air consumption (Sandvik XL and SD series).*

The most common sizes of DTH hammers for water well drilling are 6" and 8", but sizes such as 4" and 5" in the lower end are not unusual. Also 10", 12", 15" and even 18" hammer are used in water well drilling, however the operator must compromise the 350 psi (24.6 bar) operating pressure. Self-contained well drilling rigs do not have a compressor that is big enough to hold the optimum pressure in large hammers and the connection of an additional portable compressor into the air system is not feasible in most cases due to high operating costs.

In well drilling applications, the DTH hammer has the fastest penetration in hard formation. Even if the hammer operates at less than the optimum pressure level, penetration is faster than in air or mud rotary drilling.

DIRECT CIRCULATION MUD DRILLING

In the direct rotary mud method, the drilling process is similar to air drilling except the flushing fluid is water or a water/bentonite clay mixture (also called drilling mud). The air compressor is replaced by a mud pump that pumps mud through a rotating drill pipe and a bit to the bottom of the hole. The fluid picks up the cuttings and then flows upward through the space between the drill pipe and the hole wall to the ground surface. At the surface, the mud flows to the settling pit where the cuttings fall down to the bottom. From there, the clean surface mud flows through a ditch to another pit where the suction of the mud pump is located (**FIGURE** *8.2.-9.*).

When selecting the correct mud pump, two factors must be considered: mud volume for creating the necessary bailing velocity, and pressure for compensating pressure loss in the drill string and surface equipment. The following formula is used to calculate the required mud pump volume capacity:

$$BV = \frac{24.5 \cdot GPM}{D^2 - d^2}$$

where BV = bailing velocity in fpm
GPM = mud pump displacement in gpm
D = hole diameter in inches
d = drill pipe diameter in inches

The recommended bailing velocity is 80 - 150 fpm (0.41 - 0.76 m/s). A lower velocity results in a slower drilling speed. This fact must often be accepted in water well drilling. A mud pump necessary to create 80 fpm or more bailing velocity in a medium or large diameter well is too big and heavy to be mounted on a truck.

The mud pump pressure requirement depends on the hole size, drill pipe size, hole depth, mud pump displacement and type of surface equipment (in other words, everything between the first drill pipe and mud pump). Detailed pressure loss calculations are fairly complicated and usually the losses are only estimated using tables and graphs shown in well drilling manuals.

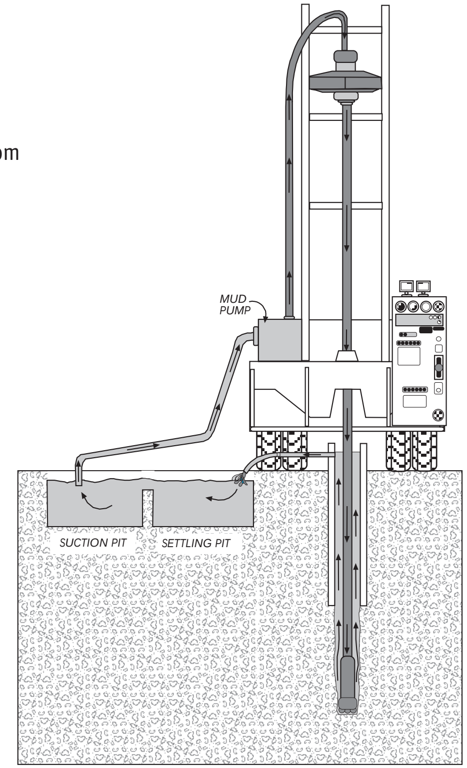


FIGURE *8.2.-9. Direct-circulation mud drilling.*

Drilling mud has the following main functions in the well drilling process:

- Drill bit cooling and lubrication
- Cuttings removal from the hole
- Hole wall support and caving prevention
- Sealing hole wall to reduce fluid loss
- Blow-out prevention (oil and gas exploration)

Correct mud properties are essential for a successful well drilling process, therefore, each well must be individually designed. Main mud drilling properties include density (specific weight), viscosity and filtration. Mud density affects the hydrostatic pressure, pumping pressure, lifting capacity of cuttings, and the stability of the hole walls. Excess density may cause damage to the aquifer by forcing solid material into the formation. Likewise, insufficient density may cause the walls to cave in.

Mud viscosity affects the pumping pressure, lifting capacity of cuttings and the settling time of cuttings in the mud pit. Mud viscosity should be low enough for rapid cuttings removal in settling bit, but high enough to ensure proper hole cleaning.

Filtration properties consist of the mud´s ability to form a thin low-permeability filter cake on the hole walls. Walling prevents drilling fluids containing solid content from entering the water bearing bed and plugging the aquifer. A thick filter cake can restrict the passage of tools.

The direct circulation mud drilling method can be used in almost any kind of formation or drilling depth. There are however places where bentonite mud has been prohibited for environmental reasons. Additionally, if the formation has many cavities and cracks, mud drilling can not be used due to the risk of circulation loss.

REVERSE-CIRCULATION MUD DRILLING

The reverse-circulation method was developed to overcome hole size limitations in direct mud drilling. As the name indicates, the difference between direct and reverse mud circulation is the direction of the fluid flow inside the drill pipe (**FIGURE** *8.2.-10.*). The fluid and cuttings are sucked through the drill pipe to the surface by a mud pump and then discharged to the settling pit. Pit fluid flows back to the bore hole and down to the bit between the drill pipe and the hole wall due to gravitational pull. Drilling fluid usually consists of water or very light mud.

The reverse circulation system is used mainly in large diameter wells in unconsolidated rock (**FIGURE** *8.2.-11.*). Whereas hole size in direct circulation is limited by the mud pump capacity, in reverse circulation even a medium-sized mud pump can create enough bailing velocity to bring the cuttings up because the inner diameter of the drill pipe is relatively small.

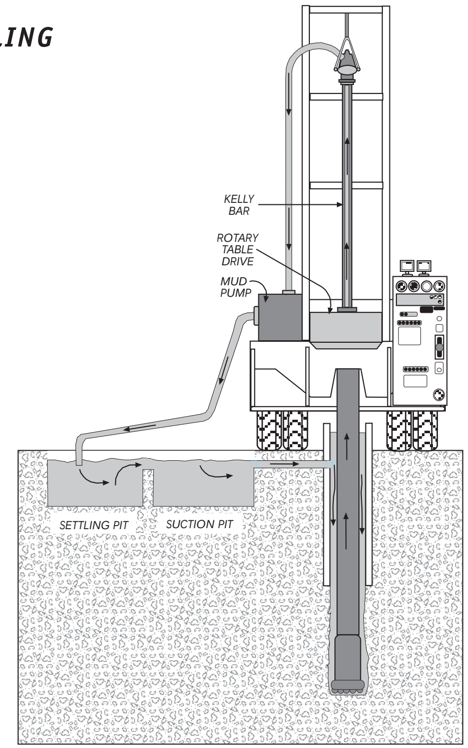


FIGURE *8.2.-10. Reverce-circulation mud drilling.*

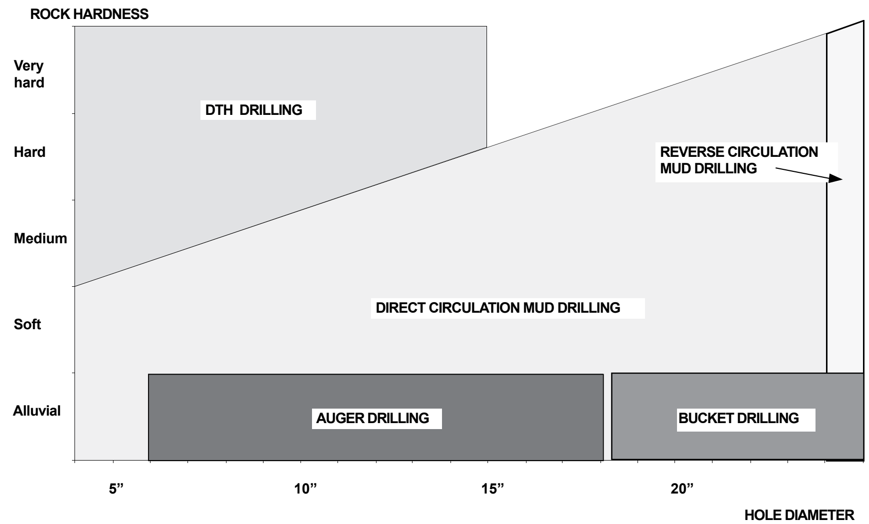


FIGURE *8.2.-11. Selection guide for rotary well drilling.*

The mud pump is usually a centrifugal type which can handle the cuttings without excessive wear. Reverse-circulation rigs are always rotary table type because they have more rotation torque and can handle larger drill pipe diameters.

Another form of reverse-circulation drilling is air-lift drilling, which is suitable for top-head drive machines (**FIGURE** *8.2.-12.*).
This system consists of a side discharge swivel that is mounted underneath the top drive, and special drill pipes equipped with built-in air channels. Instead of the centrifugal mud pump, compressed air is used to circulate the drilling fluid. Air, injected through pipe's air channels, enters the pipe and creates a flow that brings the fluid and cuttings to the surface and through the discharge swivel to settling pit.

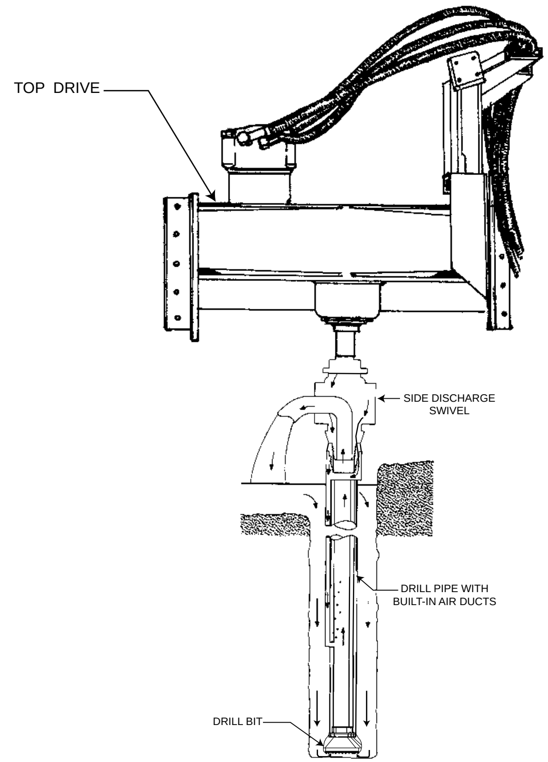


FIGURE *8.2.-12. Air-lift reverse circulation drilling.*

8.3. WELL DRILLING PROCESS

DRILLING PREPARATION

Preparing for a drilling project starts as soon as the location of the well has been decided. The location is chosen based on geological, hydrological or merely functional reasons.
Preparations include:

- Opening access road to the site
- Transportation of drilling tools and accessories (drill pipe, bits, hammers, casing, screens, bentonite, foaming agent, etc.) to the site
- Transportation of water to the site, if otherwise unavailable
- Mud pit and ditch digging (mud drilling)
- Rig set-up (leveling, raising the mast, building the starting drill string, etc.)

DRILLING

The actual drilling procedure varies a lot depending on the formation and purpose of the well. The following three main types are identified:

A) Alluvial or unconsolidated formation

Drilling is performed in several stages starting with a large diameter bit. After each stage, the casing is run down and drilling continues with a smaller bit through the casing. The depth of each stage depends on the drilling rig's holdback capacity (how much each size of casing it can handle).

B) Consolidated formation - large diameters

Drilling starts with a pilot hole, which is drilled to the final depth. The hole is then opened with a reamer bit to the final diameter. It may be necessary to ream several times with different reamer sizes to reach the final diameter. A top hole is drilled prior to the pilot hole if surface casing is used.

C) Consolidated formation - small to medium diameters

Consolidated formations are often covered with a shallow layer of soil. In order to get through this layer, the well is started driving casing simultaneously with the drilling. (Tubex overburden drilling or casing hammer). Once solid rock is reached, the drilling continues with the desired diameter to the final depth.

WELL SCREEN, CASING AND GRAVEL PACKING PLACEMENT

A well screen is a section of slotted steel casing that allows water from the aquifer to enter the well. Well screens are used in unconsolidated formation to prevent sand from entering the well. If the aquifer formation is very fine, gravel packing is placed around the screen to make the material around the screen more permeable.

Well casing has several functions in the well. It houses the pumping equipment, works as a conduit for upward flowing water and seals contaminated surface water from the well.

The well screen, casing and gravel packing are set down in the following way based on the formation (**FIGURE** *8.3.-1.*):

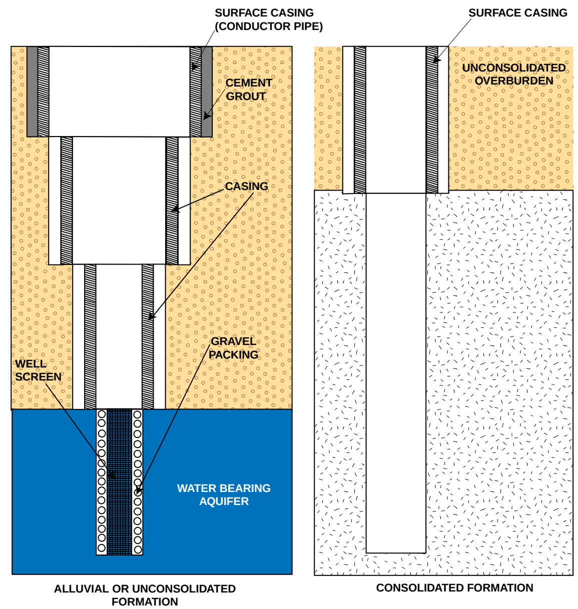


FIGURE *8.3.-1. Well designs in alluvial and consolidated formations.*

A) Alluvial or unconsolidated formation

The casing is set down after each diameter in telescopic drilling. The screen and gravel packing are set on the level of aquifer.

B) Consolidated formation - large diameters

If used, the surface casing is set after the hole has been completed. Screens and gravel packing are not used.

C) Consolidated formation - small to medium diameters

If necessary, the casing is set simultaneously with the drilling until solid rock is reached after which the casing is no longer needed. Screens and gravel packing are not used in this type of well.

WELL DEVELOPMENT

Well development repairs the damages done to the aquifer during the drilling operation so that the hydraulic properties can be restored. Additionally, it alters the basic physical properties of the formation so that more water can flow into the well. If well development is not properly performed, the aquifer's water yield may not be adequate. The life span of a water pump may also be limited in a poorly developed well due to a high clay and sand content in the water.

There are several methods of developing wells, include surging with a plunger, air lifting and jetting. In surging, the plunger forces water from the well into the surrounding formation on the downstroke. The upstroke pulls water back into the well drawing all particles small enough to pass through the screen along with it. Air lifting is similar to air lift reverse circulation except that it uses clean water. Jetting is a method in which high pressure water is pumped through small nozzles into the formation.

TEST PUMPING

Test pumping consists of determining the performance characteristics of the well and the hydraulic parameters of the aquifer. In a well performance test, yield and drawdown are measured so the capacity of the well can be calculated. An aquifer test is performed to estimate the affect of the new well to the aquifer. Test pumping also provides valuable information for correct pump selection.

PUMP INSTALLATION

The last operation in the well drilling process is pump installation. Pumps are installed in water wells to lift water to the surface and pump it to the point of use. Pump installation is often a side business for drilling contractors, but there are also companies that specialize in it. Pump selection is made by matching well performance characteristics, customer requirements and pump specification.

9.1. GENERAL

The objective of exploration drilling is to gather information about the formations below ground surface. This data is used to locate ore bodies and determine the mineral content and quantity in the subsurface layers. Methods used in exploration drilling include:

- Diamond coring
- Dual-tube reverse circulation drilling
- Cable tool drilling
- Hollow-stem auger drilling
- Rotary air and mud drilling

Cable-tool drilling, rotary air & mud drilling, and auger drilling methods were discussed earlier in chapter 8. The first two, samples were taken from the drilling cuttings. Auger drilling provides a soil sample. Diamond coring and dual-tube reverse circulation drilling are methods developed specially for exploration drilling.

9.2. METHODS

DIAMOND CORE DRILLING

Diamond coring is a method that provides core samples. The name of the method refers to the drill bit, which has small diamond inserts. These bits may also have tungsten carbide inserts or even rollers instead of diamonds. However, the method is still referred to as diamond coring. Core drills perform best in consolidatedformations.

A hollow coring bit is attached to a core barrel that collects sample into an inner tube as the bit penetrates the formation (**FIGURE** *9.2.-1.*). The inner tube is swiveled so that it stays stationary at all times. There are two ways to get the core sample out when the inner tube is full. In conventional coring, the complete core barrel must be drawn up to the surface, which means that all drill rods must be pulled out of the hole. This is very time consuming since it must be done every time the inner tube is full. In wire-line coring, the inner tube is lifted to the surface and lowered back to the barrel through the drill rods with a winch. This methods saves time because the drill rods stay in the hole.

Diamond coring drills are small in size compared to rotary drills and are therefore well suited for remote exploration sites. Some are small enough to be transported by helicopter to areas without access roads. The rig consists of a high-speed rotation unit (over 1000 rpm), a feed system, which provides smooth and consistent pressure on a bit, and a diesel engine that powers the unit. Drill rod handling is manual because it uses shorter and lighter drill pipes than rotary drilling. Coring drills are equipped with a water pump for flushing cuttings to the surface between the hole wall and drill pipe.

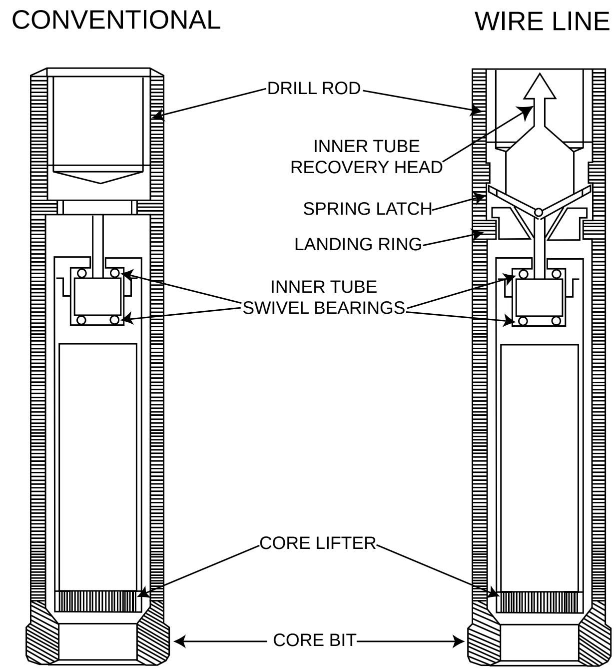


FIGURE ***9.2.-1.*** *Conventional and wire-line core barrels.*

CENTER SAMPLE RECOVERY (CSR) DRILLING

Center sample recovery drilling is a continuous sampling method that provides uncontaminated chip samples. This method is also known as reverse air or reverse dual-tube drilling because it uses a double-wall drill pipe. The drilling fluid is usually compressed air, but water, foam or bentonite mud can also be used.

Compressed air is forced between the inner and outer tube down to the bottom of the hole. The air cleans the face of a bit and brings the sample cuttings through the center hole of the inner tube up to the surface. Samples are then directed through side inlet swivel and top-drive rotary head into the discharge cyclone. After the cyclone, the samples can be split and collected as required (**FIGURE** *9.2.-2.*).

The drill rig for CSR drilling is often a standard rotary machine equipped with a side inlet swivel, cyclone and dual-tube drill pipes. Drilling can be performed with either a tricone bit or DTH hammer depending on the formation. Standard bits and hammers can be used in the drilling process, but tools designed especially for CSR drilling give better results. The normal bit size used in CSR drilling is 5 1/8″ to 5 1/2″ (130 mm - 140 mm). The dual-tube drill pipe diameter is only slightly smaller than the bit diameter (1/2″ - 1″), so that cuttings can not be flushed out between the hole wall and drill pipe. Additional tricone bit shrouds or hammer sleeves are used to block the annulus.

The superior drilling speed over core drilling has been one of CSR drilling's key success factors. CSR drilling provides samples approximately 10 times faster than coring. There are, however, cases where core drilling is considered the only alternative in sampling. The following table compares the benefits of both coring and dual-tube reverse circulation drilling:

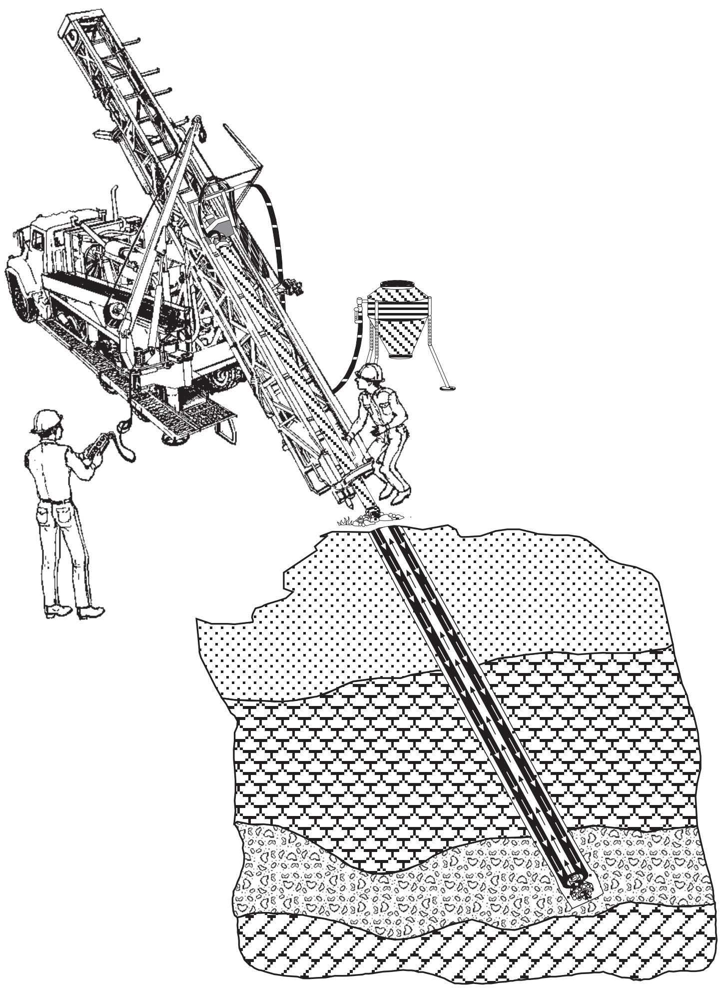


FIGURE 9.2.-2. *Dual-tube reverse circulation drilling.*

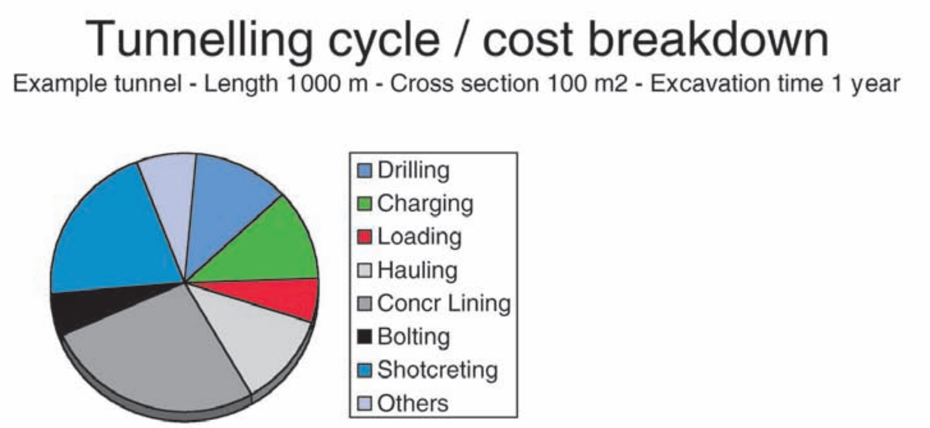
Table 9.2.-1. *Comparison between diamond coring and CRS drilling in mineral exploration.*

METHOD	*DIAMOND CORE DRILLING*	*CRS DRILLING*
SAMPLE	- Core sample - Uncontaminated or contaminated - Provides complete range of information	- Chip samples - Uncontaminated - No information on rock structure and physical properties
FORMATION	- Consolidated formations	- All formation types
DRILLING FLUID	- Water	- Compressed air - Water, foam or bentonite mud optional
DRILLING SPEED	- Slow (intermittent sampling)	- Fast (continuos sampling, possibility to use DTH hammer)
COST	- Low investment costs - High drilling costs due to slow speed and expensive bits	- High investment costs - Low drilling costs due to fast speed and longer life of drilling tools
TRANSPORTABILITY OF EQUIPMENT	- Excellent, due to size of equipment	- Good in developed areas - Poor in remote areas

10.1. PROJECT COSTS

Most costs are time related. Penalties can be very high if the project can't keep its schedule. Savings may be significant if the project is finished ahead schedule. Equipment availability is essential to finish the project in time, but it is also important to have committed personnel at the work site. Trained operators and service people as well as spare part logistics also affect the time spent on the project.

The following illustrations (**FIGURE** *10.1.-1.* and *10.1.-2.*) give an example of time and cost relations in the tunneling cycle.



Stage	Capital	Labour	Consumables and Material	Total (MUSD)
Drilling	0.15	0.30	0.15	0.6
Charging	0.05	0.20	0.35	0.6
Loading and Hauling	0.25	0.50	0.15	0.9
Bolting	0.10	0.10	0.10	0.3
Sprayed Concrete	0.05	0.15	0.80	1.0
Concrete lining	0.05	0.30	1.45	1.8
Others	0.10	0.25	0.35	0.7
Total	0.8	1.7	3.4	5.9

FIGURE *10.1.-1. The above circle illustrates different jobs in tunneling cycle. The cost breakdown figures for different jobs are appear in boxes.*

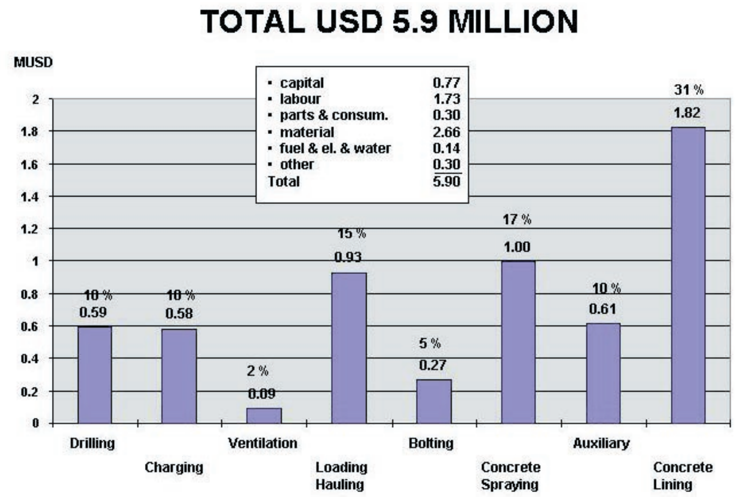


FIGURE *10.1.-2. Cost relations in tunneling cycle.*

As shown on the above chart (**FIGURE** *10.1.-2.*), concrete lining is very costly. The example is a real one-kilometer-tunnel (cross-section 100 m^2), which concrete lining cost deviation is 30-50% according to design, and 50-70% because of overbreak. Therefore the object is to plan tunnel dimensions in the right place with the right surface quality. The following matters should considered to achieve greater accuracy in tunneling:

- Optimized drilling and blasting pattern
- Optimized equipment with skilled operators
- Accurate tunnel location by using laser as reference point
- Tunnel profile line manually marked or by using TCAD or Data jumbo
- Drilling accuracy using precise instrumentation, ref. TAMROCK Measurement System TMS, TCAD, etc...
- Accurate charging with high-quality explosives and detonators
- Result follow-up-modifications to drilling and blasting pattern

Optimization is the key to a successful project. (**FIGURE** *10.1.-3.*) Should you select more expensive but more effective and faster equipment; or cheaper, less effective and slower equipment? If the majority of costs comes from wages and interest, for example, rather than from consumables, explosives etc., then it is more important to choose the more effective equipment.

When choosing equipment, quality should also be taken into account. Location accuracy, reduced overbreak, minimizing drill holes are also important.

Optimization can be attained through partnering. **FIGURE** *10.1.-4.* shows the main aspects of what a contractor can gain through partnering.

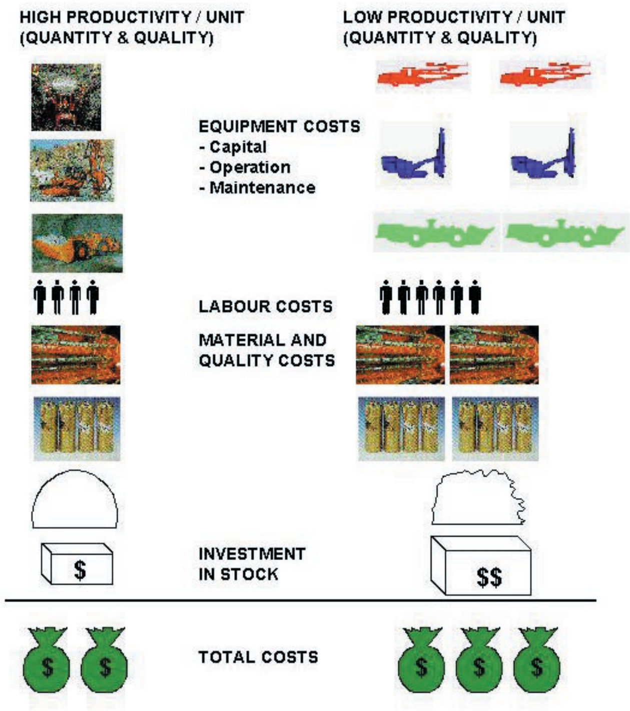


Figure 10.1.-3. *Reducing costs by maximizing productivity/unit.*

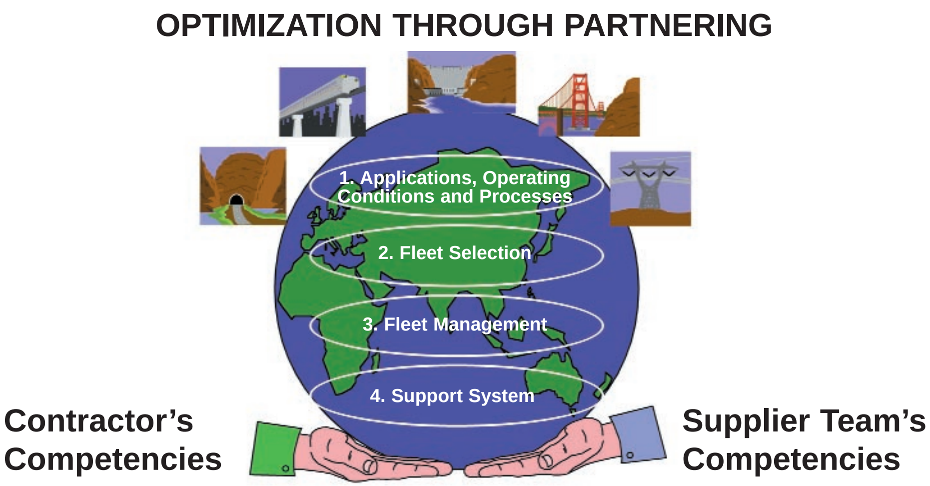


Figure 10.1.-4. *Optimization through partnering.*

10.2. TAMROCK PROJECT STUDIES

10.2.1. Excavation process recommendations

Project documents often determine the excavation method, but to optimize the excavation process many factors must be determined. The key issue is to have a balanced fleet to optimize performance. Operations can be cyclic or continuous depending on the selected rock excavation method. Factors affecting performance include rock conditions, operator experience, chosen method, hauling distance etc.

Drilling rate index (DRI) is one way to determine the rock's characteristics. A low DRI value indicates a slow drilling penetration rate and a high DRI indicates high penetration rate. DRI varies typically between 20 - 100. (See more in Chapter 2.3.)

Available time may vary from months to years depending of the size of the site and project, and it also greatly affects fleet size. Operator experience varies from beginner to professional, however, for example in face drilling a computerized system reduces the effect of operator skills.

The selected method, for example, drill & blast compared to mechanized method, also affects equipment selection. If the D&B method (example of cyclic method) is chosen, the fleet will consist of drifting jumbos, bolters, LHD loaders and dump trucks. If mechanized and continuous methods are used, equipment selection may consists of TBM or roadheader and hauling wagons for example on rails.

Hauling distances can vary from hundreds of meter to kilometers depending the purpose of the rock and site location. This determines the hauling system. Excavated rock can be used directly on the same work site for filling or is hauled to the crusher or dump area to wait future use.

10.2.2. Equipment selection

Before equipment selection can be done, the excavation method should be selected. Some factors in underground projects:

Following list is for underground fleet selection:

- Work schedule
- Rock properties
- Excavation method
- Drill and blast
- Road headers
- TBM
- Hammer

- Excavators, etc.
- Excavated sections, such as caverns, tunnels, shafts
- Size and amount of excavated sections
- Tunnel length and size such as width and height and variance of dimensions
- Advance
- Operators skills
- Operation cycle
- Bolting- done by jumbo, robolt or other
- Need for grouting/drainage holes
- Loading and haulage type and distances

Some factors affecting surface projects:

- Work schedule
- Rock properties
- Excavation method
- Purpose for excavation such as rock fill dam, production of concrete, etc.
- Size of excavated section
- Hole diameter
- Bench height
- Required block size
- Operators skills
- Operation cycle
- Loading and haulage type and distances

10.2.3 Performance and cost studies

TAMROCK has made some study programs as a tool for process recommendation, equipment selection, and performance and cost studies. See attached study sheets of tunnel and surface study programs. Drill and blast tunneling, surface drilling, bolting and underground loading study programs are very useful for calculating construction studies. There is also an available study for calculations on full-face methods such as tunnel boring machines. And Tamrock also has a long hole study specially designed for mining.

To enable performance and cost study the above-listed facts should be identified. Local unit prices for energy, explosives etc. should also be determined.

Through these studies, performance figures can be calculated for selected equipment, and the related total investment and operation costs.

Following are results from tunneling study calculations:

- Penetration rates on DRI (drilling rate index) basis
- Round drilling time
- Loading capacity
- Loading time
- Bolting capacity
- Bolting time
- Other cycle time estimates
- Charging time
- Blasting and ventilation time
- Concrete spraying time
- Total cycle time- tunnel advance rate - long-time advance rate with single and double heading Single heading: One drilling jumbo is working at one tunnel at the time. Double heading: One jumbo can be used in two tunnels or faces when drilling is done in face A the jumbo is moved to drill face B while other works (e.g. blasting, loading etc.) are under work in face A.
- Costs

Surface Rock Drilling/Capacity

Project:	EXAMPLE
Location:	QUARRY
Section:	BENCH

SITE INFORMATION

	Hole diameter 89 mm	Hole diameter 102 mm	Hole diameter 115 mm	
Rock type:	Granodiorite	Granodiorite	Granodiorite	
Drillhole diameter:	89	102	115 mm	
Bench height:	12	12	12 m	
Hole inclination:	0	0	0 deg	
Drilling rate index:	45	45	45	

DRILLING EGUIPMENT SELECTION

Type of rig:	PANTERA 900	PANTERA 900	PANTERA 900	
Rock drill:	HL 1000	HL 1000	HL 1000	
Drilling tools:	MF rods	MF rods	MF rods	
Drill rodltube size:	51	51	51	mm
Drill rodltube length:	3.66	3.66	3.66	m

WORKING ARRANGEMENTS

Work days per week:	6	6	6 days
Working days per month:	25	25	25 days
Working months per year:	12	12	12 months
Mechanical availability:	90	90	90 %
Work days per year:	290	290	290 days
Total working time per year:4640	4640	4640 h	
Effective time per year:	2784	2784	2784 h
Usage per year:	2506	2506	2506 h
Drilling time per year:	2004	2004	2004 h

DRILL RIG PERFORMANCE

Holes per set up:	1	1	1	
Rods per hole:	4 (3.73)	4 (3.77)	4 (3.81)	
Net penetration, lst rod:	1	0.8	0.7	m/min
Total time per hole:	20.6	24.1	26.8	min
Drilling capacity:	38	33	30	drm/h
	2.99	2.54	2.28	holes/h
	27	24	21	drm/engine-h
	164	142	130	drm/shift
	329	284	259	drm/day
	6579	5681	5181	drm/month
	76311	65904	60100	drm/year

Project:	EXAMPLE
Location:	QUARRY
Section:	BENCH

ESTIMATE OF EQUIPMENT AMOUNT

Rock production, total:	651854	699143	775494	m^3
				t
Excavation time:	12	12	12	months
Production/month:	64625	58262	64625	m^3/month
Need of drill holes/month:	6359	5492	5008	drm/month
No. of drilling units:	1	1	1	
One unit’s capacity is left free:0	0	0	%	
Tot drm in given prod:	76311	65904	60100	drm
Rock drill perc. h, total:	1523	1647	1724	hours
Rock drill perc. h, per unit:	1523	1647	1724	hours
Engine hours, total:	2796	2796	2796	hours
Engine hours, per unit:	2796	2796	2796	hours

SUMMARY OF BLAST DATA

Mean fragment size:	310	310	310	mm
Hole length:	12.8	12.9	13.0	m
Drill pattern:	9.1	11.4	14.0	m^2
Drill burden:	2.7	3.0	3.3	m
On surface:	2.7	3.0	3.3	m
Drill spacing:	3.4	3.8	4.2	m
Sub-drill:	0.8	0.9	1.0	m
Rock blastability index:	0.60	0.60	0.60	
Explosive:				
Bottom charge				
Type of explosive:	Dynamite	Dynamite	Dynamite	
Charge distribution:	10	10	10 %	
Column charge				
Type of explosive:	Anfo	Anfo	Anfo	
Charge distribution:	90	90	90	%
Bottom charge length:	1.20	1.19	1.18	m
Column charge length:	10.41	10.28	10.26	m
Uncharged height:	1.2	1.4	1.6	m
Bottom charge:	7	9	11	kg
Column charge:	62	80	101	kg
Total charge per hole:	68	89	112	kg
Specific charge:	0.62	0.65	0.67	kg/m^3
	0.23	0.24	0.25	kg/t
Specific drilling:	8.5	10.6	12.9	m^3/drm
	0.12	0.09	0.08	drm/m^3
	23.06	28.64	34.84	t/drm
	0.04	0.03	0.03	drm/t

Drilling time and tunnel advance estimate scenarios

Name of project:	EXAMPLE	
Location:	TUNNEL	
Excavated section:	FULL SECTION	

TUNNEL DIMENSIONS

Face area:	42.8	m^2
Height:	7.5	m
Width:	6.4	m
Profile length, excl. bottom:	18.7	m

DRILLING EQUIPMENT

Drilling system:	Power class	
Type of machine:	PARAMATIC 205 - 90	
Booms:	TB 90	
Rock drill:	HL 500 S	
Feed:	CF 145 x 14	
Feed length, total:	5995	mm
Drill steel length:	4305	mm
Max. hole length:	3.9	m
Number of booms:	2	
Coverage, height:	8.51	m
width:	12.82	m

DRILLING DATA

Hole diameter:	45	mm
Cut type:	Parallel	
Reamed hole diameter:	102	mm
Number of holes:	79 + 2	
Hole length:	3.9	m
Drilled meters:	324	m
Pull length:	93	%
Advance:	3.6	m/round
Specific drilling:	2.09	drm/m

DRILLING TIME

Type of rock:	GNEISS	
DRI:	45	
Density:	2.7	t/m^3
Rigs parane in tunnel:	1	
No of operators per rig:	1	
Operator experience:	Advanced	
Penetration:	2.1	m/min
Driling rate:	1.6	m/min
Reaming penetration:	0.9	m/min
Reaming time:	14	min
Drilling time, incl. reaming:	121	min
Auxiliary time:	15	min
Drilling cycle:	136	min

CHARGING TIME

Charge total:	243	kg
Specific charge:	1.57	kg/m^3
Charging crew:	3	
Charging cycle:	71	min/round

LOADING TIME

Eguipment:	Toro 1250 D	
Bucket size:	6	m^3
Carrying distance:	50	m
Loading + hauling capacity:	200	m^3 /h (loose)
Swell factor:	1.6	
Overbreak factor:	1.15	
Total volume:	286	m^3/round (loose)
Loading cycle:	101	min/round

BOLTING TIME

Rock bolts/row:	5	
Rock bolt spacing:	2	m
Rock bolts/round:	9	
Rock bolt length:	2	m
Type:	Cement	
Bolting cycle:	42	min

WORKING TIME

Hours per shift:	10	hours
Effective time:	8	hours/shift
Shifts per day:	2	
Work days per week:	6	days
Work days per month:	25	days
Working months per year:	12	months/year

SCHEDULE AND UNIT CYCLE

Rock bolting:	42	min
Steel ribs + wire mesh:		min
Sprayed concrete:	55	min
Drilling:	136	min
Charging:	71	min
Blast + ventilation:	30	min
Loading:	101	min
Scaling + clear:	20	min
Miscellaneous:	30	min
Total time/round:	485	min
	8.1	hours

PERFORMANCE

	Single	Double	
Transfer time:		30	min
Long term factor:	85	85	%
Rounds per day:	2	2.9	
Practical advance:	6.1	8.8	m/day
	37	53	m/week
	153	220	m/month
	1832	2637	m/year
Tunnel length:	1000	1000	m
lnterval niches:	300	300	m
Excavation time:	28	19	weeks
	6.7	4.6	months

10.3. SERVICE SUPPORT

Proper equipment selection on the construction site is very important, however, this alone can not ensure the success of a project. More important, if not critical, is equipment fleet management and service support. This chapter lists key elements related to equipment support and describes some of the service offering available from Tamrock.

In order for the selected to operate as efficiently as possible, certain elements must be ensured:

- Competent operators
- Knowledgeable servicemen and mechanics
- Planned service operation
- Timely spare parts and components
- Quality service work

These elements are achieved through proper service support.

FIGURE ***10.3.-1.*** *Competent operators, knowledgeable servicemen and mechanics are important for ensuring the high productivity of the selected equipment.*

TRAINING

The main objective of training is to enhance both operator and maintenance personnel skills, therefore, maximizing machine performance and availability. Both theoretical and hands-on training is needed to get the best result. Training is often customized according to local needs. The Tamrock classification system helps to identify present skills levels and targeted training needs.

Initial training is typically given on-site during start-up. More comprehensive training can be provided at Tamrock plants or at some of the regional training centers where cut-away components, videos and computer animation assist understanding and make the training more illustrative.

FIGURE ***10.3.-2.*** *Tamrock training enhances both operational and technical skills.*

PERIODIC MAINTENANCE

After commissioning, in which the equipment is checked, tested and adjusted to match local site conditions, it is important to follow the manufacturer´s periodic maintenance instructions to maximize equipment availability. The instructions consist of a per-shift inspection, and weekly and monthly maintenance. Most equipment operators perform the per-shift inspection, and maintenance personnel perform the weekly and monthly maintenance. Good information flow between operators and maintenance personnel is vital for optimum machine maintenance and performance.

Periodic maintenance is based on operating hours and typically a half shift per week is required to

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PERIODIC MAINTENANCE
Ranger 500, 600, 700

Rig Serial No.	Percussion hours	Signature:
Date	Diesel engine hours	

■ CHECK/TEST ▲ CHANGE/REPLACE ⌗ CLEAN AND FLUSH DH = DIESEL ENGINE HOURS
● CLEAN ○ LUBRICATE ◇ FLUID LEVEL E – EMPTY WATER TRAP
(1) REPLACE WHEN NEEDED S – SERVICE

PERIODIC MAINTENANCE								SERVICE POINT	Indicate when completed	
	EVERY							CRAWLER BASE		
Every day	50 DH	200 DH	800 DH	[illegible]	[illegible]	[illegible]	[illegible]	DIESEL ENGINE	Yes	No
◇		▲						Engine oil		
		▲						Engine oil filter		
■					⌗			Radiator hoses and connections		
	●							Radiator core		
■								Water pump for leaks		
■								Coolant temperature		
◇					▲			Engine coolant		
■								V belts		
■								Operation of air filter service indicator		
	■							Air intake lines and connections		
		●						Crankcase breather		
		●	▲					Air filter 1)		
				▲				Safety filter		
					■			Valve clearances		
					■			Fuel injector timing		
					■			Fuel ratio control		
					■			Engine vibration dampers		
					■		S	Turbo charger		
					▲			Thermostat of cooling system		
							S	Water pump		
							S	Starter motor		
							S	Alternator		
							■	Injector units		

NOTE: 1) Can be cleaned max. three times

Does this rig require cleaning ? Yes ☐ No ☐

FIGURE ***10.3.-3.*** *Properly performed periodic maintenance maximize equipment availability.*

complete the maintenance. During this time it is good to audit the machine to identify any needs for corrective maintenance. For the periodic maintenance, Tamrock has prepared spare parts kits including all parts needed to do service according to instructions. The kits make both order and shipment handling easier and faster.

It is important to keep track of what maintenance was performed and what spare parts were used on each machine. This provides the platform for machine life time control and operational costs follow-up.

SPARE PARTS LOGISTICS

The objective of spare parts logistics is to keep frequently used parts and some critical parts on hand at the site, and infrequently used parts supplied by the manufacturer with scheduled transportation.

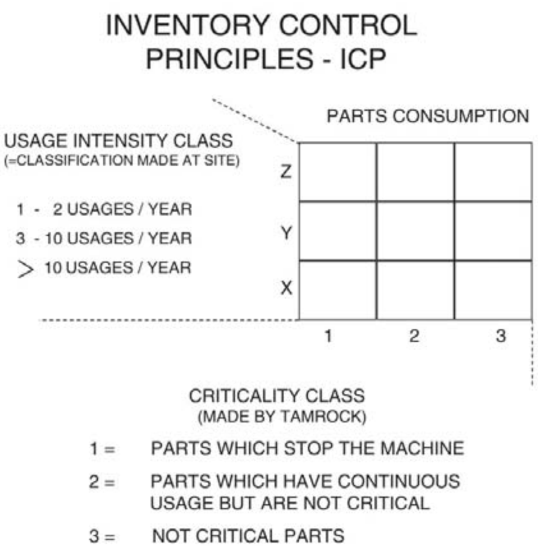

FIGURE 10.3.-4. *Spare part inventories at the site can be optimized through TAMROCK ICP.*

Tamrockís Inventory Control Principle (ICP) allows spare parts to be stocked and inventory controlled based on classification of what parts are most frequently used at the site. It is continuously followed up through the project or operation.

To start ICP, Tamrock makes on-site spare parts recommendations. The recommendations are based on estimated drilled meters or engine hours. Site conditions are taken into consideration as much as possible.

Tamrock's worldwide computer software system and Tamrock-developed electronic tools such as Rocknet, Paris, RockSite, RockDoc and RockTionary support all spare part inquiries, ordering, handling, delivering and consumption follow-up. Electronic spare part manuals and maintenance instructions make information handling easier and more illustrative.

SERVICE FACILITIES AND TOOLS

The workshop should be equipped with sufficient crane, welding machines, steam cleaning units, hand tools, hose crimping machines, grinders, presses, cutting torches, testing and measuring devices, working benches and tool panels. There should also be a dedicated place for all necessary fuel, grease, oils, lubricants and solvents. The storage facility should contain sufficient shelving with necessary bins, dividers and racks to accommodate the spare parts and components. This location shall be lockable and dry.

Where qualified and quick service set-up is needed, portable service containers provide the ideal solution for workshop, spare parts and tools storage. Tamrock service containers are equipped with a selection of tools and accessories to meet all Tamrock equipment service and maintenance requirements. The containers can be equipped with air conditioning or heating depending on the site environment.

FIGURE 10.3.-5. *Tamrock service containers include tools and accessories to meet equipment service and maintenance requirements.*

In the workshop or service container, instructions, spare part manuals and product wall charts are easily available to secure fast checking of required information.

TAMROCK SERVICE AND REBUILDING CONTRACTS

To optimize organizations on site, Tamrock offers different types of service contracts. Some machine applications are technically advanced and through these service contracts end users can avoid having trained and specialized skills required for maintaining newer equipment technology. Typically, contracts are tailor-made to match local conditions and needs.
Main contract alternatives are:

PARTS SUPPLY CONTRACT

This contract secures spare parts availability, delivery and costs required to keep the equipment running. There is usually an optimum parts inventory on site, with a present minimum reordering level and established price structure. Customs and import clearance is organized and the transportation arrangements are secured. For emergency deliveries, parts can be dropshipped direct from Tamrockís regional locations or plants. This contract can also include a software package for parts inquiries and ordering, and for inventory control and operational costs collection.

FIGURE *10.3.-6. Parts supply contracts ensure the availability of high-quality original and service parts.*

COMPONENT EXCHANGE CONTRACT

Decreasing downtime, optimizing unit operations and controlling costs are some of the benefits of the Tamrock exchange components contract. In a component exchange contract, critical spare components are stocked on-site and the periodic maintenance is done locally.
The components are exchanged as required and sent to the nearest Tamrock Center. The components are then rebuilt and tested to manufacturer standards by qualified technicians and genuine parts. The components are returned to the site complete with warranty coverage.
The contract may also include the labor for actual component changes.

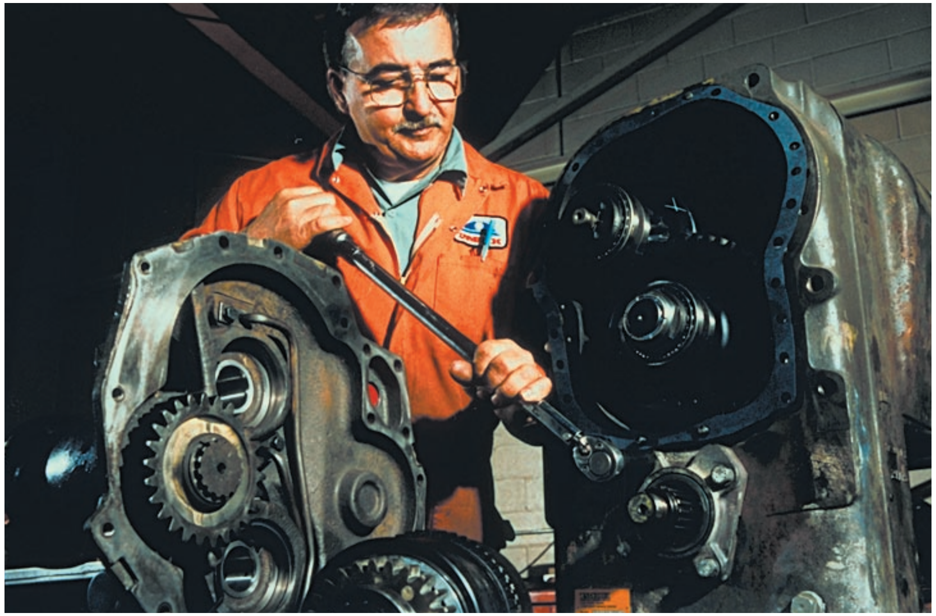

FIGURE *10.3.-7. Main drilling and loading equipment components are typically included in component exchange contracts.*

PERIODIC MAINTENANCE CONTRACT

Preventative periodic maintenance ensures that the equipment can produce according to the production schedule and that any downtime is minimized.
In this contract, Tamrock personnel make scheduled site visits based on equipment usage. They perform the periodic maintenance according to the recommendations and instructions given in the equipment service manuals. The equipment systems are adjusted to match local needs. Tamrock supplies the service parts required for each scheduled visit. The charges are fixed regardless of actual labor hours. A detailed report and recommendations for any corrective maintenance are presented to the customer. The contract can also cover lubricant supply and emergency call-out service.

FIGURE *10.3.-8. Periodic maintanance.*

SUPERVISION CONTRACT

A supervision contract ensures that the customer´s hands-on personnel are properly focused and perform exactly what is needed on the equipment. This generally is for a project-type construction where Tamrock has experienced factory trained personnel working daily with the customer organization

The supervisors ensure that maintenance programs are properly set and followed. They often work hands-on with site personnel for equipment servicing and repairs, thus provide continuos training. This contract could consist of parts inventory control and monitoring equipment operational costs with specialized software.

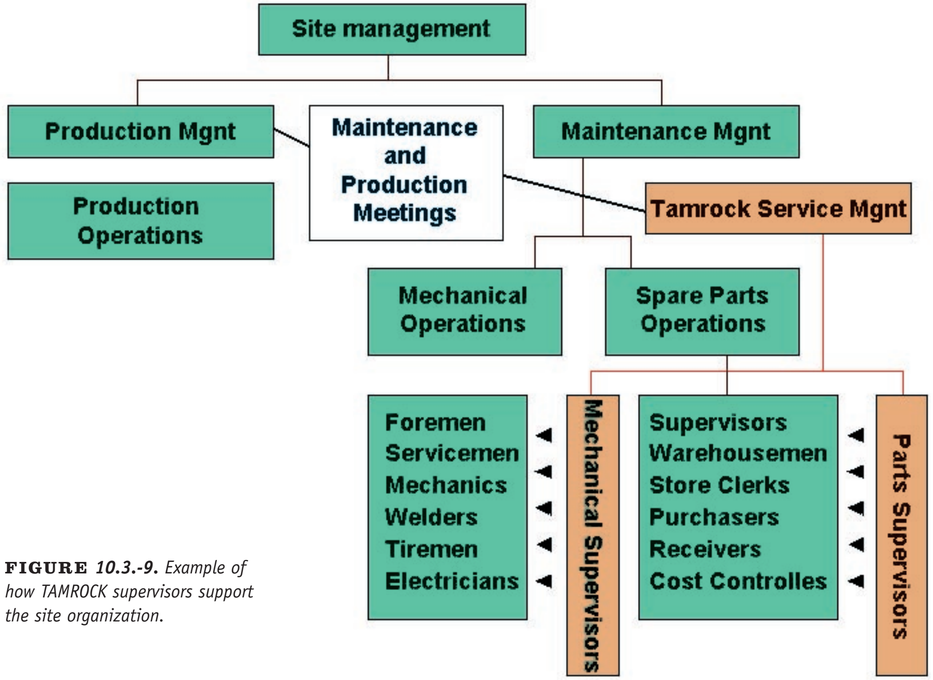

FIGURE *10.3.-9. Example of how TAMROCK supervisors support the site organization.*

Full-service contract

A full-service contract offers complete equipment support from Tamrock throughout the process. This type of contract secures "peace of mind".
It includes total equipment maintenance and repair by a full-time experienced Tamrock service organization including:

- Mechanics, servicemen, warehousemen, clerks and service managers
- Workshop containers, parts containers, tools, service and maintenance vehicles
- Computer with specialized software: monitoring costs, inventory and maintenance
- Preventative maintenance and component exchange programs are set and followed
- Complete control over site inventory of parts and components: levels, reordering and issuing.
- Drill steel and bit servicing can be included as well

FIGURE 10.3.-10. *In full-service contracts, TAMROCK joins resources with the site organization for total performance.*

Rebuilding

To extend economical machine life and secure good availability, it is worth while after certain specified operating hours to do a major machine overhaul or even rebuild the machine. Overhauling or rebuilding provides a good opportunity to upgrade the machine using Tamrock retro-fitting and upgrading packages, which include the latest components and subassemblies with all required accessories for smooth installation. Tamrock can offer on-site overhauling or rebuilding or at certified rebuilding centers, where specialized and dedicated technicians guarantee high-quality work.

There are many elements that should be considered and planned for construction site management. The high utilization and availability of the selected equipment is crucial for the success of a project. And it can be achieved through proper service support.

FIGURE 10.3.-11. *TAMROCK CHA 660 before and after rebuilding.*

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